S-K 1300 Technical Report Summary and Feasibility Study

Era Dorada Gold Project

Jutiapa, Guatemala

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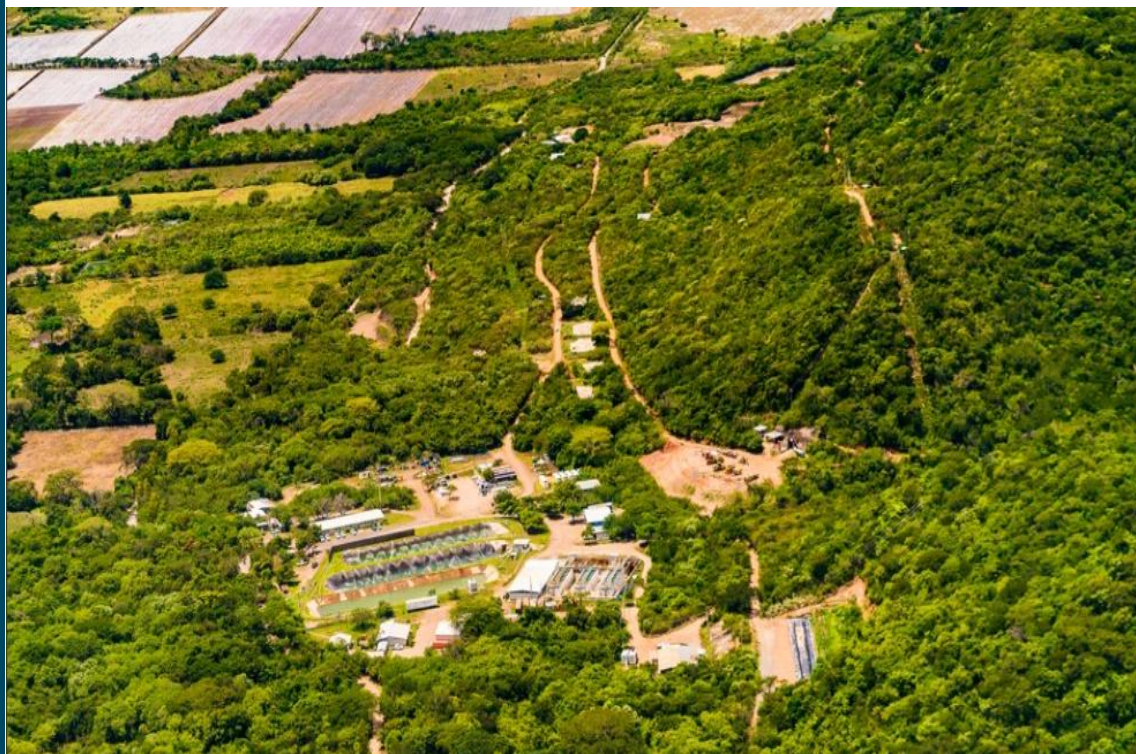
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1 EXECUTIVE SUMMARY

1.1 Introduction

In January 2025, Aura Minerals Inc ("Aura" or "the Company") completed the acquisition of the of the Era Dorada Gold Project - formerly "Cerro Blanco Gold Project" - and Mita Geothermal Project, located in Jutiapa, Guatemala, near the town of Asunción Mita and the border with El Salvador. The Era Dorada Project ("Era Dorada" or "the Project") is 100% beneficially owned by Aura. Aura is a public, NASDAQ-listed company trading under the symbol "AUGO", with its head office located at 78 SW 7th St., Miami, FL 33130, USA.

Aura commissioned Ausenco do Brasil Engenharia Ltda. (Ausenco) to prepare a Feasibility Study (FS) and associated Technical Report Summary (TRS) on the Project.

This TRS, titled "Era Dorada Project, S-K 1300 Technical Report Summary and Definitive Feasibility Study, Jutiapa, Guatemala," has been prepared in compliance with United States Securities and Exchange Commission's (SEC) Modernized Property Disclosure Requirements for Mining Registrants as described in Subpart 229.1300 of Regulation S-K, Disclosure by Registrants Engaged in Mining Operations (S-K 1300) and Item 601 (b)(96) Technical Report Summary.

The responsibilities of the engineering consultants are as follows:

- Ausenco was responsible for managing an coordinating the work related to the FS and the TRS and established an economic framework for the FS. The scope of work included the development of a conceptual flowsheet as well as detailed flowsheets, specifications, and the selection of process equipment. Ausenco also provided design oversight related to site infrastructure including the access road, power line, plant facilities, and other ancillary facilities. In addition, Ausenco designed the drystack tailings facility (DSTF) and waste rock facility (WRF), as well as the surface water management, including design of ditches, channels and ponds for stormwater control. The team's responsibilities further included estimating the process plant, general and administrative (G&A), and site services capital (CAPEX) and operating (OPEX) costs; preparing a financial model and conducting an economic evaluation including sensitivity and Project risk analyses; and developing a Project Execution Plan.
- Snowden Optiro was responsible for the mine engineering, design and scheduling, underground geotechnical design, and estimating the mining CAPEX and OPEX.
- Kirkham Geosystems Ltd. (Kirkham) was responsible for the Mineral Resource Estimate and the related geology, exploration and deposit sections of the TRS.
- Stantec Consulting Inc. (Stantec) was responsible for dewatering and injection requirements.
- BBE Company Inc. was responsible for the Cooling Plant associated with the underground mine.

1.2 Terms of Reference

The report supports disclosures by Aura in the press release named “Aura Minerals Completes Feasibility Study for the Era Dorada Project” published in December 8, 2025.

The TRS does not consider the Mita Geothermal Project, which is a separate project and subject to a separate technical report summary.

All units of measurement in this report are metric and currencies are expressed in United States dollars (symbol: US$ or currency: USD) unless otherwise stated.

Mineral Resources and Mineral Reserves are prepared in accordance with the standards and definitions of the S-K 1300.

1.3 Property Description, Location and Accessibility

The Project is located in southeast Guatemala, in the Department of Jutiapa, approximately 160 km by road from the capital, Guatemala City, and approximately 9 km west of the border with El Salvador. The nearest town to the Project is Asunción Mita, a community of about 18,500 people situated approximately 7 km west of the Project. The exploitation license covers 15.25 km^2 and lies entirely in the municipality of Asunción Mita.

The approximate center of the Project area is located at UTM coordinates X: 212,250 m E, Y: 1,587,250 m N, referenced to NAD27, Zone 16N. These coordinates correspond to the central portion of the mineral concession and are used for spatial reference in this report.

Current road access to the site is via the Pan-American Highway (Highway CA1) through the town of Asunción Mita. Existing infrastructure is in place to provide year-round access to the site. The topography is relatively flat, with rolling hills.

The climate and vegetation at the Project site are typical of a tropical dry forest environment. The elevation is between 450 and 560 masl. The wet season is typically from May to October. The average annual rainfall is 1,350 mm. Daily temperature highs reach 41°C, and lows reach 10°C. The average annual pan evaporation rate is 2,530 mm, with an annual average humidity of 62%.

The recently constructed La Barranca power substation is located a few kilometers south of Mita. The substation can supply up to 20 MW of power.

There is no record of any previous mining activity in the area; however, with the closure of Goldcorp’s Marlin Mine in late 2017, it is anticipated that a significant contingent of Guatemalan-trained labor will be available for employment at Era Dorada. As such, the Project intends to hire the majority of operations staff locally.

There is one Mineral Claim for all the project recorded in Decree DIC-CM-158-05.

A portion of the mine workforce is expected to live at the mine site in a purpose-built permanent camp, while employees living in the surrounding communities Aura will provide the transportation to and from the mine site. For employees residing in the wider Jutiapa region and areas beyond Asuncion Mita, the Company will provide transportation to and from the mine site from designated locations. There are several population centers near the Project site.

1.4 History

The Era Dorada property (formerly "Cerro Blanco") was identified by Mar-West by a sampling of densely silicified boulders. In October 1998, Mar-West's holdings in Honduras and Guatemala were purchased by Glamis Gold Ltd. In November 2006, Goldcorp Inc. became the sole proprietor of the Project through the purchase of Glamis Gold. Goldcorp undertook a comprehensive exploration program from 2006 to 2012, which included additional surface exploration, over 3.4 km of underground development, and 43,016 m of surface and underground drilling. On January 4, 2017, Bluestone agreed with Goldcorp to acquire 100% of the Project. On October 29, 2024, Aura purchased Bluestone Resources thereby acquiring 100% of the property. As of the end of 2021, Bluestone had drilled approximately 267 holes for a total of 45,725 m on the Cerro Blanco property since the acquisition from Goldcorp.

Table 1-1 summarizes the historical drilling on the property.

Table 1-1: Drilling Summary

Year	Company	Holes Drilled	Meters
1998	Mar-West	9	1,340
1999	Glamis	48	7,074
2000	Glamis	18	3,525
2002	Glamis	23	6,525
2004	Glamis	42	9,370
2005	Glamis	120	29,065
2006	Glamis	67	15,129
2007	Goldcorp	47	12,373
2008	Goldcorp	2	586
2009	Goldcorp	1	140
2010	Goldcorp	10	2,277
2011	Goldcorp	28	5,898
2012	Goldcorp	96	21,370
2017	Bluestone	8	2,324
2018	Bluestone	74	13,993
2019	Bluestone	61	8,403
2020	Bluestone	74	15,172
2021	Bluestone	50	5,833
Total		**778**	**160,397**

Source: Bluestone, 2021.

1.5 Geology and Mineralization

The Project is a classic hot springs-related, low-sulfidation epithermal gold-silver deposit comprising both high-grade vein and low-grade disseminated mineralization. The Cerro Blanco district is part of an active volcanic arc characterized by Miocene-Pliocene-aged bimodal volcanism that extends through El Salvador, Honduras, and Nicaragua.

High-grade mineralization is hosted in the Mita unit as two upward-flaring vein swarms comprising over 60 veins (North and South Zones) that converge downwards and merge into basal feeder veins. Low-grade disseminated and veinlet mineralization within and as halos around the high-grade veins is well documented in drilling since the discovery of the deposit. Most of the veins are blind to the surface and concealed by the syn-mineral Salinas Unit, a sub-horizontal sequence of volcanogenic sediments and sinter horizons approximately 100 m thick that form the low-lying hill at the Project. The Salinas cap rocks are host to low-grade mineralization associated with silicified conglomerates and contemporaneous dacite/rhyolite flow domes or cryptodomes.

Both high and low-angle banded crustiform/colloform chalcedony veins, locally with calcite replacement textures, make up the deposit, with bonanza-grade gold grades largely confined to the chalcedony-quartz veins, especially where adularia bands are prominent. High-grade mineralization occurs over a vertical profile of 400 m (150 to 450 masl). At depth, calcite-dominated veins form the limit to mineralization; nonetheless, very locally, high gold values are present in calcite-dominated veins and silicified structures containing only minor quartz veinlets.

The Salinas Group includes thin hot spring deposits, including sinters, which are genetically linked to underlying swarms of epithermal, gold-silver bearing quartz veins. The west and east sides of the Era Dorada ridge consist of flat agricultural plains characterized by Quaternary basalts, interbedded with boulder beds and sands. These rocks also appear down-faulted to lower elevations, implying major post-mineral extensional movements on such faults.

The current gold resource occurs under a small hill within an area of 400 m by 920 m. Gold-bearing structures in the Era Dorada area extend 2 km to the northwest of the gold deposit and occur largely confined within the hydrothermal alteration zone. The extensive drilling undertaken to date of the high-grade vein swarms and their surrounding low-grade mineralized envelopes and overlying mineralized cap rocks show impressive intercepts, including 203.8 m grading 2.3 g/t Au and 8.1 g/t Ag (CB20-420) and 87.2 m grading 5.9 g/t Au and 32.5 g/t Ag (UGCB18-89).

Vein textures suggest that gold and silver were introduced as one major event of multi-stage finely banded veining (originally amorphous silica) with subordinate bands of platy calcite, which is mostly pseudomorphed to cryptocrystalline silica phases. Repetitive “crack and seal” pulses and associated boiling/flashing events very close to the paleosurface are proposed as the main mechanisms for precious metal deposition. Very high-grade core intersections with coarser and more abundant sulfides, electrum, and free gold appear to represent an earlier series of events. Deportment studies indicate that approximately 99% of the gold occurs in electrum as free or exposed grains, with lesser amounts as native gold and kustelite. The lack of post-mineral structural displacement of veins and distribution of high grades over a +400 m vertical profile attest to the pristine nature of the veins at Era Dorada. The lack of inter-stage hydrothermal brecciation and coarse-grained primary quartz textures suggest that the mineralizing event was fairly short-lived and occurred very close to the paleosurface.

1.6 Exploration

To date, Aura has not completed any exploration activities on the Era Dorada property.

1.7 Sampling

The drill core from the surface and underground was stored in labeled wooden boxes at the drill site and transported to the surface core logging facility. Before core splitting and logging commence, the drill core was systematically photographed in high resolution using a tripod-mounted camera and digitally archived for reference as part of the drill database.

Logging and sampling were undertaken on-site at Era Dorada by company personnel under a QA/QC protocol developed by Bluestone. Technicians first prepared the core boxes by reviewing drill hole depth tags, reassembling broken sections, and photographing the core. Core logging to identify lithology, alteration, RQD, and sampling selection for core sawing was completed by technicians under the direction of the geologist. The typical sample lengths are 1.0 to 1.5 m with a minimum sample width of 1 m and maximum lengths of 2.0 m; sample lengths were based on the lithology and alteration. Samples are collected along the footwall, mineralized zones, and hanging walls without breaks in sampling. All data was initially captured on paper logs and later transferred to Microsoft Excel and susequently to an AcQuire/GMSuite platform. The data was then entered into MapInfo™ and MineSight™ software for geological modeling.

Specific gravity readings of all representative lithologies and vein material were taken during the various drill campaigns using the displaced water method. Samples were sealed with paraffin wax to account for natural voids/vugs.

A total of 591 channel samples were taken along representative veins exposed in the side walls of the Era Dorada underground tunnels using a portable rock saw. The sampling was undertaken across and perpendicular to the mineralized structures wherever possible and carefully surveyed with XYZ coordinates for use in 3D modeling. The samples were subject to the same QA/QC protocols as the drill core and were deemed suitable for use in calculating resources.

Samples were transported in security-sealed bags to Inspectorate Laboratories in Guatemala City for sample preparation until March 2020 and thereafter to Inspectorate Laboratories in Managua due to the closure of the Guatemalan facility. All half-core and coarse rejects are stored adjacent to the core logging facility on the Project site. The Era Dorada site is fully controlled by perimeter fencing and security.

Pulps are shipped for regular and QA/QC analysis to Inspectorate Laboratories (a division of Bureau Veritas) in Reno, Nevada, USA, and ALS Chemex in Vancouver, BC, Canada, respectively. Both are ISO 17025-accredited laboratories. th atomic absorption with gravimetric finish for values exceeding 5 g/t Au and 100 g/t Ag.

Garth Kirkham, P. Geo., has been involved with the property since in early 2017, when he performed the initial due diligence and authored the updated resource estimate for Bluestone. Mr. Kirkham first visited the property on May 8, 2017, to validate all aspects of the Project. The site visit included an inspection of the property, offices, underground vein exposures, core storage facilities, water treatment plant, and stockpiles, and a tour of major centers and the surrounding villages most likely to be affected by any potential mining operation.

Since 2017, Mr. Kirkham has visited the property numerous times for extended periods to develop and implement data-gathering and sampling methods and procedures. He also worked with Bluestone geologists to develop drill programs and supervise interpretation and modeling efforts, in addition to creating and implementing QA/QC procedures. Continued data validation and verification processes have not identified any material issues with the Era Dorada sample and assay data.

It is the opinion of the QP, Garth Kirkham, P. Geo., that the sampling preparation, security, analytical procedures, and quality control protocols used at Era Dorada are consistent with generally accepted industry best practices and are, therefore, reliable for resource estimation.

1.8 Mineral Processing and Metallurgical Testwork

Metallurgical testwork was conducted on samples from the Era Dorada deposit between April 1999 and January 2012 by Kappes, Cassiday & Associates (KCA) in Reno, NV. The most recent test program, completed in 2018 in support of this FS, was carried out at Base Metallurgical Laboratories Ltd. (BaseMet) in Kamloops, B.C.

The focus of the recent test program was to optimize the flowsheet and generate tailings for geochemistry, geotechnical and paste backfill testing. A global composite from drill core was created to run the optimization test program. The testwork included grind extraction optimization, gravity, leach optimization, tailings generation and cyanide destruction. Bulk samples from the underground workings were collected and two composites were created to represent the North and South areas of the deposit. The final flowsheet and test parameters determined in the optimization phase were used to generate tailings samples from the North and South zones for physical and chemical characterization to be used in defining DSTF and backfill applications.

Based on the results from BaseMet (2018), gold and silver doré can be produced at a primary grind size of 80% passing (P_{80}) 53 µm followed by gravity concentration, 2-hour pre-oxidation, a 36-hour cyanide leach at a sodium cyanide concentration of 500 mg/L, 6-hour carbon-in-pulp (CIP) adsorption, carbon desorption, electrowinning and refining. For the global composite, this recovery method achieved average precious metal recoveries of 96% Au and 85% Ag.

1.9 Mineral Resource Estimate

Era Dorada is a classic hot springs-related, low-sulfidation epithermal gold-silver deposit comprising both high-grade vein and low-grade disseminated mineralization. Most of the high-grade mineralization is hosted in the Mita unit as two upward-flaring vein swarms (north and south zones) that converge downwards and merge into basal feeder veins where drilling has demonstrated widths of high-grade mineralization (e.g., 15.5 m @ 21.4 g/t Au and 52 g/t Ag). Gold grades are associated with ginguru banding and carbonate replacement textures. Sulfide contents are low, typically < 3% by volume. Low-grade disseminated and veinlet mineralization in wall rocks around the high-grade veins is well documented in drilling since the discovery of the deposit, with grades typically ranging from 0.3 to 3.0 g/t Au.

The Salinas unit overlies the Mita rocks, a sub-horizontal sequence of volcanogenic sediments and sinter horizons approximately 100 meters thick, which form the low-lying hill at the Project. Low-grade disseminated, and veinlet mineralization within and as halos around the high-grade vein swarms is well documented in drilling since the discovery

of the deposit, with grades typically ranging from 0.3 to 1.5 g/t Au. The overlying Salinas cap rocks are also host to low-grade mineralization associated with silicified conglomerates and rhyolite intrusion breccias.

Mineral exploration activities conducted at Era Dorada have been performed in accordance with S-K 1300.

The mineral resource estimate reported herein was prepared by Mr. Garth Kirkham, P. Geo. The mineral resources have been estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices." There are 130,307 gold assays, totaling 153,078 m, which average 0.68 g/t, and 130,238 silver assays, totaling 153,003 m, which average 3.75 g/t. Bulk densities were assigned to individual rock types and assigned on a block-by-block basis using measurement data by lithology and mineralized veins.

The estimate was completed using MineSight™ software with a 3D block model (5 m x 5 m x 5 m). Interpolation parameters have been derived based on geostatistical analyses conducted on 1.5-meter composited Drill holes. Block grades have been estimated using ordinary kriging (OK) methodology, and the mineral resources have been classified based on proximity to sample data and the continuity of mineralization in accordance with S-K 1300 requirements.

Table 1-2: Resource Estimate using 2.25 g/t Au Cut-off Inclusive of Reserves

Resource Category	Tonnes (kt)	Au Grade (g/t)	Ag Grade (g/t)	AuEq Grade (g/t)	Contained Gold (koz)	Contained Silver (koz)	Contained AuEq (koz)
Measured							
Indicated	7,059	9.03	30.66	9.36	2,049	6,958	2,125
Measured & Indicated	7,059	9.03	30.66	9.36	2,049	6,958	2,125
Inferred	736	5.94	19.22	6.16	141	455	146

Table 1-3: Resource Estimate using 2.25 g/t Au Cut-off Exclusive of Reserves

Resource Category	Tonnes (kt)	Au Grade (g/t)	Ag Grade (g/t)	AuEq Grade (g/t)	Contained Gold (koz)	Contained Silver (koz)	Contained AuEq (koz)
Measured							
Indicated	2,460	6.36	22.76	6.61	503	1,801	523
Measured & Indicated	2,460	6.36	22.76	6.61	503	1,801	523
Inferred	736	5.94	19.22	6.16	141	455	146

Notes: The mineral resource statement is subject to the following:

1. Mineral Resources are reported in in accordance with S-K 1300.
2. Mineral resource estimates have been prepared by Garth Kirkham, P.Geo., a Qualified Person as defined by SK-1300.
3. The Mineral Resource estimate is reported on a 100% ownership basis.
4. Underground mineral resources are reported at a cut-off grade of 2.25 g/t Au. Cut-off grades are based on a assumed metal prices of US$2,500/oz gold and US$28/oz silver, and assumed metallurgical recovery, mining, processing, and G&A costs.
5. Mineral Resources are reported without applying mining dilution, mining losses, or process losses.

6. Resources are constrained within underground shapes based on reasonable prospects of economic extraction, in accordance with SK-1300. Reasonable prospects for economic extraction were met by applying mining shapes with a minimum mining width of 2.0 m, ensuring grade continuity above the cut-off value, and by excluding non-mineable material prior to reporting.
7. Metallurgical recoveries reported as the average over the life of mine and are assumed to be 96% Au and 85% Ag, respectively.
8. Bulk density is estimated by lithology and averages 2.47, 2.57 and 2.54 g/cm^3 for the Salinas, Mita and mineralized vein domains, respectively.
9. Mineral resources are classified as Indicated, and Inferred based on geological confidence and continuity, spacing of drill holes, and data quality.
10. Effective date of the Mineral Resource Estimate is November 30, 2025.
11. Tonnage, grade, and contained metal values have been rounded. Totals may not sum due to rounding.
12. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

Source: Kirkham, 2025.

In addition, there has been mixed grade material mined during the creation of the extensive, existing ramp network which has been stockpiled adjacent to North Ramp entrance. Table 1-4 shows the volume and tonnage based on an unconsolidated specific gravity of 2.0 gm/cm^3 along with gold and silver grades and metal content. These resources are classified as measured.

Table 1-4: Stockpile Resource Estimate (Measured Resource)

Volume (BCM)	Mine (t)	Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)
14,863	29,726	5.35	22.59	5,108	21,590

Source: Kirkham, 2019.

1.9.1 Key Risks and Factors That May Affect Resources

The primary factors that could materially affect the Mineral Resource estimate include:

- Commodity Prices (Gold, Silver) – Lower commodity prices will change the size and grade of the potential targets. Conversely, increased commodity prices will improve economics and resources.
- Although there is a relatively high degree of confidence related to geological continuity and grade variability, vein models and grade distributions may adjust with further data and structural interpretations.

1.10 Mineral Reserve Estimate

The Mineral Reserve estimate was completed using industry-standard methodologies and software, and the resulting Mineral Reserves are reported in accordance with S-K 1300 requirements.

The Mineral Reserve estimate was subject to Legal and Permitting constraints and other modifying factors such as the plant and infrastructure timing and capacities, the metallurgical recoveries for gold and silver, economic factors as gold and silver prices, costs and exchange rates as well as technical modifying factors derived from geotechnical constraints, mining methods and productivity.

The mine plan derived from the Mineral Reserve supports an economically viable underground operation, subject to the stated assumptions, modifying factors, and risk controls in the opinion of the QP.

The mining engineering studies for the Feasibility Sudy and Mineral Reserve definition were completed using industry-standard methodologies and software, and the resulting Mineral Reserve is reported in accordance with S-K 1300 requirements. The mine plan supports an economically viable underground operation, subject to the stated assumptions, modifying factors, and risk controls.

A method-specific gold-equivalent cut-off grade was applied in stope optimization and Reserve conversion. Key assumptions are shown in Table 1-5.

Table 1-5: Mineral Reserve Cut-off Grade

Parameter		Unit	Value	
			LH	MCF
Au price		US$/oz	2,000	2,000
Ag price		US$/oz	25	25
Project Parameters				
Au Process Recovery		%	96.00	96.00
Ag Process Recovery		%	85.00	85.00
Au Payable metal		%	99.92	99.92
Ag Process Recovery		%	99.50	99.50
TC/RC		US$/oz Au	2.21	2.21
Royalty				
Royalty NSR		% of NSR	1.05%	1.05%
Guatemalan Gov't Royalty (Gross)		% total payable metals revenue	1.00%	1.00%
OPEX Estimates	Mining (Underground)	US$/tonnes milled	100	115
	Processing	US$/tonnes milled	32	32
	Site Services	US$/tonnes milled	18	18
	G&A	US$/tonnes milled	20	20
	Total OPEX estimate	US$/tonnes milled	170	185
In-situ cutoff Au grade		g/t Au eq	2.82	3.07

Gold and silver prices assumptions were defined in the beginning of the Feasibilty Study in line with Aura guidance and were deemed adequate at the time they were established by the QP.

Assumptions for gold and silver recoveries and mining costs are based on previous studies for the deposit and are assessed as adequate, within industry standards, by the QP.

The Mineral Reserve was estimated from the final stopes and sublevel designs produced from Measured and Indicated Mineral Resources which demonstrate economic viability, incorporating relevant dilution allowances and mining recovery factors.

Any Inferred Mineral Resources enclosed in feasible Indicated Mineral Resources envelopes either inside the stope shapes, dilution material or forced mine development was treated as waste, i.e., such material carried its excavation costs and no revenue was accounted for.

The Mineral Reserve estimate is summarized in Table 1-6.

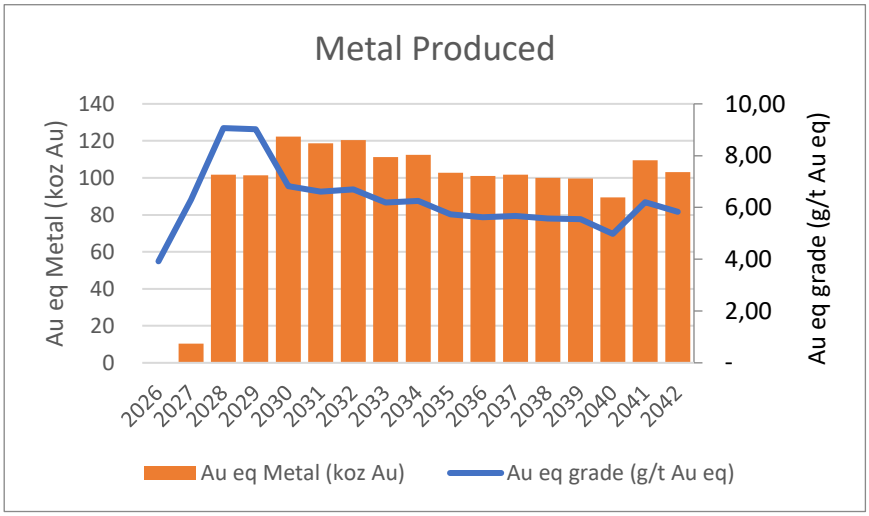
Table 1-6: Mineral Reserves

	Tonnage (kt)	Au grade (g/t)	Au metal (koz)	Ag grade (g/t)	Ag metal (koz)	Au Equiv grade (g/t)	Au Equiv metal (koz)
Proven	30	5.35	5	22.59	22	5.60	5
Probable	8,717	6.01	1,684	20.39	5,715	6.23	1,746
Proven + Probable	8,747	6.01	1,689	20.40	5,736	6.23	1,751

Mineral Reserve Notes:

1. The Mineral Reserve was estimated and classified in accordance with the USA S-K 1300 standards.
2. Mineral Reserve has an effective date of December 5, 2025. The Qualified Person for the estimate is Ruy Lacourt, BSc. Mining Engineering, MSc., Registered Member of the SME, an Associate of Snowden Optiro.
3. The Mineral Reserve was estimated using metal prices of US$2,000/oz Au and US$25/oz Ag, and metallurgical recoveries of 96% Au and 85% Ag. Underground mining costs were assumed as US$100/t (Long Hole mining) and US$115/t (Cut-and-Fill mining), with processing, site services and G&A costs as of US$32/t, US$18/t and US$20/t, respectively. Royalties comprise 1.05% NSR to the previous owners plus a 1.0% gross government royalty. Cutoff grades in gold equivalent are 2.82g/t Au eq for underground Long Hole mining and 3.07 g/t Au eq for Cut-and-fill.
4. The formula for gold equivalent is: Au eq = Au grade + 0.011 * Ag grade.
5. The Mineral Reserve is presented on a 100% ownership basis fully attributable to Aura Minerals.
6. Tonnages and grades have been rounded in accordance with reporting guidelines. Tonnages are rounded to the nearest 1,000 t, metal grades are rounded to two decimal places. Tonnage and grade are in metric units, containing gold and silver are reported as thousands of troy ounces. Totals may not sum due to rounding.
7. The existing surface stockpile (29,726 t, dry basis, at 5.35 g/t Au and 22.59 g/t Ag) were evaluated using the same economic parameters as the underground Mineral Reserve and is classified as Proven Mineral Reserve.

Figure 1-1: Gold Production and Grades



Source: Snowden, 2025.

1.1.1 Key Risks and Factors That May Affect the Mineral Reserve Estimate

The primary factors that could materially affect the Mineral Reserve estimate include:

- Vein geometry and continuity risk: Uncertainty in vein positioning may lead to increased dilution, local stope loss, or constraints on mining adjacent stopes where sub-parallel structures occur.
- Geotechnical and domain constraints: Incorrect assumptions regarding rock mass quality, stope dip limitations (particularly in Domain 1), or required pillar thickness may reduce recovery or force conversion from LH to the less productive MCF method.
- Sequencing dependency: The schedule assumes independent LH extraction within sublevels. If future geotechnical performance requires a mandatory extraction order or additional sequencing constraints, LoM production profiles could be disrupted.
- Hydrogeological constraints: Mining below the 420 level is contingent on effective dewatering, with implications for safety, development rates, ventilation/cooling requirements, and equipment availability.
- Backfill performance: Paste fill and CRF placement rates and curing times are critical to adjacent stope availability. Underperformance may delay stoping and reduce recovery.
- Dilution and recovery variability: Higher-than-planned dilution or lower recovery would negatively impact grade, metal content, and ability to meet early production targets.

1.1.2 Opportunities and Upside Potential of the Mineral Reserve

Several factors may provide upside to the Mineral Reserve inventory and LoM performance:

Resource confidence upgrade: Some dilution within stopes is currently classified as Inferred Resources and treated as zero grade. Conversion to Indicated through infill drilling is expected to increase average stope grades and contained metal, potentially supporting future Proven Reserve conversion.

Stockpile strategy flexibility: Expanding early ore stockpiling (including selective higher grades) could improve start-up flexibility and enable more productive early development. A temporary stockpile combined with the existing stockpile (29,726 t at 5.35 g/t Au and 22.59 g/t Ag) over the first three years could deliver this benefit, provided grade segregation is maintained.

Open pit potential: Near-surface Resources not included in the underground plan could potentially be converted to Mineral Reserves if a future open pit evaluation demonstrates technical and economic viability under S-K 1300 criteria. This could expand the Reserve base and be integrated into the LoM plan.

1.11 Mining Methods

The Era Dorada deposit is planned to be mined using underground methods, with production derived from a combination of sublevel Long hole stoping (LH) as the dominant method, mechanized cut-and-fill (MCF) in geotechnically or geometrically constrained areas, and minor room-and-pillar. Long hole stoping is expected to contribute approximately 98.5% of total metal production, with MCF contributing approximately 1.2% and room-and-pillar approximately 0.1%. The selection of mining methods reflects orebody geometry, vein continuity and dip, and geotechnical domain constraints, with LH preferred for productivity, safety and cost effectiveness where conditions allow.

The deposit will be accessed via two existing main declines servicing the South and North zones, supplemented by additional ramps developed at up to 15% gradient. Sublevels are spaced 20 m vertically. A panel geometry is adopted, consisting of four sublevels with 20 m plus a sill pillar with 20 m for total panel height as of 100 m, with the sill pillars recovered late in the LoM. The schedule respects operational constraints including maximum annual development advance of approximately 8,500 m, plant throughput limits, paste/CRF placement and curing cycles, and dewatering requirements below the 420 level water table.

The life-of-mine plan targets a production rate of 1,600 tonnes per day (t/d) with accelerated mine development in the first two years meant to expose high grade Mineral Resources. The planned Life of Mine (LoM) extends for 18 years, with higher metal output scheduled in the early years through prioritized access to higher-grade areas.

1.11.1 Mine Geomechanical Studies

Empirical stability assessments indicate that long hole stoping is feasible for the majority of the mine under the geometries evaluated. As for the upper, altered and more fractured rock masses closer to surface (Geotechnical Domain 1) Long hole mining remains feasible provided systematic support is installed where the orebody dips are less than 60° and, when the ore lenses dip less than 60 degrees, cut-and-fill mining is recommended.

Two-dimensional finite-element stress–strain analyses were completed for representative mine sections to guide mine design and detailed mining sequence for excavation and backfilling either with pastefill or CRF.

Crown pillar stability was assessed and a minimum thickness of 10 m is recommended.

Stability assessments of inter-stope pillars indicate acceptable factors of safety when timely backfilling is applied.

Development support recommendations were derived from the Q-system. All development headings should employ systematic galvanized Swellex type bolts, supplemented by 5-cm shotcrete as the baseline surface support. Welded mesh may be used in selected locations, and intersections may require cable bolts for larger spans.

1.11.2 Mine Hydrogeology

The groundwater system comprises five main hydrostratigraphic units, with flow strongly controlled by geological structures. Shallow alluvium is highly permeable and largely discharges to the Rio Ostua, while deeper fault zones act as preferential conduits for geothermal upflow. Groundwater temperatures can locally approach 190°C, which becomes a primary constraint for pump selection, materials, and overall water-handling design.

The groundwater model was updated in FEFLOW over an approximate 174 km² domain with 25 layers, building on prior work. A steady-state calibration to 45 observation points achieved an NRMSE around 6.1%. Transient simulations for 2008–2024 reproduce observed hydrographs and confirm the Rio Ostua as a gaining stream.

Predicted dewatering rates ramp from about 1,220 gpm in 2026 to about 6,080 gpm by 2029. Full build-out reaches roughly 10 wells by January 2031, sustaining approximately 6,080 gpm, with contingency capacity up to about 7,600 gpm to manage operational variability.

The currently permitted discharge capacity of about 5,250 gpm is below projected dewatering needs. The strategy includes two reinjection wells, approximately 1,000 gpm each, to manage peak flows, reduce reliance on surface discharge limits, and preserve operational flexibility beyond 2031.

By 2043, model results indicate annual flow reductions of approximately 12% in the Rio Ostua (about 1,564 to 1,376 L/s) and approximately 46% in the Río Tacunshapa (about 6.5 to 3.5 L/s). Treated mine effluent is expected to exceed baseflow depletion under the evaluated conditions, supporting downstream compliance.

The high-temperature regime introduces a steam-flashing risk and increases mechanical and materials demands on dewatering infrastructure. The design basis assumes ESPs with appropriate pressure and backpressure control, plus materials and components suitable for geothermal conditions.

To reduce uncertainty and strengthen the design basis, priority actions include targeted field testing and an updated structural interpretation, integration of coupled thermal–hydraulic evaluation where appropriate, optimization of well spacing and activation sequencing, expansion and diversification of disposal permitting (including reinjection), and strengthening of surface-water and groundwater monitoring tied to clear trigger actions and contingency planning. A staged underground-based dewatering approach should also be evaluated as a flexible complement to surface wells.

Hydrogeologic setting: The groundwater system comprises five main hydrostratigraphic units with flow strongly controlled by structures. Shallow alluvium is highly permeable and largely discharges to the Rio Ostua, while deeper fault zones act as preferential conduits for geothermal upflow. Groundwater temperatures can locally approach 190°C, which is a key constraint for pump selection, materials, and overall water-handling design.

Modeling basis: The groundwater model was updated in FEFLOW over an approximate 174 km² domain with 25 layers, building on prior work. A steady-state calibration to 45 observation points achieved NRMSE around 6.1%. Transient simulations for 2008–2024 reproduce observed hydrographs and confirm the Rio Ostua as a gaining stream.

Dewatering requirements: Predicted dewatering rates ramp from about 1,220 gpm (2026) to about 6,080 gpm (2029). Full build-out reaches roughly 10 wells by January 2031, sustaining approximately 6,080 gpm, with contingency capacity up to about 7,600 gpm to manage operational variability.

Discharge capacity and reinjection: The currently permitted discharge capacity of about 5,250 gpm is below projected dewatering needs. The strategy includes two reinjection wells (about 1,000 gpm each) to manage peak flows, reduce reliance on surface discharge limits, and preserve operational flexibility beyond 2031.

Hydrologic impacts: By 2043, model results indicate annual flow reductions of approximately 12% in the Rio Ostua (about 1,564 to 1,376 L/s) and approximately 46% in the Rio Tacunshapa (about 6.5 to 3.5 L/s). Treated mine effluent is expected to exceed baseflow depletion under the evaluated conditions, supporting downstream compliance.

Key risks and controls: The high-temperature regime introduces a steam-flashing risk and increases mechanical and materials demands on dewatering infrastructure. The design basis assumes ESPs with appropriate pressure and backpressure control, plus materials and components suitable for geothermal conditions.

Recommended next steps: Reduce uncertainty through targeted testing and an updated structural interpretation, integrate coupled thermal–hydraulic evaluation where needed, optimize well spacing and activation sequencing, expand and diversify disposal permitting (including reinjection), strengthen surface-water and groundwater monitoring with clear trigger actions, and evaluate staged underground-based dewatering as a flexible complement to surface wells.

1.11.3 Mine Infrastructure

The ventilation, cooling and underground pumping systems for the Era Dorada underground mine have been designed to support safe, efficient, and sustainable operations throughout the life of mine as required by the Project site high ambient temperatures, geothermal conditions, diesel equipment fleet, and planned production schedule. In the absence of Guatemalan-specific mine ventilation regulations, the design adopts criteria and standards commonly applied in the United States and Brazil, consistent with internationally accepted mining industry practices.

Ventilation airflow requirements were calculated based on the diesel equipment fleet, the number of active and inactive levels, and additional demands from fixed installations such as pumping stations. Total mine airflow requirements are forecast to peak at approximately 392 m³/s in 2031, corresponding to the period of maximum development and production activity. The main ventilation system is configured as a push–pull arrangement, with all primary fans installed on surface. The system incorporates both northern and southern ventilation circuits utilizing

existing and new raises. The design for the main fans includes both axial intake fans and centrifugal exhaust fans, with centrifugal units selected for exhaust duties due to high system resistance, elevated humidity, and corrosive air conditions. Fan installations provide sufficient capacity and redundancy to meet peak airflow demands without constraining production.

Local ventilation within development and production levels will be provided by auxiliary fans supplied with cooled intake air. Two fan configurations are specified: 75 HP units delivering approximately 20 m^3/s for production, exploration, and preparation headings, and 125 HP units delivering approximately 30 m^3/s for main development headings. Due to high underground temperatures, return air is exhausted directly to surface and is not reused at other levels.

Given the Project's geographic location and geothermal gradient (approximately 0.1 °C/m), underground heat loads represent a major design consideration. Total heat load at peak production is estimated at approximately 21 MWR, with heat transfer from surrounding rock mass and geothermal water accounting for roughly 55% of the total. The mine cooling system is designed to maintain underground working conditions below 28 °C wet-bulb temperature, consistent with industry standards for worker's safety and productivity.

The cooling strategy consists of three surface-based cooling plants installed at intake raises between 2026 and 2030. These plants provide a combined cooling capacity of approximately 15 MWR, supplying cooled intake air to the main ventilation circuits.

The underground pumping system is designed to manage water used by mine equipment, groundwater infiltration not captured by surface dewatering wells, and rainwater infiltration. Pumping requirements were based on year-by-year groundwater inflow projections provided by Stantec and the mine equipment water balance. Total underground water inflows, including a 25% contingency, are projected to peak at approximately 44.6 L/s in 2040. The Southern Zone pumping system with 50 L/s capacity and 285 kW includes three pumping stations, while the Northern Zone system with 7 L/s capacity and 38 kW has two main pumping stations, both conveying water to the site water treatment plant.

The ventilation, cooling, and underground pumping systems are considered technically sound and appropriately designed at a Feasibility Study level by the QP. Capital and operating costs have been estimated with FS accuracy and integrated into the mine plan and economic analysis. With the implementation of the proposed systems, ventilation, thermal conditions, and underground water management will not constrain mine safety, production rates, or economic performance over the life of mine.

1.12 Processing and Recovery Methods

Aura has developed a process flowsheet for their Era Dorada project to process run-of-mine (ROM) ore to produce gold (Au) and silver (Ag) doré. The metallurgical test programs summarized in Section 10, have demonstrated that gravity concentration followed by cyanide leach, carbon adsorption/desorption and electrowinning can yield an average overall recovery of 96% Au and 85% Ag. Results from this test program were used to develop the corresponding process design criteria, mechanical equipment list, flowsheets and operating costs.

The primary crushing plant will have a throughput capacity of 1,600 t/d with average life of mine (LOM) head grades of 6.0 g/t Au and 28.2 g/t Ag. The crushing circuit will operate at an availability of 75%, resulting in an operating rate

of 88.9 t/h. The grinding, gravity, leaching and CIP circuits will operate 24 hours per day, 365 days per year at an availability of 92%, resulting in an hourly throughput of 72.5 t/h. Carbon elution and regeneration will process 4 t of loaded carbon daily to produce gold (Au) and silver (Ag) doré.

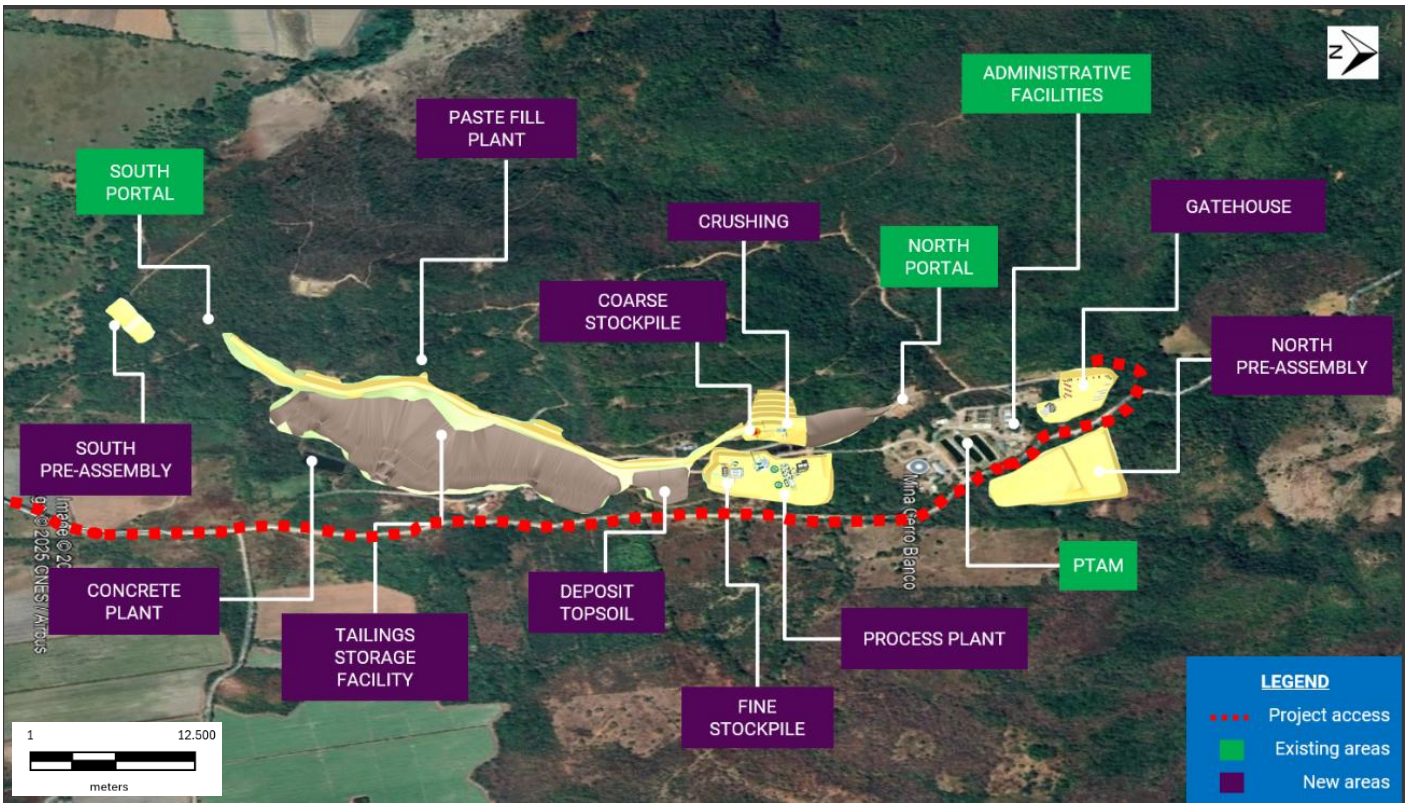
The primary crushing circuit will reduce the ROM ore to a product size P_{80} of 87.8 mm and the two-stage grinding circuit will target a final P_{80} grind size of 53 µm. Centrifugal gravity concentrators will be fed by a portion of the hydrocyclone underflow, to recover any gravity recoverable gold and silver. Hydrocyclone overflow passes over a trash screen with the undersize thickened with the underflow pumped to the leach circuit with the gold- and silver cyanide complexes adsorbed onto activated carbon in the CIP (carbon in pulp) circuit. The loaded carbon is recovered from the first CIP tank via the loaded carbon screen. Loaded carbon will be washed with hydrochloric acid and eluted with the pressure Zadra process with a strong caustic cyanide solution at 140°C. The resulting pregnant gold and silver solution will be passed through electrowinning, and recovered to a precious metal sludge which will be refined in an induction furnace to produce gold and silver doré. CIP circuit tailings will be treated with the SO_2/Air cyanide destruction process, and the final tailings will be pressure filtered to a moisture content of 19% and transferred to a dry stack tailings facility or mixed with cement to produce paste to be deposited underground.

1.13 Infrastructure

1.13.1 Introduction

The Era Dorada Project is a gold mining initiative located in southeastern Guatemala, within the municipality of Asunción Mita, department of Jutiapa, approximately 160 km from Guatemala City and 9 km from the border with El Salvador. The concession area covers 15.25 km² entirely within Asunción Mita, with the nearest town being Asunción Mita itself, a community of about 17,500 inhabitants situated 7 km from the siteFigure 15-1: Mine Era Dorada – Location.

Figure 1-2: Infrastructure Layout Plan



Source: Ausenco, 2025.

1.13.2 Site Access

Current access to the project site is through Asunción Mita, via narrow streets and a gravel road crossing the Grande de Mita River over the El Achotal bridge, limited to 27 t, which is inadequate for continuous transport of heavy equipment.

To meet construction and operational requirements, a new access road has been designed, connected to the CA1 Pan-American Highway, suitable for bidirectional traffic at 50 km/h, and upgrades to existing rural roads.

Within the site, the main roads will comprise the access road to the industrial complex and the connector road between the North and South portals, supplemented by auxiliary roads for drainage and reinjection wells, all within property boundaries and security fencing. Ore will be transported from the North portal to the crusher, while waste rock will be hauled from the South portal to the designated dump, with temporary construction roads adapted from existing routes as needed.

1.13.3 Building Infrastructure

The definition of building types for the project considered construction feasibility, cost-benefit ratio, functionality, execution and plant operation timelines, as well as the use of existing structures. All buildings must comply with seismic standards NSE-2-2024 and NSE-3-2024, applicable to Seismic Zone 4.1 in Guatemala, adopting a PGA of 0.40 g for stability analyses. Four main typologies were established:

(i) prefabricated thermo-acoustic modular buildings for administrative and support facilities, delivered complete and ready for use;

(ii) steel structures for the mine workshop, vehicle wash, reagent and cyanide storage, with masonry up to 2.0 m and metal roofing;

(iii) masonry buildings for new electrical substations, constructed with cast-in-place concrete, block masonry, and metal roofs; and

(iv) renovations of existing buildings, maintaining their original typology, including masonry structures and steel roofing, with adaptations using thermo-acoustic panels and glass where required.

1.13.4 Geotechnical Facilities

1.13.4.1 General Considerations

The Era Dorada Project requires engineered facilities for the disposal of tailings, waste rock, topsoil, and low-grade ore. Tailings will be filtered and deposited as paste backfills underground and in Dry Stack Tailings Facilities (DSTFs). Waste rock will be placed underground as cemented or loose rock fill, with remaining volumes stored in surface Waste Rock Dumps (WRDs). These facilities are designed to ensure long-term stability, environmental compliance, and operational efficiency.

1.13.4.2 Site Characterization

Geotechnical investigations identified alluvial, colluvial, and volcanic deposits with moderate strength and low permeability in fine-grained soils. Groundwater occurs at approximately ten meters depth. The site is located in a tectonically active region, requiring seismic-resistant design in accordance with AGIES 2018 and GISTM standards. Tailings are non-plastic, predominantly silt-sized, with low hydraulic conductivity and no potential for acid generation. Waste rock is considered potentially acid generating, requiring liner systems and drainage controls.

1.13.4.3 Design and Construction Considerations

Foundations will be prepared by removing unsuitable materials and installing geomembrane liners with subsurface drainage. Conservative geometries have been adopted for seismic stability: DSTFs with 3H:1V slopes and WRDs with 2H:1V slopes. Water management systems will separate contact and non-contact flows, supported by diversion channels and ponds. Construction will involve controlled placement and compaction of materials, with continuous monitoring through instrumentation and quality control surveys.

1.13.4.4 Facility Capacities, Stability and Performance

Two DSTFs are planned: the first with a capacity of approximately 498,000 m^3 and the second with 2.16 Mm^3. Waste rock will be stored in WRD 1 (145,000 m^3), WRD 2 (67,000 m^3), and a new WRD with 616,000 m^3 capacity. Additional areas include a topsoil deposit of 6,800 m^3 and a low-grade ore stockpile of 61,500 m^3.

Slope stability analyses under static, earthquake, and post-earthquake conditions achieved required factors of safety. Reinforcement with geogrids is planned for DSTFs under seismic loading. WRDs demonstrated stability under all scenarios, with localized foundation treatment required for WRD 1.

1.13.5 Water Management

The surface water management infrastructure was designed to segregate contact and non-contact runoff, directing potentially contaminated flows to containment basins for treatment, while clean water is conveyed through drainage channels to controlled discharge points. Perimeter channels, energy dissipation structures, crossings, and a diversion channel were defined to ensure proper stormwater management. The water balance evaluated inflows, uses, storage, and discharge, verifying compliance with water rights and identifying the need for capacity expansion. The treatment infrastructure includes dedicated systems for mine water, process water, potable water, and sanitary wastewater.

1.13.6 Electrical Power

The project's electrical system has been designed to ensure reliable and scalable power supply from construction through full operation. Initially, power will be provided by a 17 MW diesel plant operating at 4.16 kV under a lease arrangement for the first three years. Permanent power supply will come from a utility substation in Asunción Mita, connected to the site via a 69 kV overhead transmission line approximately 8.6 km long, scheduled for commissioning in 2030. The architecture includes a main substation with step-down transformers, medium-voltage switchgear, primary and secondary distribution networks, as well as emergency and redundancy systems, ensuring operational flexibility, safety, and compliance with industry best practices.

1.13.7 Fuel

The Era Dorada project currently has an on-site diesel storage facility consisting of two tanks of approximately 37,500 L each, within a concrete containment area, which meets the current demand for electric power generators and for fueling underground mine vehicles and equipment through a mobile fueling station. With the planned expansion, diesel consumption will increase due to on site power generation during the initial years of operation, an expanded fleet for ore transport and handling, and operation of equipment for waste and tailings management. To meet this demand, additional infrastructure for fuel supply, logistics, distribution, and storage will be contracted, ensuring supply for underground mining fronts and ore handling at the processing plant and storage areas. Light vehicles powered by gasoline or other fuels will be refueled off-site in Asunción Mita, Guatemala.

1.14 Market Studies and Contracts

Mineral resource estimations were conducted using a reference gold price of US$2,000/oz. For the evaluation of project economics, gold and silver prices were defined as a price vector over the life of the project, reflecting long-term consensus forecasts compiled from more than 20 investment banking institutions.

The applied metal assumptions are summarized in Table 1-7.

Table 1-7: Gold and Dilver Pricing

Precious Metals	2025	2026	2027	2028	LT
Gold (US$/oz)	$3,368	$3,930	$3,827	$3,689	$3,140
Silver (US$/oz)	$37.5	$45.2	$42.8	$40.0	$36.9

No contracts have been entered into at the report effective date for the Era Dorada project.

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

1.15.1 Environmental Considerations

The Era Dorada Project is licensed to proceed with the operation of the underground mine based on the implementation of the beneficiation and support infrastructure, as presented in the Environmental Impact Assessment (EIA) approved in 2007 by the Ministerio de Ambiente y Recursos Naturales (MARN) of Guatemala. The EIA remains valid, and its associated license is renewed whenever necessary. The continuity of activities depends on compliance with the requirements of current licenses, evidencing the commitment to legal and regulatory compliance. The project maintains up-to-date records of environmental licensing and monitoring.

1.15.2 Closure and Reclamation Considerations

The EIA approved for the Era Dorada Project includes a Conceptual Mine Closure Plan, which establishes guidelines for the decommissioning of facilities after operations have ceased. Although Guatemalan law requires the official presentation of the plan three years before the end of activities, during the 2019 Feasibility Study, the plan was reportedly reviewed for Aura's internal management purposes.

1.15.3 Permitting Considerations

The Era Dorada Project is being developed in accordance with the scope approved in the EIA of 2007, which includes the underground mining operation and the respective environmental licenses in force. However, as the project progresses and new technical and operational demands are identified, the submission of new permitting applications will be required.

1.15.4 Social Considerations

The social strategy of the Era Dorada Project seeks to strengthen the relationship with local communities through transparency, traceability and active participation. The Social Management Plan (SMP) as presented in the 2007 EIA guides actions aimed at community development and impact mitigation and provides a strategy and framework that can be periodically reviewed and revised as conditions require.

1.16 Capital and Operating Costs

The capital and operating cost estimates presented in this report provide substantiated costs that support the feasibility study of Aura's Era Dorada project. The estimates considered the beneficiation plant with an average gold production rate of 104,113 oz/year.

The capital and operating cost estimate aligns with Class 3 guidelines for an FS-level estimate with an accuracy range of +/-15% as specified by the Association for the Advancement of Cost Engineering International (AACE International). The estimate, developed in Q4 2025, is based on the proposed design for the Project, on Ausenco's budgetary quotations, in-house project and study database and Aura's inputs.

All capital and operating cost estimates are presented in US dollars (symbol: US$, currency: USD) and Guatemalan Quetzal (currency: Q, currency: GTQ). The exchange rate applied is:

Guatemalan Quetzal (GTQ) to US dollars (USD): GTQ7.60 = USD1.00.

1.16.1 Capital Cost Estimate

The following costs and scope items are excluded from the capital cost estimate:

- scope changes, project schedule changes, and other associated costs;
- any facilities or structures not included in the project scope;
- tax benefit analysis; and
- demolition or decontamination costs for existing site.

The total capital cost for the Era Dorada Project is US$382.11 million, of which US$197.75 million is for the Plant and US$4.74 million for the tailings, waste rock and stockpiles and US$179.64 of mining costs (with contingency).

The capital cost summary is presented in Table 1-8.

Table 1-8: Capital Cost Summary

Description	Initial Total Cost (MGTQ)	Initial Total Cost (MUSD)
Equipment	186.11	24.49
Turn-key Packages	199.67	26.27
Materials	87.77	11.55
Construction and Erection	313.67	41.27
Laboratory	20.69	2.72
Pre-Production Power Plant Fuel	20.65	2.72
Land Compensation	16.99	2.24
Topographic Survey	2.09	0.27
Soil Resistivity Testing	1.19	0.16
Site Clearing	0.85	0.11
Owner Costs	265.44	34.93
Indirect Costs	210.67	27.72
Plant Contingency	177.05	23.30
Tailings, Rock Waste and Stockpiles	31.75	4.18
Tailings, Rock Waste and Stockpiles Contingency	4.23	0.56
Mining	1204.95	158.55
Mining Contingency	160.29	21.09
Project Total	**2,904.05**	**382.11**

Note: Values may not sum correctly due to rounding.

1.16.2 Operating Cost Estimate

Operating costs include the ongoing costs of operations related to processing, tailings and waste rock disposal, water treatment stations, as well as general and administrative activities. Table 1-9 provides a summary of the operating costs across all phases of operation, expressed on a USD/t ROM basis.

Operation estimated to start with a four-month ramp-up period.

Table 1-9: Operating Cost Summary (USD/t ROM basis)

Item	Year -1 (Ramp-up)	Year 1 (Ramp-up)	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t
Labor	34.4	21.0	20.5	20.5	12.8	12.8	12.8	12.8	12.8
G&A	30.7	18.7	18.3	24.4	15.3	15.3	15.3	15.3	15.3
Laboratory	9.5	4.5	4.4	4.4	2.8	2.8	2.8	2.8	2.8
Access Maintenance	8.9	5.0	4.8	4.9	3.0	3.0	3.0	3.0	3.0
Mobile Equipment Fleet	5.7	3.2	3.1	3.1	1.9	1.9	1.9	1.9	1.9
Reagents	7.6	7.6	7.6	7.6	7.6	7.6	7.6	7.6	7.6
Consumables	7.1	5.6	7.8	7.5	6.3	6.3	6.3	6.3	6.3
Maintenance, Fuel and Lubricants	8.4	5.1	5.0	5.0	3.1	3.8	3.8	3.8	3.8
Power	89.4	81.4	90.0	95.7	63.2	20.8	20.8	20.8	20.8
Water Treatment	15.1	10.8	10.5	14.8	9.4	11.1	11.3	11.3	11.3
Tailings and Rock Waste Piles	31.9	11.3	2.2	11.1	7.0	7.0	7.0	7.0	7.0
Mining Costs	122.5	71.6	80.4	108.8	86.8	86.7	87.7	71.3	73.7
Total Operating Costs	**371**	**246**	**255**	**308**	**219**	**179**	**180**	**164**	**166**

Item	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t
Labor	13.3	13.2	13.2	13.2	13.2	13.2	14.1	14.1	19.5
G&A	15.3	15.3	15.3	15.3	15.3	14.7	16.3	16.3	23.8
Laboratory	2.8	2.8	2.8	2.8	2.8	2.8	3.0	3.0	4.8
Access Maintenance	3.0	3.0	3.0	3.0	3.0	3.0	3.2	3.2	5.2
Mobile Equipment Fleet	2.0	1.9	1.9	1.9	1.9	1.9	2.1	2.1	3.3
Reagents	7.6	7.6	7.6	7.6	7.6	7.6	7.6	7.6	7.6
Consumables	6.3	6.3	6.3	6.3	6.3	6.3	6.5	6.5	7.6
Maintenance, Fuel and Lubricants	3.8	3.8	3.8	3.8	3.8	3.8	4.0	4.0	6.3
Power	20.9	20.8	20.8	20.8	20.8	20.8	22.2	22.1	34.8
Water Treatment	11.4	11.5	11.5	11.5	11.6	11.6	12.3	12.3	19.3
Tailings and Rock Waste Piles	7.0	7.0	7.0	7.0	7.0	7.0	7.4	7.4	14.4
Mining Costs	62.1	61.3	61.3	62.3	62.8	62.9	58.1	56.2	74.1
Total Operating Costs	**155**	**155**	**154**	**156**	**156**	**156**	**157**	**155**	**221**

Note: Values may not sum correctly due to rounding.

Common to all operating cost estimates are the following assumptions:

- Cost estimates are based on Q4 2025 pricing, without allowances for inflation.
- Costs are expressed in USD, using the exchange rate of GTQ7.60 = USD1.00.
- Equipment and materials will be purchased as new.
- Reagent consumption rates were determined by metallurgical test results.

1.16.3 Sustaining Capital Cost Estimate

Plant sustaining costs consider the purchase and assembly of the main electrical substation, power transmission line 69 kV, Asunción Mita's electrical connection substation, underground mine water treatment and decontamination station, as well as tailings and waste rock piles and owner costs.

The sustaining capital costs summary is presented in Table 1-10.

Table 1-10: Sustaining Capital Costs

Description	Year -1 (MUSD)	Year 1 (MUSD)	Year 2 (MUSD)	Year 3 (MUSD)	Year 4 (MUSD)	Year 5 (MUSD)	Year 6 (MUSD)	Year 7 (MUSD)	Year 8 (MUSD)
Plant Costs	-	9.45	-	4.91	-	-	-	-	-
Off-Site Infrastructure Costs	-	-	-	3.3	-	-	-	-	-
Tailings, Rock Waste and Stockpiles	-	1.95	9.05	-	-	3.66	-	-	3.66
Indirect Costs	0.27	0.06	0.05	0.02	-	0.02	-	-	0.02
Mine Costs	6.07	53.71	27.12	26.89	18.37	8.34	8.95	10.98	6.07
Plant Contingency	0.05	1.9	-	0.99	-	-	-	-	-
Off-Site Infrastructure Contingency	-	-	-	0.66	-	-	-	-	-
Tailings, Rock Waste and Stockpiles Contingency	-	0.39	1.82	-	-	0.74	-	-	0.74
Mine Contingency	1.21	10.74	5.42	5.38	3.67	1.67	1.79	2.2	1.7
Project Total	**7.62**	**78.2**	**43.46**	**42.15**	**22.04**	**14.42**	**10.74**	**13.18**	**14.62**

Description	Year 9 (MUSD)	Year 10 (MUSD)	Year 11 (MUSD)	Year 12 (MUSD)	Year 13 (MUSD)	Year 14 (MUSD)	Year 15 (MUSD)	Year 16 (MUSD)	TOTAL (MUSD)
Plant Costs	-	-	-	-	-	-	-	-	14.36
Off-Site Infrastructure Costs	-	-	-	-	-	-	-	-	3.3
Tailings, Rock Waste and Stockpiles	-	-	-	-	-	-	-	-	18.32
Indirect Costs	-	-	-	-	-	-	-	-	0.44
Mine Costs	3.13	6.22	3.34	6.82	11.52	5.4	2.17	0.49	208.03
Plant Contingency	-	-	-	-	-	-	-	-	2.94
Off-Site Infrastructure Contingency	-	-	-	-	-	-	-	-	0.66
Tailings, Rock Waste and Stockpiles Contingency	-	-	-	-	-	-	-	-	3.68
Mine Contingency	0.63	1.24	0.67	1.36	2.3	1.08	0.43	0.1	41.61
Project Total	**3.75**	**7.46**	**4.01**	**8.19**	**13.82**	**6.48**	**2.6**	**0.58**	**293.34**

Note: Values may not sum correctly due to rounding.

1.17 Economic Analysis

1.17.1 Economic Summary

The results of the economic analyses discussed in this section represent forward-looking information as the results depend on inputs that are subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here.

Forward-looking statements in this Report include, but are not limited to, statements with respect to future metal prices and concentrate sales contracts, assumed currency exchange rates, the estimation of Mineral Reserves and Mineral Resources, the realization of Mineral Reserve estimates including the achievement of the dilution and recovery assumptions, the timing and amount of estimated future production, costs of production, capital expenditures, costs and timing of the development of ore zones, permitting time lines, requirements for additional capital, government regulation of mining operations, environmental risks, unanticipated reclamation expenses and title disputes.

The Project was evaluated using an 5% discounted cash flow (DCF) analysis on a non-inflated, post-tax basis. The cash flows consist of approximately 1 years of pre-production costs and 17 years of operations. Cash inflows consist of annual revenue projections for the mine calculated at considering a price scenario as presented in Section 16. Cash outflows include capital costs, operating costs, royalties, and taxes, which are subtracted from the inflows to arrive at the annual cash flow projections.

The financial model is based on the Mineral Reserves outlined in Section 11, the mining rates and assumptions discussed in Section 12 and 13; and processing rates and recovery methods discussed in Section 14 and capital and operating costs in Section 18, respectively.

- Initial capital costs are estimated to be US$382.1 million. Over the LOM sustaining capital is estimated to be US$293.3 million.
- LOM operating costs are estimated to be US$1,543 million.
- Closure and reclamation costs are estimated to be US$17.2 million.
- LOM royalties are estimated to be US$369.5 million.
- For the treatment & refining charges and transportation costs, the LOM costs are estimated to be US$17.9 million.
- The pre-tax NPV discounted at 5%, is US$1,535.2 million, the internal rate of return (IRR) is 38.5%, and payback period is 2.7 years. On a post-tax basis, the NPV discounted at 5% is US$1,344.5 million the IRR is 35.6%, and payback period is 2.8 years. A cash flow summary is included below in Table 1-11.

Table 1-11: Economic Analysis Summary

General	Unit	LOM Total Value/Average
Gold realized Price	US$/oz	$3,177
Silver realized Price	US$/oz	$37.2
Mine Life	years	16.8
Production – LOM		
Ore to Plant	kt	8,747
Total Recovered Gold	koz	1,620.4
Total Payable Gold	koz	1,618.7
Total Recovered Silver	koz	4,876.4
Total Payable Silver	koz	4,852.0
Operating Costs		
Mining Cost	US$/t	$71
Processing Cost	US$/t ore	$87.92
G&A Cost	US$/t ore	$10.54
Refining & Transport Cost	US$/oz	$11.06
Total Operating Costs	US$/t ore	$176.37
Cash Costs *	US$/oz Au	$993.1
AISC **	US$/oz Au	$1,178.0
Capital Costs		

General	Unit	LOM Total Value/Average
Initial Capital	US$M	$382.1
Sustaining Capital	US$M	$293
Closure Capital	US$M	$17.2
Financials - Pre Tax		
NPV (5%)	US$M	$1,535.2
NPV (0%)	US$M	$2,701.5
NPV (10%)	US$M	$904.0
IRR (%)	%	38.5%
Payback (years)	years	2.7
Financials - Post Tax		
NPV (5%)	US$M	$1,344.5
NPV (0%)	US$M	$2,386.8
NPV (10%)	US$M	$781.1
IRR (%)	%	35.6%
Payback (years)	years	2.8

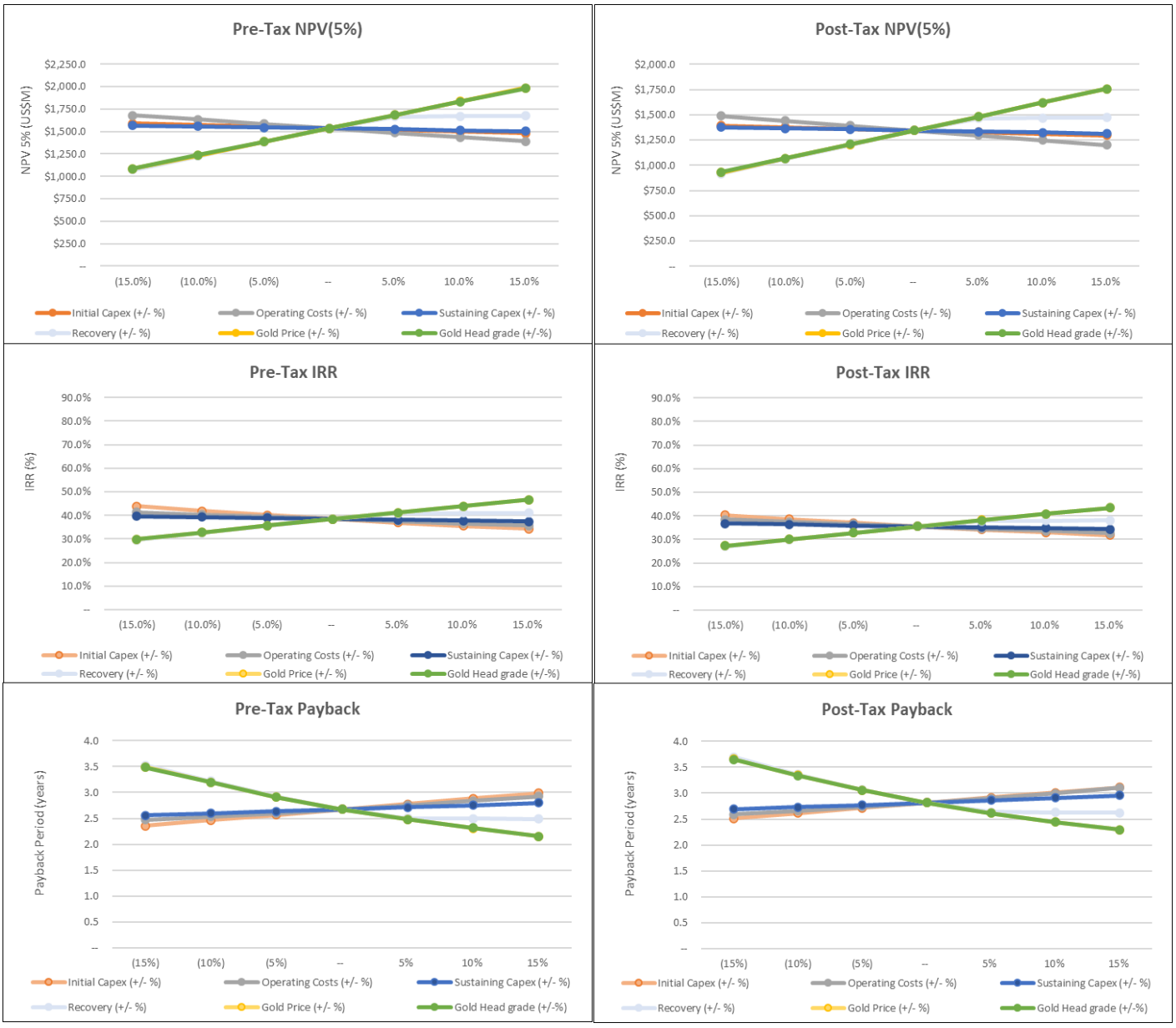
* Cash costs consist of mining costs, processing costs, mine-level G&A and refining charges and royalties.
** AISC includes cash costs plus sustaining capital, closure cost and salvage value.

1.17.2 Sensitivity Analysis

A sensitivity analysis was conducted on pre-tax and post-tax NPV and IRR of the Project, examining the following variables: gold price, gold head grade, gold recovery, sustaining capital costs, initial capital costs, and operating costs. The analysis revealed that the Project is most sensitive to gold price followed by gold head grade, with lesser sensitivity to changes in initial capital costs, operating costs, and sustaining capital costs, as shown in Figure 1-3.

Table 1-12 presents the findings of the pre-tax sensitivity analysis, and Table 1-13 shows the post-tax results.

Figure 1-3: Sensitivity Analysis Pre-Tax and Post-Tax



Source: Ausenco, 2025.

Table 1-12: Pre-Tax Sensitivity

Au Price (US$/oz)	Pre-Tax NPV(5%)	Initial Capex (US$M)		Opex (US$M)		Sustaining Capex (US$M)		Au Recovery (%)	
		(20%)	20%	(20%)	20%	(20%)	20%	(20%)	20%
(25%)	$786	$858	$715	$981	$592	$830	$743	$319	$898
(10%)	$1,236	$1,307	$1,164	$1,430	$1,042	$1,279	$1,192	$678	$1,366
0%	$1,535	$1,607	$1,464	$1,729	$1,341	$1,579	$1,492	$918	$1,678
10%	$1,835	$1,906	$1,763	$2,029	$1,640	$1,878	$1,791	$1,157	$1,990
25%	$2,284	$2,355	$2,213	$2,478	$2,090	$2,328	$2,240	$1,517	$2,458
Au Price (US$/oz)	**Pre-Tax IRR(%)**	**Initial Capex (US$M)**		**Opex (US$M)**		**Sustaining Capex (US$M)**		**Au Recovery (%)**	
		(20%)	20%	(20%)	20%	(20%)	20%	(20%)	20%
(25%)	23.8%	28.7%	20.1%	27.9%	19.5%	25.2%	22.4%	13.4%	26.1%
(10%)	32.8%	39.3%	28.1%	36.6%	28.9%	34.3%	31.3%	21.5%	35.3%
0%	38.5%	46.0%	33.1%	42.2%	34.7%	40.1%	37.0%	26.5%	41.2%
10%	44.0%	52.5%	37.9%	47.6%	40.3%	45.6%	42.4%	31.2%	46.8%
25%	52.0%	61.9%	44.9%	55.5%	48.4%	53.6%	50.4%	38.1%	55.0%

Table 1-13: Post-Tax Sensitivity

Au Price (US$/oz)	Post-Tax NPV(5%)	Initial Capex (US$M)		Opex (US$M)		Sustaining Capex (US$M)		Au Recovery (%)	
		(20%)	20%	(20%)	20%	(20%)	20%	(20%)	20%
(25%)	$655	$717	$594	$847	$471	$698	$613	$241	$757
(10%)	$1,069	$1,132	$1,005	$1,261	$875	$1,111	$1,025	$558	$1,189
0%	$1,344	$1,408	$1,281	$1,537	$1,152	$1,387	$1,302	$775	$1,476
10%	$1,620	$1,683	$1,557	$1,813	$1,428	$1,663	$1,578	$996	$1,763
25%	$2,034	$2,097	$1,970	$2,228	$1,841	$2,077	$1,991	$1,327	$2,194
Au Price (US$/oz)	**Post-Tax IRR(%)**	**Initial Capex (US$M)**		**Opex (US$M)**		**Sustaining Capex (US$M)**		**Au Recovery (%)**	
		(20%)	20%	(20%)	20%	(20%)	20%	(20%)	20%
(25%)	21.5%	25.8%	18.3%	25.6%	17.4%	22.9%	20.1%	11.8%	23.6%
(10%)	30.1%	35.9%	25.9%	34.1%	26.0%	31.6%	28.6%	19.3%	32.5%
0%	35.6%	42.3%	30.7%	39.4%	31.6%	37.1%	34.0%	24.0%	38.1%
10%	40.8%	48.5%	35.3%	44.5%	37.0%	42.4%	39.3%	28.6%	43.5%
25%	48.4%	57.3%	42.0%	52.0%	44.8%	50.0%	46.8%	35.2%	51.3%

1.18 Interpretations and Conclusions

Based in the assumptions and parameters presented in this report, the FS study shows positive economics (i.e., US$1,344 million post-tax NPV (5%) and 35.6% post-tax IRR). This FS presents a project that is ready for submission for financial and other support necessary for initiation.

1.19 Recommendations

The financial analysis of this FS demonstrates positive economics. It is recommended to continue developing the project through additional studies. The items required to be completed in advance of, and as inputs to, a execution stage are indicated in the respective sections below.

Table 1-14: Recommended Work Program - Summary

Program Component	Unit Cost (USD)	Estimated Total Cost (MUSD)
Geology and resource estimates		7.75
Mining methods		0.26
Metallurgical Testing		0.60
Hydrogeology		0.15
Infrastructure facilities		0.30
Water management		0.10
Environmental studies		0.40
Total		**9.56**

2 INTRODUCTION

2.1 Basis of Technical Report

In January 2025, Aura Minerals Inc ("Aura" or "the Company") completed the acquisition of the of the Era Dorada Gold Project - formerly "Cerro Blanco Gold Project" - and Mita Geothermal Project, located in Jutiapa, Guatemala, near the town of Asunción Mita and the border with El Salvador. The Era Dorada Project ("Era Dorada" or "the Project") is 100% beneficially owned by Aura. Aura is a public, NASDAQ-listed company trading under the symbol "AUGO", with its head office located at 78 SW 7th St., Miami, FL 33130, USA.

Aura commissioned Ausenco do Brasil Engheharia Ltda. (Ausenco) to prepare a Feasibility Study (FS) and associated Technical Report Summary (TRS) on the Project.

This TRS, titled "Era Dorada Project, S-K 1300 Technical Report Summary and Definitive Feasibility Study, Jutiapa, Guatemala," has been prepared in compliance with United States Securities and Exchange Commission's (SEC) Modernized Property Disclosure Requirements for Mining Registrants as described in Subpart 229.1300 of Regulation S-K, Disclosure by Registrants Engaged in Mining Operations (S-K 1300) and Item 601 (b)(96) Technical Report Summary.

The responsibilities of the engineering consultants are as follows:

- Ausenco was responsible for managing an coordinating the work related to the FS and the TRS and established an economic framework for the FS. The scope of work included the development of a conceptual flowsheet as well as detailed flowsheets, specifications, and the selection of process equipment. Ausenco also provided design oversight related to site infrastructure including the access road, power line, plant facilities, and other ancillary facilities. In addition, Ausenco designed the drystack tailings facility (DSTF) and waste rock facility (WRF), as well as the surface water management, including design of ditches, channels and ponds for stormwater control. The team's responsibilities further included estimating the process plant, general and administrative (G&A), and site services capital (CAPEX) and operating (OPEX) costs; preparing a financial model and conducting an economic evaluation including sensitivity and Project risk analyses; and developing a Project Execution Plan.liStantec Consulting Inc. (Stantec) was responsible for dewatering and injection requirements.lBBE Company Inc. was responsible for the Cooling Plant associated with the underground mine.lQualifications and ResponsibilitieslThe Qualified Persons (QPs) preparing this report are specialists in the fields of geology, exploration, Mineral Resource and Mineral Reserve estimation and classification, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics. lNone of the QPs or any associates employed in the preparation of this report has any beneficial interest in Aura Minerals and neither are insiders, associates, or affiliates. The results of this report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Aura Minerals and the QPs. The QPs are being paid a fee for their work in accordance with normal professional consulting practices. lThe following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the SK-1300, and are members in good

standing of appropriate professional institutions/associations. The QPs are responsible for the specific report sections as follows in Table 2-1.

Table 2-1: QP Responsibilities

Qualified Person	Company	QP Responsibility/Role	Report Section(s)
Garth Kirkham	Aura	Property Description Accessibility, climate, local resources, infrastructure and physiography History Geological setting, mineralization and deposit Deposit Types Exploration Drilling Sample preparation, analyses and security Data Verification Mineral Resource Estimates Adjacent Properties	1.1, 1.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.9, 2 (contribution), 3, 4, 5, 6, 7, 8, 9, 11, 20, 22.1, 22.2, 22.3, 23.1, 24, 25.1, and 25.2
Tommaso Roberto Raponi	Ausenco	Mineral processing and metallurgical testing Processing and recovery methods Capital, Operating and Sustaining Cost	1.8, 1.12, 1.13, 1.14, 1.16, 1.16.2, 1.16.3, 1.17, 1.18, 1.19 (contribution), 2 (contribution), 10, 14, 15.1, 15.2, 15.3, 15.7, 15.8, 15.9, 16, 18.1, 18.3, 18.4, 19, 22.3, 22.6, 22.7.3, 22.7.4, 22.8, 22.10, 22.11, 22.12, 22.13.1.6, 22.13.1.7, 22.13.1.8, 22.13.2.2, 22.13.2.6, 22.13.2.7, 22.13.2.8, and 25.4.
Aleksandar Spasojevic		Dry Stacking Storage Facility Waste Storage Facilities Top Soil	1.13.4, 1.19 (contribution), 15.4, 22.7.1, 22.12 (contribution), 23.7 and 25 (contribution)
Jonathan Cooper		Water Supply and Management	1.13.5, 15.5, 15.6, 22.7.2, 22.12 (contribution) and 23.8

Qualified Person	Company	QP Responsibility/Role	Report Section(s)
James Millard		Environmental studies, permitting, and plans, negotiations, or agreements with local individuals or groups	1.15, 17, 22.8, 22.12 (contribution), 23.9 and 25.3
Ruy Lacourt	Snowden Optiro	Mineral Reserve Estimates Mining Methods Mining Capital, Operating and Sustaining Cost Dewatering	1.10, 1.11, 1.16 (contribution), 1.19 (contribution), 2.2 (contribution), 12, 13, and 18.2, 22.4, 22.5, 22.13.1.3, 22.13.2.3, 23.4; 23.5 and 23.6

2.2 Site Visit Details

1) Garth Kirkham, P. Geo., has been involved with the property since its acquisition in early 2017, when he performed the initial due diligence and authored the updated resource estimate for Bluestone. Mr. Kirkham first visited the property on May 8, 2017, to validate all aspects. The site visit included an inspection of the property, offices, underground vein exposures, core storage facilities, water treatment plant, and stockpiles, and a tour of major centers and the surrounding villages most likely to be affected by any potential mining operation.

    a. Since 2017, Mr. Kirkham has visited the property numerous times for extended periods to develop and implement data gathering and sampling methods and procedures. He also worked with Bluestone geologists to develop drill programs and to supervise interpretation and modeling efforts in addition to creating and implementing QA/QC procedures.

    b. From September 21 to 22, 2017, Mr. Kirkham inspected the progress of the recommended historic drill core rehabilitation program and initiated structural studies.

    c. From April 24 to 28, 2018, Mr. Kirkham's site visit focused on advancing the planning of sampling and drilling along with supporting lithological and structural modeling.

    d. From February 16 to 22, 2020, Mr. Kirkham provided guidance on the planning and development of advanced drilling and sampling, as well as grade vein modeling.

    e. From January 10 to 15, 2021, Mr. Kirkham assisted with validating drill and sample data, refining high-grade models, reviewing low-grade models, and providing guidance for the finalization of the open pit bulk tonnage resource scenario.

2) Ruy Lacourt, BSc. Mining Engineering, MSc., Registered Member of the SME, Associate Executive Consultant of Snowden Optiro and Qualified Person visited the project from July 7 to July 9 2025. The site visit included inspection of the property, underground works and associated surface infrastructure, core storage facilities, workshops and offices.

3) Tommaso Roberto Raponi visited and inspected the property September 16, 2025. Mr. Raponi inspected the existing infrastructure, location of future processing plant, reviewed diamond drill core and reviewed permitting status with site staff.

4) Aleksandar Spasojevic visited and inspected the property September 16, 2025. Dr. Spasojevic inspected the future locations of WRSF/DSTF, portals of the existing underground structure and location of future processing plant.

QPs not listed above have not visited or inspected the property. Personal inspections by these QPs are not required to complete their responsibilities.

The QPs are satisfied that no unauthorized access or other work has been conducted on the property based on the site security including site access via a paved road through a locked security gate combined with the fact that the site is continuously manned by company personnel.

Finally, the QPs also review publicly available information on the Company and its activities including the audited financial statements of the Company, which the QPs are satisfied do not point to any additional work being conducted on the property.

2.3 Sources of Information

This report is based on information collected by the QPs during site visits and on additional information provided by Aura Minerals throughout the course of the activiteis. Other information was obtained from the public domain. Ausenco has no reason to doubt the reliability of the information provided by Aura Minerals. This technical report is based on the following sources of information:

- Discussions with Aura Minerals's on-site personnel, including the site General Manager and Environmental Manager.
- Inspection of the site.
- Review of exploration data collected by Aura Minerals.
- Previous studies completed by Bluestone and Aura Minerals.
- Additional information from public domain sources.
- Previous studies have been published on Era Dorada since 2017 as follows:
    - Preliminary Economic Assessment (March 20, 2017)
    - Preliminary Economic Assessment Update (June 2, 2017)
    - Feasibility Study (January 29, 2019)
    - Preliminary Economic Assessment Update (February 28, 2021)
    - Preliminary Economic Assessment Update (June 30, 2021)

- Initial Assessment and Technical Report (December 31, 2024)

Four of the technical reports and resource estimates were for an underground mining scenario while two resource estimates were for the open pit scenario. All five NI43-101 technical reports are filed on the System for Electronic Document Analysis and Retrieval (SEDAR+) whilst the December 31, 2024 SK-1300 Technical Report Summary is filed on EDGAR.All estimates were authored by Qualified Person, Garth Kirkham, P.Geo.

2.4 Currency, Units, Abbreviations, Rounding and Definitions

The units of measure used in this report are as per the International System of Units (SI) or "metric", except for Imperial units that are commonly used in industry (i.e., ounces (oz.) for the mass of precious metals, US gallons per minute (gpm) for water flow rates).

This report includes technical information that required 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, the QPs do not consider them to be material.

All dollar figures quoted in this report refer to US dollars (US$ or USD) unless otherwise noted.

A list of abbreviations and acronyms is provided in Table 2-2, and units of measurement are listed inTable 2-3 Table 2-2.

Table 2-2: Abbreviations and Acronyms

Abbreviation	Description
AA	atomic absorption spectroscopy
Au	gold
Az	azimuth
BIF	banded iron formation
BBWi	bond ball mill work index
CAD:USD	Canadian-American exchange rate
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIM Definition Standards	CIM Definition Standards for Mineral Resources and Mineral Reserves 2014
CIP	carbon in pulp
CoG	cut-off grade
CRM	certified reference material
CWi	Bond crusher work index
DCIP	direct current resistivity and induced polarization
DDH	diamond drill hole
E-GRG	extended gravity recoverable gold

Abbreviation	Description
EM	electromagnetic
FA	fire assay
FET	federal excise tax
FoS	Factor of Safety
FS	feasibility study
G&A	general and administration
GPR	gross production royalty
GQCV	greenstone-hosted quartz-carbonate vein deposits
GRAV	gravimetric finish method
ICP	inductively coupled plasma
ICP-OES	inductively coupled plasma - optical emission spectrometry
ID2	inverse distance squared
ID3	inverse distance cubed
IOCG	iron oxide copper gold
IP	induced polarization
IRGS	intrusion-related gold system
ISO	International Organization for Standardization
LIDAR	light detection and ranging
LUP	land use permit
MCF	mechanized cut and fill
MRE	mineral resource estimate
NAD 83	North American Datum of 1983
NI 43-101	National Instrument 43-101 (Regulation 43-101 in Quebec)
NN	nearest neighbour
NSR	net smelter return
NTS	national topographic system
OK	ordinary kriging
PEA	preliminary economic assessment
PFS	prefeasibility study
PGE	platinum group elements
QA/QC	quality assurance/quality control
QP	qualified person (as defined in National Instrument 43-101)
ROM	run of mine
RQD	rock quality designation

Abbreviation	Description
SAG	semi-autogenous grinding
SCC	Standards Council of Canada
SD	standard deviation
S_d-BWI	micro hardness or bond ball mill work index on SAG ground material
SEDEX	sedimentary exhalative deposits
SG	specific gravity
TMF	tailings management facility
UG	underground
UTM	Universal Transverse Mercator coordinate system
UV	ultraviolet
VLF-EM	very low frequency electromagnetic
VMS	volcanogenic massive sulphide

Table 2-3: Units of Measurement

Abbreviation	Description
%	percent
% w/w solids	percent solids by weight
CAD	Canadian dollar (currency)
C$	Canadian dollar (as symbol)
$/t	dollars per metric ton
°	angular degree
°C	degree Celsius
µm	micron (micrometer)
cm	centimeter
cm^3	cubic centimeter
ft	foot (12 inches)
g	gram
g/cm^3	gram per cubic centimeter
g/L	gram per liter
g/t	gram per metric ton (tonne)
GTQ	Guatemalan Quetzal (as currency)
h	hour (60 minutes)

Abbreviation	Description
ha	hectare
kg	kilogram
kg/t	kilogram per tonne
km	kilometer
km^2	square kilometer
kW	kilowatt
kWh/t	kilowatt-hour per tonne
L	liter
lb	pound
m, m^2, m^3	meter, square meter, cubic meter
M	million
Ma	million years (annum)
masl	meters above mean sea level
mm	millimeter
Moz	million (troy) ounces
Mt	million tonnes
MW	megawatt
oz	troy ounce
oz/t	ounce (troy) per tonne
oz/ton	ounce (troy) per short ton (2,000 lbs)
ppb	parts per billion
ppm	parts per million
Q	Guatemalan Quetzal (as symbol)
t	metric tonne (1,000 kg)
ton	short ton (2,000 lbs)
t/d	tonnes per day
USD	US dollars (currency)
US$	US dollar (as symbol)

This report includes technical information that required 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, the QPs do not consider them to be material.

3 PROPERTY DESCRIPTION AND LOCATION

3.1 Introduction

The Project is located in southeast Guatemala, in the Department of Jutiapa, approximately 160 km by road from the capital, Guatemala City (Figure 3-1), and approximately 9 km west of the border with El Salvador. The nearest town to the Project is Asunción Mita, a community of approximately 18,500 people situated approximately 7 km west of the Project. The exploitation license covers 15.25 km2 and lies entirely in the municipality of Asunción Mita.

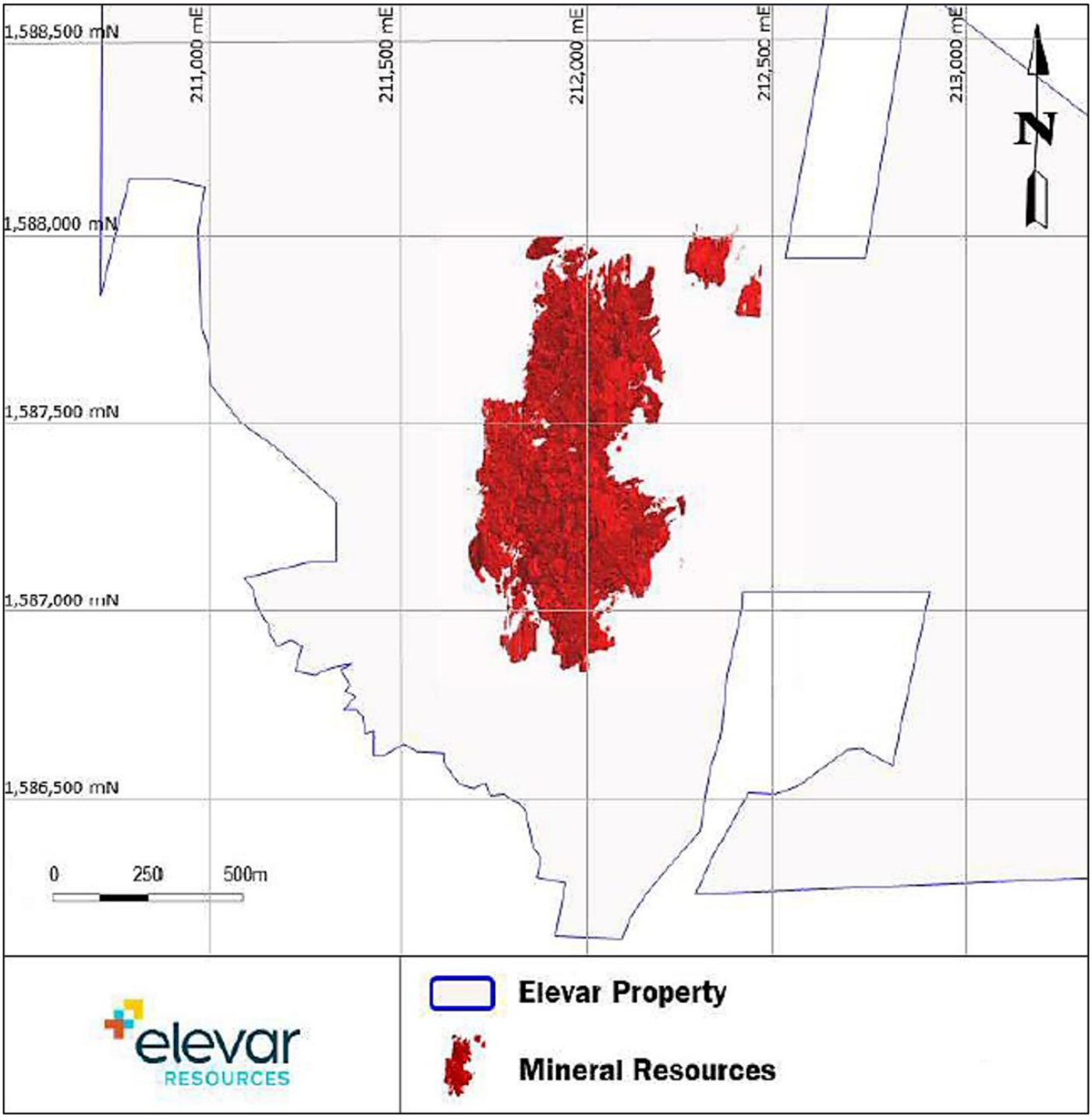
Figure 3-1: Project Location Map



Source: Bluestone, 2021.

The location of the mineral resources relative to the property boundary is shown in Figure 3-2.

Figure 3-2: Location of Mineral Resources Relative to Property Boundary



Source: Bluestone, 2022.

3.2 Property Description and Tenure

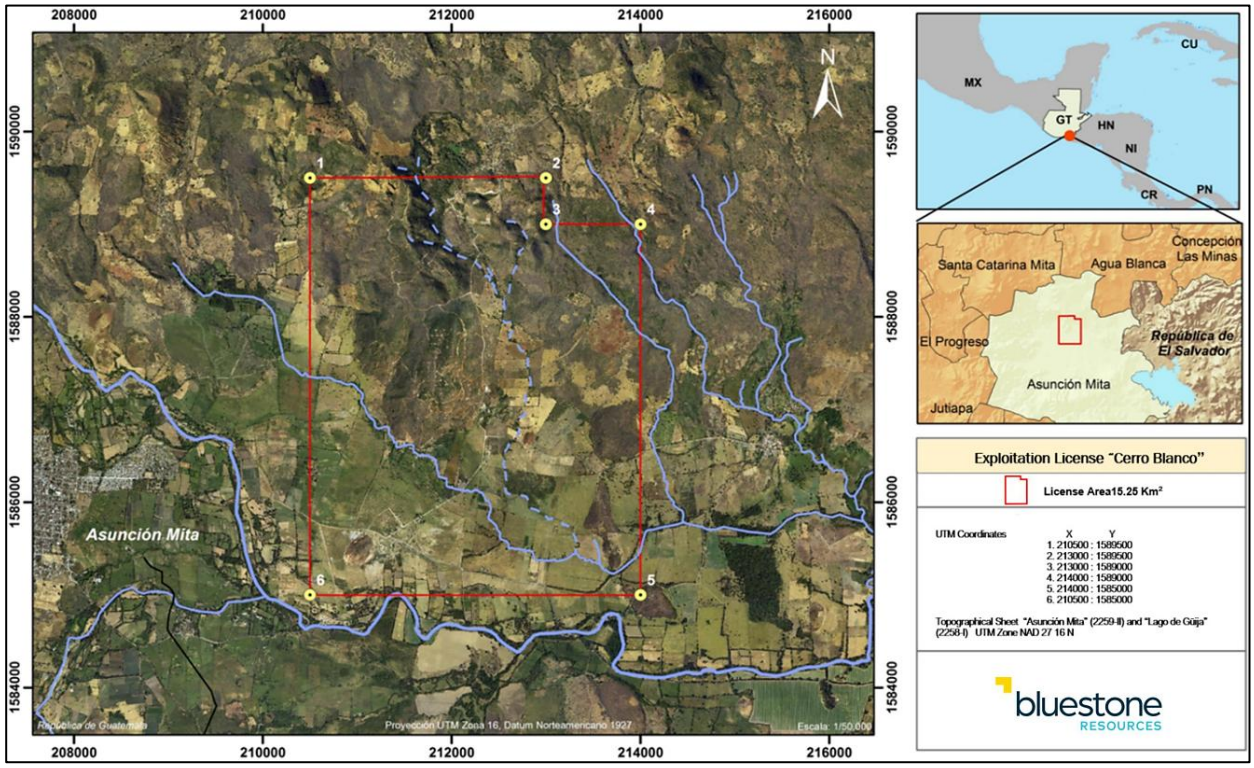
The coordinates of the 15.25 km^2 exploitation license are recorded in Decree DIC-CM-158-05 and are shown in Figure 3-3. The perimeter of the area is described as having the UTM X and Y coordinates referenced to NAD27, Zone 16N shown in Figure 3-3.

Table 3-1: Coordinates of Exploitation License "Era Dorada"

UTM East	UTM North
210500	1589500
213000	1589500
213000	1589000
214000	1589000
214000	1585000
210500	1585000

Source: Bluestone, 2022.

Figure 3-3: Era Dorada Exploitation License Coordinates



Source: Bluestone, 2019.

The company holds a mining exploitation license valid for a 25-year term expiring in September 2032, which may be renewed for an additional period equal to the original duration. The license, covering a concession area of 15 km², grants full rights to extract and commercialize minerals and is distinct from an exploration license, therefore, no conversion into another type of mining right is required, as the process has already been completed. During the current term, there are no obligations for annual payments or periodic updates, and the license remains valid and in good standing. To maintain compliance, the company must adhere to all obligations established under applicable regulations, including the Mining Law. In addition, Aura holds economic ownership rights for a period of 100 years, as established in a signed contract.

3.3 Royalties

The Project is subject to two royalties, both of which have been included in the economic analysis and cash flow model. Table 3-2 outlines the assumed royalty terms.

Table 3-2: Royalty Assumptions

Parameter	Unit	Value
Guatemalan Government Royalty	% NSR	1.00
Third-Party Royalty	% NSR	1.05*

Note: *1.05% royalty has been grossed up to account for country withholding tax. Source: Bluestone, 2021.

3.4 Environmental

The Project is following Guatemala environmental laws and regulations and has all necessary permits to proceed with developing the underground mine and construction of the process facilities, subject to future operations adhering to the conditions of the existing permits.

However, the Project design has changed since 2007 and requires permit amendments. Additionally, new baseline studies and permits are necessary for infrastructure components such as power lines.

The current permits and permit amendments are presented in Section 17.

There are currently no known environmental liabilities related to the Era Dorado project.

3.5 Discussion

There are no known significant factors and risks that may affect access, title, or the right or ability to perform work on the property.

4 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

4.1 Access

Current road access to the site is via the Pan-American Highway (Highway CA1) through the town of Asunción Mita. Existing infrastructure is in place to provide year-round access to the site. The topography is relatively flat, with rolling hills.

The Era Dorado project is situated in the municipality of Asunción Mita, in the Jutiapa Department of southeast Guatemala, about 160 km by road southeast of Guatemala City. This corresponds to roughly a 2.5-hour drive from the capital via the Pan-American Highway (CA-1).

The project site itself lies approximately 6–7 km west of the Pan-American Highway (CA-1) and about 5–7 km from the town of Asunción Mita.

The concession area is also near the Guatemala–El Salvador border, roughly 9 km west of the border.

Primary access to the project area is via the Pan-American Highway (CA-1). This major regional highway is the primary route connecting the project area with Guatemala City to the northwest and other local towns to the southeast. Use of CA-1 through Asunción Mita provides all-season access to the region.

Secondary access road to the site is from CA-1 near Asunción Mita, a project access road of approximately 5–6 km reaches the Era Dorado site. This road includes crossings such as planned/constructed bridges (e.g., over the Rio Grande de Mita) as part of infrastructure works tied to the project.

Year-round road access is possible via existing roads and infrastructure in the area provide year-round vehicular access from the highway to Asunción Mita and onward to the project site, regardless of seasonal weather.

As part of project development, additional works such as upgraded access road and bridge improvements have been planned or constructed to facilitate transport of materials, equipment, and personnel.

Guatemala has 400 km of coastline, claims territorial waters extending 22 km from its shore, and maintains an exclusive economic zone reaching 370 km offshore. Hurricanes and tropical storms sometimes affect coastal regions.

The five main ports in Guatemala, in the event of the necessity for shipping materails and consentrate, and their main activities are listed below:

- Atlantic ports
  - Puerto Santo Tomás de Castilla (containers)
  - Puerto Barrios

- Pacific ports
  - Puerto San José (liquids)
  - Puerto Quetzal (multi-use)
  - Puerto Champerico (fishing)

Puerto Santo Tomás de Castilla is the most important port on the Atlantic coast of Guatemala. This cargo terminal can handle a variety of cargo (e.g., containers and roll-on, roll-off (RoRo)), as well as general and liquid bulk cargo, passenger ships, vehicle carriers, and barges. The port facilities are approximately 290 km northeast of Guatemala City. The total distance from Santo Tomás de Castilla to the Project site is approximately 440 km.

Puerto Quetzal, which is the most important port on the Pacific Coast, has the most modern installations. It is mainly a dry bulk cargo terminal; however, it also handles containers, RoRo, general bulk cargo, and liquid bulk cargo. The port facilities are about 100 km South of Guatemala City. The distance from Puerto Quetzal to the Project site using the coastal highway is approximately 310 km. Puerto Quetzal is 2,050 nautical miles from Los Angeles.

These two ports handle nearly 80% of the sea traffic to Guatemala. Guatemala's Empresa Nacional Portuaria is a state-owned corporation of the Guatemalan port facilities.

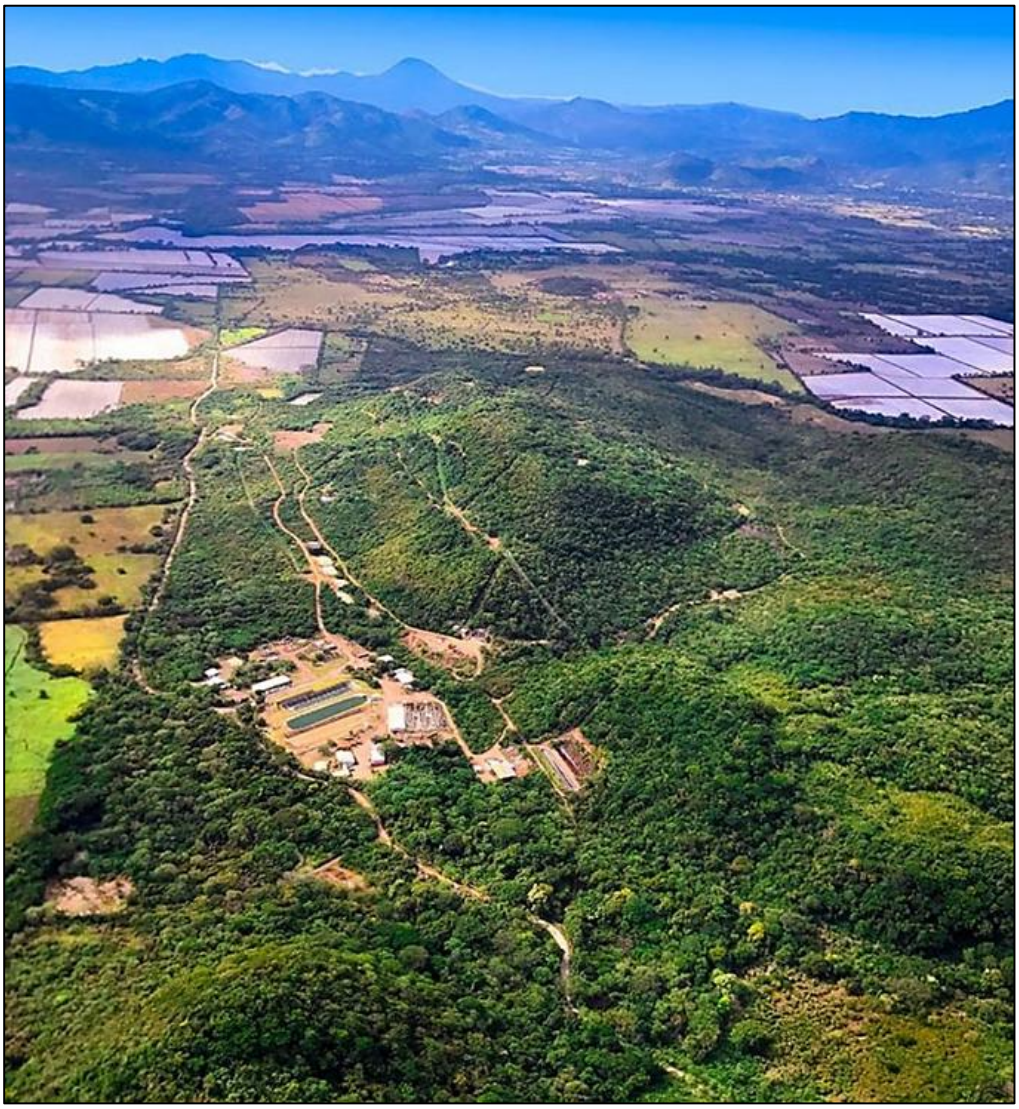
4.2 Climate

The climate and vegetation at the Project site are typical of a tropical dry forest environment. The wet season is typically from May to October. The average annual rainfall is 1,350 mm. Daily highs reach 41°C, and lows reach 10°C. The average annual pan evaporation rate is 2,530 mm, with an annual average humidity of 62%. Classified as Zona Oriental, the principal characteristics of the region are a deficiency of rain for much of the year with high ambient daytime temperatures.

4.3 Physiography

The Project is located on a hill with two peaks. The surrounding areas are relatively flat with minimal undulation. A photo showing the typical landscape around the mine property is included in Figure 4-1.

Figure 4-1: Typical Landscape in the Project Area, Looking South



Source: Bluestone, 2022.

Most of the vegetation in the Project area loses its foliage because of a lack of precipitation to support growth during the winter months of November through April.

The Project occurs within a south-southwest trending ridge that extends from higher ground to the north, outward into the basin and floodplain deposits of the Rio Ostua. The elevation of the upper part of the ridge is in excess of 600 masl. The elevation of the basin and floodplain deposits is about 460 to 490 masl.

The west side of the ridge is flanked by a south-southeast-trending perennial drainage called Rio Tancushapa. The east side of the ridge is flanked by a seasonal drainage called Quebrada El Tempisque, which also trends to the south-southeast. These drainages join to the south-southeast of the Project area and flow into the Rio Ostua about 4 km down gradient.

The regional area is generally hilly to mountainous, with broad flood plains formed by some of the larger streams and rivers. Three dormant volcanoes are within sight of the Project area: Ixtepeque to the north, Suchitan to the northwest, and Las Viboras to the southwest.

4.4 Flora and Fauna

The area around Asunción Mita, Guatemala, near the Era Dorado gold-silver project, lies within the tropical dry forest (subtropical dry forest) of eastern Guatemala, characterized by seasonal rainfall and prolonged dry periods. Vegetation consists mainly of deciduous trees, thorny shrubs, grasses, and secondary growth adapted to drought, with common species including ceiba, cedar, conacaste, guayacán, morro, zapotón, madre cacao, and various fruit and hardwood trees, alongside agricultural crops such as mango and varieties of mellon. Environmental surveys in the project area have identified on > 80 plant species across multiple families. Fauna reflects a dry-forest and agro-ecosystem setting, dominated by birds, along with reptiles (including iguanas and snakes), small mammals such as raccoons and rabbits, amphibians, and diverse insects. Streams and rivers draining the area form part of the Ostúa - Lake Güija - Lempa watershed, supporting fish and aquatic invertebrates and linking the local ecosystem to downstream habitats in both Guatemala and El Salvador. Overall, biodiversity is moderate but ecologically important, typical of Central America's dry corridor and sensitive to land-use and water-quality changes.

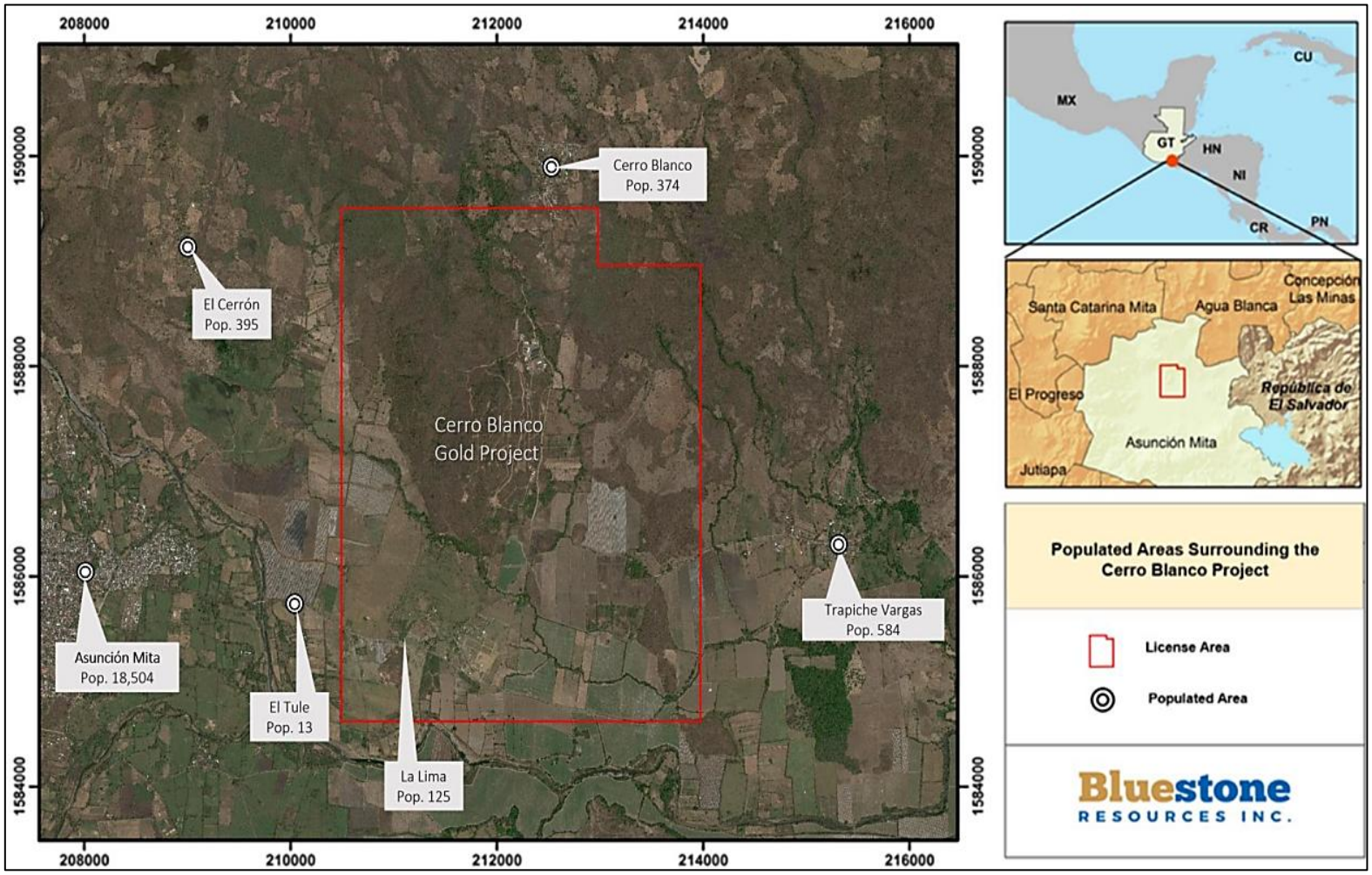
4.5 Local Resources and Infrastructure

The Project is situated in proximity to a number of communities, the largest one being Asunción Mita, with a population of approximately 18,500 people.

The local mine workforce is expected to live in the surrounding communities and provide their own transportation to and from the mine site due to the proximity of the population centers relative to the Project site (Figure 4-2). Employees from distant areas further than Jutiapa and expatriate employees will be housed in the on-site camp.

La Barranca power substation is located south of Asunción Mita, approximately ten kilometers to the west of the Project. The substation has a capacity to supply up to 20 MW of power.

Figure 4-2: Population Centers near the Project Area



Source: Bluestone, 2022.

5 HISTORY

5.1 Regional History

There is no evidence of exploration activity on the Era Dorada property (formerly “Cerro Blanco”) before 1997. Mar-West Resources Ltd. (Mar-West), a Canadian exploration company, had been working in adjacent Honduras since 1995 and expanded their gold prospecting activities into southern Guatemala in 1997. The Cerro Blanco property was identified by Mar-West by sampling densely silicified boulders, in some cases cut by chalcedonic veinlets, during an initial reconnaissance evaluation of an area known for active hot springs. Traverses over the hill at Cerro Blanco yielded surface rock assays of 1 to 3 g/t Au. An exploration concession was subsequently applied for and granted in late 1997. Mar-West drilled nine reverse circulation (RC) holes from April to June 1998, which tested near-surface potential to shallow depths of 100 to 150 m. At least seven holes contained one or more intercepts of 5 to 15 m grading 1 to 5 g/t Au, with the occasional 10 to 20 g/t Au interval, and were sufficient to justify continued exploration on the property.

In October 1998, Mar-West’s holdings in Honduras and Guatemala were purchased by Glamis Gold Ltd. (Glamis) primarily to acquire the San Martin deposit in Honduras. Mar-West geologists continued to manage the Cerro Blanco exploration program through March 1999. The sinter area was soil sampled and trenched, and drilling was advanced to hole 19 when geophysical orientation surveys were undertaken. A further 331 Drill holes were completed up until 2006.

Goldcorp became the sole proprietor of the Cerro Blanco Gold Project through the purchase of Glamis in November 2006. Goldcorp undertook a comprehensive exploration program from 2006 to 2012, including additional surface exploration, over 3.4 km of underground development, and 43,016 m of surface and underground drilling. Exploration activities at the Cerro Blanco property by Goldcorp included the following:

- Surface soil geochemistry
- Surface rock geochemistry
- Surface geological mapping
- Underground chip sampling
- Surface and underground diamond drilling.

Several unpublished feasibility studies were completed by Goldcorp from 2011 to 2014. Kappes, Cassidy & Associates (KCA) and Golder Associates (Golder) completed an FS for the Project in May 2012. After this initial FS, Goldcorp issued a new geological model and requested KCA and Golder to update the FS in 2013 using a revised mine design, mine development, mine operation, and capital costs. In 2014, an internally updated FS was produced with optimized mine stope parameters and the mine schedule and costing information that was updated by Maptek.

On January 4, 2017, Bluestone entered into an agreement with Goldcorp Inc. (Goldcorp) to acquire 100% of Minerales Entre Mares de Guatemala, S.A. (Entre Mares, or EM), which was Goldcorp's indirect wholly owned Guatemalan

subsidiary which holds a 100% interest in Cerro Blanco. On successful closure of the deal, Entre Mares became a wholly owned subsidiary of Bluestone, a Canadian company headquartered in Vancouver, British Columbia.

In January 2025, Aura completed the acquisition of the Cerro Blanco Project and Mita Geothermal Project, located in Jutiapa, Guatemala, near the town of Asunción Mita and the border with El Salvador. The Cerro Blanco Project is 100% beneficially owned by Aura.

5.2 Data Validation History

Historical core logging, sampling, and QA/QC procedures were reviewed by Golder in 2014.

Core samples were collected from quarter-sawn NQ core, and selected drill hole collars were surveyed using a GPS. Assayed gold and silver grades were found to be consistent with those reported by Goldcorp. Golder was satisfied that the drill hole data was collected in a manner consistent with industry best practice standards.

As part of the core logging data verification, Golder compared a selection of core logs against half-core stored at the Project site. Five half-core Drill holes were reviewed from the North and South deposits. The Excel files were reviewed first, and Drill holes were selected that represented the typical mineralization style for each deposit. In addition, ten verification samples were taken from these Drill holes. Each verification sample was a half-core sample sawed in half again, with the quarter sample sent for analysis and the other quarter returned to the core racks. Table 5-1 summarizes the samples selected for core logging review and verification sampling.

Table 5-1: Verification Samples

Drill hole ID	Duplicate Sample No.	Original Sample No	From (m)	To (m)	Deposit	Metal Analyzed	Rock Type
CB-152	205873	82225	128	129	North	Au, Ag	Lapilli Tuff
CB-152	205874	82226	129	130	North	Au, Ag	Lapilli Tuff
CB-200	205884	407101	156	157	South	Au, Ag	Quartz Tuff
CB-200	205885	407102	157	158	South	Au, Ag	Quartz Tuff
CB-241	205891	404849	111.4	112.6	South	Au, Ag	Conglomerate
CB-241	205892	404850	112.6	113.5	South	Au, Ag	Fault
CB-254	205895	414397	100.5	102	South	Au, Ag	Volcaniclastic sediments
CB-254	205896	414398	102	103.5	South	Au, Ag	Volcaniclastic sediments
CB-10-15	205871	435941	135	136.23	North	Au, Ag	Lapilli Tuff
CB-10-15	205872	435943	136.23	137.46	North	Au, Ag	Lapilli Tuff

Source: Goldcorp, 2014.

Samples were sawed and bagged under Golder's supervision and were transported off-site via helicopter and plane to Canada and then by ground transportation to ALS Chemex laboratories in Sudbury for sample preparation and analysis.

A comparison of the Excel files against the drill core indicated an excellent match between the core logs and the retained core.

Table 5-2 is a list of the drill hole collar surveys completed by Golder.

Table 5-2: Drill hole Collar Survey (NAD 27 Zone 16N)

Drill hole ID	Golder		Cerro Blanco	
	Easting	Northing	Easting	Northing
C 10 08	212015.1	1587867	212009	1587748
C 11 12	211906.8	1587714	211904	1587605
C 11 15	211969.7	1587769	211966	1587655
C 11 18	211866.4	1587405	211873.2	1587297
C 11 21	211901.6	1587414	211898.9	1587307
C 151	212025.1	1587821	212020.8	1587707
C 247	211985.5	1587315	211978.8	1587202

Eight drill sites were visited, with multiple Drill holes located at some sites. Casings had been removed for most Drill holes. The data collected was a mixture of pre-Goldcorp Drill holes (2006 or earlier) and drilling completed by Goldcorp during 2010 and 2011. All Drill holes from the surface were grouted to prevent water flow into the underground workings.

Approximately 5% of the Drill holes (20 holes) were subjected to data verification checks by Golder. The 20 selected holes, summarized in Table 5-3, included a variety of historical data as well as some of the more recent holes. The data verification checks consisted of the following:

- Comparison of final assays to the original laboratory certificates.
- Analysis of external laboratory duplicate assays by generating XY scatter plots.
- Review of downhole survey measurements to identify anomalous changes to hole orientation.

Table 5-3: Drill holes Selected for Data Verification

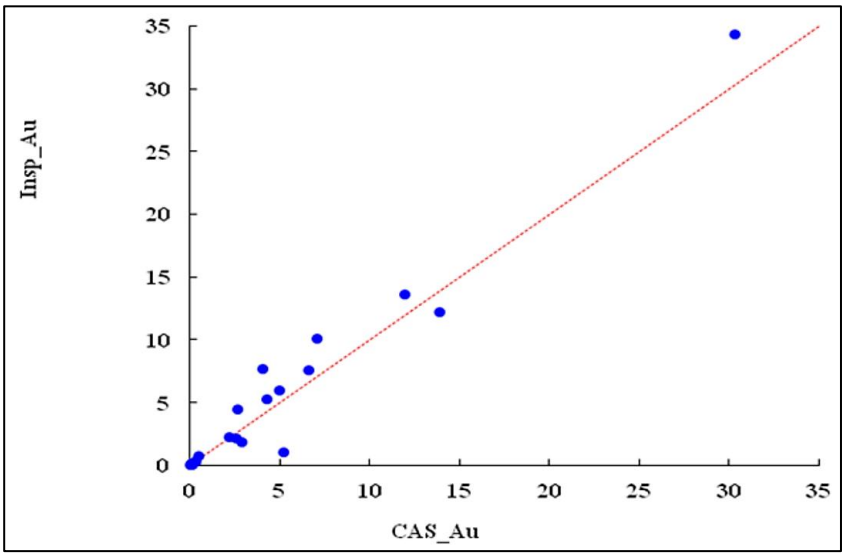
Drill hole ID	
CB-012	CB-200
CB-016	CB-227
CB-063	CB-244
CB-078	CB-247
CB-095	CB-305
CB-10-02	CB-309
CB-120	CB-314
CB-142	CB-345
CB-146	CB-357
CB-151	CB-362

Source: Goldcorp, 2014.

For the 20 holes reviewed, the comparison of final assays to the original assay certificates did not identify any material differences in assay values.

External laboratory duplicate assays were reviewed to assess the reliability of the primary assay laboratory. XY scatter plots were generated for each of the 20 holes. With the exception of a few outliers, the majority of the data compared well. Figure 5-1 illustrates an example of the XY scatter plots used to compare assay results.

Figure 5-1: Example of XY Scatter Plot for Hole CB34



Source: Goldcorp, 2014.

5.3 Historic Technical Reporting

Several technical reports have been published on Era Dorada since 2017 in six technical reports, as follows:

- Preliminary Economic Assessment (March 20, 2017)
- Preliminary Economic Assessment Update (February 28, 2021)
- Preliminary Economic Assessment Update (June 30, 2021)
- Initial Assessment and Technical Report (December 31, 2024)

Four of the technical reports and resource estimates were for an underground mining scenario while two resource estimates were for the open pit scenario. All five NI43-101 technical reports are filed on the System for Electronic Document Analysis and Retrieval (SEDAR+) whilst the December 31, 2024 SK-1300 Technical Report Summary is filed on EDGAR.

6 GEOLOGICAL SETTING, MINERALIZATION AND DEPOSIT

6.1 Introduction

The geology of Guatemala comprises rocks that are divided into two tectonic terrains due to the collision between the North American, Caribbean, and Cocos tectonic plates during the Upper Cretaceous, 70 to 90 million years ago. The Maya Block to the north is characterized by igneous and metamorphic basement rocks overlain by late Palaeozoic metasediments. Mesozoic red beds, evaporites, and marine limestones overlie these rocks, and a karst landscape formed in the thick limestone units across the north of the country. By contrast, southern Guatemala, south of the Motagua Valley, belongs to the Chortis Block, representing the northern part of the Caribbean Plate. This region forms an active volcanic arc termed the Central American Volcanic Arc (Figure 6-1), which continues from the Guatemala-Mexico border along the Pacific side of Central America into central Costa Rica, with most of the major eruptive events having occurred in the Tertiary and Quaternary.

Figure 6-1: Location of Era Dorada and other Deposits in the Central American Volcanic



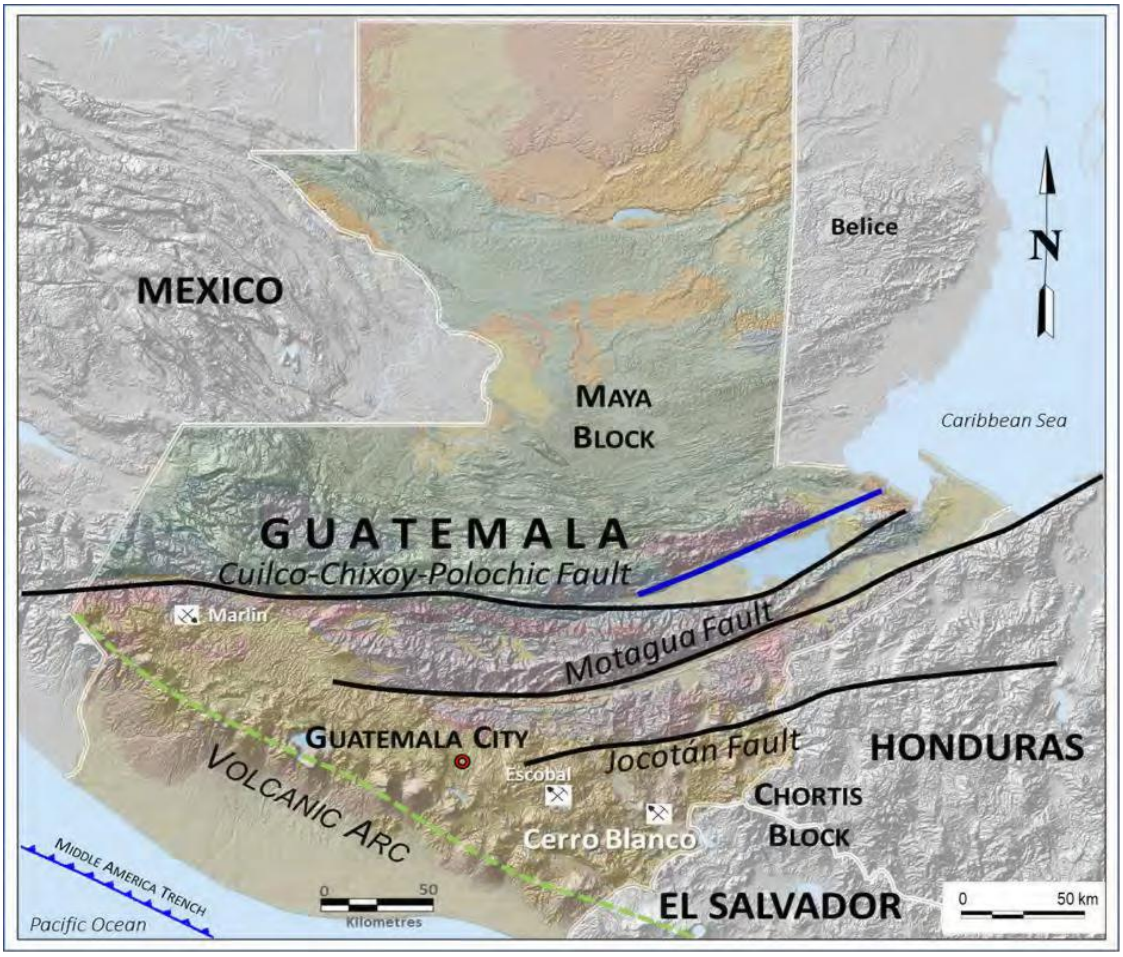
Source: Bluestone, 2020.

6.2 Regional Geology of Southern Guatemala

Southern Guatemala, El Salvador, Honduras, and Nicaragua are located within the Chortis continental crustal block. The tectonic event that sutured the Chortis block to the North American craton took place between 66 and 70 million years ago along the east-west-striking Polochic-Montagua fault system that crosses southern Guatemala (Figure 6-2). Three regional east-west trending, left-lateral transform faults form the plate collision boundary, defined by the Polochic, Motagua, and Jocotan fault systems from north to south. Nearer the Cerro Blanco deposit, other major regional structures that strike north-northeast, such as the Jalpatagua and Ipala faults, are important local structures.

A large group of granitic stocks and batholiths intruded the suture zone south of the Polochic-Montagua fault with ages of 35 to 85 million years. These broadly brackets, both temporally and spatially, the collision event (Donnelly et al., 1990).

Figure 6-2: Regional Structural Map of Guatemala



Source: Bluestone, 2021.

The Jocotan Fault is generally considered the southernmost major suture-related fault. It is an east-west fault with considerable Late Cretaceous dip-slip movement (south side down), but it had little or no Tertiary transcurrent movement. Era Dorada is located about 50 km south of the Jocotan Fault.

The ancestral Middle America Trench developed at this time. The Pacific Oceanic plate is subducted beneath Central America and is the principal driving force for volcanic and intrusive igneous activity throughout Central America along this boundary trench. The earliest documented volcanic outpouring on the Chortis block was the Paleocene (about 55 to 65 million years ago) (Pindell and Barrett, 1990).

In Costa Rica and Panama, a series of west-northwest-trending (arc-parallel) back-arc basins developed. These accumulated tuffaceous sediments continuously from the Eocene (about 55 million years) to the present (Donnelly et al., 1990). The principal periods of Andean-style calc-alkaline volcanism in the Chortis block include the Paleocene-Eocene (relatively minor), Oligocene (major), and Miocene-Pliocene (the biggest) (Pindell and Barrett, 1990).

The Polochic-Montagua suture was reactivated as a sinistral (left-lateral) transform fault that displaced the Chortis block 130 km eastward with respect to the North American craton. Movement took place from 6 to 10 million years ago (Deaton and Burkart, 1984). An associated extension was accommodated by a series of north-south grabens across southern Guatemala and western Honduras. Back-arc rift basins developed adjacent to northwest-striking normal faults all along western Central America. The Nicaraguan Rift began to form about 7 million years ago and continues to subside today. Bimodal, rhyolite-basalt volcanism began during this event and, by 7 million years ago, was widespread throughout the western half of the Chortis block.

A large number of Central American gold deposits, including Marlin and Era Dorada, occur within a narrow belt parallel to the western Central American coast from southern Guatemala through to Panama. The geology of Guatemala comprises rocks that are divided into two tectonic terrains due to the collision between the North American, Caribbean, and Cocos tectonic plates during the Upper Cretaceous, 70 to 90 million years ago. The Maya Block to the north is characterized by igneous and metamorphic basement rocks overlain by late Palaeozoic metasediments. Mesozoic red beds, evaporites, and marine limestones overlie these rocks, and a karst landscape formed in the thick limestone units across the north of the country. By contrast, southern Guatemala, south of the Motagua Valley, belongs to the Chortis Block, representing the northern part of the Caribbean Plate. This region forms an active volcanic arc termed the Central American Volcanic Arc (Figure 6-1), which continues from the Guatemala-Mexico border along the Pacific side of Central America into central Costa Rica, with most of the major eruptive events having occurred in the Tertiary and Quaternary.

This metallogenic belt follows the volcanic arc, and precious metal deposits are clearly related in space and time to Miocene-Pliocene extensional tectonics and associated bimodal basalt-rhyolite volcanism. Published age dates cluster between four and eight million years. Argon-argon dating (40Ar-39Ar) of vein adularia from Era Dorada returned a date of 4.93 ± 0.47 Ma.

6.3 Local Geology

The Project deposit is a classic hot springs-related, low-sulfidation quartz-chalcedony-adularia-calcite vein system. It was localized along a structural corridor created during the late Miocene- Pliocene tectonic extension within the active

Central American Volcanic Arc. Deep penetrating faults and local bimodal igneous activity drove the Cerro Blanco hydrothermal system and the formation of the gold deposit.

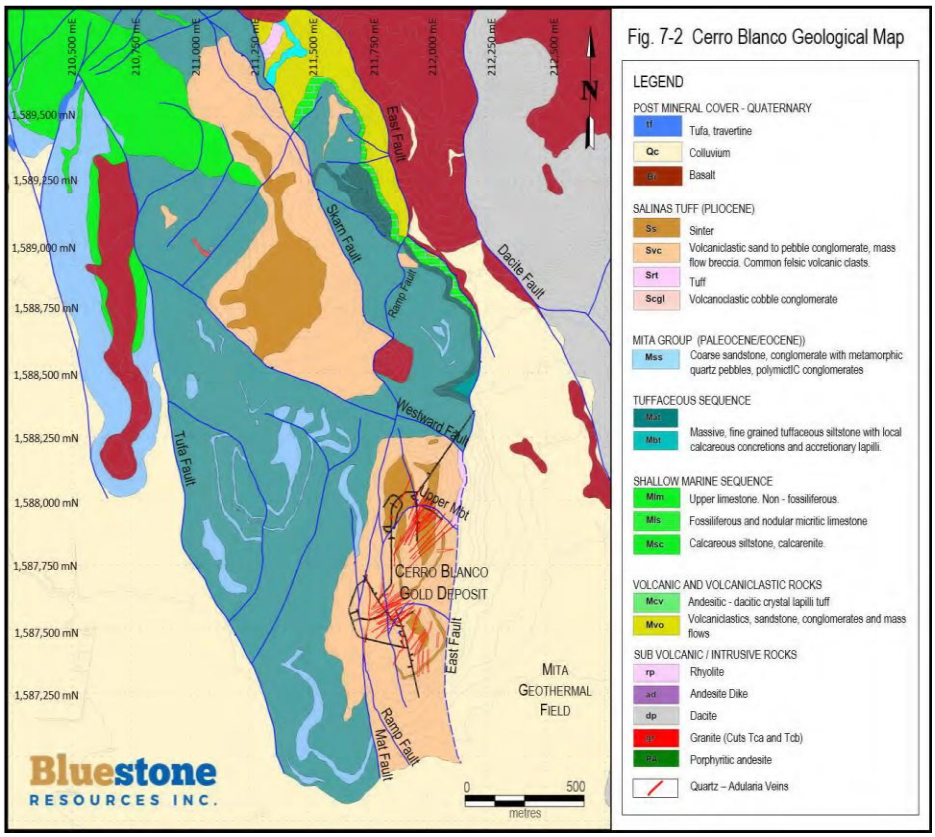
The Project lies within the volcanic province, with the principal rock units being Tertiary volcanic, volcaniclastics, and sediments, including ignimbrites, siltstone, limestones, and conglomerates, that are intruded by andesitic and rhyolitic dykes. Recent basalt lava flows form the youngest rocks in the area in addition to locally derived volcanic sediments.

The gold- and silver-bearing veins and upper unit of silicified sediments (Salinas unit) occupy a north-trending graben bounded by a fault (termed the East Fault), representing a major structural feature that separates the main Era Dorada gold deposit from the Mita geothermal field immediately to the east.

To the north, the graben is concealed beneath Quaternary basalt flows, and to the south, it is concealed by recent alluvium. Rhyolite/dacite domes underlie the extreme northeast portion of the district. Active hot springs occur immediately south of Cerro Blanco hill.

Figure 6-3 shows the simplified geological map for Cerro Blanco.

Figure 6-3: Geological Map of Era Dorada

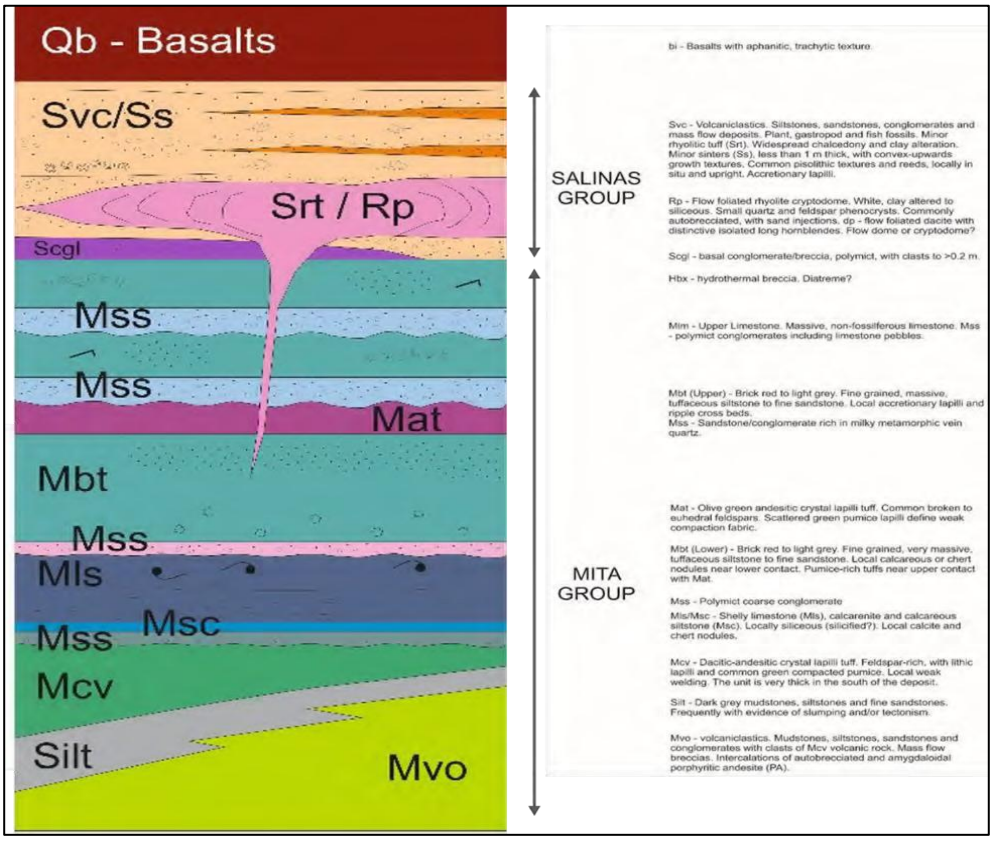


Source: Pratt and Gordon, 2019.

6.3.1 Lithology

The oldest rocks at Cerro Blanco Gold Project, intersected in deep Drill holes, belong to the Mita Group (Pliocene-Miocene). This group exhibits a great variety of volcanic and sedimentary rocks with important marker beds that are crucial for understanding complex structural geology. Thicknesses seem fairly constant, with little evidence of growth faulting or internal unconformities during their accumulationFigure 6-4Figure 6-4.

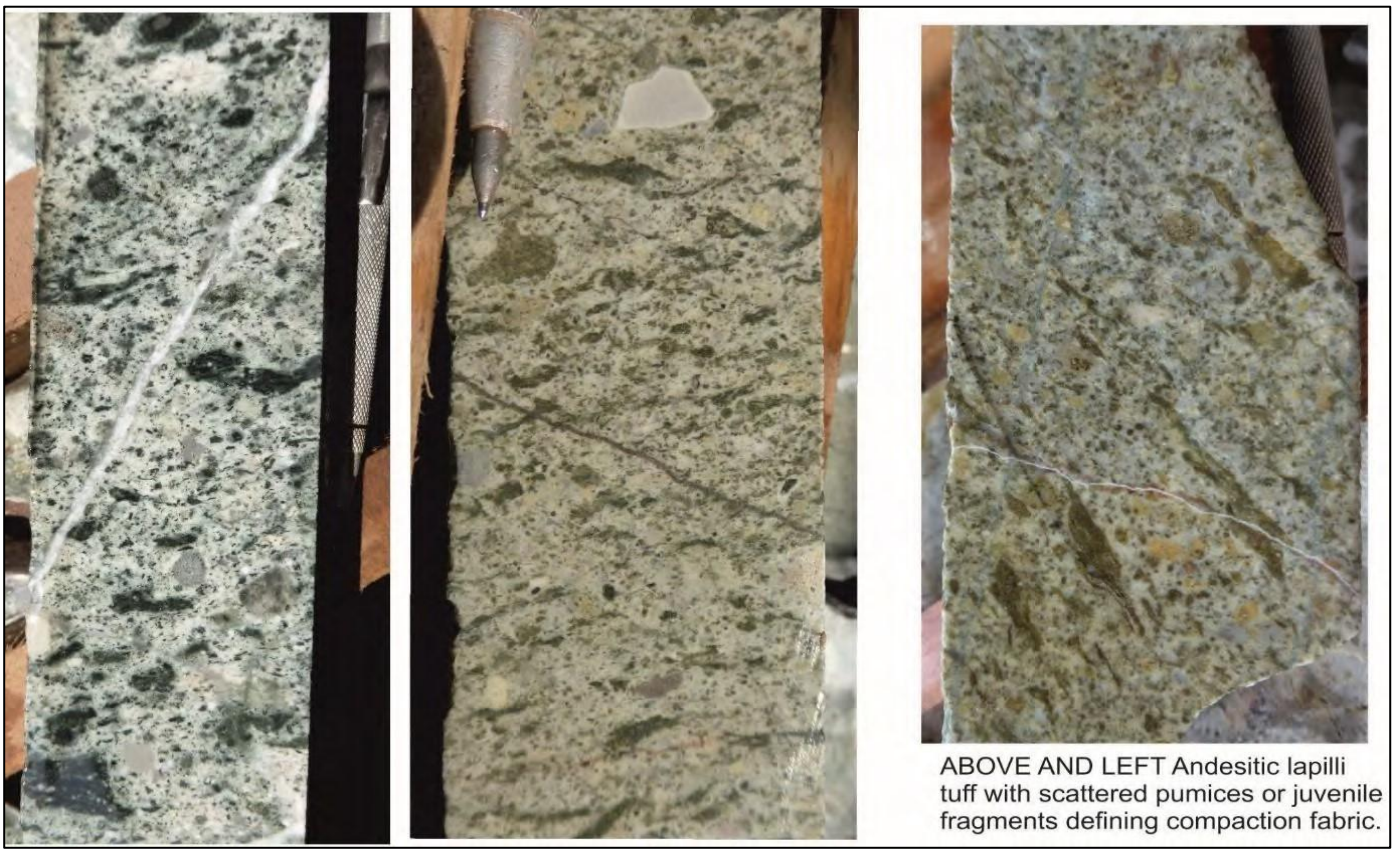
Figure 6-4: Lithostratigraphy and Lithology Codes at Era Dorada



Source: Pratt and Gordon, 2019.

The deeper parts of the Mita Group are dominated by volcaniclastic rocks (Mvo, mass flow deposits, conglomerates) with intercalated auto-brecciated and amygdaloidal porphyritic andesites (lithology code PA). There is a distinctive unit of dark grey siltstones and fine sandstones (Silt), frequently with syn- sedimentary disruption. The sequence is capped by a major unit of andesitic-dacitic tuff (Mcv) (Figure 6-5), which erupted in a single event. This is at least 50 m thick and rich in broken crystals and small pumice lapilli. It shows a weak compaction fabric or welding (refer to the photographs in Figure 6-5).

Figure 6-5: Examples of Andesitic Lapilli Tuff (Mcv)



Source: Pratt and Gordon, 2019.

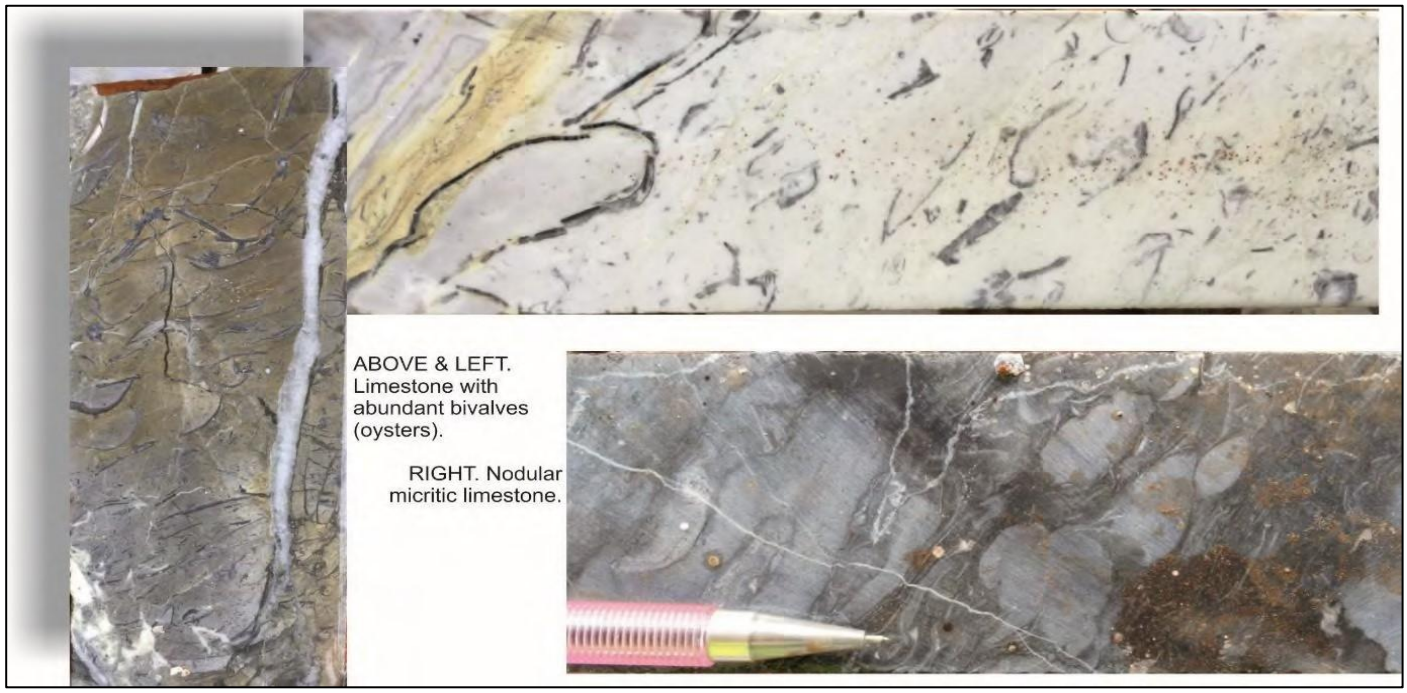
The tuff is overlain by sandstones (Mss), followed by a nodular micritic to shelly, oyster-rich limestone (Mls, Figure 6-6), which is the most distinctive rock at Era Dorada. This limestone sequence is about 20 m thick and includes calcarenites (Msc).

The limestone is overlain by a thick sequence of relatively massive, brick-red to light grey siltstone and fine sandstone (Mbt). This distinctive rock has local accretionary lapilli, horizons of flaser and ripple cross-bedded fine sandstone, and local calcareous concretions. The Mbt sequence is divided into lower and upper parts by an andesitic crystal tuff (Mat). It is also punctuated by intervals of clean, well-sorted, fine-grained conglomerate (Mss). These can be rich in metamorphic vein quartz pebbles and dark grey schist, indicating a metamorphic hinterland. In the north part of the property, there is a second major package of limestone (Mlm) (Figure 6-3), in turn overlain by further massive siltstones (Mbt).

The Mita Group is overlain by the Salinas Group (Svc). This is a complex sequence of interbedded plant-rich siltstones, mudstones, sandstones, conglomerates, mass flow deposits, phreatic breccias, and hot spring sinters. The Salinas unit, of probable Pliocene age, was previously considered to unconformably overlie the Mita unit, which was then assigned

to the Eocene-Oligocene. The presence of the unconformity is certainly suggested by the structural culmination defined by the Mita limestone. However, thin sinter horizons are observed interbedded with siltstone at the top of the Mita unit, a situation that requires that the Mita and Salinas are part of a single, uninterrupted succession. This interpretation implies that the Mita part of the succession was in place before the mineralization commenced, whereas the overlying Salinas part accumulated during the mineralization event (Sillitoe, 2018).

Figure 6-6: Examples of Limestones (Mls)



Source: Pratt and Gordon, 2019.

The syn-mineral Salinas unit is believed to have accumulated progressively in a low-relief graben characterized by a shallow groundwater table. The Salinas conglomerate was presumably derived by erosion of the flanking horst blocks as relief was created during the active faulting. The topographic inversion required to explain the current prominent position of the graben fill is ascribed to the silicic character of the Salinas unit and its consequent resistance to erosion.

Where the paleo-groundwater table intersected the paleosurface, siliceous sinter was precipitated—a situation that must have prevailed on several occasions for relatively protracted time intervals to produce the main sinter horizons. The presence of abundant reed casts in the sinter shows that its formation encroached on marshy ground (Figure 6-7). Where the paleo-groundwater table was several meters below the paleosurface, a conglomerate in its immediate vicinity was silicified, and the vadose zone above it was subjected to steam-heated alteration. The steam-heated alteration, containing cristobalite, kaolinite, and possible alunite (an advanced argillic assemblage), was the product of acidic solutions formed by the condensation of ascendant H2S-bearing steam into downward-percolating

groundwater. The overall result is an interlayered sequence of sinter, silicified conglomerate, and steam-heated alteration (Figure 6-7).

The Salinas Group is characterized in the mineralized area by widespread chalcedonic alteration, which can make identifications difficult, and elsewhere by strong clay alteration. In some places, rock fragments have concentric chalcedony coats (pisoliths), implying they accumulated in a hot spring pool. Silicified reed fragments are common and locally upright in their original growth position. Rare gastropods were observed.

Figure 6-7: Silicified Reed Fragments



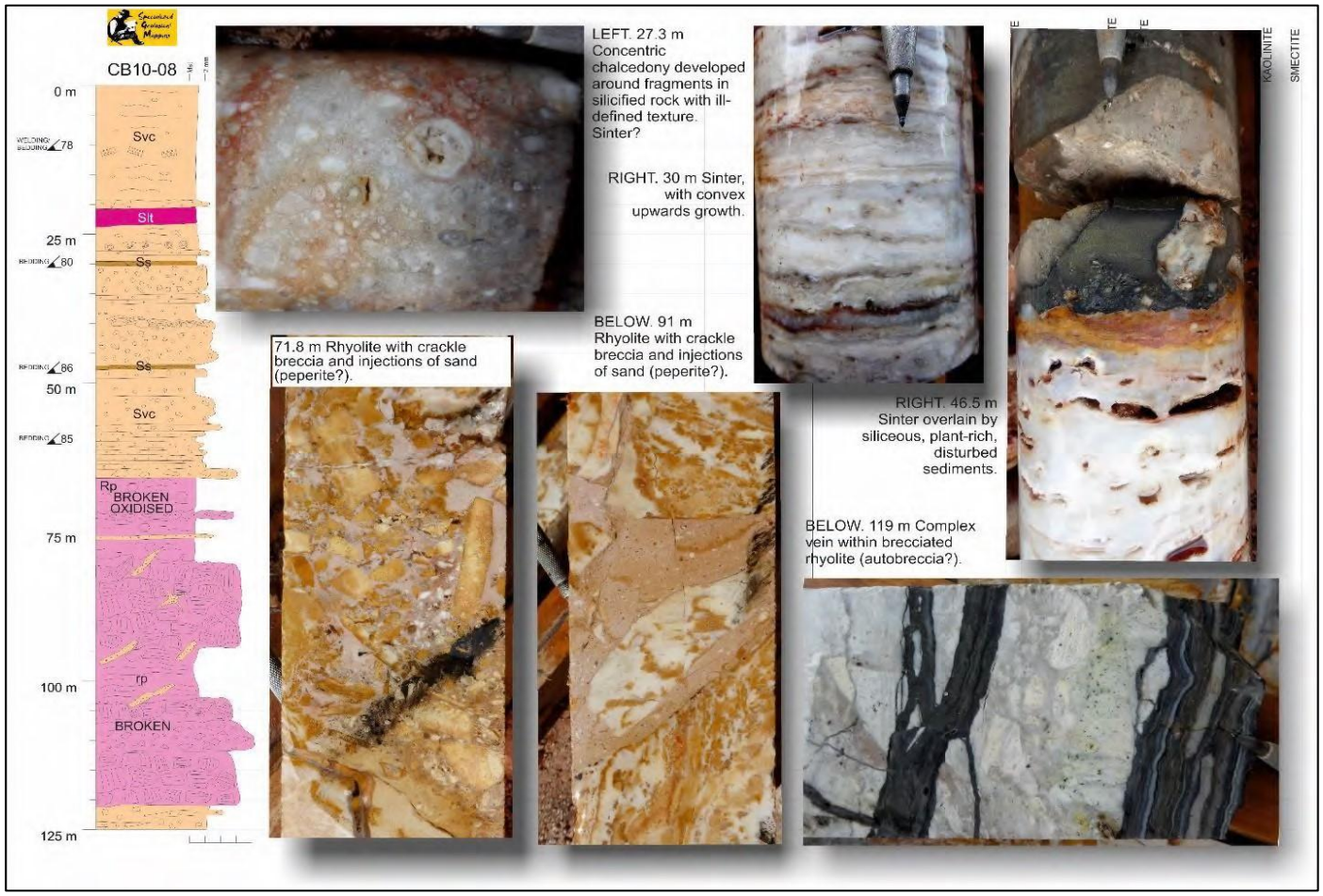
Source: Pratt and Gordon, 2019.

The sequence also includes rhyolitic tuffs and a rhyolite cryptodome / flow dome (Rp), both with bipyramidal, embayed quartz crystals. A dacite cryptodome or flow dome (dp) also crops out around the Era Dorada village and is observed in drill holes in the hanging wall of the East Fault (Figure 6-4). It has no quartz crystals but distinctive, isolated, long hornblende phenocrysts. Sediment dykes, common in geothermal districts, where they form the feeders to sand and mud volcanoes, are common in the Salinas Group.

A typical log of the Salinas Group, shown in the photographs in Figure 6-8, includes a body of rhyolite, possibly a cryptodome since probable properties were seen at the contacts.

The highest stratigraphic part of the Salinas Group, at least 60 m thick and above the sinters, is cut in the graben in the hanging wall of the East Fault. It comprises lacustrine siltstones and volcaniclastic sandstones. The rocks are plant-rich and contain rare fish fossils and brine shrimp/ostracods (e.g., drill hole CB332).

Figure 6-8: Example Drill Log from the Salinas Group



Source: Pratt and Gordon, 2019.

The Salinas Group includes common mass flow or hydrothermal breccias. Their geometry is frequently unclear; it is uncertain if they are dykes or aprons of phreatic (explosion) breccia ejected from hot springs. Some contain sinter clasts, confirming phreatic eruptions. Underground, the South Ramp is dominated by hydrothermal breccias (Hbx), with polymict clasts up to 0.5 m in diameter. This may be the north margin of a south-dipping diatreme. Successive cross-sections show it extending progressively deeper towards the south.

Quaternary basalts (bi), with a felted, trachytic texture, crop out in the north of the Era Dorada property and occur in the low graben on either side of the horst. They are clearly lava flows. Around the village of Cerro Blanco, they in-fill the paleo-topography formed by a large dacite flow dome. It is unclear if this topography is erosional or the original hummocky shape of the dacite flow. The basalts include flow-foliated and autobrecciated types.

The youngest rocks comprise alluvium, and in a few places, modern travertine and tufa occur at springs around the flanks of Cerro Blanco hill (Figure 6 9). The tufa cements colluvial blocks of the siliceous sinter (Salinas Group) are modern and should not be confused with sinter. They imply probable karst formation and dissolution of limestone.

Figure 6-9: Recent Travertine Exposure



Source: Pratt and Gordon, 2019.

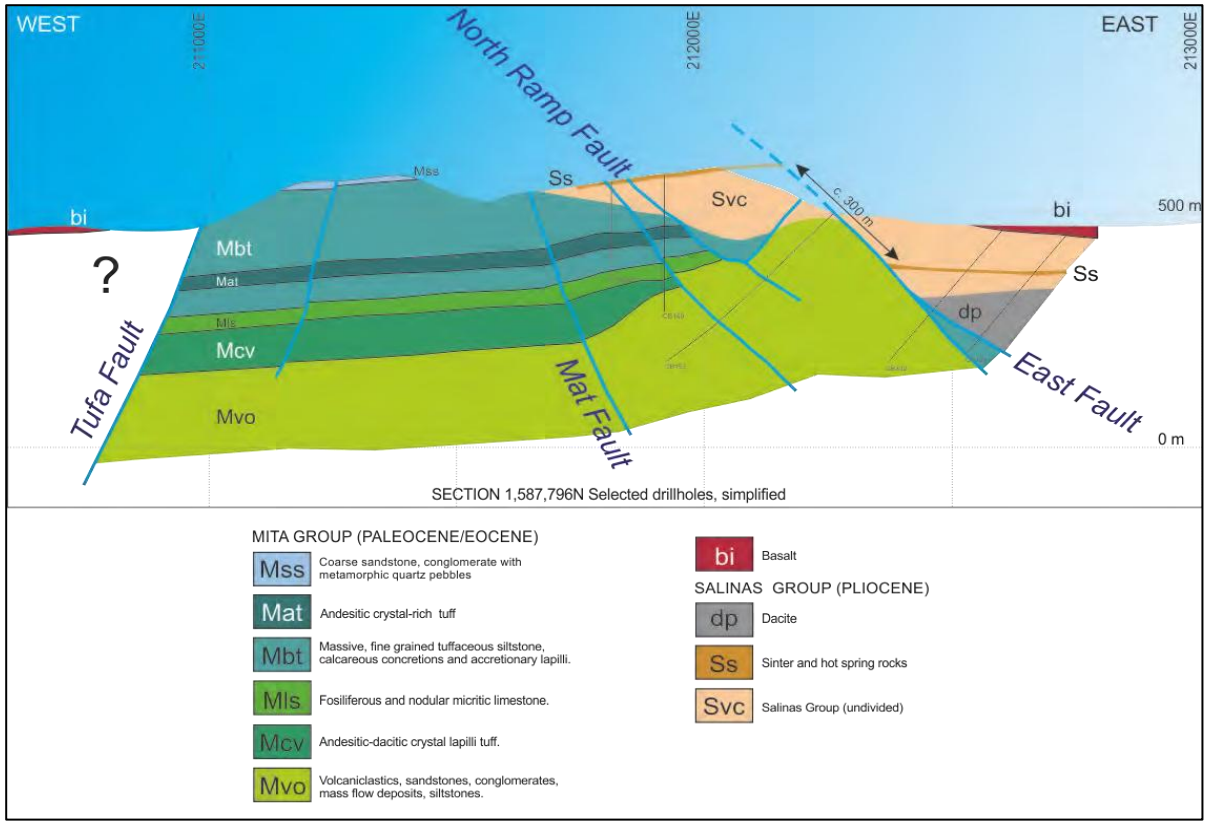
Discordant igneous intrusions are rare at Cerro Blanco, but a few thin rhyolite (Rp) and aphanitic andesite (ad) dykes are observed.

6.3.2 Structure

The gold mineralization at the Project is hosted within a broadly north-south-striking graben. The East Fault (Figure 6-10), also referred to as the “East Horst Fault” in previous studies, is cut by several and observed in the drill core as a broad zone of post-mineral cataclasite developed in Mita siltstone; however, the structure appears to control a linear rhyolite body, suggesting that it was also active during the mineralizing event. This fault may be listric and made up of several strands. Holes CB332 and CB329, in the section below, show narrow wedges of ‘exotic’ lithologies along the fault zone, including limestone (Mls) and conglomerate (Mss). The apparent displacement, shown by the offset of the sinter (Ss), is about 300 m.

The immediate footwall of the East Fault, which hosts the gold-bearing quartz veins, is structurally complex. A deep geothermal drill hole (MG-07) shows gold mineralization in the probable down-dip extension of the East Fault at 634-640 m downhole depth.

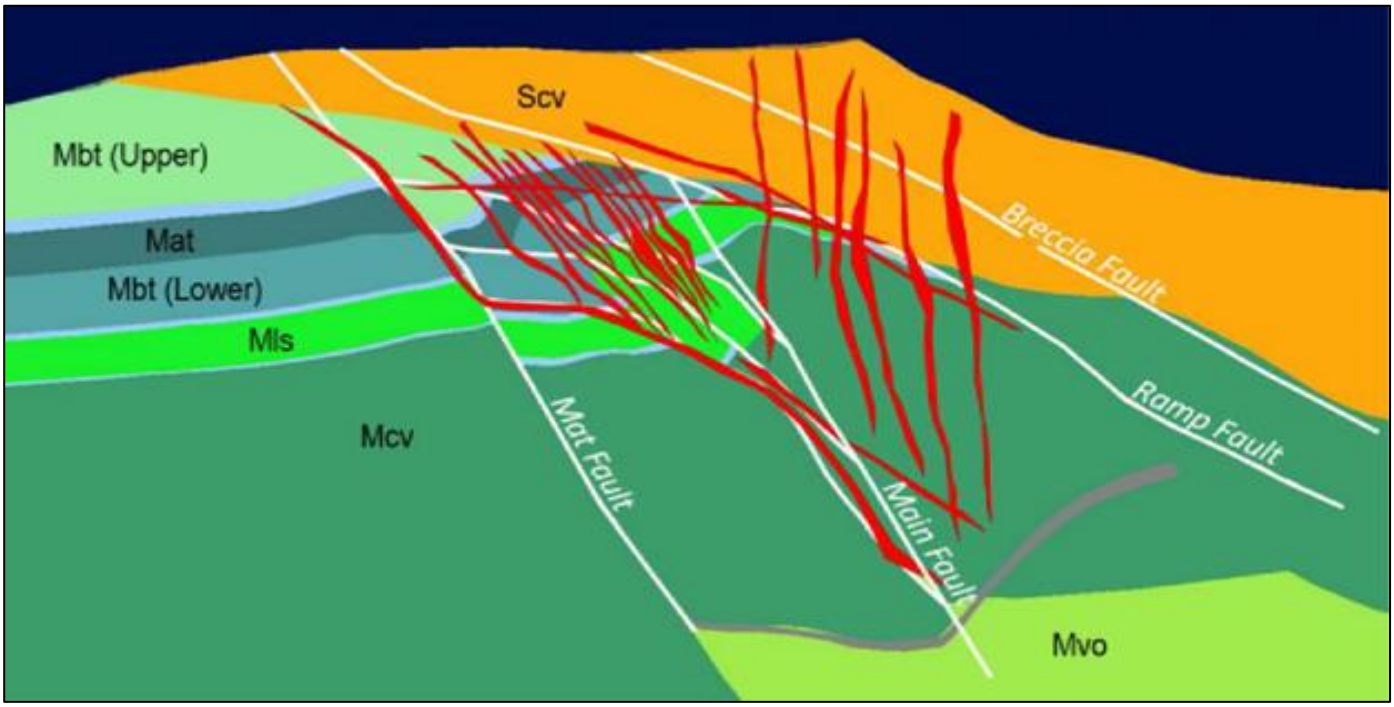
Figure 6-10: Simplified West-West Cross-Section Across Era Dorada



Note: Note: Many and some lithostratigraphic units and faults were omitted to conserve clarity. Source: Pratt and Gordon, 2019.

The Cerro Blanco property has a complex history of faulting. The structural control on mineralization is unusual for low-sulfidation epithermal vein deposits, which normally comprise a single, relatively continuous vein. At Era Dorada, there are sheeted vein swarms that resemble a duplex. Figure 6-11 indicates the typical complexity in an east-west section. Note that the thickness of the Mcv west of the Mat Fault and the Mvo to the east is an artifact of Leapfrog software and is overstated. Veins are shown in red, and faults in white.

Figure 6-11: East-west cross-section of the South zone, Era Dorada looking North



Source: Bluestone, 2021.

Simplistically, the structural history is comprised of the following:

1. Sedimentation of the Mita Group in a basinal to shelf environment, with periodic incursions of calc-alkaline volcanism (mostly waterlain andesitic tuffs and andesite flows, and their volcaniclastic equivalents). Some of the beds appear turbiditic (silt), implying moderate water depth. Some metamorphic clasts imply a metamorphic hinterland.
2. A compressive episode formed a series of broadly north-south-striking, west-verging folds cored by Mita Group rocks, in particular, the Mvo and Mcv. These folds were associated with west-verging reverse faults and resulted in local overturned limbs. There may have been a component of strike-slip, with the development of a positive flower structure at the restraining bend in a major north-south strike-slip fault. There is evidence that most of the gold-bearing veins developed at this stage. The controlling structures for the vein swarms are in the footwall of the East Fault and apparently steeper (e.g., the Main Fault, see Figure 6-10).

3. Major extensional faulting with downthrows to the east of up to several hundred meters. These include the Ramp and East faults (see above). These faults may have been active during deposition of the Salinas Group (Svc), possibly growth faults. Metamorphic clasts in the Salinas Group imply continued input from a metamorphic hinterland. The offset of Quaternary basalts implies that the faults may still be active (neotectonic). These faults have the greatest surface expression, reflected in the modern topography by the Cerro Blanco ridge and flanking low-relief alluvial plains.

Most of the gold-bearing veins are constrained between the Mat Fault in the west and the East Fault, and evidence suggests that most veins at this stage developed along early pre-mineral faults. The Mat fault is interpreted to be a major early structure and hosts the principal footwall vein (VS-101) in the South Zone for some of its length. The lack of continuity of major veining up into the Salinas suggests that much of the faulting had ceased by the time of the Salinas deposition, except for the Cross, Ramp, and East Faults.

Some of these faults may represent syn-volcanic growth faults typical of near-surface epithermal settings that represent shallow, low displacements that manifest as larger pre-mineral faults at depth with increased displacements.

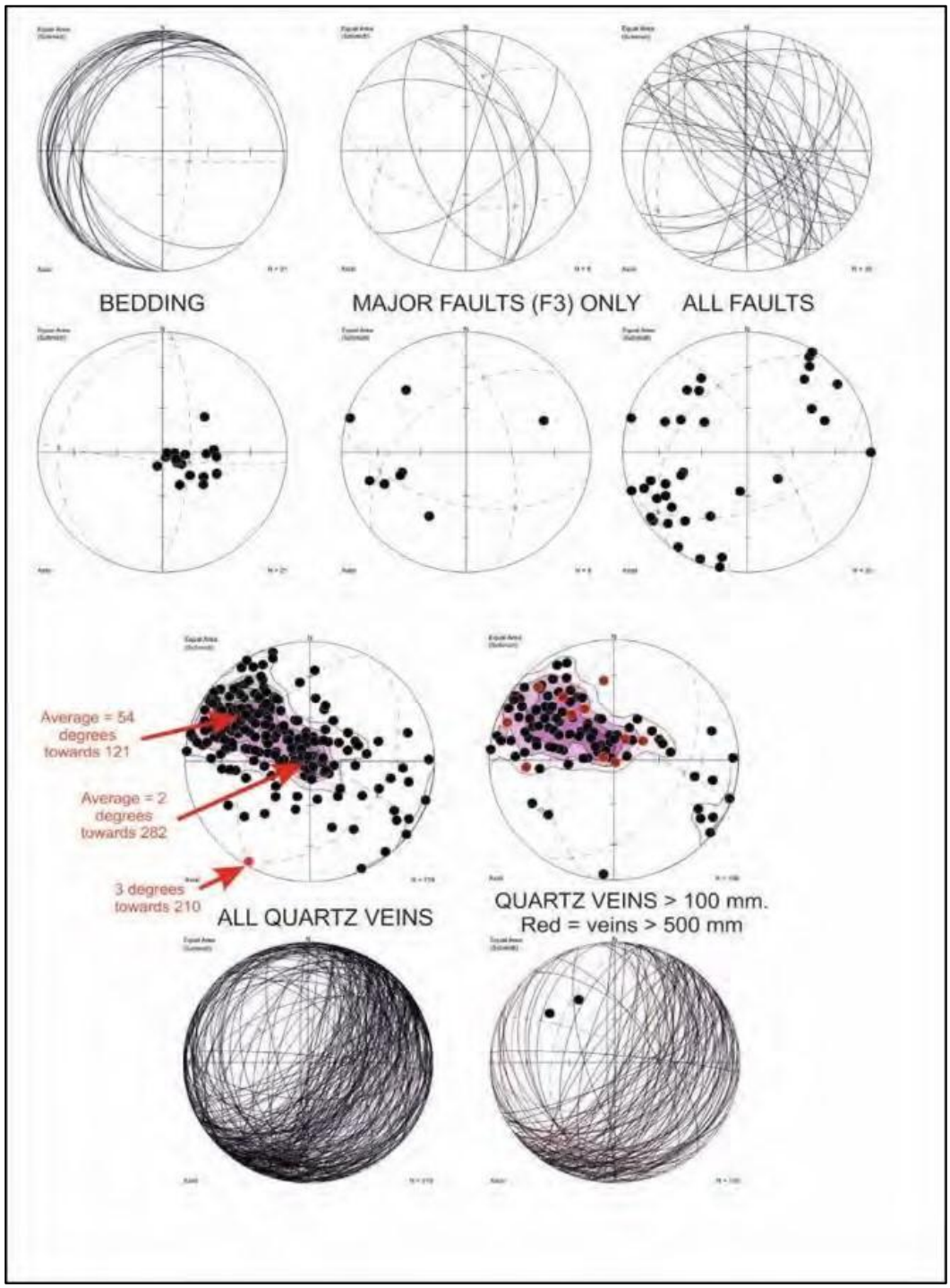
In the southeast portion of the South Zone, narrow sub-vertical gold-bearing veins extend into the Salinas and possibly represent a progression from the early compressive to more extensional conditions by the end of the Salinas deposition. Drilling demonstrates that a large chunk of stratigraphy is missing in the area separating the north and south zones of the deposit. This comprises the Mls + Mbt (lower) and Mat. A northwest-striking, southwest-dipping fault ("Upper Mbt Fault") is inferred. It is unclear if this terminates into the major Ramp Fault or vice versa. The throw on the Upper Mbt Fault seems to decline towards the north, and the stratigraphy is increasingly preserved in the footwall. Together, the Ramp and Upper Mbt faults define a triangular-shaped block that seems to have slid out southwards. Explaining the geometry, in terms of tectonic regime, is difficult, but a reactivated, extensional flower structure is one possible explanation.

Faults are difficult to map underground and in drill core because they are largely quite narrow (centimeter scale) and 'sealed' by silica; they generally do not form the zones of poor rock quality that typify post-mineral faults (though there are exceptions, for example along the East and Cross faults). This is reflected underground by the general lack of wall rock support. Figure 6-12 shows structural measurements from the underground workings for faults and veins. However, most understanding of the principal faults comes from 3D modeling, based on offsets of the lithostratigraphy and the marker beds.

The underground workings display numerous swarms of quartz veins. There are examples of conjugate veins and veins refracting through different lithologies (competency control). Examples are shown in Figure 6-13.

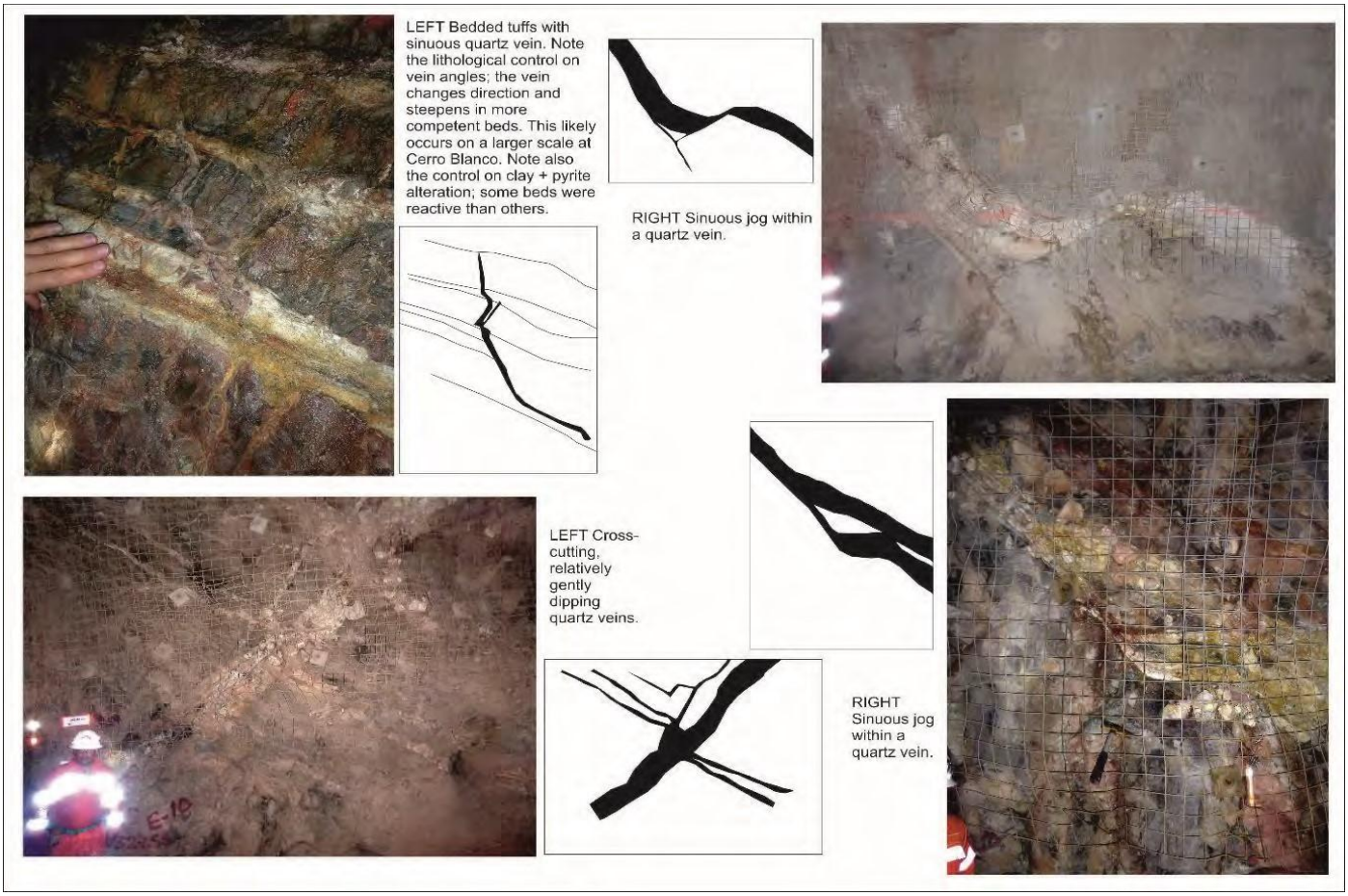
The gold-bearing veins at Era Dorada are focused in the footwall to the west of the steep Main Fault (also referred to as the Main Zone); in particular, they are concentrated in the uplifted blocks and west-verging folds of basement volcanic rock (Mcv and Mvo). The Upper Mbt lithostratigraphic unit seems to have been less favorable for veining, explaining the relative gap in veining between the North and South ramps. Likewise, the veins tend to pinch out in the Salinas Group (though some do make it to the surface and carry low grades).

Figure 6-12: Stereograms (Equal Area) Showing Poles & Great Circles for Faults & Veins



Note: All measured underground. Dots on the great circle plots represent slickensides. Source: Pratt and Gordon, 2019.

Figure 6-13: Photographs with Sketches of Veins Exposed Underground



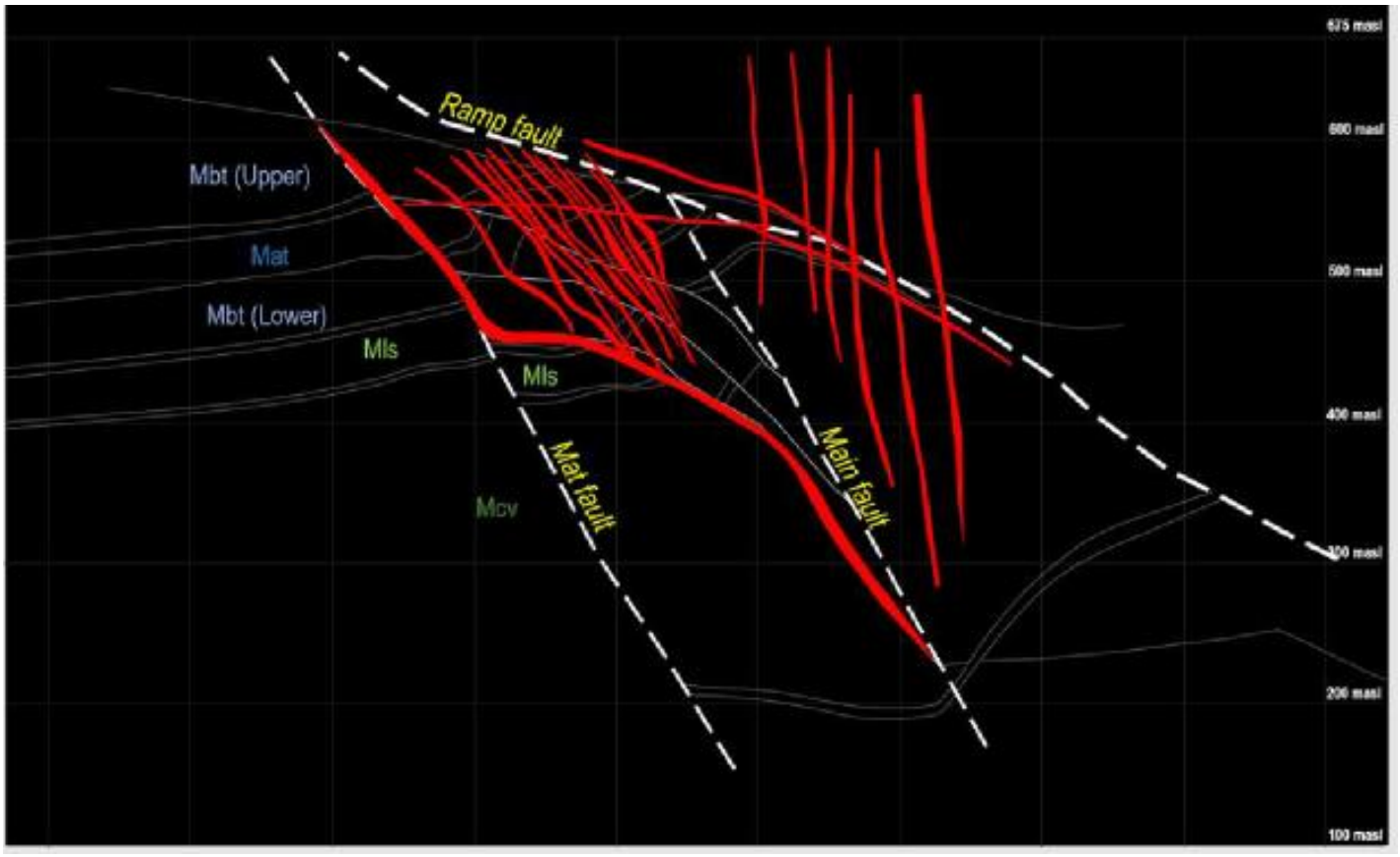
Source: Pratt and Gordon, 2019.

In section view, the veins clearly form lozenge-like duplexes and sheeted swarms, one in the South Ramp, the other in the North Ramp. Figure 6-14 is a cross-section across the South Ramp. Vein wireframes were generated in Leapfrog using core logging, alpha angles (angle between the core axis and vein) in non-oriented drill core, assay data, and underground mapping. They show a distinct branching and converging of relatively shallow veins into a steeper zone (Main Fault). Most veins are also constrained to the footwall of the Ramp Fault and the hanging wall of the steeper Mat Fault.

Sheeted veins and lozenge-shaped duplexes are also obvious in the map (plan) view. Figure 6 15 shows a series of horizontal slices at different elevations. The gap between the South and North resource areas mostly comprises the triangular wedge Upper Mbt stratigraphy between the Upper Mbt and Ramp faults. This seems to have been unfavorable for veining.

Underground mapping supports the 3D modeling; it shows a similar steepening and converging of veins into the Main Fault/Zone. Individual veins become thicker and more closely spaced along the Main Fault. The way individual veins swing into and intersect with the Main Fault creates ore shoots that plunge approximately 30° south.

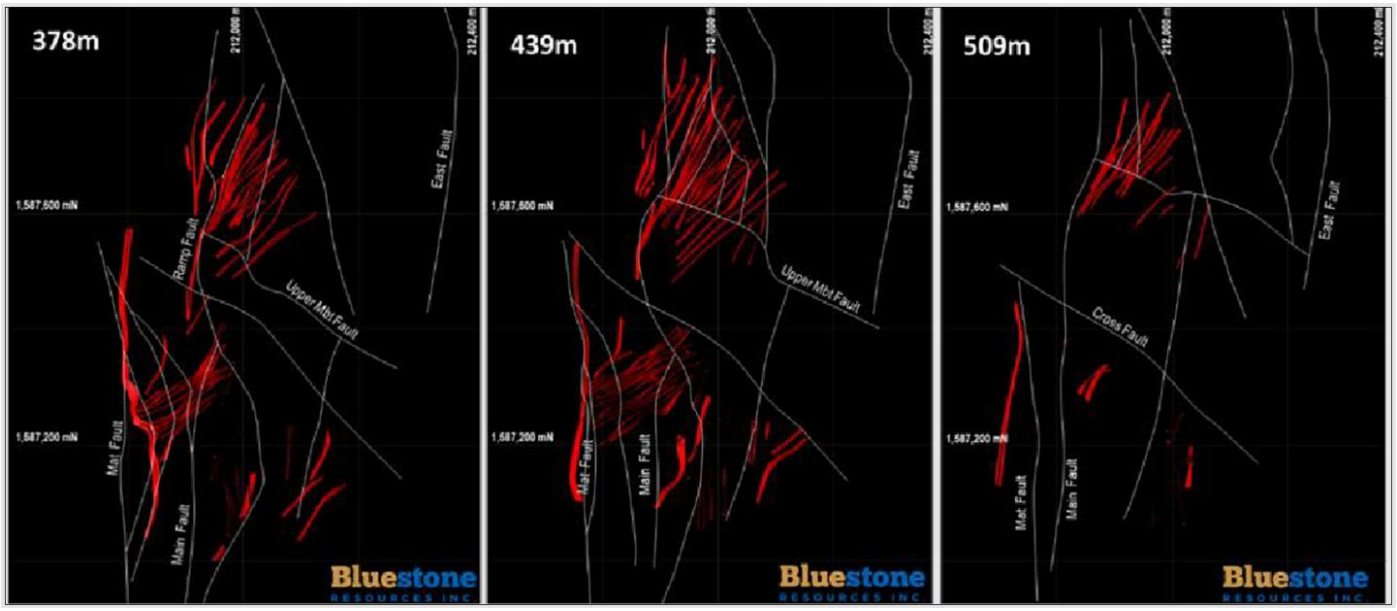
Figure 6-14: Annotated, Vertical East-West Cross-Section across the South Ramp (looking North)



Source: Bluestone, 2020.

Sheeted veins and lozenge-shaped duplexes are also obvious in map (plan) view. Figure 6-15 shows a series of horizontal slices at different elevations (North is up, Red = veins, Blue = faults). The gap between the South and North resource areas mostly comprises the triangular wedge Upper Mbt stratigraphy between the Upper Mbt and Ramp faults. This seems to have been unfavorable for veining.

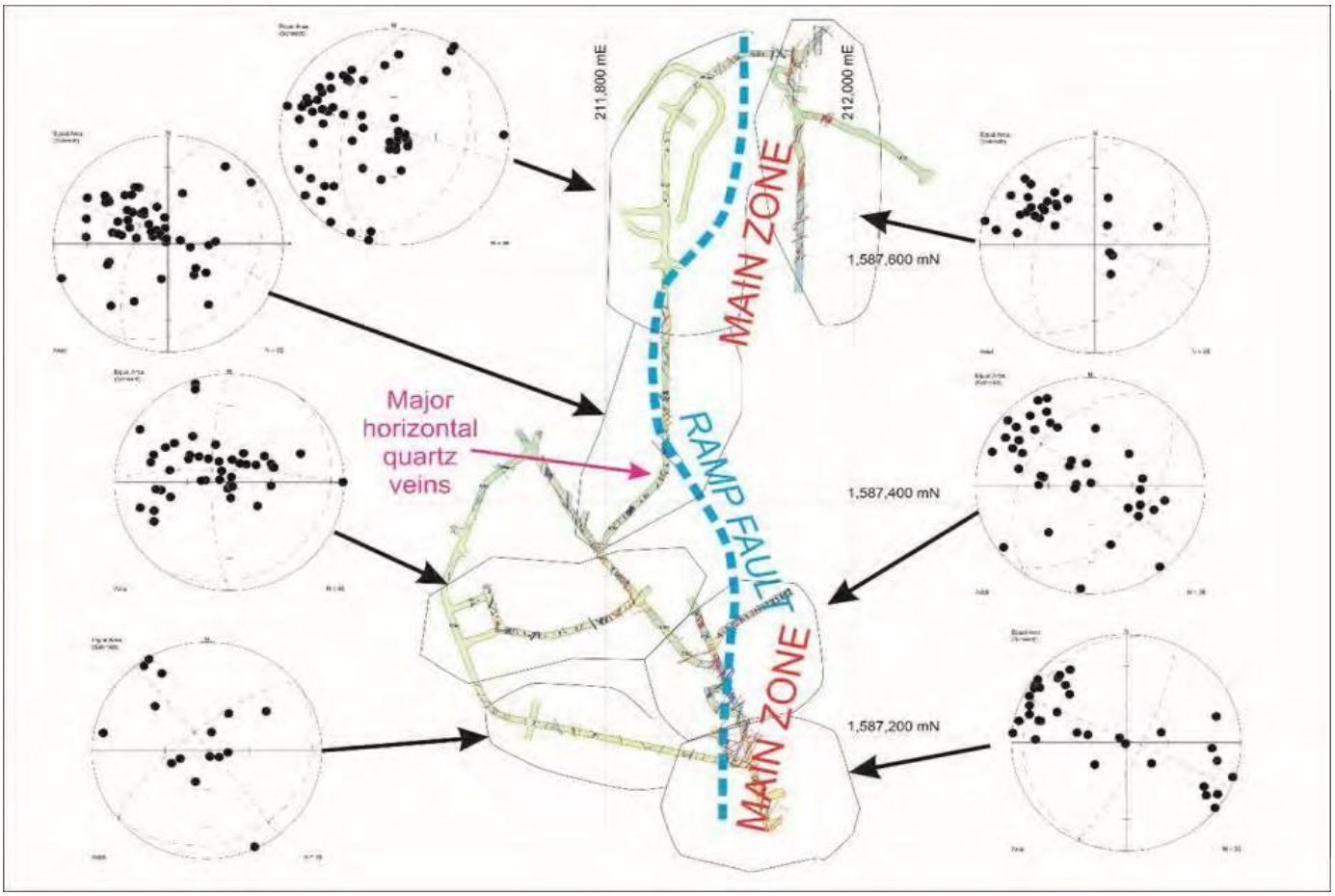
Figure 6-15: Horizontal Slices at Different Elevations through Era Dorada



Note: North is up. Red – veins; blue – faults. Source: Bluestone, 2020.

There are some secondary (conjugate) vein directions, but stereograms for sub-areas (Figure 6-16) show consistent patterns: steeper veins are mostly in the east and shallow veins in the west. A swarm of thick, sub-horizontal veins occurs in the immediate footwall of the Ramp Fault. The cumulative thickness of the veins exceeds 3 m. The flat veins clearly imply reverse (compressive) movement on the Ramp Fault. Clearly, the major faults played an important role in partitioning vein development.

Figure 6-16: Stereograms for More Detailed Sub-Areas in Underground Mapping



Source: Pratt and Gordon, 2019.

The stress regime during vein formation can also be calculated from conjugate veins. The stereogram for all quartz veins measured underground shows the intersection between the two principal vein directions is sub-horizontal). The dominant extension direction seems to have been vertical, which is highly unusual; epithermal veins generally develop during horizontal extension. The predominance of horizontal veins in the west supports the idea of vertical extension.

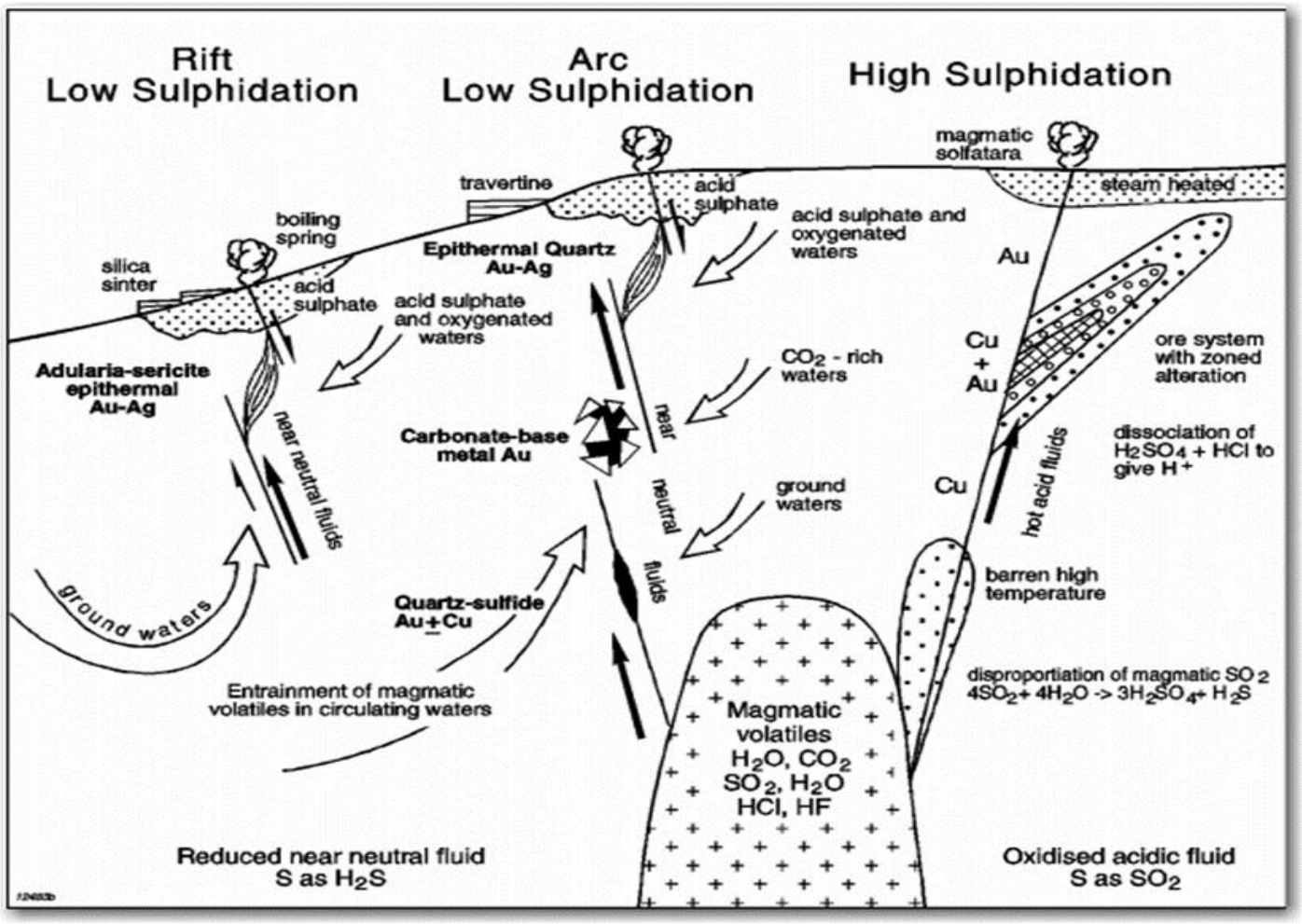
Field observations, 3D modeling, and stereograms, therefore, imply that the veins developed during compression rather than extension, at least in the initial stages of mineralization. This fits with the overall compressional geometry of the west-verging folds and reverse faults, later reactivated as normal, extensional faults. Recently discovered steeply dipping/vertical veins in the hanging wall of the south zone possibly record this change from a more compressional to extensional regime during the latter part of the mineralizing event. As some steep veins cut the Salinas Group and the sinters are contemporaneous with hydrothermal activity; this suggests that the hydrothermal/geothermal activity spanned the change from compressional to extensional tectonics.

6.4 Deposit Type

The low sulfide content and near absence of base metals in the Era Dorada veins confirm it as a classic hot springs-related, low-sulfidation epithermal deposit. In common with most low-sulfidation deposits, it appears to be linked to compositionally bimodal, basalt-rhyolite volcanism, the hallmark of intra- and back-arc rift settings worldwide. The hydrothermal system seems likely to have been initiated during rhyolite dyke and cryptodome emplacement, at the base of the Salinas unit, with the rhyolitic magma and magmatic input to the mineralizing fluid both being derived from the same deep parental magma chamber.

Arc-related low-sulfidation gold deposits occur at the highest crustal levels, most removed from inferred intrusion source rocks. Figure 6-17 shows the generalized deposit model.

Figure 6-17: Generalized Deposit Model Schematic



Source: Corbett and Leach, 1998.

Adularia-sericite epithermal gold-silver deposits characteristically occur as banded fissure veins and local vein/breccias, which comprise predominantly colloform banded quartz, adularia, quartz pseudomorphing carbonate, and dark sulphidic material termed ginguro bands. Examples of adularia-sericite epithermal gold-silver deposits include Waihi and Golden Cross, Pajingo, Vera Nancy, Cracow, Hishikari, Sado, Konamai, Tolukuma, Toka Tindung, Lampung, Chatree, Cerro Vanguardia, Esquel, El Peñon.

At near surficial levels, many are capped by eruption breccias and sinter deposits. Eruption (phreatic) breccias, which form by the rapid expansion of depressurized geothermal fluids, are characterized by intensely silicified matrix and generally angular fragments, including sinter, host rock, and local surficial plant material. Although sinter deposits formed distal to fluid upflows commonly associated with eruption breccias, sinters tend to be barren with respect to gold but may be anomalous in other elements such as boron, arsenic, and antimony.

Although cooling and traditional boiling models still hold for the deposition of gangue minerals (adularia, quartz pseudomorphing platy calcite, and chalcedony) and some gold, mixing of rising pregnant fluids with oxygenated or collapsing acid sulfate (low pH), groundwater is also favored as a mechanism for the development of characteristic bonanza gold-silver grades. Adularia-sericite vein systems are silver-rich, with gold-to-silver ratios greater than 1:10 being common.

Wall rock alteration formed as halos to veins occurs as sericite (illite) grading to peripheral smectite clays with associated pyrite and chlorite, and this alteration grades to more marginal chlorite-carbonate (propylitic) alteration. Low-temperature acid waters developed by the condensation of volatiles in the vadose zone contribute towards the formation of surficial acid sulfate alteration comprising silica (chalcedony, opal), kaolin, and local alunite, and these acid sulfate waters are interpreted to collapse to deeper levels and so aid in mineral deposition.

Structure and host rock competency are important mineralization controls in adularia-sericite vein systems. High-grade mineralized shoots often develop in dilational jogs or flexures in through-going veins where veins of greater thickness and higher gold grade develop and the intersections of fault splays. Bonanza-grade material may also develop at preferred sites of fluid quenching at rock competency changes. Recent studies (e.g., Rhys et al., 2020) attest that fault systems in very shallow epithermal systems characterized by sinter, lacustrine sediments, and hydrothermal breccias, similar to Era Dorada may represent syn- volcanic low-displacement growth faults that manifest as larger displacement pre-mineral faults at depth.

The connection between modern hot spring deposits and ancient hydrothermal systems, some with gold mineralization, has long been recognized (Lindgren, 1933). Epithermal mineral deposits are defined as those that develop close to the Earth's surface (within 1,000 m). They developed from fluids like those in modern geothermal systems. Sillitoe and Hedenquist (2003) defined the three types of epithermal deposits: high, intermediate, and low sulfidation. The low-sulfidation variant commonly occurs in rift settings, with bimodal volcanism in young, often Tertiary, volcanic arcs (e.g., Henley and Ellis, 1983). It is commonly associated with maar volcanoes, diatremes, and felsic flow domes.

Era Dorada shows all the characteristics of a completely preserved, non-eroded epithermal deposit. The occurrence of hot springs (sinters, silicified reeds, pisoliths) directly above the presumed feeder veins at Era Dorada implies a high water table and swampy conditions (cf. McLaughlin, California). In areas of high topographic relief, outflow springs (sinter) are usually found several kilometers from the upflow zones. The widespread occurrence of lacustrine and fluvial

clastic sediments in the Salinas Group and accretionary lapilli, typical of water-rich pyroclastic surges, supports this interpretation. Sedimentation probably kept up with subsidence. Mudstone dykes and geopetal structures—open fractures filled by horizontally bedded chalcedonic and Sulfide-rich sediment—reinforce the interpretation.

6.5 Era Dorada Deposit Geology

The Era Dorada deposit is a classic hot springs-related, low-sulfidation quartz-adularia-calcite vein system. It is localized along a complex fault intersection created during the late Miocene-Pliocene tectonic extension within the active Central American volcanic arc. Local igneous activities that drove the Era Dorada hydrothermal system include a vesicular andesite dike swarm and mineralization stage rhyolite/dacite flow dome eruption and cryptodome intrusion.

The Era Dorada vein systems are best developed (widest and most continuous) between the 300 masl to 500 masl elevation ranges. Principal host rocks include a lithic tuff—calcareous shallow marine-volcaniclastic sequence and, to a lesser extent, the overlying volcaniclastic-hydrothermal breccia sequence of probable Pliocene age. Vein zones often appear to transition to barren calcite beneath the ±300 m elevation in the northern half of the deposit. To the south, high-grade quartz-adularia-calcite vein zones continue at least another 100 m down to 200 m elevation. Some veins remain open at depth.

Massive chalcedonic silicification, referred to as a “silica cap,” dominates the conglomerates of the Salinas unit. Silica-flooded volcaniclastics and phreatic breccia are interbedded with chalcedonic silica sinter from the present surface to depths of ±100 m. Silicification also occurs in the underlying Mita as irregular envelopes, up to several meters wide, around the main veins as well as in the upper part of the limestone horizon as jasperoid. The red-bed siltstone is partially bleached and altered to a grey-green, illite, and smectite-bearing rock. Chlorite, in addition to illite and smectite, is a prominent alteration mineral in the ignimbrite, where it is concentrated in the fiamme.

Wall rock alteration, to a large extent, determines geotechnical rock hardness and presents contrasting resistivity and electrical chargeability characteristics that could be exploited across the district in the search for new gold occurrences beneath thin colluvial or basalt cover.

6.6 Mineralization

The Era Dorada gold deposit occurs within a large hydrothermal alteration zone covering an area of about 5 km long and 1 km wide. This zone exhibits the effects of strong, pervasive hot spring-type hydrothermal alteration.

Gold mineralization is hosted within a broadly north-south striking sequence of westerly-dipping siltstones, sandstones, and limestones (Mita Group) that are capped by silicified conglomerates and argillaceous sediments with contemporaneous dacite/rhyolite flow domes or cryptodomes (Salinas Unit). The Salinas rocks are syn-mineral and believed to have accumulated progressively in a low-relief graben characterized by a shallow groundwater table. The Salinas conglomerate was presumably derived by erosion of the flanking horst blocks as relief was created during active faulting. The topographic inversion required to explain the current prominent position of the graben fill is ascribed to the silicic character of the Salinas unit and its consequent resistance to erosion.

The west and east sides of the Era Dorada ridge consist of flat agricultural plains characterized by Quaternary basalts, interbedded with boulder beds and sands. These rocks also appear down-faulted to lower elevations, implying major post-mineral extensional movements on such faults, and they may be neotectonic (active).

The current gold resource occurs under a small hill and is confined within an area of about 400 m x 800 m. Gold and silver occur almost exclusively in quartz-dominated veins of low-sulfidation epithermal origin and in low-grade disseminated mineralization within the Salinas conglomerates and rhyolites. The highest grades are hosted by high to low-angle banded chalcedony veins, locally with calcite replacement textures.

Gold-bearing structures in the Project area extend 3 km to the northwest of the gold deposit and occur largely confined within the hydrothermal alteration zone. Exposures are poor and locally covered by alluvium and post-mineral rocks. Gold-bearing structures extend at least 1 km south and southwest of the deposit under valley fill and post-mineral rocks.

Geothermal well MG7, located about 0.5 km east of the deposit, encountered a 27 m zone averaging 6.3 g/t Au and 22 g/t Ag at a depth of 634 m. The upper 6 m of this zone averages 23.9 g/t Au and 79 g/t Ag. Although the geometry is uncertain and the sampling methodology of the drill cuttings cannot be determined, possibly this vein material was caught up in a fault crush zone/splay within the East Fault (much like the other exotic lithologies seen within the fault zone), or conversely, represents a separate mineralized system distinct from the main deposit.

6.6.1 Vein Zones

Petrographic descriptions of four vein zones by Economic Geology Consulting (Thompson et al., 2006) concluded that the veins consist of crustiform banded chalcedony, quartz, adularia, calcite, sulfides, and visible gold. The samples represent a range of almost 300 m in elevation. Bladed calcite or pseudomorphs after bladed calcite (lattice blade texture) were observed in all four samples. Bladed calcite is a rapid depositional texture, common when calcite precipitates from boiling fluids. A wide variety of recrystallization textures in quartz and chalcedony may also indicate changing fluid conditions and periodic boiling. Figure 6 18 shows a high-grade intercept in drill hole CB-20-430 with banded chalcedony-adularia-acanthite and visible gold that assayed 144 g/t Au and 282 g/t Ag.

Observations suggest that mineralization occurred as one principal multi-stage event as banded vein material, dominated by cryptocrystalline and originally amorphous silica phases (jigsaw quartz and chalcedony) characteristic of both the north and south zone vein swarms. Colloform banding with gel-like precursor textures is common, and observations from drill core suggest that banding is characteristic of high-grade zones, with coarser crustiform and crystalline bands more associated with lower-grade veins. Higher grades are associated with fine-grained (<100 µm) electrum, kustelite, and acanthite concentrated in bands of fine- to very fine-grained jigsaw quartz (crystallized amorphous silica, Albinson, 2019). Gold-silver minerals are accompanied by the rare presence of tetrahedrite and chalcopyrite.

Repetitive “crack and seal” pulses and associated boiling/flashing events very close to the paleosurface are suggested as the main mechanisms for precious metal deposition. The higher-grade, often bonanza-grade core intersections with coarser and more abundant sulfides, electrum, and free gold appear to represent an earlier series of events. Multistage banding can be very finely repetitive down to 5 to 10 µm widths for individual bands. Soft sediment-type deformation is commonly visible in the bands with mamillary colloform bands deformed into flame-like textures due to the

deformation of the bands by turbulent fluid flow. Sulfides and electrum are present mainly in the fine- or very fine-grained jigsaw quartz bands. Adularia-rich bands are not easily visible with the hand lens and are very fine-grained.

Figure 6-18: High-grade Drill hole Intercept Hole CB20-430 – 144 g/t Au, 282 g/t Ag (227.3 to 228.9 m)



Source: Bluestone, 2020.

The lack of inter-stage hydrothermal brecciation and coarse-grained primary quartz textures suggest that the mineralizing event was a fairly short-lived event that occurred very close to the paleosurface. The lack of post-mineral structural displacement of veins and distribution of high grades over a +300 m vertical profile attest to the pristine nature of the veins.

Underground observations include the following:

- Vein zones are best developed throughout the model between elevations of 300 and 500 m. This elevation range roughly coincides with the Mcv contact beneath and the Salinas contact above. Thus, the principal host rocks are the Mita Group sandstones, calcareous sediments, and overlying tuffs. The quartz veins at Era Dorada occur mainly within Mita sediments and Mcv tuffaceous rocks. These moderate to steep veins are associated with a subsidiary conjugate set of low-angle veins. The majority of veins appear tThe majority of veins appear to stop at the Salinas contact, with the exception of sub-vertical veins in the southeast part of the south zone that cut the Salinas and continue to surface. Vein zones occur as two upward-flared arrays that appear to converge downwards and merge with basal master veins around the contact with the Mcv. The south zone vein array is the better-formed. High gold grades locally persist at least down to the 200 m elevation, notably in the southern third of the model, where

at least one vein merges with the main footwall feeder structure. In several locations north of 1,587,400N, pass beneath high-grade quartz veins but encounter only massive barren calcite. This is an indication that the bottoms of productive veins have been found at those locations. Within vein zone envelopes, individual veins do not form a random stockwork but tend to run parallel or sub-parallel to the main structural trends. The definition of economic mineralization depends on the vein thickness, grade, and spacing. The structural control of the veins is discussed above. Most individual veins exposed in the underground workings do not exceed 1 to 2 m; much thicker veins, up to 7 m width, do appear in the vicinity of the north zone ramp (Figure 6-19) and in deeper levels of the south zone. Closely spaced veins or zones of convergence form wide zones of high-grade mineralization (Figure 6-19).

Figure 6-19: View of Veins VN-05, 06, 07 in the North Ramp Underground Workings



Note: Section assayed 20.4 m grading 18.9 g/t Au and 33.2 g/t Au. Source: Bluestone, 2020.

Erro! Fonte de referência não encontrada. shows vein textures associated with gold mineralization; they include bladed calcite, a classic indicator of boiling fluids, subsequently replaced by quartz or leached to give a skeletal framework. Other classic textures include crustiform banding, bands of cream-pinkish euhedral adularia, and quartz with minor dark grey silver Sulfides/sulphosalts. Inspection of vein textures suggests that gold and silver were introduced as one major event of multistage finely banded veining (originally amorphous silica) with subordinate bands of platy calcite that is mostly pseudomorphed to cryptocrystalline silica phases

Figure 6-20: Examples of Vein Textures from Era Dorada



Source: Bluestone, 2020. Note: Section assayed 20.4 m grading 18.9 g/t Au and 33.2 g/t Au. Source: Bluestone, 2020.

Figure 6-21 shows vein textures associated with gold mineralization; they include bladed calcite, a classic indicator of boiling fluids, subsequently replaced by quartz or leached to give a skeletal framework. Other classic textures include crustiform banding, bands of cream-pinkish euhedral adularia, and quartz with minor dark grey silver Sulfides/sulphosalts. Inspection of vein textures suggests that gold and silver were introduced as one major event of multistage finely banded veining (originally amorphous silica) with subordinate bands of platy calcite that is mostly pseudomorphed to cryptocrystalline silica phases.

Figure 6-21: Examples of Vein Textures from Era Dorada



Source: Bluestone, 2020.

Many veins and siliceous rocks (rhyolite/dacite) at Era Dorada display siliceous mudstone/sandstone dykes. There are also common geopetal structures, late cavities filled by horizontally banded siliceous sediments of hydrothermal origin mixed with vein gangue (Figure 6-22). These “fossil spirit levels” indicate proximity to the paleosurface and are confirmed by the presence of sinter immediately above.

It is unusual to see epithermal veins developed immediately beneath sinter, although other examples do exist (e.g., McLoughlin, California), implying the topography at the time of mineralization was low and the water table was very high. This is supported by the presence of accretionary lapilli in the Salinas Group and Mbt siltstones; they are typical of wet phreatic-dominated eruptions and pyroclastic surges. Diatremes and rhyolite flow domes are also typical in this environment.

In summary, the principal control on gold mineralization at Era Dorada was probably the boiling level in a hydrothermal system. The best grades are associated with boiling textures. At many low-sulfidation epithermal deposits, the vertical interval of economic grade is restricted to the former boiling level. This can be less than 100 m. These boiling levels form flat ore shoots. There are occurrences of high gold grade down to 640 m (downhole depth) in a geothermal hole (MG-07).

Figure 6-22: Example of Geopetal Structure

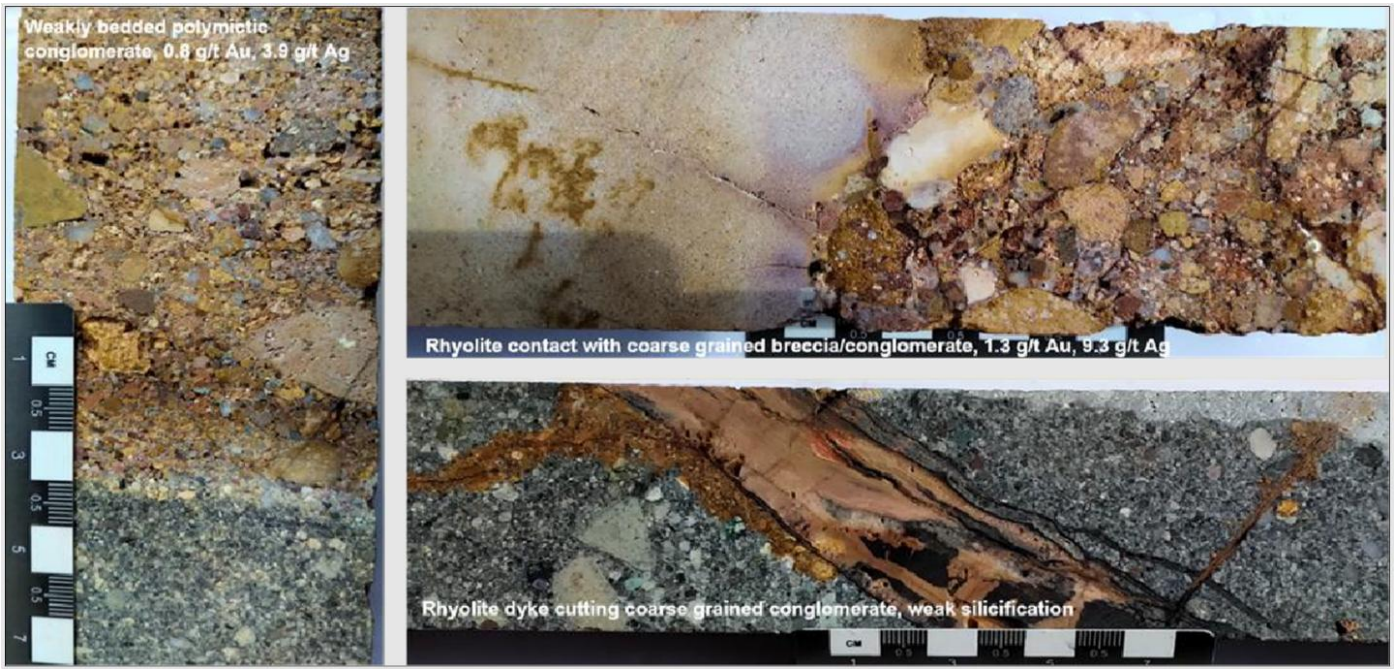


Source: Pratt and Gordon, 2019.

6.6.2 Disseminated Mineralization

The Salinas unit shows widespread and low-grade disseminated gold mineralization associated with weak to strongly silicified polymictic conglomerates and altered rhyolite breccias and flows. Mineralization grading of 0.2 to 2 g/t Au is pervasive and present in variably silicified bedded conglomerates and appears to be driven by intrusive rhyolite dykes and breccias (Figure 6-23). Locally, parts of the base of the Salinas are marked by an aphanitic rhyolite body, probably a cryptodome, given it is underlain by narrow rhyolite dykes. The thicker Sinter horizons do not contain significant gold values, nor do strongly argillic-altered lithologies and fault gouge zones.

Figure 6-23: Salinas Unit – Examples of Disseminated Mineralization Rock Types, Salinas Unit



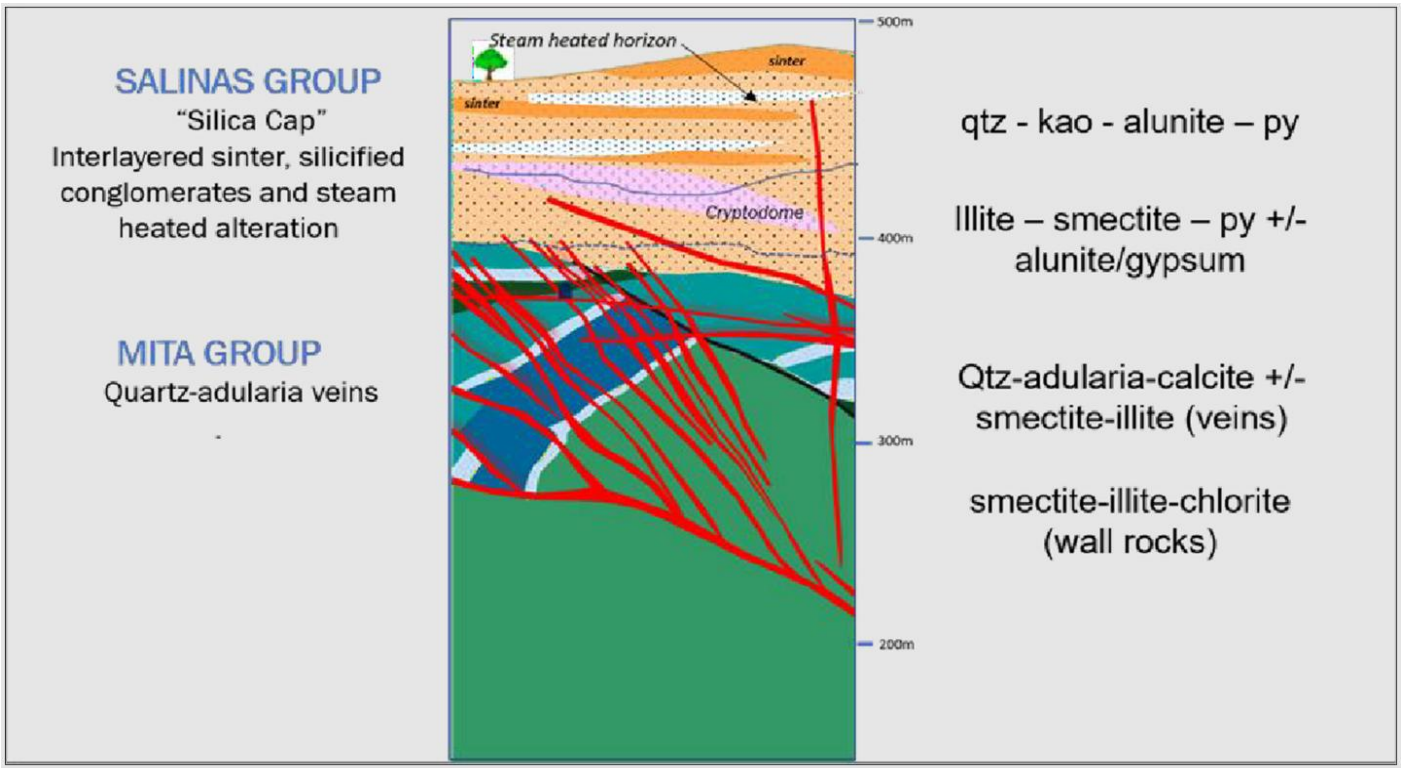
Source: Bluestone, 2020.

6.6.3 Hydrothermal Alteration

Many low-sulfidation epithermal vein deposits have significant, mechanically weak halos of illite/smectite + pyrite + sphene/leucoxene; however, the wall rocks at Era Dorada are generally only weakly clay altered and have a very low Sulfide content (Figure 6-24). Most clay alteration is concentrated along some late faults, for example, the East and Cross faults, and within some of the hydrothermal breccias, particularly the phreatic breccias in the Salinas Group.

A study using drill core hyperspectral imaging spectroscopy in the 500 nm to 2,500 nm wavelength range and detailed petrographic, SEM, and EDS studies revealed two paragenetic stages of vein formation (Savinova, 2020). The main auriferous veins consist of multi-stage crustiform and colloform bands that are characterized by paragenetic Stage one equilibrium assemblage of quartz (chalcedony)-adularia-calcite- ankerite. Sulfides are located mostly in ginguro bands that consist of fine-grained pyrite, chalcopyrite, tetrahedrite, and acanthite. Stage two of the paragenesis is characterized by intense overprinting of the quartz-adularia veins by montmorillonite and interstratified illite. Locally, bladed calcite is replaced by quartz. Hydrothermal alteration in the proximal zone of the sedimentary and volcanoclastic wall rocks is characterized by quartz-adularia-illite-montmorillonite. Wall rock-hosted illite suggests a temperature of formation >230°C. The distal alteration zone is marked by illite-chlorite-calcite.

Figure 6-24: Vertical Alteration Profile through Era Dorada



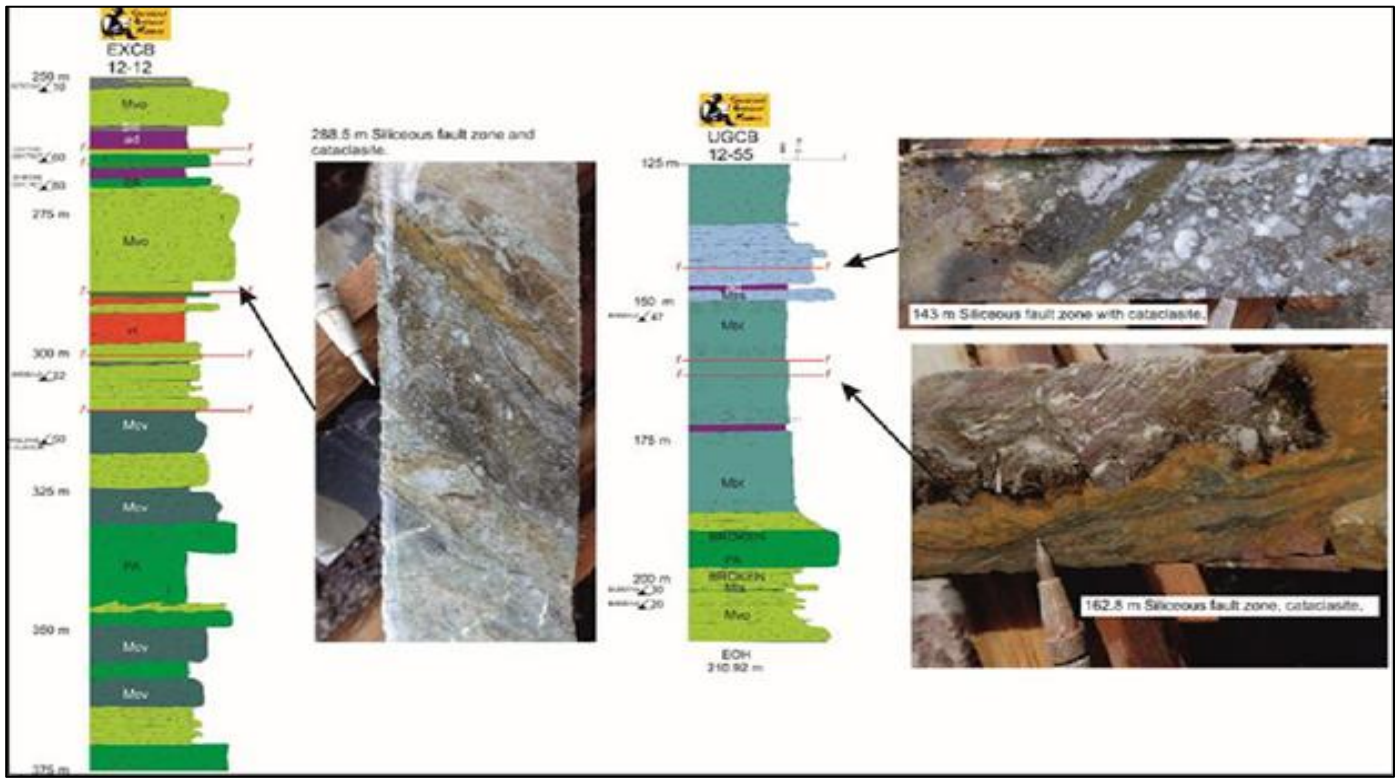
Source: Savinova, 2020.

Silicification is widely developed within the Salinas and more selectively in the underlying Mita Group, where it occurs as irregular envelopes, up to several meters wide, around the main veins as well as in the upper part of the limestone horizon as jasperoid. The most impressive alteration feature at Era Dorada is the large "silica cap" hosted in the Salinas sediments, typically beginning at or below 400 m elevation and continuing upward to the surface. Most silica is directly related to hot spring activity; the sinters and pisolithic beds contain abundant silica (although it is possible that some had carbonate precursors). However, there are also numerous beds of sandstone, conglomerate, and mass flow deposits in the Salinas Group that are highly siliceous and locally flooded by chalcedony and fine-grained pyrite. These rocks are black when fresh, white, and limonite stained when oxidized. Exposures around the Era Dorada ridge show that this silicification can be very capricious and replaced abruptly and laterally by smectite-rich clay alteration.

Where the paleo-groundwater table was several meters below the paleosurface, a conglomerate in its immediate vicinity was silicified, and the vadose zone above it was subjected to steam-heated alteration. The steam-heated alteration, containing cristobalite, kaolinite, and alunite (an advanced argillic assemblage), was the product of acidic solutions formed by the condensation of ascendant H2S-bearing steam into downward-percolating groundwater. The overall result is an interlayered sequence of sinter, silicified conglomerate, and steam-heated alteration.

Many faults at Era Dorada are sealed by silica and are pre-mineral. Examples are shown in Figure 6-25.

Figure 6-25: Examples of Sealed, Silicified Fault Zones



Source: Pratt and Gordon, 2019.

The boiling hydrothermal fluids that formed the Era Dorada vein system produced an even larger volume of intensely altered wall rock. Alteration types and zoning are typical of low-sulfidation epithermal systems. The remnant sinter above the deposit suggests that the Era Dorada system remains largely intact.

Silicification continues locally down to 300 m elevation along fault zones and in favorable rock types. Overall, the Era Dorada silica cap averages 400 m wide and is up to 150 m deep for at least a kilometer in strike. Within 50 m to 100 m of the surface, silicification is manifested by opaline silica flooding in the fragmental Svc and Rp units. At depth, very fine-grained quartz replacement of Mita Group calcareous sediments (locally forming jasperoid) and tuffs dominate. The Mcv crystal lithic tuff is generally only silicified near contacts with overlying sediments and along fault zones.

Silicification typically yields outward to moderate to strong sericitic alteration above 400 or 450 m elevation. At deeper levels, silicified zones grade outward and downward into large volumes of clay-sericite- pyrite±calcite alteration in Mita Group sediments and tuffs. Pyrite contents are commonly in the range of 1-3%, locally reaching 5%.

The Mcv is pervasively sericite-chlorite-pyrite±calcite altered virtually everywhere it has been drilled. Sericite dominates closer to mineralized faults and higher. Chlorite-calcite dominates outward and at depth. Pyrite is ubiquitous but generally less than 0.5%.

7 EXPLORATION

7.1 Exploration

As of the end of 2021, Bluestone had drilled approximately 267 holes for a total of 45,725 m on the Era Dorada property since acquiring it from Goldcorp. Table 7-1 summarizes historical drilling on the property.

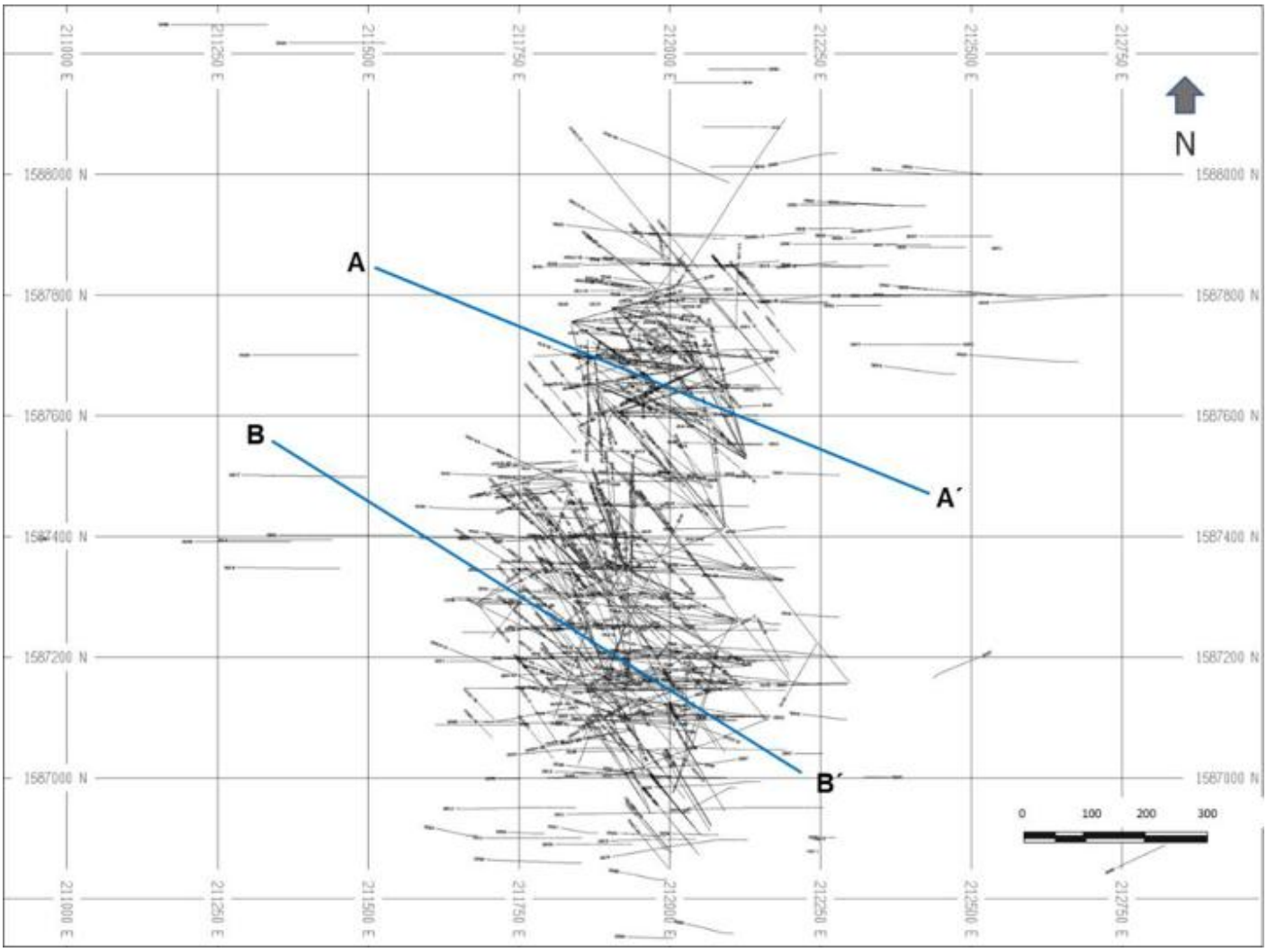
Table 7-1: Drilling Summary

Year	Company	Holes Drilled	Meters
1998	Mar-West	9	1,340
1999	Glamis	48	7,074
2000	Glamis	18	3,525
2002	Glamis	23	6,525
2004	Glamis	42	9,370
2005	Glamis	120	29,065
2006	Glamis	67	15,129
2007	Goldcorp	47	12,373
2008	Goldcorp	2	586
2009	Goldcorp	1	140
2010	Goldcorp	10	2,277
2011	Goldcorp	28	5,898
2012	Goldcorp	96	21,370
2017	Bluestone	8	2,324
2018	Bluestone	74	13,993
2019	Bluestone	61	8,403
2020	Bluestone	74	15,172
2021	Bluestone	50	5,833
Total	**778**	**160,397**	

Source: Kirkham, 2021.

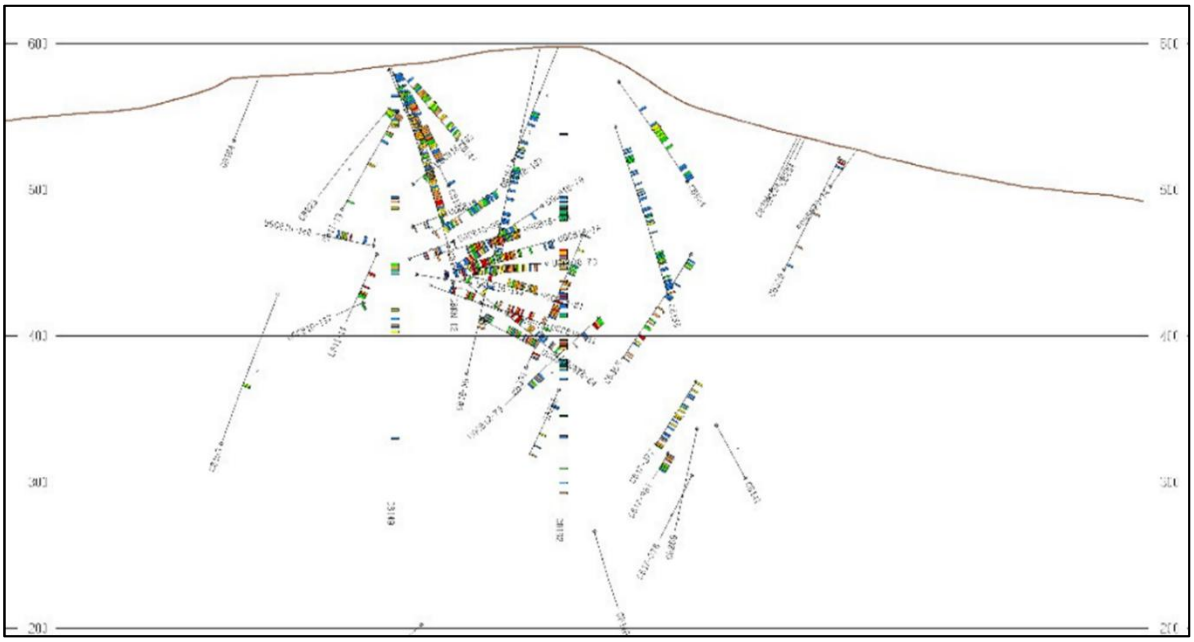
Figure 7-1 shows a plan view of drill hole locations. Figure 7-2 and Figure 7-3 show representative section views of the drilling along with gold assay data and topography.

Figure 7-1: Plan view of Drill hole Locations



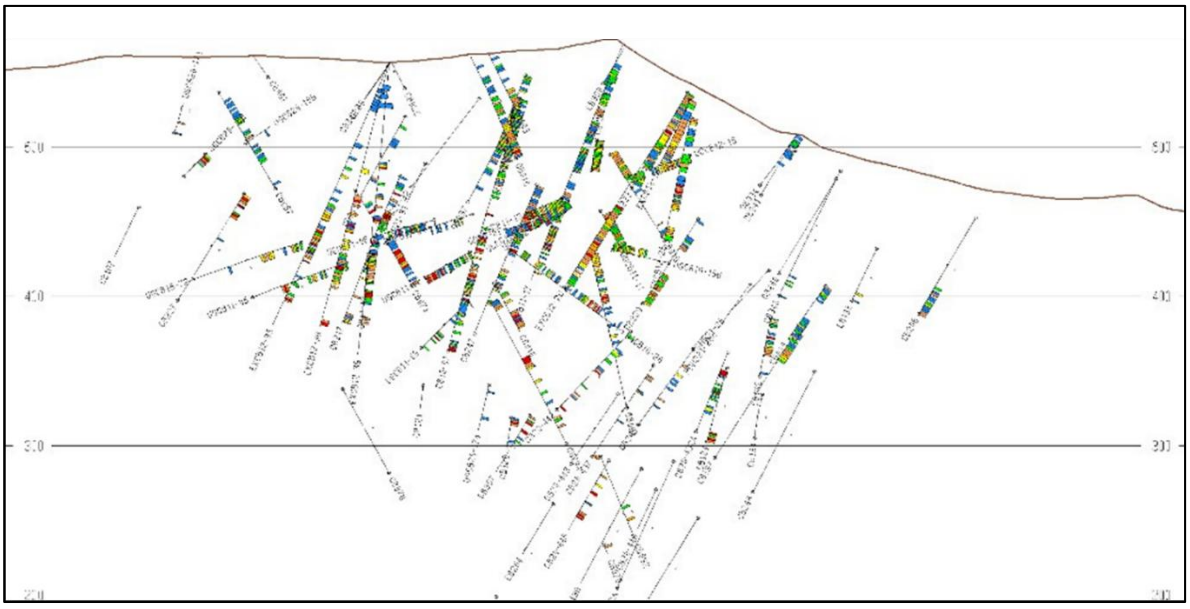
Source: Kirkham, 2021.

Figure 7-2: Section View A-A' (Azimuth 110°)



Source: Kirkham, 2021.

Figure 7-3: Section View B-B' (Azimuth 110°)



Source: Kirkham, 2021.

7.2 Goldcorp & Glamis Drilling (Pre-2017)

Prior to Bluestone's ownership, reverse-circulation (RC) and diamond drilling (DD) was carried out. Many early holes were collared using RC size core before switching to NQ size core. Collar data from these historical programs was surveyed with a differential global positioning system (GPS), and down-hole survey measurements were taken with either a single-shot Sperry-Sun camera system or a multi-shot Flexit instrument.

Many of the earlier drill holes by previous operators were not drilled perpendicular to the strike and dip of the veining, and therefore, drilled widths of many veins were not representative. The most common vein intersections occur from between 0° and 60° to the core access. These intervals are thought to belong to steep to moderately dipping vein sets. These core intervals would be longer than the true thickness of the actual veining. Intersections ranging from 60° to 90° to the core axis are less common and are believed to belong to flat to near-flat vein structures. These vein intervals would be closer to the true thickness of the veining but still longer than the true thickness. Only vein intervals drilled perpendicular to the strike and dip of the veining would represent the true thickness of the vein. Based on previous reports from Glamis Gold, the ratio to the true thickness of the vein on average is about 1.73 (i.e., every 1.7 m represents 1 m of true vein thickness).

7.3 Data Validation

Historical core logging, sampling, and quality assurance/quality control (QA/QC) procedures were first reviewed and documented by Golder in 2014. Ten core samples were collected from one-quarter sawn NQ core, and selected drill hole collars were surveyed using a GPS. Assayed gold and silver grades were found to be consistent with those reported by Goldcorp. Golder was satisfied that the drill hole data was collected in a manner consistent with industry best practice standards.

As part of the core logging data verification, Golder compared a selection of core logs against half-core stored at the project site. Five half-core Drill holes were reviewed from the North and South deposits. The Microsoft Excel files were reviewed first, and Drill holes were selected that represented the typical mineralization style for each deposit. In addition, ten verification samples were taken from these Drill holes. Each verification sample was a half-core sample sawed into quarters, with one-quarter sample sent for analysis and the other returned to the core racks. Table 7-2 on the following page summarizes the samples selected for core logging review and verification sampling.

Samples were sawed and bagged under Golder's supervision and were transported off-site via helicopter and plane to Canada and then by ground transportation to ALS Chemex Laboratories in Sudbury for sample preparation and analysis. A comparison of the Excel files against the drill core indicated an excellent match between the core logs and the retained core. Table 7-3 provides a list of the drill hole collar surveys completed by Golder.

Eight drill sites were visited, with multiple Drill holes located at some sites. Casings had been removed for most Drill holes. The data collected was a mixture of pre-Goldcorp Drill holes (2006 or earlier) and drilling completed by Goldcorp during 2010 and 2011. All Drill holes from the surface were grouted to prevent water flow into the underground workings.

Table 7-2: Verifications Samples

Drill Hole ID	Duplicate Sample No.	Original Sample No	From (m)	To (m)	Deposit	Metal Analysed	Rock Type
CB-152	205873	82225	128	129	North	Au, Ag	Lapilli Tuff
CB-152	205874	82226	129	130	North	Au, Ag	Lapilli Tuff
CB-200	205884	407101	156	157	South	Au, Ag	Quartz Tuff
CB-200	205885	407102	157	158	South	Au, Ag	Quartz Tuff
CB-241	205891	404849	111.4	112.6	South	Au, Ag	Conglomerate
CB-241	205892	404850	112.6	113.5	South	Au, Ag	Fault
CB-254	205895	414397	100.5	102	South	Au, Ag	Volcaniclastic Sediments
CB-254	205896	414398	102	103.5	South	Au, Ag	Volcaniclastic Sediments
CB-10-15	205871	435941	135	136.23	North	Au, Ag	Lapilli Tuff
CB-10-15	205872	435943	136.23	137.46	North	Au, Ag	Lapilli Tuff

Source : Goldcorp. 2014.

Table 7-3: Drill Hole Collar Survey (NAD 27 Zone 16N)

Drill Hole ID	Golder		Cerro Blanco	
	Easting	Northing	Easting	Northing
C 10 08	212015.1	1587867	212009	1587748
C 11 12	211906.8	1587714	211904	1587605
C 11 15	211969.7	1587769	211966	1587655
C 11 18	211866.4	1587405	211873.2	1587297
C 11 21	211901.6	1587414	211898.9	1587307
C 151	212025.1	1587821	212020.8	1587707
C 247	211985.5	1587315	211978.8	1587202

Source: Goldcorp, 2014.

Approximately 5% of the Drill holes (20 holes) were subjected to data verification checks by Golder. The 20 selected holes, summarized in Table 7-4, included a variety of historical data as well as some of the more recent holes. The data verification checks consisted of the following:

- comparison of final assays to the original laboratory certificates
- analysis of external laboratory duplicate assays by generating XY scatterplots

- review of downhole survey measurements to identify anomalous changes to hole orientation.

For the 20 holes reviewed, the comparison of final assays to the original assay certificates did not identify any material differences in assay values.nExternal laboratory duplicate assays were reviewed to assess the reliability of the primary assay laboratory. XY scatterplots were generated for each of the 20 holes. With the exception of a few outliers, the majority of the data compared well. Table 7-4 illustrates an example of the XY scatterplots used to compare assay results.

Table 7-4: Drill Hole Selected for Data Verification

Drill Hole IDs	
CB-012	CB-200
CB-016	CB-227
CB-063	CB-244
CB-078	CB-247
CB-095	CB-305
CB-10-02	CB-309
CB-120	CB-314
CB-142	CB-345
CB-146	CB-357
CB-151	CB-362

Source: Goldcorp, 2014.

7.4 Bluestone Drilling (2017-2021)

Drilling completed by Bluestone between 2017 and 2020 was a combination of surface and underground diamond core drilling. Underground channel sampling was also performed and included in the resource estimation.

Drills were operated by Continental Drilling of Guatemala. The surface drilling was performed using two Hydracore 1000 portable drill rigs, one of which was replaced later in the program by a Boart Longyear LM-75 belonging to Bluestone, which was later converted for underground drilling. During the height of the drill program, five LM-75s were operative. Drill holes were developed by drilling a larger diameter (HQ) core at the early stage of the hole and then decreasing to NQ and/or BQ size if the drilling conditions became difficult.

Core recoveries were high, and by utilizing several drill core sizes, Bluestone was able to ensure drill hole target completion. To date, 89 holes have been drilled from the surface and 128 holes from underground.

Drill hole collars were surveyed using a total station (coordinate system UTM NAD 27 Zone 16N). In-hole drill surveying for azimuth and dip was completed using the Reflex EZ-Shot system approximately every 25 m down-hole. Orientation of the drill core was performed throughout Bluestone's drill.

7.5 Significant Assay Results

Table 7-5 provides a selection of significant drill hole intervals from the Era Dorada drill hole database. Drill hole intervals are reported as actual core lengths, and many may not represent the true thickness.

Table 7-5: Gold & Silver Samples from the Drill Hole Database

Hole	Company	From	To	Length (m)	Au (g/t)	Ag (g/t)
CB-012	Mar-West/ Glamis	99.50	108.50	9.00	13.7	46.5
CB-012	Mar-West/ Glamis	141.50	147.50	6.00	12.9	75.8
CB-012	Mar-West/ Glamis	195.50	198.50	3.00	3.0	8.0
CB-012	Mar-West/ Glamis	236.00	237.50	1.50	13.0	6.0
CB-016	Mar-West/ Glamis	192.55	195.35	2.80	3.3	0
CB-063	Glamis	88.50	99.00	10.50	4.4	25.7
CB-063	Glamis	114.00	126.00	12.00	3.2	21.0
CB-063	Glamis	183.00	186.00	3.00	7.7	20.0
CB-063	Glamis	196.50	199.50	3.00	4.2	25.0
CB-063	Glamis	207.00	210.00	3.00	18.7	20.0
CB-063	Glamis	225.00	228.00	3.00	37.3	75.0
CB-063	Glamis	241.50	244.50	3.00	5.1	3.5
CB-078	Glamis	158.20	161.40	3.20	3.4	4.1
CB-078	Glamis	242.10	245.10	3.00	3.5	4.7
CB-078	Glamis	248.10	273.75	25.65	66.1	42.2
CB-078	Glamis	299.25	303.75	4.50	4.7	17.7
CB-078	Glamis	338.25	345.75	7.50	10.8	17.4
CB-095	Glamis	155.00	158.00	3.00	3.7	204.9
CB-095	Glamis	179.00	182.00	3.00	17.8	7.4
CB-095	Glamis	233.00	236.00	3.00	88.0	98.6
CB-10-02	Goldcorp	117.50	120.30	2.80	14.7	79.5
CB-10-02	Goldcorp	135.75	139.50	3.75	12.9	91.8
CB-10-02	Goldcorp	146.00	149.00	3.00	9.5	79.6
CB-10-02	Goldcorp	168.86	173.00	4.14	26.2	144.8
CB-10-02	Goldcorp	197.00	200.00	3.00	20.3	19.9
CB-120	Glamis	219.00	238.50	19.50	17.5	20.3
CB-120	Glamis	246.00	249.00	3.00	8.8	20.6
CB-142	Glamis	163.50	171.50	8.00	16.0	72.2

Hole	Company	From	To	Length (m)	Au (g/t)	Ag (g/t)
CB-142	Glamis	196.20	204.50	8.30	19.2	11.7
CB-142	Glamis	302.75	306.00	3.25	19.3	14.3
CB-146	Glamis	80.30	86.00	5.70	14.0	196.8
CB-146	Glamis	109.00	112.40	3.40	10.3	78.9
CB-146	Glamis	118.90	130.00	11.10	70.4	226.3
CB-146	Glamis	139.00	143.00	4.00	12.4	35.4
CB-146	Glamis	149.00	152.00	3.00	3.7	8.0
CB-146	Glamis	156.00	159.00	3.00	21.1	30.6
CB-146	Glamis	182.00	185.00	3.00	4.2	2.5
CB-151	Glamis	162.40	165.50	3.10	25.6	152.8
CB-151	Glamis	172.90	179.30	6.40	13.6	24.7
CB-151	Glamis	327.50	330.50	3.00	5.0	5.5
CB-200	Glamis	117.00	120.00	3.00	5.7	26.0
CB-200	Glamis	144.00	147.00	3.00	5.0	13.0
CB-200	Glamis	152.00	161.00	9.00	7.5	13.6
CB-200	Glamis	165.00	168.50	3.50	16.7	212.9
CB-227	Glamis	117.34	124.96	7.62	15.4	20.6
CB-227	Glamis	131.00	134.00	3.00	5.6	22.0
CB-244	Glamis	90.00	99.00	9.00	10.3	57.0
CB-244	Glamis	139.50	142.50	3.00	4.2	4.0
CB-244	Glamis	234.00	237.00	3.00	22.5	21.0
CB-247	Glamis	135.00	138.00	3.00	3.5	25.5
CB-247	Glamis	159.00	162.00	3.00	4.0	4.5
CB-247	Glamis	231.00	234.00	3.00	6.8	15.7
CB-247	Glamis	240.00	243.00	3.00	28.6	98.5
CB-305	Glamis	86.00	90.00	4.00	5.0	9.5
CB-305	Glamis	138.00	141.50	3.50	5.5	21.3
CB-309	Glamis	128.50	132.00	3.50	3.5	8.6
CB-309	Glamis	183.00	186.70	3.70	130.1	304.6
CB-309	Glamis	193.50	196.50	3.00	40.3	17.0
CB-314	Glamis	99.50	102.50	3.00	5.3	11.0
CB-314	Glamis	111.50	119.50	8.00	8.3	19.9
CB-314	Glamis	124.50	127.50	3.00	24.2	113.6

Hole	Company	From	To	Length (m)	Au (g/t)	Ag (g/t)
CB-314	Glamis	131.50	134.50	3.00	13.6	30.7
CB-314	Glamis	140.50	143.50	3.00	11.8	45.0
CB-314	Glamis	151.50	154.50	3.00	3.7	15.0
CB-314	Glamis	175.50	178.50	3.00	85.6	386.9
CB-314	Glamis	186.00	189.00	3.00	4.2	12.5
CB-345	Glamis	231.70	234.70	3.00	13.1	20.8
CB-345	Glamis	315.50	318.50	3.00	5.8	6.7
CB-357	Glamis	63.00	66.00	3.00	5.5	33.3
CB-357	Glamis	140.00	143.00	3.00	3.4	2.7
CB-357	Glamis	159.00	162.50	3.50	4.0	2.7
CB-357	Glamis	184.00	187.00	3.00	3.6	22.0
CB-357	Glamis	192.50	195.50	3.00	46.4	126.3
CB-357	Glamis	200.00	206.20	6.20	12.6	6.3
CB-357	Glamis	217.50	220.80	3.30	4.3	5.0
CB-362	Glamis	128.50	131.50	3.00	4.2	6.0
CB-362	Glamis	219.00	222.20	3.20	4.5	6.0
CB17-376	Bluestone	221.90	224.40	2.50	17.1	33.0
CB18-386	Bluestone	243.80	246.47	2.63	5.1	5.6
CB18-388	Bluestone	37.70	41.00	3.30	8.6	3.5
CB18-389	Bluestone	104.70	110.00	5.30	7.9	35.1
CB18-390	Bluestone	164.27	169.57	5.30	16.0	29.1
CB18-393	Bluestone	253.60	261.50	7.90	16.5	18.4
CB18-394	Bluestone	110.60	128.00	17.40	7.0	65.2
CB18-395	Bluestone	46.30	51.00	4.70	5.8	4.2
CB18-396	Bluestone	103.08	108.15	5.07	7.1	24.7
CB18-396	Bluestone	167.14	181.41	14.27	16.2	20.6
UGCB18-71	Bluestone	0.00	27.69	27.69	5.5	17.1
UGCB18-71	Bluestone	0.00	27.69	27.69	5.5	17.1
UGCB18-72	Bluestone	88.10	90.00	1.87	7.6	23.5
UGCB18-73	Bluestone	6.00	23.00	17.00	5.1	17.2
UGCB18-73	Bluestone	37.19	43.13	5.94	5.2	10.3
UGCB18-73	Bluestone	13.20	16.85	3.65	19.3	59.4
UGCB18-74	Bluestone	37.62	41.23	3.61	9.0	28.5

Hole	Company	From	To	Length (m)	Au (g/t)	Ag (g/t)
UGCB18-74	Bluestone	54.40	56.39	1.99	21.3	63.4
UGCB18-75	Bluestone	45.72	51.22	5.50	7.3	60.9
UGCB18-76	Bluestone	12.61	47.10	34.49	5.8	18.6
UGCB18-76	Bluestone	12.61	16.53	3.92	26.8	84.4
UGCB18-79	Bluestone	11.31	20.82	9.51	5.6	33.9
UGCB18-80	Bluestone	47.77	53.25	5.48	9.3	105.3
UGCB18-80	Bluestone	85.95	88.47	2.52	13.9	85.2
UGCB18-81	Bluestone	100.50	105.07	4.57	20.8	46.9
UGCB18-81	Bluestone	122.18	125.20	3.02	11.2	13.1
UGCB18-82	Bluestone	71.16	81.18	10.02	15.0	32.5
UGCB18-84	Bluestone	53.33	56.08	2.75	44.7	39.9
UGCB18-85	Bluestone	52.34	59.12	6.78	24.6	92.8
UGCB18-85	Bluestone	70.05	71.13	1.08	21.2	60.9
UGCB18-86	Bluestone	23.50	30.50	7.00	17.2	94.9
UGCB18-86	Bluestone	33.35	37.19	3.84	9.1	28.9
UGCB18-86	Bluestone	43.55	51.81	8.26	32.7	79.6
UGCB18-87	Bluestone	97.74	98.81	1.07	16.0	26.8
UGCB18-88	Bluestone	43.00	52.20	9.22	9.8	29.9
UGCB18-88	Bluestone	62.20	64.20	2.00	9.8	35.7
UGCB18-89	Bluestone	50.72	65.72	15.00	16.7	105.4
UGCB18-89	Bluestone	92.01	101.37	9.36	14.3	68.5
UGCB18-91	Bluestone	12.90	15.85	2.95	17.9	27.6
UGCB18-92	Bluestone	36.80	58.20	21.40	9.6	34.9
UGCB18-92	Bluestone	112.30	117.60	5.40	12.8	10.8
UGCB18-93	Bluestone	10.30	11.30	1.00	24.5	32.2
UGCB18-94	Bluestone	98.10	100.30	2.20	7.2	15.7
UGCB18-95	Bluestone	6.40	7.60	1.20	8.9	49.2
UGCB18-95	Bluestone	14.10	15.60	1.50	12.2	27.3
UGCB18-96	Bluestone	39.40	52.40	13.00	11.5	48.6
UGCB18-96	Bluestone	56.40	61.40	5.00	7.1	30.5
UGCB18-98	Bluestone	108.20	110.60	2.30	9.9	8.7
UGCB18-98	Bluestone	115.20	116.20	1.00	28.6	112.0
UGCB19-126	Bluestone	32.20	43.00	10.20	13.1	25.0

Hole	Company	From	To	Length (m)	Au (g/t)	Ag (g/t)
UGCB19-143	Bluestone	57.00	66.00	9.00	8.4	53.2
UGCB19-144	Bluestone	98.80	106.70	7.50	19.0	44.3
UGCB19-147	Bluestone	62.80	76.50	13.70	11.2	78.0
UGCB19-152	Bluestone	39.60	41.90	2.30	49.2	42.0
UGCB19-155	Bluestone	75.30	82.30	7.00	11.9	18.0
UGCB19-157	Bluestone	132.30	139.30	7.00	10.7	131.5
CB19-410	Bluestone	222.40	233.90	11.50	8.5	7.1
CB19-411	Bluestone	215.90	225.40	9.50	7.2	16.0
UGCB20-174	Bluestone	120.83	128.20	7.40	14.9	54.9
UGCB20-176	Bluestone	128.30	142.40	14.10	24.9	38.6
UGCB20-179	Bluestone	61.30	73.10	11.90	86.3	364.9
UGCB20-179	Bluestone	68.60	73.10	4.20	194.0	810.4
CB20-180	Bluestone	170.60	175.93	5.40	334.7	538.8
CB20-181	Bluestone	210.60	215.70	5.10	75.7	32.8
CB20-188	Bluestone	177.70	186.74	9.00	26.0	26.8
CB20-191	Bluestone	24.80	126.20	101.40	2.4	9.6
CB20-420	Bluestone	179.50	195.00	15.50	21.6	51.7
CB20-427	Bluestone	215.80	218.90	3.00	19.1	15.0
CB20-429	Bluestone	22.90	212.14	189.30	0.8	2.5
CB20-430	Bluestone	227.30	236.47	9.30	34.6	66.9
CB20-433	Bluestone	75.60	293.20	217.60	1.4	5.6
CB20-433	Bluestone	293.10	314.30	21.20	11.2	11.7
CB20-442	Bluestone	263.50	292.10	28.60	11.6	12.3
CB20-442	Bluestone	282.60	28.88	6.30	29.0	30.1
CB20-444	Bluestone	54.60	166.30	111.80	2.1	12.5
CB20-444	Bluestone	136.50	143.56	9.50	7.6	55.6
CB20-449	Bluestone	43.30	158.20	114.90	2.5	13.4
CB21-460	Bluestone	114.60	172.21	57.60	3.1	9.9
CB21-469	Bluestone	1.52	141.73	140.20	1.1	8.2
CB21-487	Bluestone	85.30	92.90	7.60	30.2	85.5

8 SAMPLE PREPARATION, ANALYSES, AND SECURITY

8.1 Sampling Method & Approach

8.1.1 Sampling Preparation, Analyses & Security (prior to November 2006)

Prior to Goldcorp taking ownership of the Project in November 2006, all previous drilling, sampling, and assaying were under the control of Glamis.

All sample data used in the Era Dorada mineral resource calculations was produced by either diamond drilling (DD) or reverse-circulation (RC) drilling. Drilling contractors were hired to supply the drilling equipment and perform the work under the direct supervision of owner-field personnel.

The Glamis drill hole program used a variable combination of sample collection, as follows:

- Double-tube HQ core in the upper reaches of the hole switching to double-tube NQ core deeper in the hole.
- RC drilling in the upper reaches of the hole above the water table and/or the anticipated mineralization zone, switching to a double-tube NQ core deeper in the hole.
- RC drilling for the entire hole.

Rotary samples collected from the 4¾ inch, face-sampling, hammer-drilled RC holes were initially collected in a five-gallon bucket. The weight was then recorded, and the sample was placed into the hopper of a Gilson splitter. The process was repeated until the entire 1.5 m sample was collected. The total weight was recorded on the sample sheet along with the sample identification and the time of day collected. Weights were only recorded for the dry portion of the drill hole. The Gilson splitter was set to split the sample into two halves, with one half retained and the other wasted. The remaining 50% was placed into the hopper again, and another 50% split was made. The two samples were placed into pre-labeled plastic sample bags, one for assay and the other for storage. An air hose and nozzle were provided for cleaning the Gilson splitter, pan, and buckets. A geologist was assigned to the rotary rig to supervise sample collection and log geology. A chip tray was created as a permanent record of each hole.

The core was collected and placed in wooden core boxes. The core was washed to obtain a clean surface for geological and geotechnical logging and placed in a covered logging facility. All core was photographed on print film. The core was sawn longitudinally with a diamond saw and half the core, on a nominal 1.5 m interval broken at lithologic boundaries, and was placed in pre-labeled plastic bags.

The other half was retained for inspection or additional tests as warranted. Splits from the core holes were shipped to a facility operated by CAS Laboratories (CAS Honduras) in Tegucigalpa, Honduras. The unused core was retained for inspection on-site.

Samples were transported from Era Dorada to the independent laboratory in Tegucigalpa, Honduras, by CAS personnel, and all sample preparation and analyses were conducted at CAS Honduras. There is no evidence from Honduran accreditation authorities indicating that CAS Laboratories (or CAS de Honduras, S. de R.L.) in Tegucigalpa was accredited as a certified assay laboratory under international standards (e.g., ISO/IEC 17025) by the Organismo Hondureño de Acreditación (OHA) or similar accreditation body.

Reject samples and pulps were stored at the CAS Honduras facility. Samples were analyzed for gold using a 30 g pulp with a fire assay atomic absorption (AA) finish. Samples that ran over 1.0 g/t Au from this method were re-analyzed for both gold and silver using a 30 g pulp fire assay with gravimetric finish.

Glamis had established a limited QA/QC program focused on coarse reject and pulp reject checks. A frequency of one in 20 pulps was systematically submitted to the Chemex Laboratories in Nevada for gold and silver analysis in addition to coarse rejects.

The drill samples were initially quick-logged to locate and mark significant changes in volcanic stratigraphy. Each volcanic unit was then described, and the location of the structure and their orientations, the percentage of quartz veining, and the type of alteration were recorded.

Standard logging conventions were used to capture information from the drill sample. Detailed, daily logging was transcribed onto log sheets and independently entered into Excel spreadsheets. The geologist checked data entry before the data was merged with the main database.

Detailed core logging was done by capturing data in four tables: lithology, alteration, Sulfide type, and geotechnical information. Lithology was captured using standardized abbreviations. The alteration was captured as a numeric value corresponding to the alteration type. The visible Sulfide types were captured as a total modal percentage and as relative ratios. Structural data was captured in the “comments/structures” table in the database, as the type and angles taken related to the core axis are displayed in an area as a graphical representation. The geotechnical data recorded rock quality designation (RQD) data for the core portion of the hole.

All independent laboratories used in the Project employed quality control procedures and protocols that included duplicates, standard reference materials, and blanks. These were available to Glamis but were not included in assay reports.

8.1.2 Sample Preparation, Analyses & Security (Goldcorp 2010 through 2012)

Drilling completed by Goldcorp (2010 to 2012) was a combination of surface and underground diamond core drilling. Drills were operated by both contract and Goldcorp personnel. The Goldcorp underground drill rig (Boart Longyear LM-75) was used on the surface and converted for underground drilling. Drill holes were developed by drilling a larger diameter (HQ) core at the early stage of the hole, decreasing to NQ and/or BQ if the drilling conditions became difficult.

Drill recovery was high, and by utilizing several drill core sizes, Goldcorp was able to ensure drill hole target completion. Drill hole collar surveys were completed using a GPS Trimble system (UTM NAD 27 Zone 16N). In-hole drill surveying for azimuth and dip was completed using the Reflex EZ-Shot system approximately every 50 m along the drill hole.

Drill cores (surface and underground) were stored in wooden labeled boxes from the drill and transported to the surface core logging facility at the Era Dorada surface core facility.

Technicians first prepared the core boxes by reviewing drill hole depth tags and reassembling broken sections (from zones of poor recovery).

Core logging to identify lithology, alteration, RQD, and sampling selection for core sawing was completed by geologists or technicians under the direction of the geologist. Sampling was also completed by Goldcorp personnel, which included technicians and geologists. The typical sample lengths were 1.0 to 1.5 m with maximum lengths of 2.0 and 3.0 m; sample lengths were based on the lithology and alteration. Logs and the sample database indicated that low-grade and high-grade gold and silver samples were of the same lengths and were not broken out separately or collected in a way that caused sample bias. Samples were collected along the footwall, mineralized zones, and hanging walls without breaks in sampling. Blanks were inserted by Goldcorp personnel when a core sample was submitted. All data was initially collected on paper logs and later transferred to Excel files. This data was then entered in MapInfo™ and MineSight™ software for geological modeling.

The core selected for analysis was transported to Inspectorate Laboratories in Guatemala City for sample preparation. Samples were prepared at the Inspectorate (Guatemala) by crushing and pulverizing the drill core to 100 g pulp samples.

One pulp sample was sent to Goldcorp's Marlin Mine for gold assaying (fire assay with AA or gravimetric finish) and silver assaying (AA or AA with gravimetric finish). The second pulp sample was sent to the Inspectorate Laboratory in Reno, Nevada, for gold assaying (fire assay with AA or gravimetric finish) and silver assaying (AA or AA with gravimetric finish). The Marlin Mine assays were completed quickly, which assisted the geologists in developing the drilling program. The Inspectorate assays were used for the purposes of mineral resource modeling and estimation.

Inspectorate America Corporation (located in Sparks, NV, near Reno) has historically been used by mining companies as an independent analytical laboratory for mineral sample analyses and check assays (e.g., fire assays for gold) in technical reports. Inspectorate (now part of Bureau Veritas) has been described in mining technical reports as an independent laboratory that held ISO/IEC 17025:2005 accreditations.

The QA/QC program employed at the Project was under the direction of Goldcorp. Blank samples were inserted by Goldcorp geologists prior to shipping to the Inspectorate at a frequency of 1 in 25 sample submissions. No duplicates of coarse rejects or standards were included in the QA/QC program at Era Dorada; however, it was recommended that duplicates of the coarse rejects be analyzed and compared and that standards be inserted into the QA/QC sample stream for future drilling campaigns. All analytical results were provided to Goldcorp staff and stored first in Excel and later in MapInfo™ and MineSight™ software. All half-core samples collected by both Goldcorp and Glamis are stored adjacent to the core logging facility on the Project site. The Era Dorada site is fully controlled by perimeter fencing and security. All samples removed from the site were under the control of Inspectorate Laboratories.

8.1.3 Sampling Preparation, Analyses & Security (Bluestone 2017 to 2021)

The drill core from the surface and underground was stored in labeled wooden boxes (Figure 8-1) at the drill site and transported to the surface core logging facility. Before core splitting and logging commence, the drill core was systematically photographed in high resolution using a tripod-mounted camera and digitally archived for reference as part of the drill database.

Figure 8-1: Example of Core Box Photography



Source: Bluestone, 2019.

Logging and sampling were undertaken on-site at Era Dorada by company personnel under a QA/QC protocol developed by Bluestone. Technicians first prepared the core boxes by reviewing drill hole depth tags, reassembling broken sections, and photographing the core. Core logging to identify lithology, alteration, RQD, and sampling selection for core sawing was completed by technicians under the direction of the geologist. Sampling was also completed by Bluestone technicians. The typical sample lengths are 1.0 to 1.5 m with a minimum sample width of 1 m and maximum lengths of 2.0 m; sample lengths were based on the lithology and alteration. Samples are collected along the footwall, mineralized zones, and hanging walls without breaks in sampling. All data was initially captured on paper logs and later transferred to Microsoft Excel. The data was then entered into MapInfo™ and MineSight™ software for geological modeling.

Specific gravity readings of all representative lithologies and vein material were taken during the various drill campaigns using the displaced water method. Samples were sealed with paraffin wax to account for natural voids/vugs.

A total of 591 channel samples were taken along representative veins exposed in the side walls of the Era Dorada underground tunnels using a portable rock saw. The sampling was undertaken across and perpendicular to the mineralized structures wherever possible and carefully surveyed with XYZ coordinates for use in 3D modeling. The samples were subject to the same QA/QC protocols as the drill core and were deemed suitable for use in calculating resources. Figure 8-2 shows a saw-cut channel sample across a mineralized vein in the South Ramp of the Era Dorada underground workings.

Figure 8-2: Example of Underground Channel Sample



Source: Bluestone, 2019.

Samples were transported in security-sealed bags to Inspectorate Laboratories in Guatemala City for sample preparation until March 2020 and thereafter to Inspectorate Laboratories in Managua due to the closure of the Guatemalan facility. Samples were prepared at the Inspectorate by crushing and pulverizing the drill core down to 85%, passing -75 µm. Pulps were weighed and individually packaged into 100 g envelopes and shipped for analysis. Both coarse rejects and pulp were stored for future use and utilized in Bluestone's QA/QC program. All half-core and coarse rejects are stored adjacent to the core logging facility on the Project site. The Era Dorada site is fully controlled by perimeter fencing and security.

Pulps are shipped for regular and QA/QC analysis to Inspectorate Laboratories (a division of Bureau Veritas) in Reno, Nevada, USA, and ALS Chemex in Vancouver, BC, Canada, respectively. Both are independent ISO 17025-accredited laboratories. Gold and silver were analyzed by a 30 g charge with atomic absorption with gravimetric finish for values exceeding 5 g/t Au and 100 g/t Ag.

All analytical results were provided to Bluestone by respective laboratory secure servers in Excel, .csv, and .pdf formats (certificates). Bluestone database files are stored and managed in Access and Excel formats before being transferred to MapInfoTM and MineSightTM software.

During Q3 and Q4 2020, the Cerro Blanco database was transitioned to the AcQuire/GMSuite platform, providing an enhanced, secure, and high standard of data management.

8.2 Quality Assurance & Quality Control

8.2.1 QA/QC Performance & Discussion for Samples prior to 2017

Field blanks of non-mineralized material were inserted into the sample series every 25 samples (4%) to test for any potential carry-over contamination that might occur in the crushing phase of sample preparation due to poor cleaning practices. A total of 1,390 blanks were analyzed, with 558 performed at Inspectorate Laboratories, 302 at CAS Honduras, and 530 at the Marlin Mine laboratory. An analysis of the Inspectorate blanks resulted in five fails or 0.01%, with one re-failing on resample. This appears to be the result of sample misclassification as both the original and resample are relatively high grade. The CAS Honduras results showed eight fails or 0.03%, with four of those failing on resample. There may have been some cleaning issues at CAS Honduras, although it was not widespread or significant. The blanks from the Marlin Mine laboratory resulted in 14 fails or 0.03%, which is not significant. Considering that the Marlin Mine assaying was utilized for fast turnaround to guide the program and not for resource estimation purposes, this fail rate does not pose an issue.

Core duplicate samples were used to evaluate analytical precision and to determine if any biases exist between laboratories that may affect the overall assay database. The core duplicate samples were quarter-spilt cores sampled on-site and sent to Inspectorate Laboratories and CAS Honduras. A total of 1,060 samples with gold values >2 g/t were selected in the drill hole database through hole CB-222. Of those, a total of 797 samples were submitted for check analyses, with 618 samples being submitted to the Inspectorate for checks of original CAS Honduras analyses, while 179 samples were submitted to CAS Honduras for checks of original Inspectorate analyses. The 618 Inspectorate duplicate check samples show the CAS Honduras original samples to be 3% higher in gold and 16% higher in silver on an individual basis and 3% and 2.8% higher in gold and silver, respectively, on an overall basis.

The 179 CAS Honduras duplicate check samples show the Inspectorate original samples to be 1.5% lower in gold and 27% lower in silver on an individual basis and 6.8% and 11.4% lower in gold and silver, respectively, on an overall basis.

Duplicate analyses from both labs show high variation in individual gold values, potentially attributable to the nugget effect, particularly for higher-grade samples. However, on average, the samples show a better correlation, which has greater implications on a global or resource scale. The CAS Honduras check samples appeared to show a relatively small grade bias.

Standards are used to test the accuracy of the assays and to monitor the consistency of the laboratory over time. Neither Glamis nor Goldcorp employed the use of standards. It was recommended that a QA/QC program be implemented during all future drill programs that include the insertion and analysis of standards, blanks, and duplicates, as well as umpire assays.

8.2.2 QA/QC Performance & Discussion of Results (Bluestone 2017 to 2021)

Since 2017, Bluestone has implemented a comprehensive QA/QC program employing industry standards and best practices for all its drill core and channel sampling. This includes the insertion of blind-certified reference materials (blanks and standards) into the sample stream, in addition to field blanks. Furthermore, duplicate analysis of pulps and coarse rejects was performed at a second laboratory to independently assess the analytical precision and accuracy of each sample batch as they were received from the laboratory. Additionally, pulp and coarse rejects were systematically submitted to ALS Chemex Laboratories in Vancouver for check analysis and additional quality control.

A total of 7,652 control samples (Table 8-1) were assigned for QA/QC purposes, accounting for approximately 20% of the total samples taken during the program.

Table 8-1: Quantity of Control Samples by Type (Bluestone 2017 to 2021)

Control Type	Number
Standards	1,602
Field Blanks	685
Pulp Blanks	859
Pulp and Coarse Reject Duplicates	4,506
Total	7,652

Source: Bluestone, 2021.

Standards are used to test the accuracy of the assays and to monitor the consistency of the laboratory over time. A variety of certified standards of various gold grades were purchased from CDN Laboratories (Table 8-2) and inserted by the logging geologists.

Table 8-2: Summary of Standards (Bluestone 2017 to 2021)

Control Sample	Au PPM	Standard Deviation	Analysis
CDN-GS-16	16.48	0.315	Fire Assay Gravimetric
CDN-GS-11B	11.04	0.44	Fire Assay Gravimetric
CDN-GS-6F	6.79	0.15	Fire Assay Gravimetric
CDN-GS-6E	6.06	0.16	Fire Assay Gravimetric
CDN-GS-5T	4.76	0.105	Fire Assay AA Finish
CDN-GS-1W	1.063	0.038	Fire Assay AA Finish
CDN-GS-1T	1.08	0.05	Fire Assay AA Finish
CDN-GS-1X	1.299	0.06	Fire Assay AA Finish
CDN-BL-10	<0.01	-	Fire Assay AA Finish
FIELD BLANKS	<0.01	-	Fire Assay AA Finish

Source: Bluestone, 2021.

Field blanks are non-mineralized materials sourced locally that are inserted into the sample series every 20 samples (5%). Field blanks are inserted to test for any potential carry-over contamination that might occur in the crushing phase of sample preparation due to poor laboratory cleaning practices.

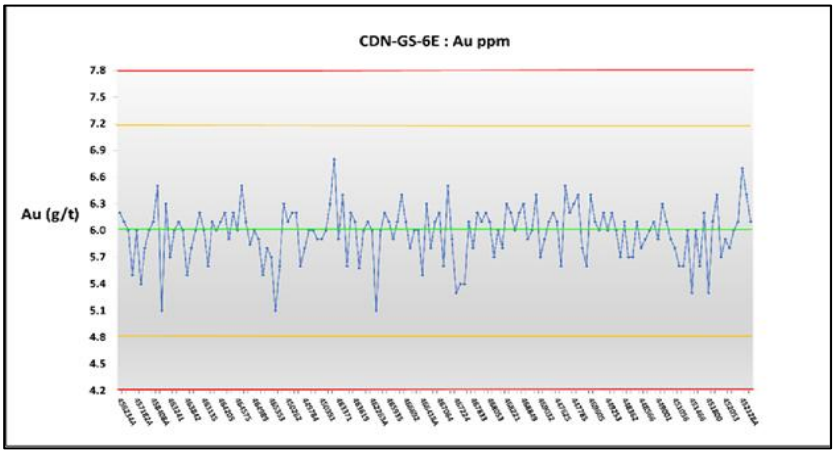
Duplicate analysis of pulps and quarter-core are used to evaluate the analytical precision and to determine if any biases exist between laboratories. Duplicate analysis of coarse rejects is used to analyze preparation errors. Table 8-3 shows the QA/QC sample insertion rate.

QA/QC assay results were checked by a Bluestone database QA/QC manager on a batch-by-batch basis for analytical or batch errors. No evidence of obvious analytical bias was noted. Figure 8-3 shows a control plot for standard CDN-GS-6E.

Table 8-3: Bluestone QA/QC Sample Insertion Rates

Batch Size – 45 Samples	Minimum Insertion Rates	Notes
Standards	1 every 20	Inserted according to the estimated grade of mineralization before, within, or immediately after a mineralized interval. Insertion at regular intervals avoided.
Field Blanks	1 every 20	Usually inserted at the end of mineralized runs to measure carry-over
Pulp Blanks	1 every 20	Usually inserted at the end of mineralized runs to measure carry-over
Pulp Duplicates	1 every 20	Undertaken at the second laboratory with the same analytical technique. High- and low-grade mineralized samples are usually chosen
Coarse Duplicates	1 every 20	Normally choose mineralized samples, used to measure laboratory sample preparation

Figure 8-3: Batch Plot of Standard CDN-GS-6E

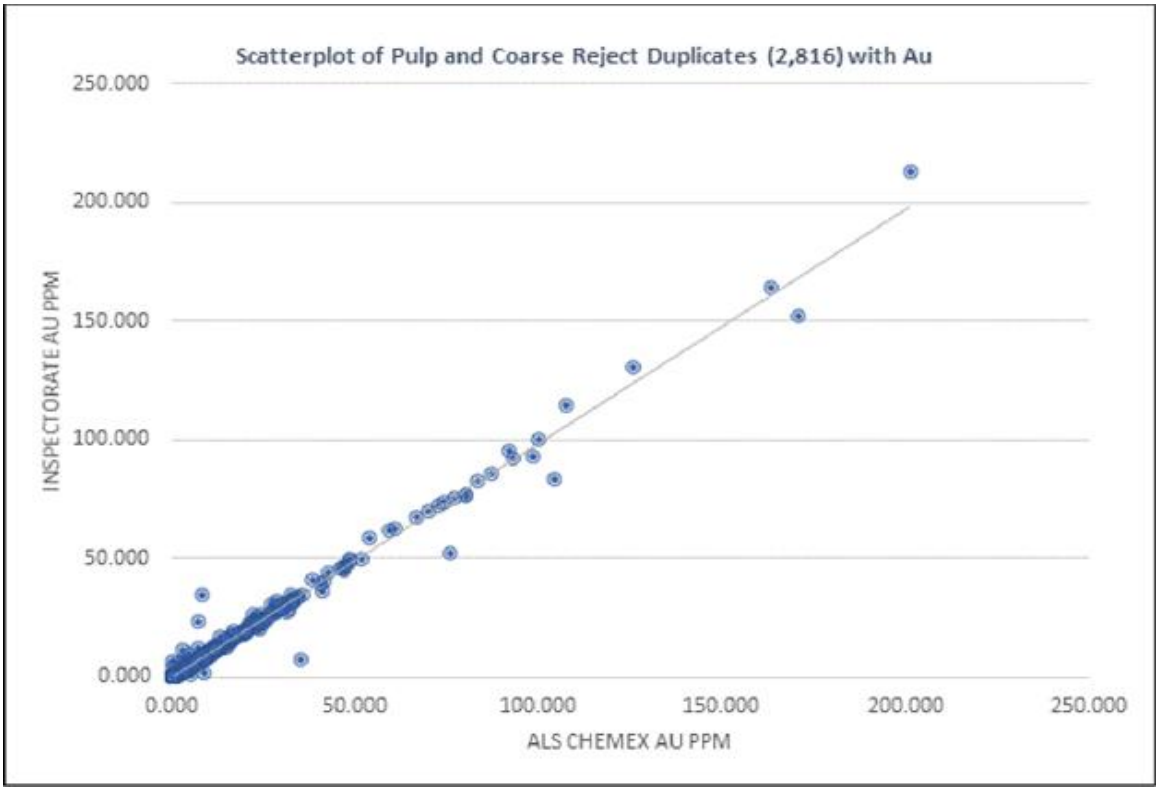


Source: Bluestone, 2020.

Except for one standard, the performance of the control samples was very good, reflecting the overall high quality of the analysis. Standard CDN-GS5T (4.76 g/t Au) utilized early in the Bluestone drill program plotted consistently along the highest acceptable threshold for fire assay with instrumental finish. Check analysis at both the Inspectorate and ALS Chemex laboratories gave similar results. As lower-grade CRM / blanks and the laboratories' internal QA/QC procedures ruled out any calibration issues, the use of this particular standard was discontinued.

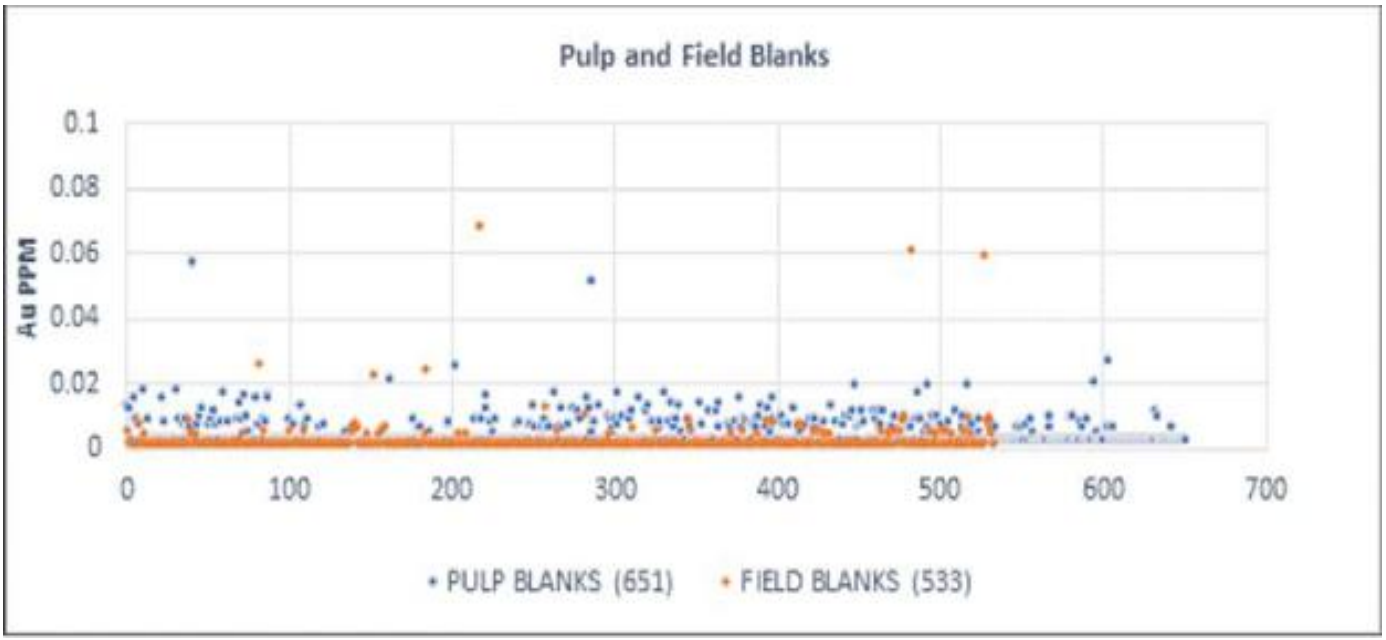
Duplicates of pulp and coarse rejects were sent to ALS Chemex in Vancouver for check gold analysis with the analysis at the principal laboratory, Inspectorate Laboratories in Reno. As shown in Figure 8-4, the results indicate a very good correlation at both low and high gold levels and excellent reproducibility between the two laboratories, with a correlation coefficient of 0.993. The results can be interpreted as a reflection of the micron-sized nature of the gold and the lack of coarse, nuggety gold in the Era Dorada deposit. Analyses of both pulp and field blanks (Figure 8-5) consistently yielded gold values near or below the detection limit of the primary laboratory. No sample contamination was detected.

Figure 8-4: Plot of pulp & coarse reject duplicates (Bluestone 2017-2021)



Source: Bluestone, 2021.

Figure 8-5: Pulp & Field Blanks (Bluestone 2017 to 2021)



Source: Bluestone, 2021.

It is the opinion of the QP, Garth Kirkham, P. Geo., that the sampling preparation, security, analytical procedures, and quality control protocols used by Bluestone are consistent with generally accepted industry best practices and are, therefore, reliable for the purpose of resource estimation.

The Qualified Person is of the opinion that the sample preparation, security, and analytical procedures are adequate for the purpose of mineral resource estimation as presented within this Technical Report Summary.

9 DATA VERIFICATION

9.1 Introduction

A site visit is a critical part of the due diligence process that ensures mineral disclosures are accurate, independently verified, and based on sound technical observations. Multiple site visits were conducted by several of the QP, as detailed in Section 2.2. The purpose of a site visit in the context of Securities and Exchange Commission Regulation S-K Subpart 1300 is to provide qualified third-party review and verification of the geological, technical, and operational aspects of a mineral property. These site visits consisted of underground investigations of mineralized and non-mineralized headings, as well as an inspection of the surface core logging, sampling, storage areas, and existing infrastructure.

9.2 Geology, Drilling & Assaying

Garth Kirkham, P. Geo., has been involved with the property since its acquisition in early 2017, when he performed the initial due diligence and authored the updated resource estimate for Bluestone. Mr. Kirkham first visited the property on May 8, 2017, to validate all aspects. The site visit included an inspection of the property, offices, underground vein exposures, core storage facilities, water treatment plant, and stockpiles, and a tour of major centers and the surrounding villages most likely to be affected by any potential mining operation.

The QP performed an independent verification of the data, observations, and interpretations for Era Dorada. This included confirmation sampling procedures, drilling methods, core logging, and QA/QC practices. Inspected drill cores, outcrops, underground workings, and surface trenches to corroborate reported geological models, historical data, and reporting. Additionally, this involved a thorough examination of mining infrastructure access, along with an extensive review of environmental and social conditions. Identification and evaluation of risk in support of the mineral resource/ estimates.Since 2017, Mr. Kirkham has visited the property numerous times for extended periods to develop and implement data gathering and sampling methods and procedures. He also worked with Bluestone geologists to develop drill programs and to supervise interpretation and modeling efforts in addition to creating and implementing QA/QC procedures.

From September 21 to 22, 2017, Mr. Kirkham inspected the progress of the recommended historic drill core rehabilitation program and initiated structural studies.

From April 24 to 28, 2018, Mr. Kirkham's site visit focused on advancing the planning of sampling and drilling along with supporting lithological and structural modeling.

From February 16 to 22, 2020, Mr. Kirkham provided guidance on the planning and development of advanced drilling and sampling, as well as grade vein modeling.

From January 10 to 15, 2021, Mr. Kirkham assisted with validating drill and sample data, refining high-grade models, reviewing low-grade models, and providing guidance for the finalization of the open pit bulk tonnage resource scenario.

Continued data validation and verification processes have not identified any material issues with the Era Dorada sample and assay data. Mr. Kirkham is satisfied that the assay data is of suitable quality to be used as the basis for this resource estimate.

During Q3 and Q4 2020, the Era Dorada drill and assay database was switched over to the AcQuire - GMSuite platform hosted by CSA Global, providing an enhanced and more secure standard of data management.

Mr. Kirkham is confident that the data and results are valid and can be relied upon. Mr. Kirkham is also confident that the methods and procedures used are reliable. It is the opinion of Mr. Kirkham that all work, procedures, and results have adhered to best practices and industry standards.

The Qualified Person is of the opinion that the data is adequate for the purposes used within this Technical Report Summary.

9.3 Metallurgical Data and Test Results

Metallurgical test data was verified through a review of previous testwork reports as no current metallurgical testing has been performed. Metallurgical testing used for process design has been completed at specialist laboratories Pocock Industrial and Base Metallurgical Laboratories Ltd.. Each laboratory has their own QA/QC procedures, which they adhere to in performing their testing on samples. All metallurgical data was verified and is adequate for this technical report as required by S-K 1300 guidelines.

There have been no limitations on the author on his verification of any of the data presented in this report. The author's opinion is that all data presented in this report are adequate for the purposes Mineral Resource estimation of this report and is presented so that it is not misleading.

10 MINERAL PROCESSING AND METALLURGICAL TESTING

10.1 Introduction

Historical metallurgical testing was performed on Era Dorada samples by Kappes, Cassiday & Associates ("KCA") between 1999 and 2012, with auxiliary testing being carried out by SGS Lakefield Research Ltd., Carson GeoMIn inc., Pocock Industrial Inc., Phillips Enterprises Inc. and CyPlus GmbH. The most recent test program, completed in 2018 in support of this Feasibility Study, was carried out at Base Metallurgical Laboratories Ltd. ("BaseMet") in Kamloops, BC. A full breakdown of the results for each metallurgical test program can be found in Table 10-1.

Table 10-1: Metallurgical Testwork Summary

Year	Laboratory/Location	Laboratory Certification	Relationship to the Registrant	Testwork Performed
1999	Kappes, Cassiday & Associates	None listed on website	Independent	Cerro Blanco Project, Results of Cyanide Leach Tests
2000	Kappes, Cassiday & Associates	None listed on website	Independent	Cerro Blanco Project, Results of Cyanide Bottle Roll Tests
2000	Kappes, Cassiday & Associates	None listed on website	Independent	Cerro Blanco Project, Bottle Roll Tests
2002	Kappes, Cassiday & Associates	None listed on website	Independent	Cerro Blanco Project, Results of Leaching Tests and Gravity Concentration Tests
2005	SGS Lakefield Research Ltd	Conforms to the requirements of the ISO/IEC 17025 standard for specific registered tests. I	Independent	Cerro Blanco North Zone Samples for Met Testing at SGS Lakefield
2005	Kappes, Cassiday & Associates	None listed on website	Independent	Cerro Blanco Project
2005	Carson GeoMIn Inc	No information available	Independent	Mineralogy of Ore Composites and Related Cyanide Tailings from the Cerro Blanco Gold Project

Year	Laboratory/Location	Laboratory Certification	Relationship to the Registrant	Testwork Performed
2006	Kappes, Cassiday & Associates	None listed on website	Independent	Cerro Blanco Project
2006	Kappes, Cassiday & Associates	None listed on website	Independent	Cerro Blanco Project
2011	Phillips Enterprises LLC	-	Independent	Comminution Tests, Cerro Blanco
2011	Pocock Industrial Inc	None listed on website	Independent	Sample Characterization and PSA, Flocculant Screening, Gravity Sedimentation, Pulp Rheology, Vacuum Filtration and Pressure Filtration Studies Conducted for Kappes, Cassiday & Associates Cerro Blanco Project
2012	Kappes, Cassiday & Associates	None listed on website	Independent	Cerro Blanco Project, Report of Metallurgical Testwork, January 2012
2018	Base Metallurgical Laboratories Ltd	None listed on website	Independent	BL0246: Generation of Cyanide Detox Tailings – Cerro Blanco Project

This section will only discuss the results used as the basis for the process design and recovery method presented in Section 14. This discussion will include a summary of the results from the BaseMet (2018) test program, as well as key historical test results related to comminution and solid/liquid separation.

Based on the results from BaseMet (2018), gold and silver doré can be produced with a target primary grind size of 80% passing (P_{80}) of 53 µm followed by gravity concentration, a 36-hour cyanide leach, 6-hour carbon-in-pulp (CIP) adsorption, carbon desorption and refining process. For the Global Composite, this recovery method achieved precious metal recoveries of 96% Au and 85% Ag.

10.2 Metallurgical Testwork

10.2.1 Legacy Testwork

All metallurgical testing referenced in this study is considered legacy. Only relevant metallurgical testing to the current process design will be described in this section of the report.

10.2.2 KCA (2012) Sample Selection

In April 2011, KCA received six pallets from the Era Dorada Project, previously Cerro Blanco Project. The pallets contained a total of 55 cloth bags containing half split HQ and PQ drill core material from five samples. A portion from each sample was then utilized in the generation of a Master Composite. The amount of each individual sample utilized

to generate the Master Composite was determined by then Cerro Blanco personnel. The head assay results for each sample are summarized in Table 10-2. Supporiting lithological descriptions for the samples listed in Table 10-2 and Table 10-3 are provided in Section 6.3.1.

Table 10-2: Head Assays for KCA (2012)

KCA Sample No.	Description	Average Au Assay (g/t)	Average Ag Assay (g/t)
48901	MbT	9.40	46.25
48902	Mcv	4.47	5.79
48903	Svc	6.52	44.71
48904	Msc	5.07	38.79
48905	Cbx	4.59	18.06
48907	Master Composite	7.70	37.86

10.2.2.1 Phillips Enterprises (2011) Comminution Results

After bottle roll leach testing, portions of material from each of the individual samples were submitted to Phillips Enterprises LLC in Golden, Colorado for comminution testing. Testwork was completed to determine the Bond ball mill and rod mill work indices for grinding specific energy calculations, and the Bond abrasion index for estimating grinding mill consumables. The results of the testwork are summarized in Table 10-3. The averages from the five samples were used as the design criteria to size the primary and secondary ball mills and to estimate mill operating costs.

Table 10-3: Comminution Test Results from Phillips Enterprises (2011)

KCA Sample No.	Description	Bond Rod Mill Work Index (kWh/t)	Bond Ball Mill Work Index (kWh/t)	Bond Abrasion Index (g)
48901	MbT	17.08	20.27	0.1931
48902	Mcv	13.91	16.37	0.1035
48903	Svc	18.26	22.24	0.3280
48904	Msc	16.90	21.45	0.3286
48905	Cbx	15.52	18.95	0.2461
Average		**16.33**	**19.86**	**0.2399**

For a complete description of the Lithologies tested, see Table 11-1.

10.2.2.2 Pocock Industrial (2011) Solid/Liquid Separation Results

Two bulk leach tests were conducted on a milled portion of the Master Composite sample for solid/liquid separation test purposes. The combined tailings sample was subject to detoxification and the slurry was packaged and submitted to Pocock Industrial, Inc. (Pocock) for detoxification analysis and solid/liquid separation testing. KCA also delivered a ground sample to Pocock that had not been leached.

Solid/liquid separation tests were conducted on the ground sample and tailings samples delivered from KCA. The purpose for conducting the testwork was to generate data for solid/liquid separation equipment design and sizing criteria. All testing was conducted by Pocock Industrial at their laboratory facility located in Salt Lake City, Utah during October 2011.

The following are the key findings from the study that were used in the process design:

- The minimum flocculant dose anticipated varied by individual sample and thickener type or application, but was in the overall range of 35 to 55 g/t for the ground sample and 30 to 55 g/t for the tailings samples in the tested pH range,
- For conventional thickener sizing, Pocock recommended a minimum unit area design basis of 0.30 to 0.40 $m^2/t/d$ for ground sample, and 0.25 to 0.35 $m^2/t/d$ for the tailings sample,
- Dynamic thickening tests conducted on the samples indicated a hydraulic net feed loading rate design basis in the maximum range of 3.1 – 4.3 $m^3/m^2 \cdot h$ for ground sample, as well the tailings sample to achieve optimal performance,
- The overall maximum underflow density range for the Leached and Detoxed material was 53 to 57% solids by weight based on fully sheared data (but this could be limited to 53 to 55% solids by weight with rake torque considerations based on un-sheared data), and
- Pressure filter testing for the ground and tailings samples acieved a filtration rate of 174 $kg/h/m^2$ with 18.3% moisture at pH 8.5 using a 30 mm depth filter chamber.

10.2.2.3 BaseMet (2018) Test Program

The primary objective of the test program was to generate tailings from a bulk sample for downstream geotechnical and environmental studies. The bulk sample was separated into north and south areas of the deposit and prepared to create two bulk composites.

e north and south were tested using the optimized flowsheet to confirm gold and silver extractions. Limited process optimization testwork was also conducted to further the understanding and optimization of the processing characteristic in support of this Feasibility Study.

The study included sample preparation, interval assaying, gravity concentration, cyanide leach optimization and bulk cyanide leaching to produce material for continuous cyanide destruction testwork. A single Global Composite was

constructed from drill core intervals to carry out the gravity concentration, cyanide leach and cyanide destruction testing.

Samples were received on April 6, 2018 by BaseMet in two forms. Approximately 90 kg arrived as cut drill core (1/4 and 1/2 core) and about 590 kg arrived as bulk rock. In total, 180 individual interval samples were received.

The Global Composite was created using the individual drill core. The drill-core was initially inspected and weighed. Each interval was then individually stage crushed to a nominal 3.36 mm (6 mesh). The crushed material was blended, and a 250 g sample was riffled split and pulverized for subsequent assaying.

A representative sub-sample of the Global Composite was removed during sample preparation and pulverized. The head assay results are shown in Table 10-4.

Table 10-4: Head Assays for BaseMet (2018)

Composite	Au (g/t)	Ag (g/t)	Cu (%)
Global Composite 1	4.21	23	0.007
Global Composite 2	5.65	21	0.007
Average	4.93	22	0.007

1.1.2.1.1 Gravity Concentration Results

One-kilogram test charges were ground in a laboratory rod mill to three target P_{80} grind sizes of 50 µm, 75 µm, and 100 µm before passing through a laboratory Knelson MD-3 centrifugal gravity concentrator. Knelson concentrates were then panned to reject entrained gangue, targeting a 0.1% to 0.5% mass recovery. Gravity concentration results are presented in Table 10-4 and indicate moderate gravity recoverable gold. Gravity results follow the general trend of improving performance as the grind size is reduced from 100 µm to 50 µm with the test series average recovery of 19% Au. Based on these values, a gravity concentration circuit was included. Expected plant scale gravity mass recoveries would be around 0.05%. Actual plant scale gravity gold and silver recoveries would be less than projected in testing.

Table 10-5: Gravity Concentration Results for BaseMet (2018)

Test No.	Test Type	Grind Size (µm)	Mass Recovery (%)	Au Recovery (%)	Ag Recovery (%)
4	Gravity / Leach	50	0.317	22.5	6.3
10	Gravity / Leach	50	0.186	21.1	6.9
11	Gravity / Leach	50	0.230	15.1	6.5
17	Gravity / Leach	53	0.319	20.8	9.4
18	Gravity / Leach	53	0.301	17.9	8.5
19	Gravity / CIL	53	0.274	16.7	6.0
20	Gravity / CIL	53	0.326	21.7	9.1

Test No.	Test Type	Grind Size (µm)	Mass Recovery (%)	Au Recovery (%)	Ag Recovery (%)
21	Gravity / Leach	75	0.185	16.3	5.4
2	Gravity / Leach	75	0.239	29.8	16.5
6	Gravity / Leach	75	0.270	14.7	5.4
7	Gravity / Leach	75	0.314	20.7	6.4
8	Gravity / Leach	75	0.398	20.0	6.6
3	Gravity / Leach	75	0.480	17.7	10.2
12	Gravity / Leach	75	0.291	15.9	4.1
5	Gravity / Leach	100	0.534	15.6	10.2

1.1.2.1.2 Bottle Roll Leach Results

Leaching testwork was carried out using two different methods. The first method used direct cyanide leaching on fresh milled product, the second on gravity tailings. All tests were completed in closed bottles on rolls, allowing constant agitation of the pulp as the sample leached for 72 hours. Cyanide levels, dissolved oxygen (DO) and pH were monitored and controlled throughout each test. Kinetic leach solution sampling was done at 2, 6, 24, 48 and 72 hours.

The optimization testwork focused on the effect of leach time, pre-oxidation, lead nitrate addition and primary grind size on gold recovery and leach kinetics. The results are summarized in Table 10-6.

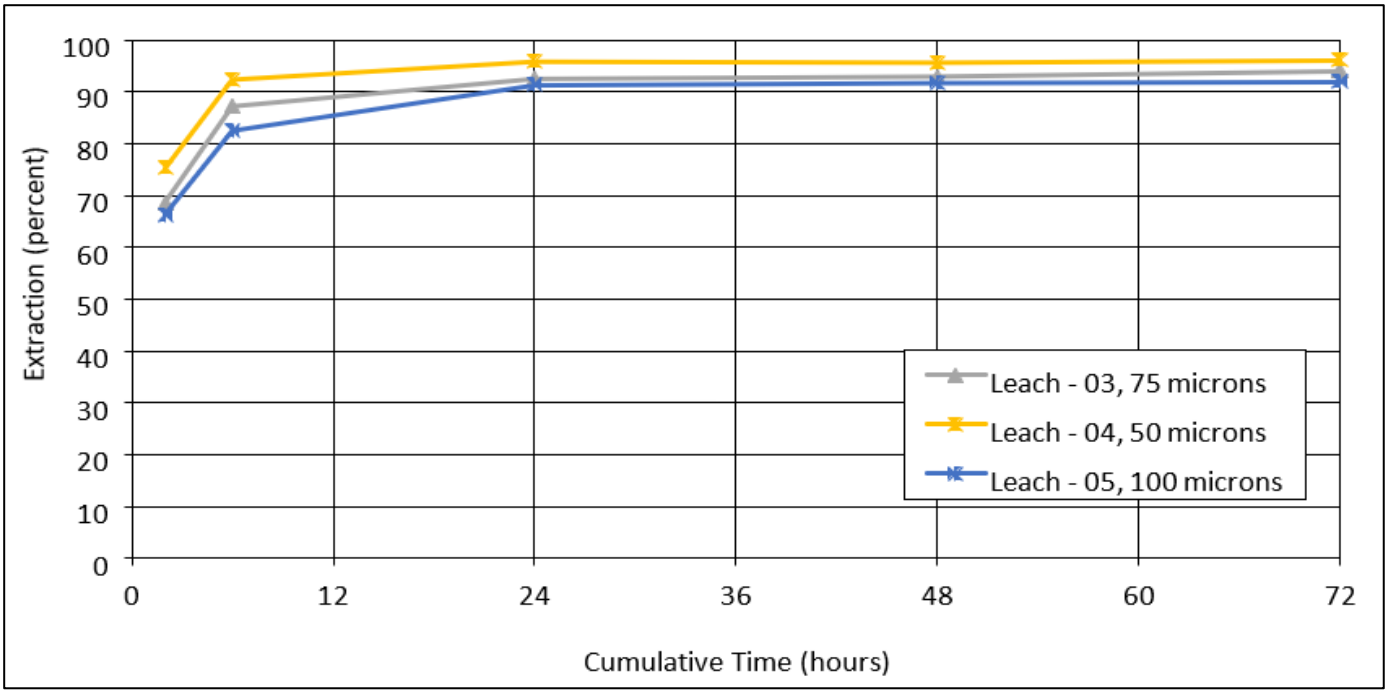
Table 10-6: Bottle Roll Leach Results for BaseMet (2018)

Test No.	Grind Size (µm)	Consumption		Gravity Au Recovery (%)	Cumulative Gold Extraction (%)				Final 72 h Recovery	
		NaCN (kg/t)	Lime (kg/t)		2 h	6 h	24 h	48 h	Au (%)	Ag (%)
4	50	0.84	0.87	22.5	75.3	92.4	95.9	95.7	96.1	92.4
10	50	0.36	1.17	21.1	78.9	92.4	94.4	95.7	97.5	69.6
11	50	0.52	1.02	15.1	81.5	92.7	94.0	96.5	97.3	78.6
17	53	0.52	1.22	20.8	91.9	93.2	94.2	95.2	95.9	88.4
18	53	0.50	1.12	17.9	82.2	91.6	96.6	95.1	96.1	92.3
19	53	0.86	1.33	16.7	87.7	94.9	98.1	97.4	94.5	70.8
20	53	0.60	1.50	21.7	80.7	90.9	94.1	92.8	96.3	69.7
21	53	0.28	0.96	16.3	80.9	90.1	91.5	91.6	94.7	67.2
2	75	0.82	0.86	29.8	61.2	78.5	91.9	94.1	94.7	86.9
6	75	0.82	0.84	14.7	82.9	90.4	92.4	93.1	94.2	82.9
7	75	1.00	0.71	20.7	77.8	90.8	92.9	93.7	94.4	84.2

Test No.	Grind Size (µm)	Consumption		Gravity Au Recovery (%)	Cumulative Gold Extraction (%)				Final 72 h Recovery	
		NaCN (kg/t)	Lime (kg/t)		2 h	6 h	24 h	48 h	Au (%)	Ag (%)
8	75	0.46	0.82	20.0	75.0	88.3	92.7	93.2	93.6	83.1
3	75	0.76	0.89	17.7	68.6	87.2	92.5	93.0	94.0	93.2
12	75	0.20	1.00	15.9	80.6	89.3	91.7	93.8	95.6	65.2
5	100	0.58	0.71	15.6	66.3	82.4	91.2	91.7	91.9	82.7
1	75	2.98	0.50	No Gravity	3.2	10.1	88.4	92.2	93.1	84.7
9	75	0.90	0.77	No Gravity	67.4	86.5	92.6	92.1	94.4	86.3

A decrease in grind size was found to improve gold and silver exractions. As grind size decreased from a P_{80} of 100 µm (Test #5) to a P_{80} of 50 µm (Test #4), gold and silver extractions improved by 4.2% and 9.7% respectively (72-hour leach time). A P_{80} grind size of 50 µm was selected for design. Figure 10-1 shows gold extraction versus time at the different grind sizes.

Figure 10-1: Effect of Grind Size on Gold Extraction



Source: BaseMet, 2019.

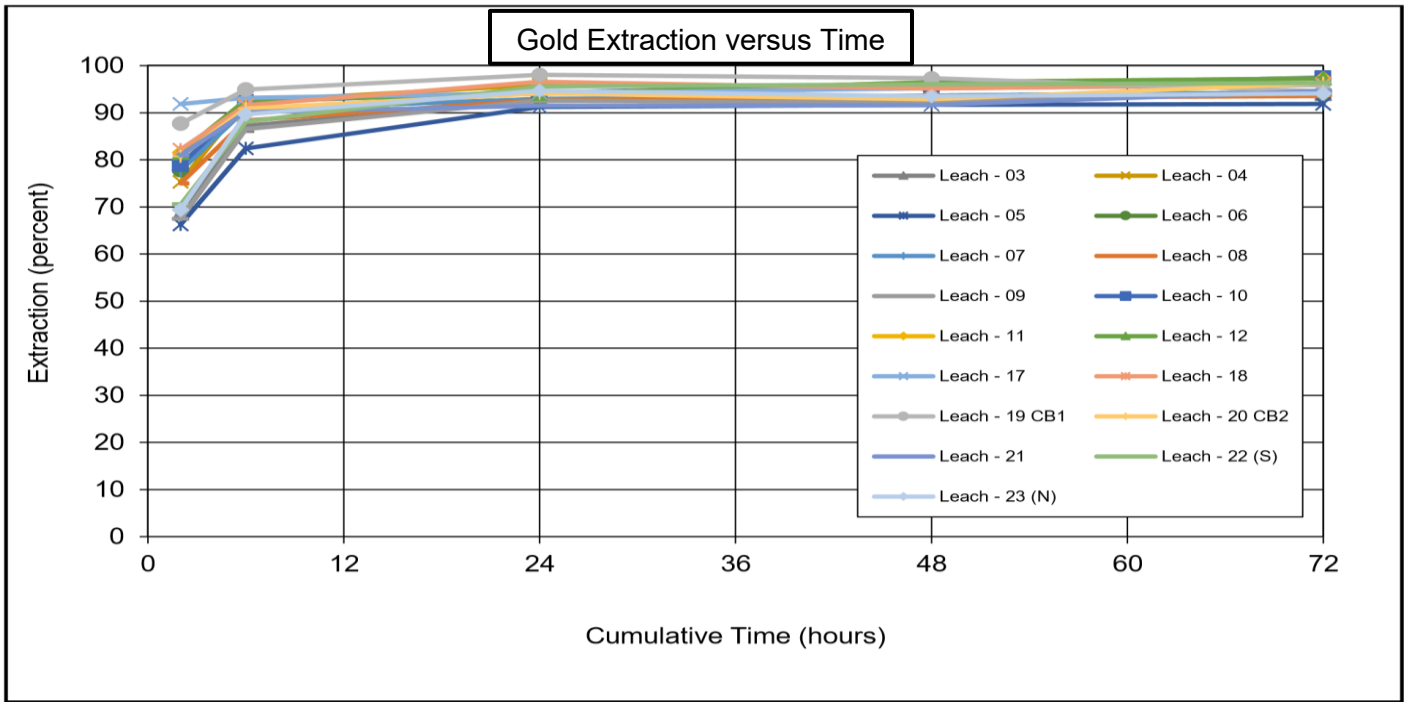
The addition of lead nitrate may be effective at improving silver extraction, based on this sample. Tests completed at a P_{80} of 50 µm showed an increase in silver extraction from 78.6% (Test #11) to 88.4% (Test #17) with the addition of lead nitrate. But in contrast, tests completed at a P_{80} of 75 µm showed a decrease in silver extraction from 86.9% (Test #2) to 82.9% (Test #6) with the addition of lead nitrate.

Pre-oxidation of the slurry with oxygen should be incorporated into the process design. Test #1 did not include pre-oxidation and resulted in a measured DO of below 1 mg/L until after six hours of leaching. This had a significant impact on initial leach rates in the first 24 hours. When pre-oxidation was incorporated under similar conditions in Test #12, leach kinetics improved considerably. Two hours of pre-oxidation was incorporated into the process design.

Tests were conducted using the optimized flowsheet and testwork parameters to investigate gold and silver extraction at 40ºC, with site treated (CB-1), untreated site water (CB-2) and bulk rock sample composites from the North and South deposits. Gold extractions results ranged from 94 to 96% and silver between 77% and 92%.

The general trend for all tests shows that there is minimal advantage to leaching after 48 hours. Considering the increased costs associated with leaching to 72 hours, a leach time of 48 hours was selected to ensure adequate gold and silver recovery. The gold extraction vs time curves are shown in Figure 10-2.

Figure 10-2: Gold Extraction as a Function of Time



Source: BaseMet, 2019.

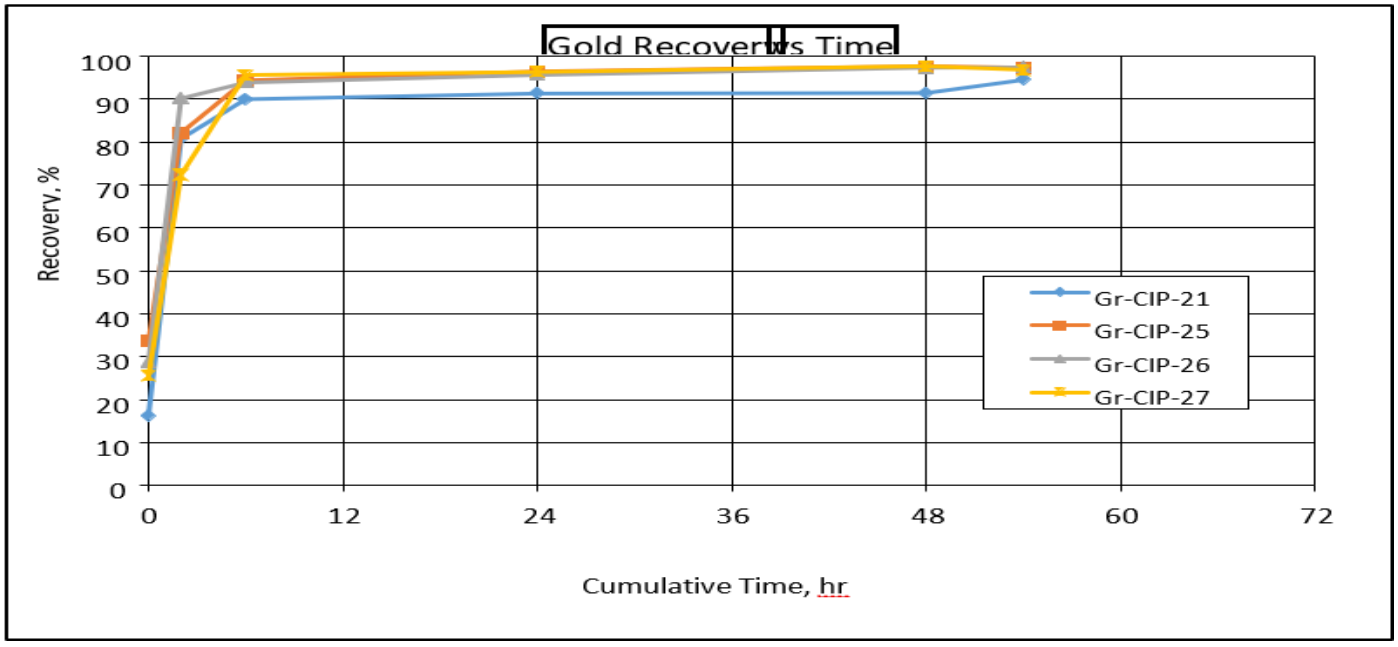
Overall, the conditions used in Test #17 produced the best results. After gravity concentration and 48 hours of leaching, overall metal extractions of 95.2% Au and 85.4% Ag were achieved. The conditions for Test #17 are summarized in Table 10-7.

Table 10-7: BaseMet (2018) Leach Test #17 Operating Conditions

Condition	Unit	Value
Target Grind Size P_{80}	µm	50
Gravity Concentration Included	Y/N	Yes
Operating pH	-	10.5
Lead Nitrate Addition	g/t	250
Sodium Cyanide Concentration	mg/L	500
Pre-Oxidation Time	h	2
Optimal Leach Time	h	48
Sodium Cyanide Consumption after 48 hours	kg/t	0.40
Lime Consumption after 48 hours	kg/t	1.22

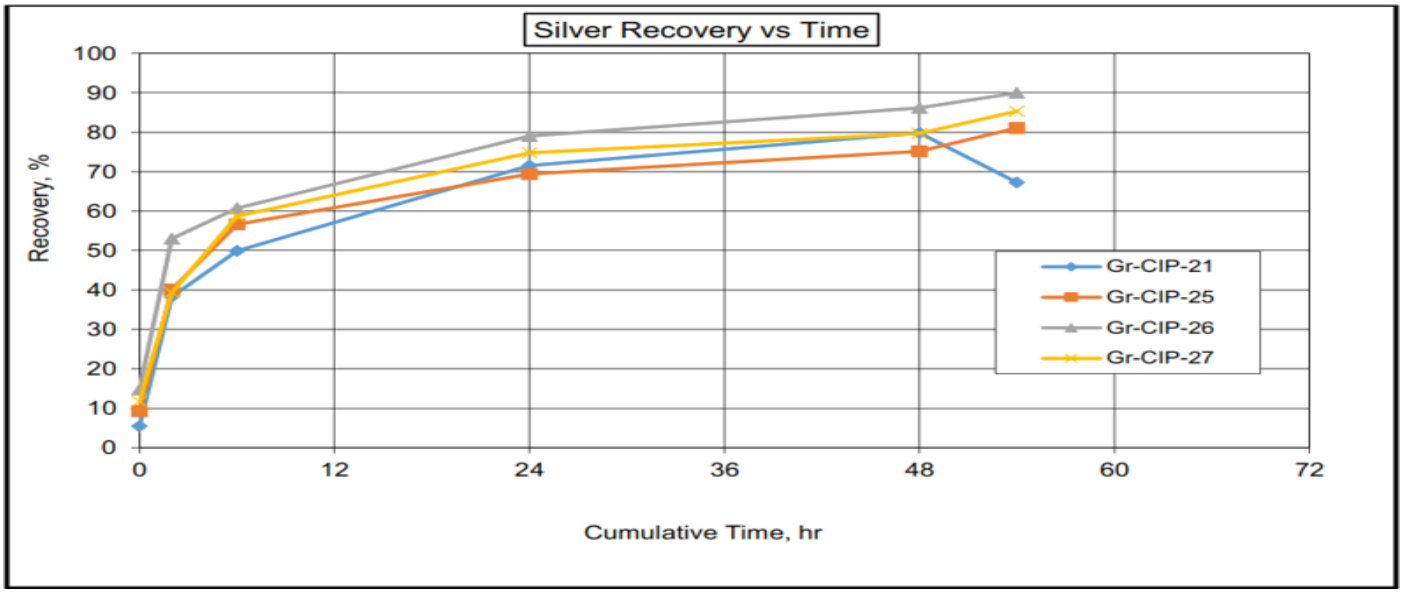
A global composite was further tested to determine the adsorption of gold and silver on carbon. Test, CIP21 was carried out at a carbon concentration was 25 g/L for six hours following the 48-hour leach. The overall recovery for gold was 94.7% and 67.2% silver. Based on the results an additional three tests, CIP-25, 26 and 27, were completed at 50 g/L carbon. Tests CIP-26 and CIP-27 included the addition of 250 g/t lead nitrate. The three additional tests produced higher recoveries for both gold and silver. The addition of lead nitrate appears to improve silver leach kinetics and final recovery. The results and testwork parameters from the four tests were used to develop the process design criteria and projected recoveries for the leach/CIP circuits. The recovery curves for gold and silver vs time are illustrated in Figure 10-3 and Figure 10-4. The addition of carbon to batch leach tests does not replicate the counter current movement of carbon that occurs in plant scale CIP operations. The leach test results are described as recoveries since the activated carbon added adsorbed dissolbed gold and silver.

Figure 10-3: Gold Recovery as a Function of Time



Source: BaseMet, 2019.

Figure 10-4: Silver Recovery as a Function of Time



Source: BaseMet, 2019.

1.1.2.1.3 Cyanide Destruction Test Results

Feed for the cyanide destruction testwork was created from bulk leach tests. To determine cyanide species, a representative pulp sample was taken and filtered. The filtrate was then submitted for analysis. The cyanide solution from the produced pulp contained 283 mg/L total cyanide (CN_T), 270 mg/L weak acid dissociable cyanide (CN_{WAD}), 10.2 mg/L Fe, and 0.2 mg/L Zn.

Continuous cyanide destruction testwork was completed to produce a treated product using the SO_2/air process, targeting less than 5 mg/L CN_{WAD}. A batch test (CND-B1) was conducted on the leached pulp to produce a starting pulp with a low residual CN_{WAD}. A series of continuous cyanide destruction tests were then completed to establish the cyanide destruction circuit design criteria and understand the effect of reagent dosage on the oxidation of cyanide. The SO_2/air process does not directly reduce CN_T.

The cyanide pulp produced during the test program responded well to the SO_2/air cyanide destruction process, producing a treated pulp with < 1 mg/L CN_{WAD} and <4 mg/L CN_T. The results are shown in Table 10-8. The conditions used in CND-C7 were incorporated into the process design for the cyanide destruction circuit.

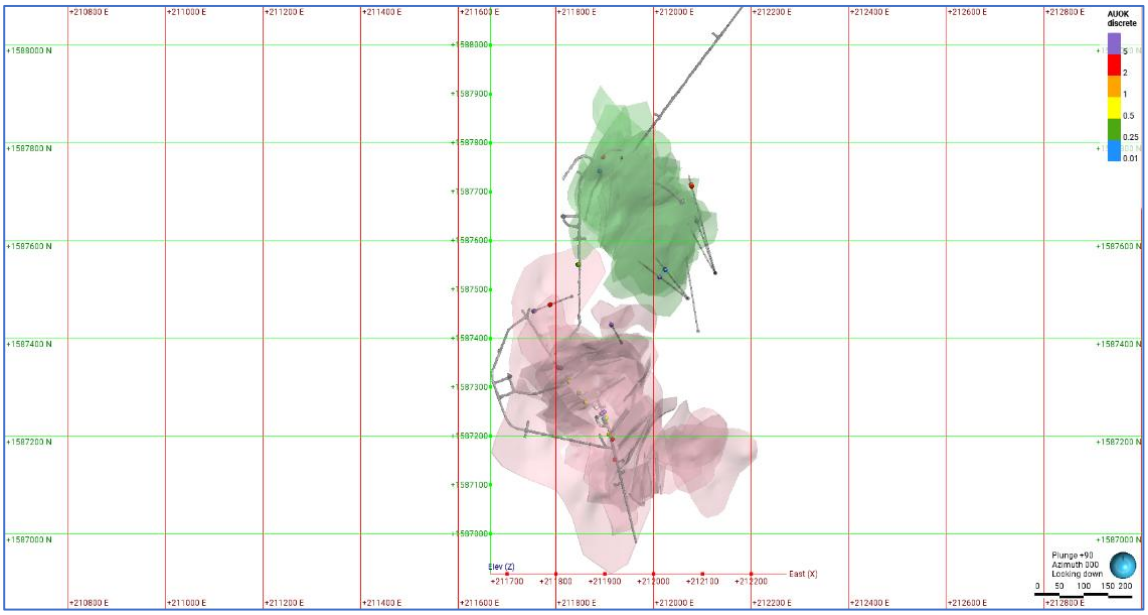
Table 10-8: Cyanide Destruction Results for BaseMet (2018)

Test No.	Retention Time (mins)	pH	Final Solution Composition				Reagent Addition (g/g CN_{WAD})		Cu (mg/L of solution)
			CN_T (mg/L)	CN_{WAD} (mg/L)	Cu (mg/L)	Fe (mg/L)	SO_2 Equiv.	Lime	
CND-C1	90	8.5	4.59	1.8	1.02	< 0.1	7	4.6	100
CND-C2	90	8.5	2.96	0.17	0.25	< 0.1	5.5	2	100
CND-C3	90	8.3	0.49	0.22	0.39	0.1	4	2.2	100
CND-C4	90	8.4	2.94	0.14	0.47	< 0.1	4	1.6	50
CND-C5	90	8.5	3.02	0.24	1.2	< 0.1	4	1.4	25
CND-C6	90	9.0	18.3	4.18	16.5	5	4	-	0
CND-C7	60	8.5	3.56	0.48	3.93	1.1	4	0.8	25

1.2 Metallurgical Variability

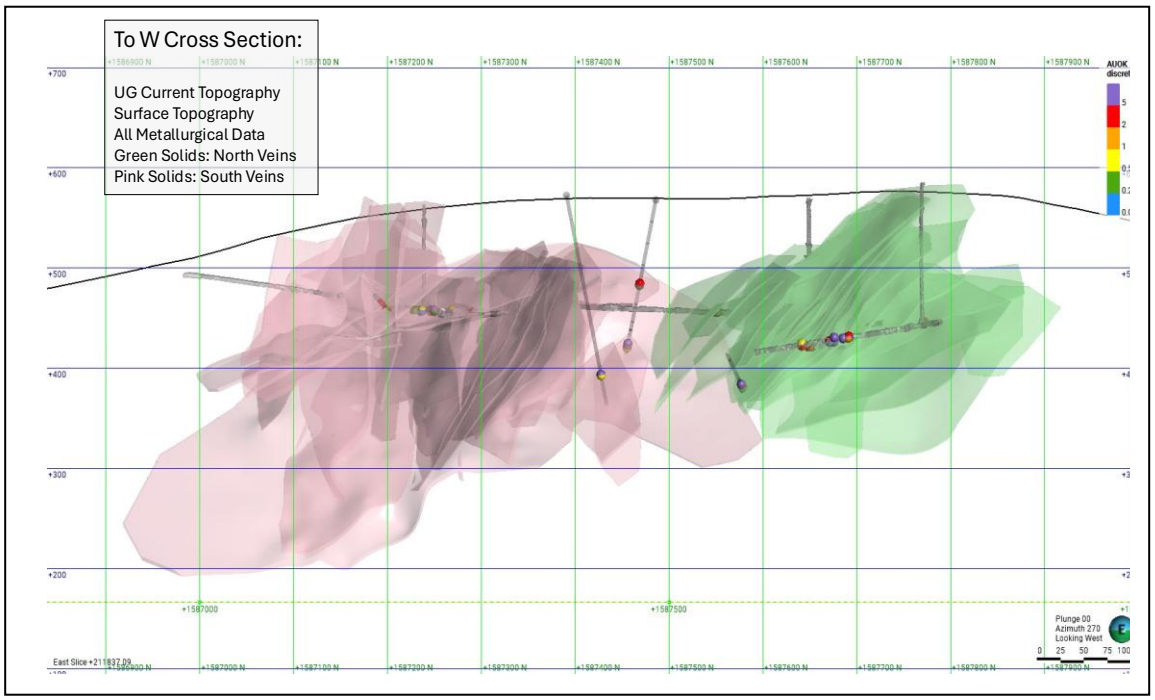
The samples for the 2018 BaseMet metallurgical test program were collected from drill holes that intercepted the North and South ore bodies of the mineral deposit. Figure 10-5 and Figure 10-6 below illustrate the sample locations in relation to the mine plan. The intervals selected provide spatial, grade and lithological representation for the global composites. This is suitable for the intended uses of these samples.

Figure 10-5: BaseMet (2018) Sample Location (plan view)



Source: Aura,2025.

Figure 10-6: BaseMet (2018) Sample Location (section view)



Source: Aura, 2025.

10.3 Comments on Mineral Processing and Metallurgical Testing

Previously completed metallurgical testing demonstrates that Era Dorada samples are free milling, resulting in high gold and silver leach extractions. The samples show some amenability to gravity concentration upstream of leaching. Samples require a finer grind to achieve the design leach extractions with a P_{80} of 53 µm. The presence of soluble sulphides requires the addition of lead nitrate to ensure sufficient dissolved oxygen and cyanide in solution to achieve the required leach extractions. Leach tailings slurry was found to be amenable to cyanide detox using the SO_2/air process to achieve acceptable concentrations of CN_{WAD}. Filtration testing of detox slurry showed that pressure filtration provided suitable moisture content for tailings disposal with acceptable filtration duties.

Limited information is available on the presence of deleterious elements in the more recent samples tested. Arsenic is likely to occur considering groundwater is treated for arsenic removal. Further testing of pre-production samples is recommended to confirm the quantity of arsenic present and if treatment is required. Testing should also include mercury to confirm that mercury abatement equipment is not required in the carbon elution and regeneration and refining areas of the plant.

10.4 Recovery Estimates

Preliminary estimates of gold and silver recovery are summarized in Table 10-9. Test Gr-CIP-21 was not included in the average for silver since there was insufficient carbon (25 g/L compared to 50 g/L) in solution to recover all the silver. These projections are based on the results from BaseMet's (2018) CIP test results. The economic results presented in Section 22 are based on an average gold recovery of 96% and silver recovery of 85%. These estimates included plant losses.

Table 10-9: Preliminary Recovery Projections

CIP Test No.	Recovery	
	Au (%)	Ag (%)
Gr-CIP-21	94.7	67.2
Gr-CIP-25	97.4	81.1
Gr-CIP-26	97.5	90.0
Gr-CIP-27	96.9	85.4
Overall Recovery Projections (including plant losses)	**96**	**85**

11 MINERAL RESOURCE ESTIMATES

11.1 Introduction

This section describes the work undertaken by Kirkham Geosystems Ltd (KGL), including key assumptions and parameters used to prepare the mineral resource models for Era Dorada, together with appropriate commentary regarding the merits and possible limitations of such assumptions.

Era Dorada is a classic hot springs-related, low-sulfidation epithermal gold-silver deposit comprising both high-grade vein and low-grade disseminated mineralization. Most of the high-grade mineralization is hosted in the Mita unit as two upward-flaring vein swarms (north and south zones) that converge downwards and merge into basal feeder veins where drilling has demonstrated widths of high-grade mineralization (e.g., 15.5 m 21.4 g/t Au and 52 g/t Ag). Bonanza gold grades are associated with ginguru banding and carbonate replacement textures. Sulfide contents are low, typically < 3% volume.

The Mita rocks are overlain by the Salinas unit, a sub-horizontal sequence of volcanogenic sediments and sinter horizons approximately 100 m thick that form the low-lying hill at the project. Low-grade disseminated and veinlet mineralization within and as halos around the high-grade vein swarms is well documented in drilling since discovery of the deposit, with grades typically ranging from 0.3 to 1.5 g/t Au. The overlying Salinas cap rocks are also host to low-grade mineralization associated with silicified conglomerates and rhyolite intrusion breccias.

The mineral resource has a footprint of 800 x 400 m between elevations of 525 and 200 m above sea level (masl). The mineral resource estimate is the result of 141,969 m of drilling by Bluestone and previous operators (1,256 drill holes and channel samples by Bluestone). The 3.4 km of underground infrastructure allowed for underground mapping, sampling, and over 30,000 m of underground drilling that enhanced the current understanding and validation of the Era Dorada geological model. The mineral resource estimate is based on a scenario that considers open pit mining methods and therefore requires improved and refined geological models of the lithologic units. These broad mineralised lithologies are host to the high-grade veins that have been the focus of the potential underground mining scenario. The resulting domain models and estimation strategy were designed to accurately represent the grade distribution.

Several resource estimates have been published on Era Dorada since 2017 in four technical reports, as follows:

- Preliminary Economic Assessment (March 20, 2017)
- Preliminary Economic Assessment Update (June 2, 2017)
- Feasibility Study (January 29, 2019)
- Preliminary Economic Assessment Update (February 28, 2021)
- Preliminary Economic Assessment Update (June 30, 2021)

- Initial Assessment and Technical Report (December 31, 2024)

Four of the technical reports and resource estimates were for an underground mining scenario while two resource estimates were for the open pit scenario. All five NI43-101 technical reports are filed on the System for Electronic Document Analysis and Retrieval (SEDAR+) whilst the December 31, 2024 S-K 1300 Technical Report Summary is filed on EDGAR.All estimates were authored by Qualified Person, Garth Kirkham, P.Geo.

11.2 Data

The drill hole database was supplied in electronic format (i.e., Microsoft Excel and Access) by Aura. This included collars, down hole surveys, lithology data and assay data (i.e., grams per tonne of gold and silver, and down hole "from" and "to" intervals in metric units). Lithology group and description information was provided, along with abbreviated alpha-numeric and numeric codes (see Table 11-1). Figure 11-1 shows the plan view of drill holes with collars. A total of 130,238 assay values and 55,285 lithology values were supplied for the project. Validation and verification checks were performed during import to confirm there were no overlapping intervals, typographic errors, or anomalous entries.

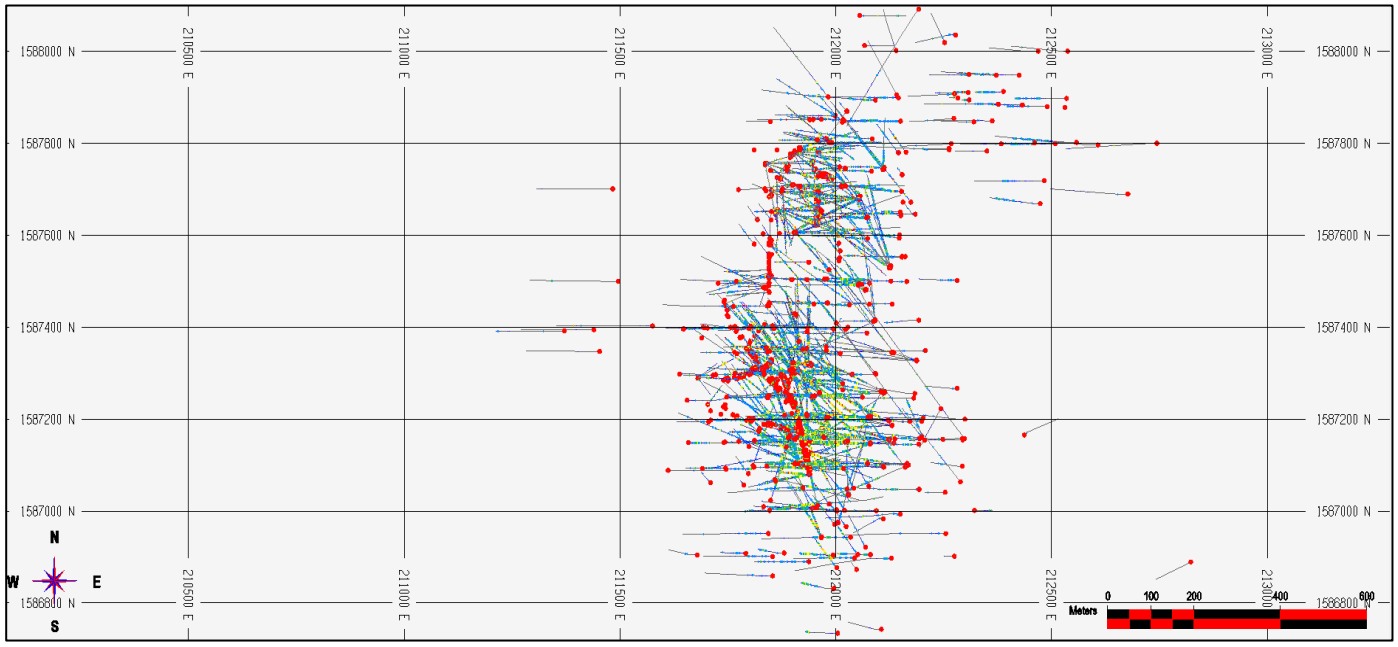
Table 11-1: Lithology Units and Codes

Lithology	Code	Code B	Lithology Group	Lithology Description
Qc	10	1	Post-Mineral Cover Rock - Quaternary	Colluvium
Qb	11	1.1		Basalt Flows
Bi	20	2	Cross-Cutting Rock Types	Basaltic Intrusive Dikes
Cbx	30	3		Collapse Breccia
Dp	180	18		Dacite
Gr	40	4		Granite
Ad	50	5		Andesite Dike
Rp	60	6		Quartz Eye Rhyolite
Vt	70	7		Vein
Stock	71	7.1		Stockwork
Hbx	72	7.2		Hydrothermal Breccia
RF	80	8		Rhyolite Flow
SZ	81	8.1		Shear Zone
Ss	90	9	Salinas Group	Sinter
Svc	91	9.1		Volcanic Sediments
Srt	92	9.2		Quartz Eye Rhyolite
Sfx	93	9.3		Phreatic Breccia
Slt	94	9.4		Siltstone
Sct	95	9.5		Ash Tuff

Lithology	Code	Code B	Lithology Group	Lithology Description
Scgl	96	9.6		Conglomerate
Mss	100	10	Mita Group	Sandstone
Mat	101	10.1		Andesite Tuff
Mlt	102	10.2		Crystal Tuff
Mbt	103	10.3		Lapilli Tuff
Msc	104	10.4		Calcareous Limestone
Mls	105	10.5		Limestone
Mcv	106	10.6		Quartz Latite Crystal Lithic Tuff
Mvo	107	10.7		Conglomerate
Mlm	190	19		Upper Limestone
Silt	108	10.8		Siltstone - mudstone
PA	130	13		Porphyritic andesite
Tcb	110	11	Tempisque Volcanic Complex	Basalt-dominated
Tca	111	11.1		Andesite-dominated

Source: Kirkham, 2025.

Figure 11-1: Plan View of Drill holes



Source: Kirkham, 2025.

11.3 Data Analysis

Table 11-2 shows statistics of gold and silver assays for each of the lithologic units. It should be noted that the total number of values from section to section vary depending on the parameter being analysed and the value for reporting these varied data sub-sets is to detect and investigate issues or anomalies. Included for the statistical analysis, there are 130,307 gold assays (153,078 m) total, which average 0.68 g/t, and there are 130,238 (153,003 m) silver assays by lithology logged, which average 3.75 g/t. The maximum gold assay is 1,380 g/t, while the maximum silver assay is 8,656.7 g/t. It is important to note that 73 gold assays are greater than 100 g/t and 54 silver assays are greater than 500 g/t which may be a reflection of the non-nuggety nature of the mineralization present at Era Dorada.

Table 11-2: Statistics for Weighted Gold and Silver Assays

Code	Metal	Valid	Length (m)	Max (g/t)	Mean (g/t)	CV
Total	AU	130,307	153,077.8	1,380.0	0.68	9.9
	AG	130,238	153,003.0	8,656.7	3.75	11.1
All	AU	131,215	154,481.6	1,380.0	0.69	9.8
	AG	131,146	154,406.9	8,656.7	3.78	11.0

Source: Kirkham, 2025.

Table 11-3 above shows intervals that intersect the high grade are primarily encountered within the Vt unit, as would be expected. The Vt unit which represents the majority of the very high-grade populations, has 7,554 gold (3,716.8m) and 7,553 (3,716.7 m) silver assay intersections, resulting in an average grade of 9.94 g/t Au and 38.92 g/t Ag. The coefficient of variation is relatively high with 3.3 for gold and 4.0 for silver. These are reviewed once compositing and cutting is applied which will reduce the CV to reasonable values. Also, of particular interest within the Cross-cutting group are the Stock which shows 2,899 values (3,714 m) with 1.64 g/t gold and 8.11 g/t silver and HBX shows 1592 values (1,067 m) with 1.08 g/t gold and 6.94 g/t silver, respectively. The grades within the Stock and Hbx intervals display very high variability due to a small number of very high-grade outliers. These values are fairly widely distributed within the Salinas and Mita units which may positively skew the grades within the low-grade envelopes. However, as they are disseminated and to be treated within the domains, they will be cut appropriately to ensure that they reasonably represent the estimated grades.

Table 11-3: Statistics for Weighted Gold and Silver Assays for Quaternary and Cross-cutting Rock Types

Code	Lith	Code	Metal	Valid	Length (m)	Max (g/t)	Mean (g/t)	CV
10	Qc	10	AU	787	1,271.0	5.1	0.05	2.9
			AG	786	1,270.6	35	0.97	2.1
11	Qb	11	AU	144	214.7	0.06	0.01	0.4
			AG	144	214.7	1	0.83	0.4
30	Cbx	30	AU	4,016	4,466.6	1,380.0	0.78	14.3
			AG	4,016	4,466.6	2,194.0	3.86	5.4
40	Gr	40	AU	419	685.1	0.246	0.01	1.5
			AG	419	685.1	2.3	0.81	0.5

Code	Lith	Code	Metal	Valid	Length (m)	Max (g/t)	Mean (g/t)	CV
50	Ad	50	AU	1,780	2,268.6	313.97	0.47	13.3
			AG	1,780	2,268.6	801.2	2.73	7.8
60	Rp	60	AU	2,899	3,714.1	46.3	0.22	2.9
			AG	2,899	3,714.1	241	2.12	2.9
70	Vt	70	AU	7,554	3,716.8	1,380.0	9.94	3.3
			AG	7,553	3,716.7	4,677.8	38.92	4.0
71	Stock	71	AU	2,383	2,214.9	148.75	1.64	3.7
			AG	2,383	2,214.9	409	8.11	2.6
72	Hbx	72	AU	1,592	1,067.4	266.09	1.08	7.9
			AG	1,591	1,067.3	969	6.94	4.7
80	RF	80	AU	5,494	6,923.0	150.7	0.28	9.2
			AG	5,489	6,919.0	8,656.7	5.11	26.6
81	SZ	81	AU	36	31.5	8.4	0.27	3.0
			AG	36	31.5	55.7	2.49	2.2

Source: Kirkham, 2025.

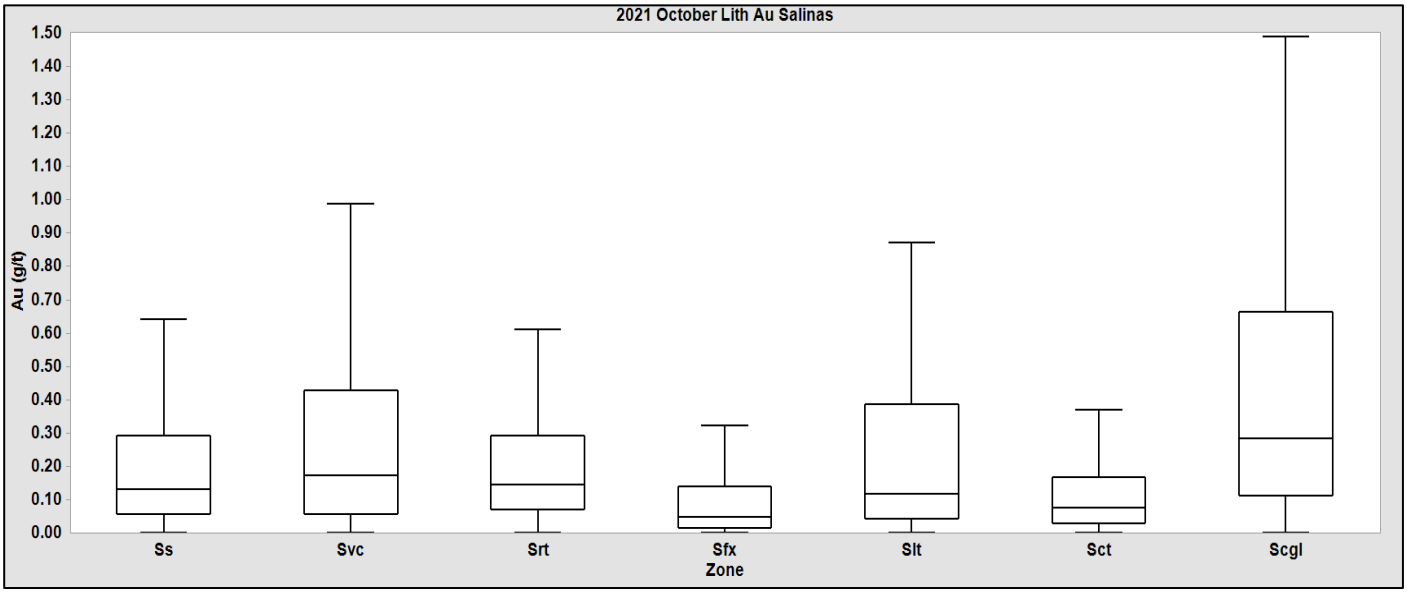
Table 11-4: Statistics for Weighted Gold & Silver Assays for the Salinas Group Rocks

Code	Lith	Metal	Valid	Length (m)	Max (g/t)	Mean (g/t)	CV
90	Ss	AU	4,200	6,269.7	15.67	0.27	2.1
		AG	4,198	6,269.2	187.8	1.52	2.7
91	Svc	AU	19,081	24,245.9	131.6	0.48	4.0
		AG	19,032	24,189.7	1,346.9	3.41	3.9
92	Srt	AU	1,215	1,522.1	16.47	0.27	2.3
		AG	1,215	1,522.1	88	2.38	2.1
93	Sfx	AU	1,495	2,334.3	194.7	0.34	10.8
		AG	1,495	2,334.3	267.4	2.59	4.4
94	Slt	AU	273	399.3	9.06	0.40	2.2
		AG	273	399.3	74	1.47	4.0
95	Sct	AU	242	347.7	3.57	0.19	1.9
		AG	242	347.7	32	1.42	1.6
96	Scgl	AU	3,189	3,481.8	157.43	0.71	3.8
		AG	3,189	3,481.8	1,552.0	4.15	5.7
Total		AU	29,695	38,600.8	194.7	0.45	4.4
		AG	29,644	38,544.1	1,552.0	3.04	4.3

Source: Kirkham, 2025.

Table 11-4 lists the statistics for the Salinas Group rocks units with the predominant unit being the Volcanic Sediments (Svc) showing mean gold and silver grades of 0.48 g/t and 3.41 g/t, respectively with relatively high variability (CV) of 4.0 and 3.9. It is apparent from logging and modeling of the Salinas that the Sinter (Ss) and the Basal Conglomerate (Scgl) illustrate consistency and continuity. In addition, the Sinter has relatively lower grades with a mean of 0.27 g/t gold while the Basal Conglomerate results show higher grades with a mean of 0.71 g/t gold as illustrated Figure 11-2. Therefore, observations and statistical analysis supports the resultant domaining for the Salinas of the Sinter, Basal Conglomerate and the remaining sedimentary units with the Volcanic Sediments (Svc) the predominant rock type.

Figure 11-2: Box Plot Gold Assays for the Salinas Group Rocks



Source: Kirkham, 2025.

Table 11-5: Statistics for Weighted Gold & Silver Assays for the Mita Group Rocks

Code	Lith	Metal	Valid	Length (m)	Max (g/t)	Mean (g/t)	CV
100	Mss	AU	10,292	12,214.2	368.33	0.33	12.0
		AG	10,292	12,214.2	2,405.90	2.37	10.8
101	Mat	AU	5,303	6,472.7	105.647	0.45	7.1
		AG	5,303	6,472.7	1,257.0	2.70	5.9
102	Mlt	AU	3,387	4,703.9	62.059	0.36	5.7
		AG	3,387	4,703.9	419	2.26	4.7
103	Mbt	AU	22,353	24,157.8	1,380.0	0.61	10.0
		AG	22,353	24,157.8	2,863.0	3.71	7.0

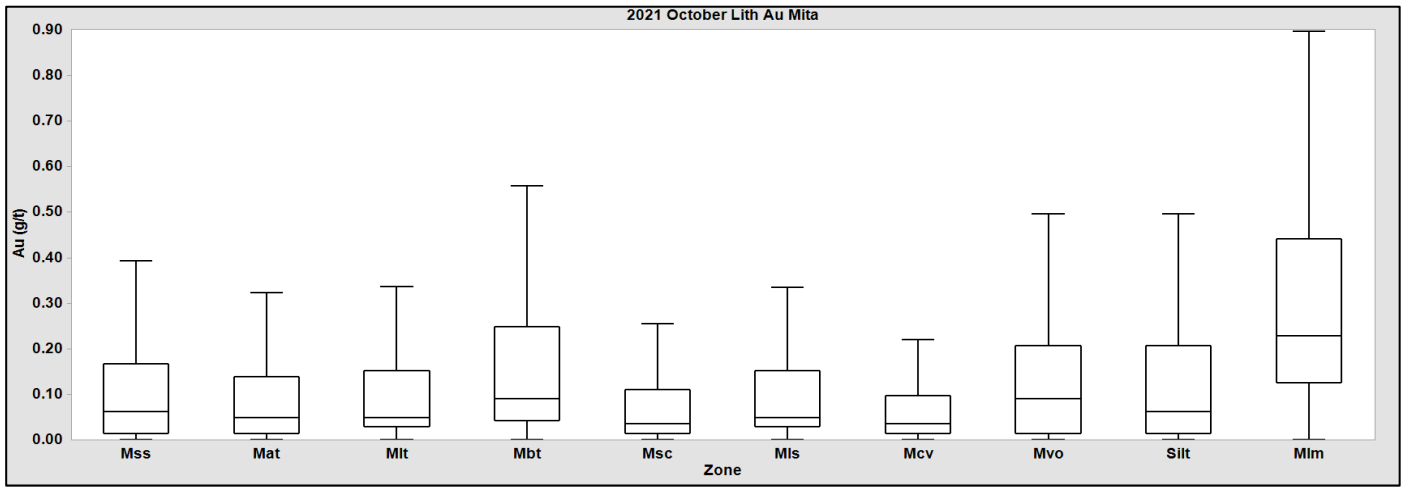
Code	Lith	Metal	Valid	Length (m)	Max (g/t)	Mean (g/t)	CV
104	Msc	AU	3,183	3,988.7	180.73	0.34	8.4
		AG	3,183	3,988.7	624.6	2.20	5.5
105	Mls	AU	2,750	2,981.8	163.3	0.59	6.7
		AG	2,750	2,981.8	1,202.00	3.73	6.8
106	Mcv	AU	21,432	28,724.3	287.13	0.32	9.9
		AG	21,422	28,710.9	997.7	1.49	4.5
107	Mvo	AU	2,488	2,192.1	210.3	0.53	7.5
		AG	2,488	2,192.1	271	1.94	2.9
108	Mlm	AU	988	852.2	45	0.37	3.4
		AG	988	852.2	50.6	2.08	1.7
120	Silt	AU	2	6.1	0	0.00	n/a
		AG	2	6.1	0	0.00	n/a
130	PA	AU	497	388.5	132.9	0.29	7.0
		AG	497	388.5	125	1.56	2.4
190	Mlm	AU	98	73.0	14.9	0.86	2.6
		AG	98	73.0	101	6.08	1.7
Total		AU	72,773	86,755.4	1,380.0	0.43	9.9
		AG	72,763	86,742.0	2,863.0	2.49	7.5

Source: Kirkham, 2025.

Table 11-5 lists the statistics for the Mita Group rocks units with the predominant unit being the Sandstone (Mss), Crystal Lithic Tuff (Mcv) and Lapilli Tuff (Mbt) units showing mean gold grades of 0.33 g/t, 0.61 g/t, 0.32 g/t and silver grades of 2.37 g/t, 1.49 g/t, 3.71 g/t, respectively. It is noted that the variability is very high with CV's ranging from 4.5 to 12.0. It is again clear from logging and modeling of the Mita that the Mbt and the Mcv represent the main stratigraphic units which are distinct and significant showing consistency and continuity throughout Era Dorada.

Figure 11-3 shows that the Lapilli Tuff (Mbt), Conglomerate (Mvo) and Siltstone (Silt) are statistically similar, and the Upper Limestone (Mlm) is statistically different from all of the other Mita rock units. All other rock units are statistically similar as shown in Figure 11-3. Further analysis and modeling for the purpose of grouping and domaining takes these observations and conclusion into account.

Figure 11-3: Box Plot Gold Assays for the Mita Group Rocks



Source: Kirkham, 2025.

Table 11-6: Statistics for Weighted Gold & Silver Assays

Code	Lith	Metal	Valid	Length (m)	Max (g/t)	Mean (g/t)	CV
110	Tcb	AU	37	49.5	0.05	0.01	1.3
		AG	37	49.5	1	0.39	1.0
111	Tca	AU	697	1,096.8	1.33	0.03	3.1
		AG	697	1,096.8	13	0.77	1.2

Source: Kirkham, 2025.

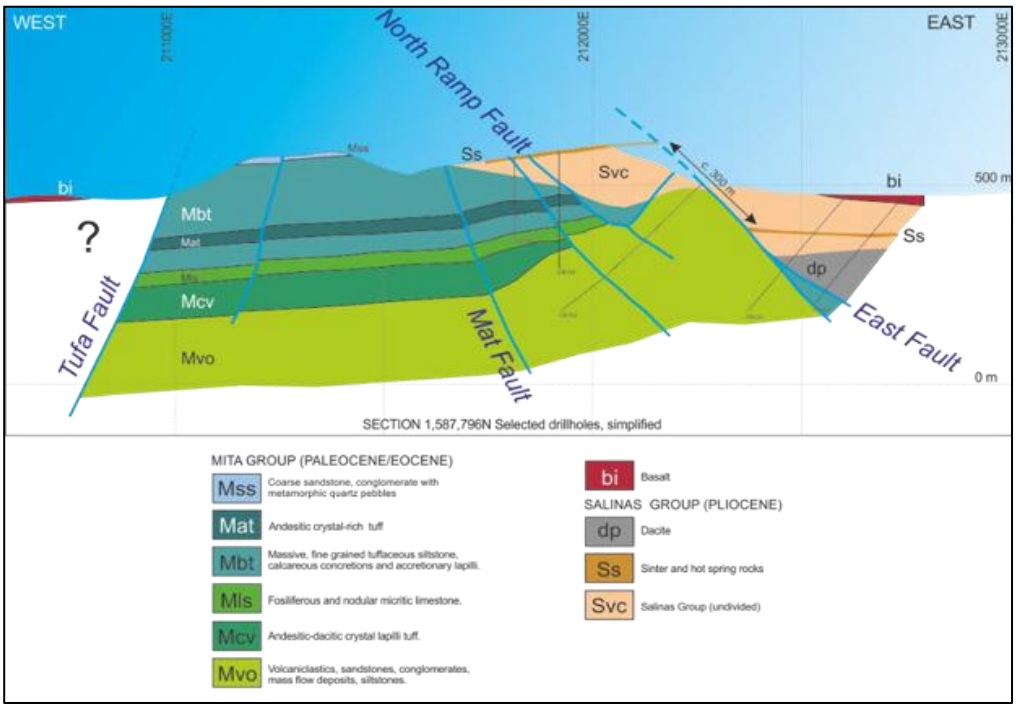
Table 11-6 above shows intervals that intersect Tempisque Volcanic Complex are primarily treated as waste.

11.4 Geology & Domain Model

A three-phased modeling approach was taken to creating geology and estimation domains which included a lithostratigraphic model, detailed vein modeling, and domain modeling to estimate low-grade host rock solids within the Salinas and the Mita lithology units.

The lithology models were completed using the lithology codes within the database as shown in Figure 11-4.

Figure 11-4: Section View Schematic of Lithology for the Era Dorada Deposit



Source: Kirkham, 2025.

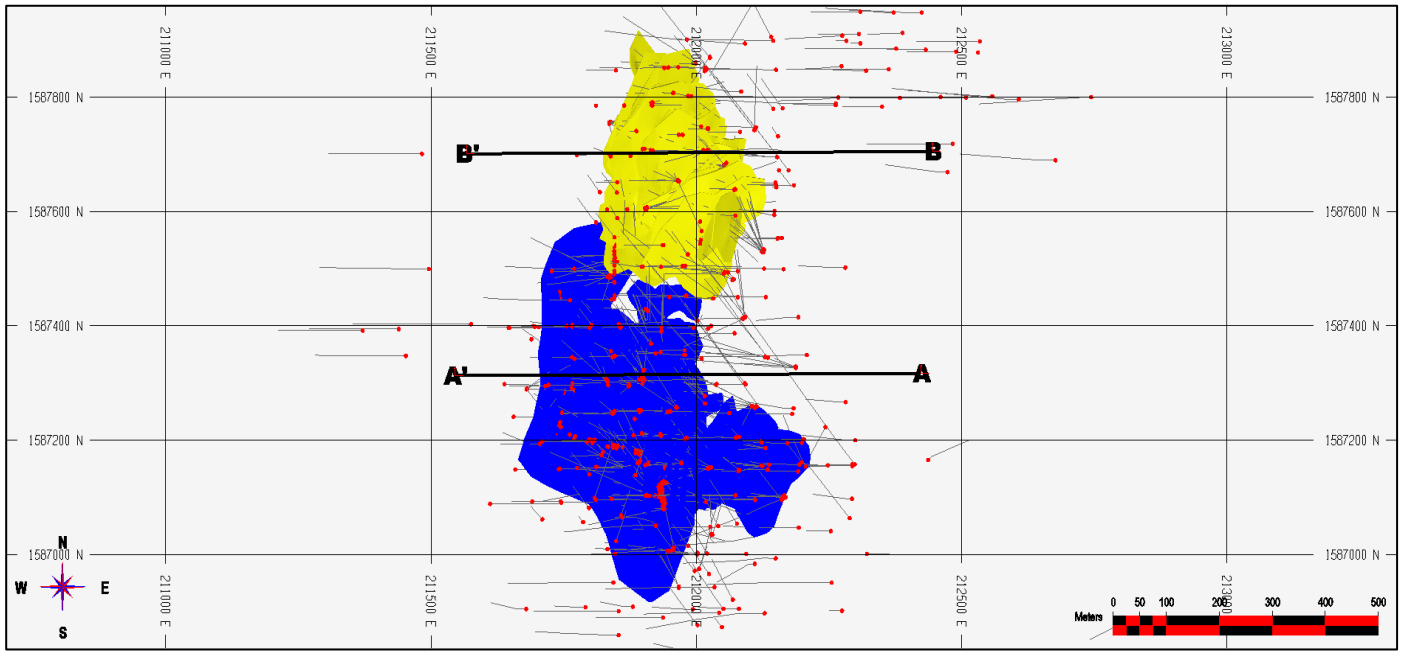
The models were created from first principals within LeapFrogTM and refined in MineSightTM for statistical analysis and to be used for the estimation process. Figure 11-4 illustrates the sectional interpretation of the main significant lithology units, namely the Salinas and Mita Group rock units. In addition, logging showed that within the Salinas, there appeared to be zones of gouge potentially related to fault zones termed TBX that were determined to require modeling so that they could be masked out of the domain models.

In addition, solid models of each of the individual veins were created and are displayed in plan in Figure 11-5 with the north veins in yellow and the south veins in blue, respectively. In preparation for the creation of the vein models, a comprehensive structural model was developed that incorporated the current drilling, underground sampling, mapping, and extensive re-logging of drill core. The models were also created from first principals using the lithostratigraphic models and the structural modeling as guides by Bluestone staff within LeapFrogTM under the supervision of the independent QP. This was done utilising the current and re- logged data, and from sectional interpretations that were subsequently wireframed based on a combination of lithology and gold grades.

Once completed, intersections were inspected, and all of the solids were then manually adjusted to match the drill intercepts. Once the solid models were edited and complete, they were used to code the drill hole assays and composites for subsequent statistical and geostatistical analysis. The solid zones were utilised to constrain the block model, by matching assays to those within the zones.

The orientation and ranges (distances) utilised for the search ellipsoids used in the estimation process were omni-directional and guided the strike and dip of the lithologic solids for the low-grade domains and by the highly constrained vein solids for the high-grade domains shown in Figure 11-5. The vein models were employed to estimate the high-grade structures on a partial block basis that are to be combined with the low-grade component to derive the whole block diluted grade for each block.

Figure 11-5: Plan View of Drill holes & Vein Solids



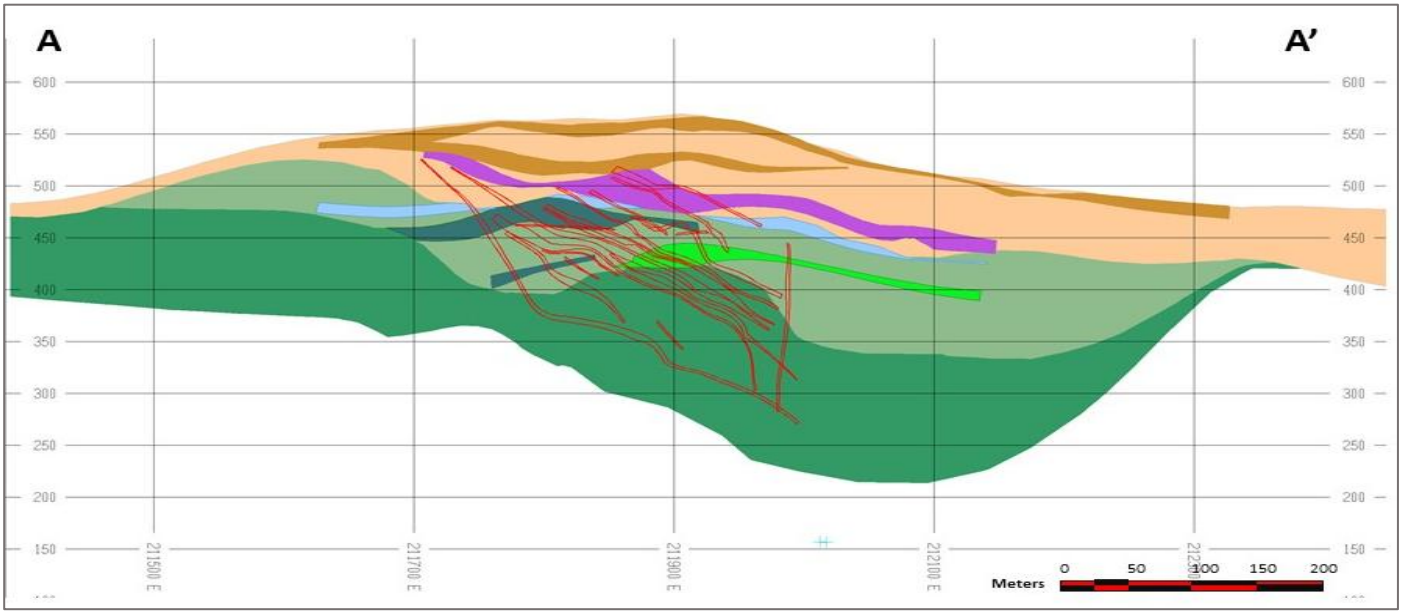
Note: Yellow – north veins, blue – south veins. Source: Kirkham, 2025.

The low-grade estimation domains were created using lithology. The methodology was to determine which lithology units could be segregated or grouped based on grade profiles and it was determined that the Salinas be modeled as Salinas, Sinter, Basal Conglomerate. Within the Mita Group, the moderately mineralised volume that envelops that North and South vein clusters are predominantly the Mbt and Mcv units.

Figure 11-6 and Figure 11-7 illustrate the estimation domains in the north and south, respectively, which include the veins, Salinas and Mita units.

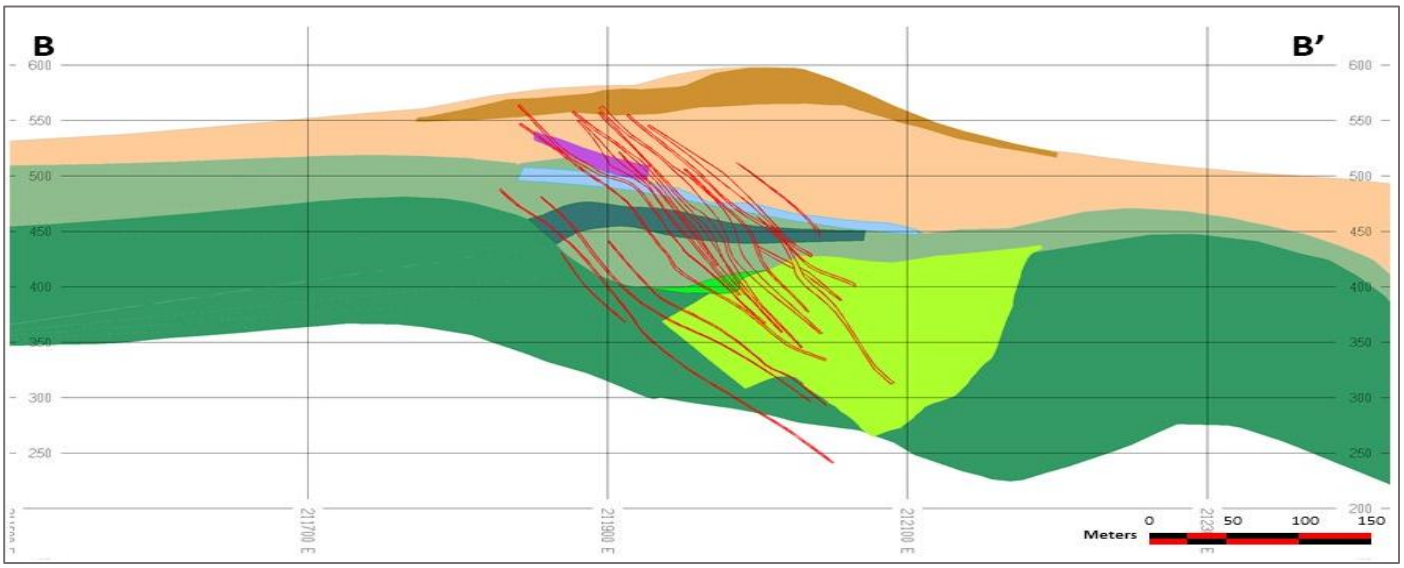
The solids were coded into the composite database in separate fields so as accurately account for the low- and high-grade components of each block along with the waste.

Figure 11-6: South Area Section A-A' View of Drill holes, Vein Solids with Salinas and Mita Units



Legend: Vein Solids – red; Sinter – brown polygons; Salinas – beige; Scgl conglomerate – purple; Mss – pale blue; Mat – sapphire blue, Mbt – pale green, Mls – bright green; Mcv – dark green. Source: Kirkham, 2025.

Figure 11-7: North Area B-B' Section View of Vein Solids with Salinas and Mita Units



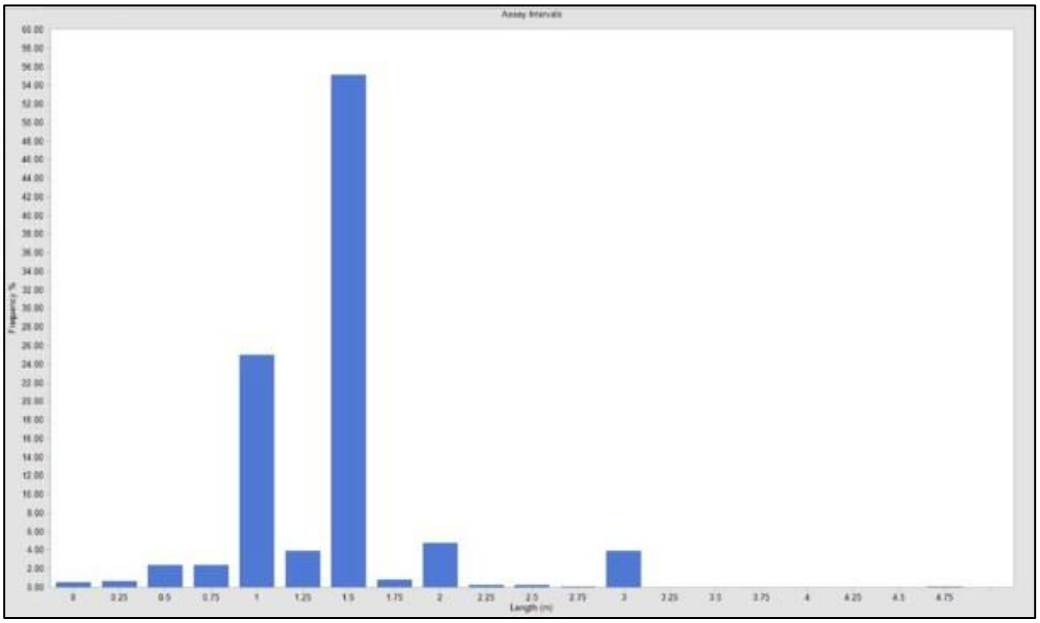
Legend: Vein Solids – red; Sinter – brown polygons; Salinas – beige; Scgl conglomerate – purple; Mss – pale blue; Mat – sapphire blue, Mbt – pale green, Mls – bright green; Mcv – dark green. Source: Kirkham, 2025.

11.5 Composites

It was determined that the 1.5 m composite lengths offered the best balance between supplying common support for samples and minimising the smoothing of grades. Figure 11-8 shows a histogram illustrating the distribution of the assay interval lengths for the complete database with 90% of the data having interval lengths greater than 1.5 m while Figure 11 9 shows the histogram of for the assay intervals limited to within the high-grade veins where 97.5% are less than or equal to 1.5 m; 16% less than or equal to 1.0 m and 2% less than or equal to 0.5 m. To determine whether there may be selective sampling an analysis of high-grade gold samples versus assay interval lengths was performed. The scatterplot of Figure 11 10 for samples within the high-grade veins shows that the assay intervals and corresponding gold grade have the same distribution and illustrate that there is not a high-grade bias within the small intervals and sample selectivity is not occurring.

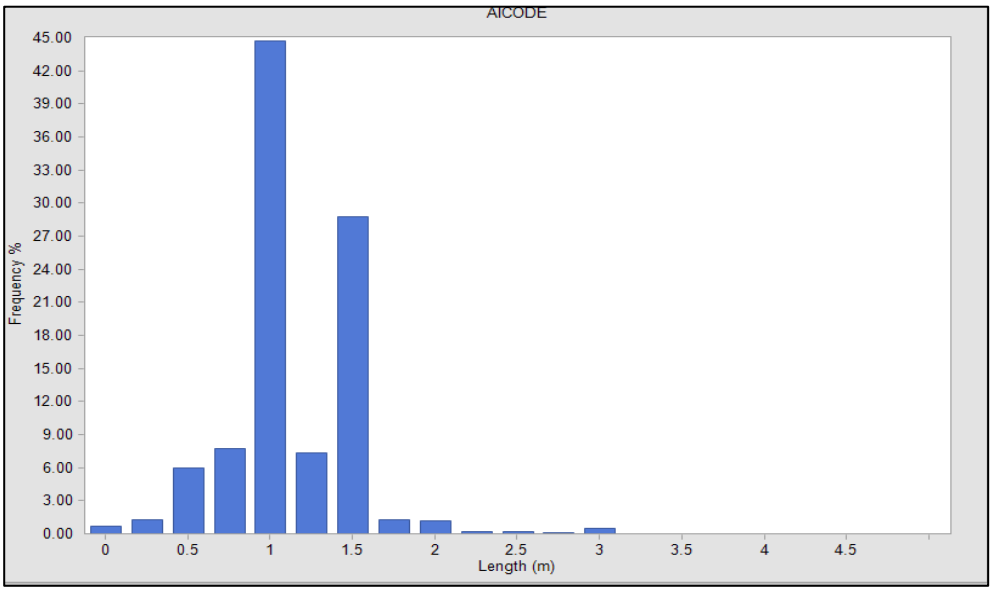
The 1.5 m sample length also was consistent with the distribution of sample lengths. It should be noted that although 1.5 m is the composite length, any residual composites of greater than 0.75 m in length and less than 1.5 m remained to represent a composite, while any composites residuals less than 0.75 m were combined with the composite above.

Figure 11-8: Histogram of Assay Interval Lengths in Meters



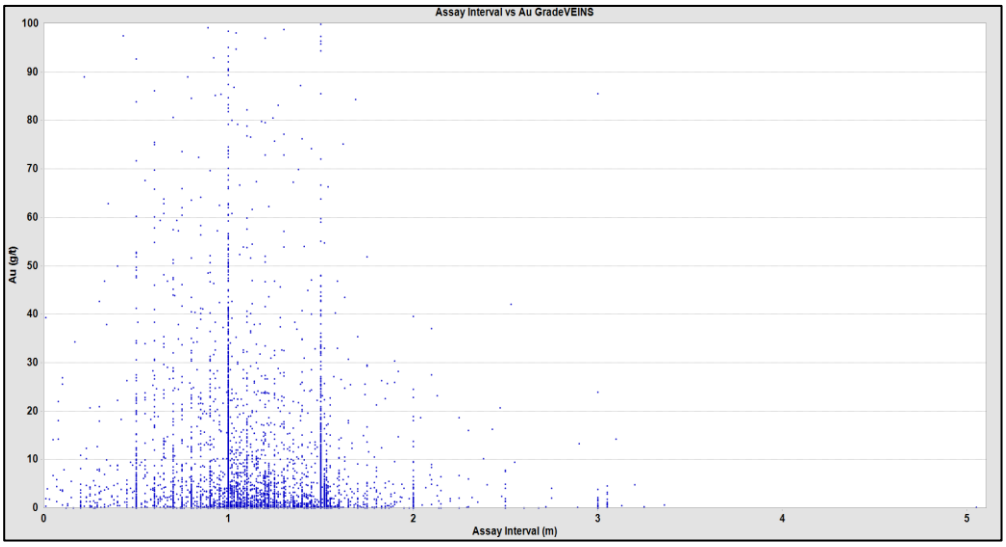
Source: Kirkham, 2025.

Figure 11-9: Histogram of Assay Interval Lengths within Veins in Meters



Source: Kirkham, 2025.

Figure 11-10: Scatterplot of Assay Interval Lengths within Veins in Meters versus Gold Grade

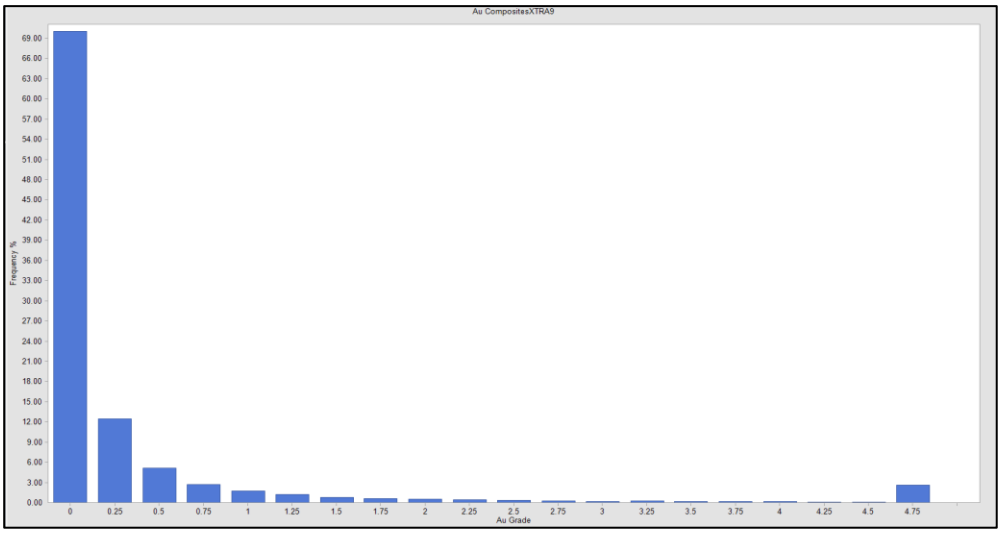


Source: Kirkham, 2025.

Figure 11-11 and Figure 11-12 show histograms of the gold composite values for all composites and for those that are assigned to the high-grade veins, respectively.

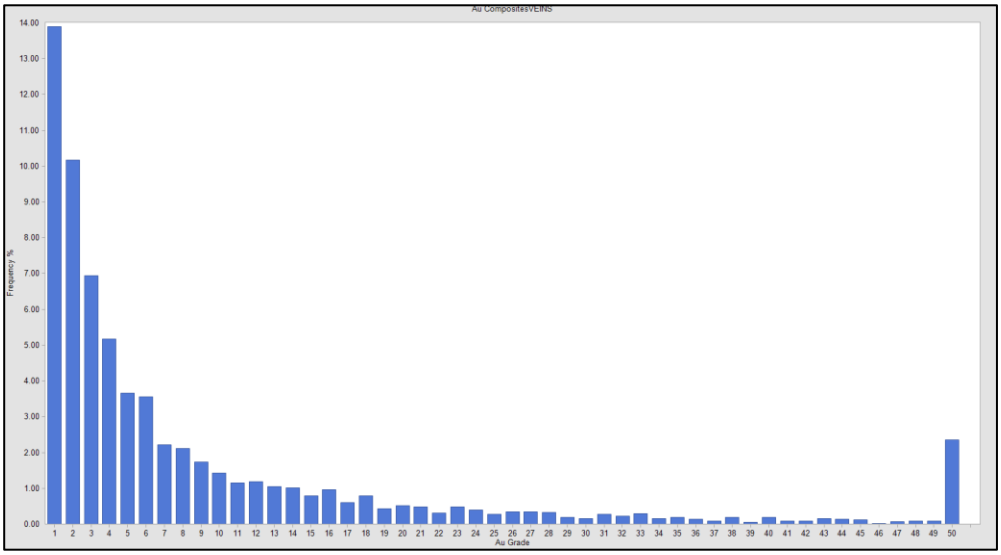
Figure 11-13 and Figure 11-14 show histograms silver composite values for all composites and for those that are assigned to the high-grade veins, respectively. The composite data demonstrates log-normal distributions in both cases.

Figure 11-11: Histogram of Gold Composite Grades (g/t)



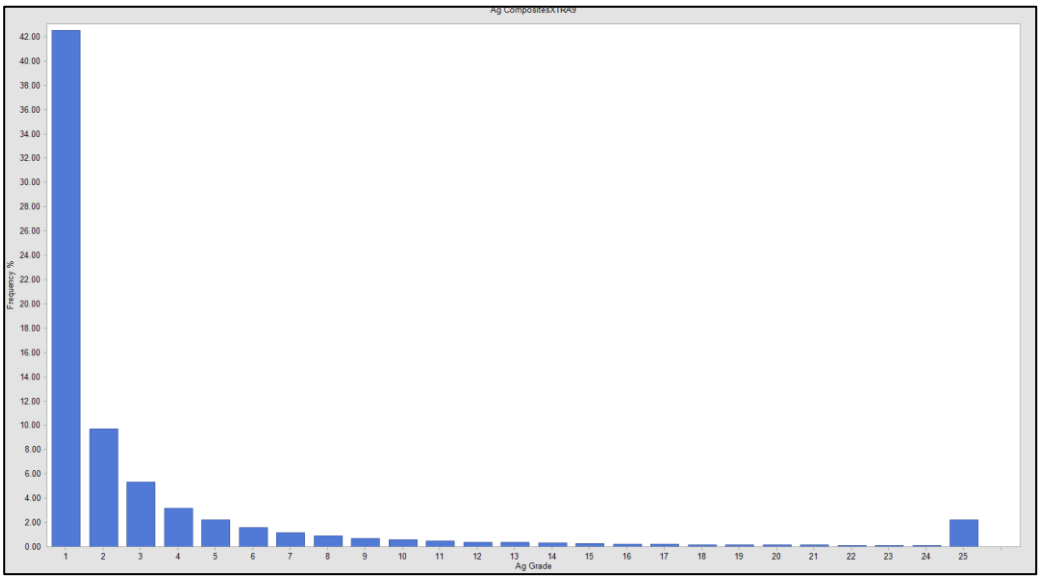
Source: Kirkham, 2025.

Figure 11-12: Histogram of Gold Composite Grades (g/t) with Vein Zones



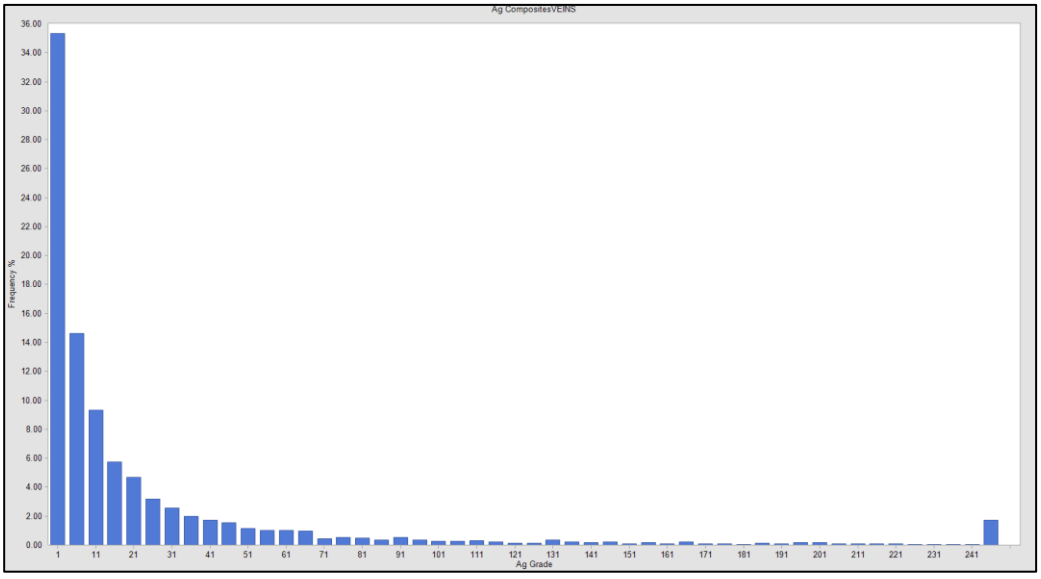
Source: Kirkham, 2025.

Figure 11-13: Histogram of Silver Composite Grades (g/t)



Source: Kirkham, 2025.

Figure 11-14: Histogram of Silver Composite Grades (g/t) with Vein Zones



Source: Kirkham, 2025.

11.5.1 High-Grade Composite Analysis

The high-grade veins for north and south were grouped for statistical, geostatistical and estimation purposes by location and orientation in addition to relative grade profile. The results of these groupings are shown in Table 11-7 where there are two vein groups in the north and six groups in the south.

Table 11-7: Vein Groupings for Derived for Statistical, Geostatistical and Estimation

Vein Domains Group	Vein Ranges
VN Group 1	VN1-VN16, VN21-VN23, VN25
VN Group 2	VN17, VN18-VN20, VN24, VN26-VN30
VS Group 11	VS101 - VS103, NS121
VS Group 12	VS105-VS118
VS Group 13	VS119-VS120
VS Group 14	VS122-VS128
VS Group 15	VS132-VS138
VS Group 16	VS130-VS131, VS139

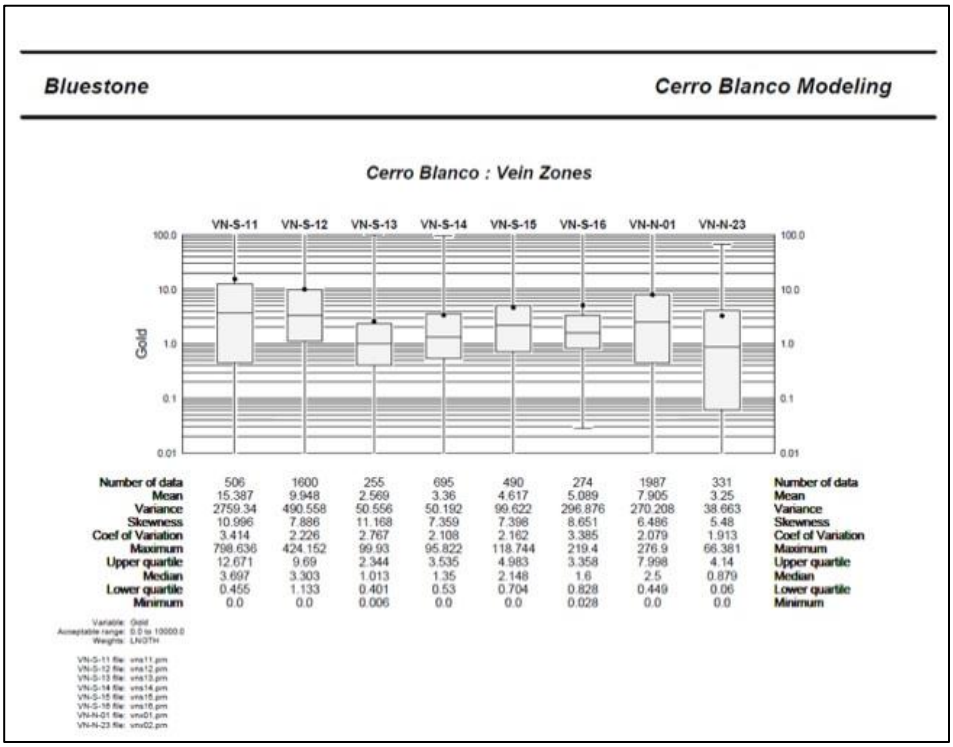
Source: Kirkham, 2025.

Statistical analysis Figure 11-15 and Figure 11-16 show the box plots and basic statistics for the grouped gold and silver composites, respectively, for the high-grade vein domains. Table 11-8 and Table 11-9 show the basic statistics for the 1.5 m gold and silver composite grades within the mineralised domains, respectively. There is a total of 6,107 composites or specifically 3,791 in the north zone and 2,316 in the south zone composites with 30 veins in the north and 36 veins in the south.

The weighted average gold grades for the north zone is 7.97 g/t and 7.28 g/t in the south zone with coefficients of variation (CVs) being 3.2 and 2.1, respectively. Silver grades range from 31.6 g/t in the north and 26.8 g/t in the south with CV's being 3.4 to 3.4, respectively. CVs or variability is typically high for precious metal deposits primarily due to the nuggety nature particularly within epithermal veins; Grade limiting a cutting will further reduce the CVs.

The box plots and statistics show that the mean gold grade very consistent between the north and the south zones. However, the spread (i.e., SD or standard deviation) and therefore the variability (i.e., CV) are higher in the south zone. This may be due to significant outlier grades in the south which has a maximum composite value of 792.3 g/t Au which is in the very high-grade volume in VS-101 versus 276.9 g/t in VN-6 in the north. Similarly, the mean silver grades are higher in the south versus the north at 31.57 g/t and 26.77 g/t, respectfully. In addition, the silver grades have similar distribution characteristics, not only north and south but also within the individual vein groupings, with their being approximately a 4:1 ratio Ag:Au. Furthermore, variability is also significantly greater in the south which is partially due to significant outlier grades in the south where the maximum composite value is 3,540 g/t Ag in the South within VS-106 versus 1,257 g/t in the north within VN-5.

Figure 11-15: Box Plot of Gold Composites for Veins



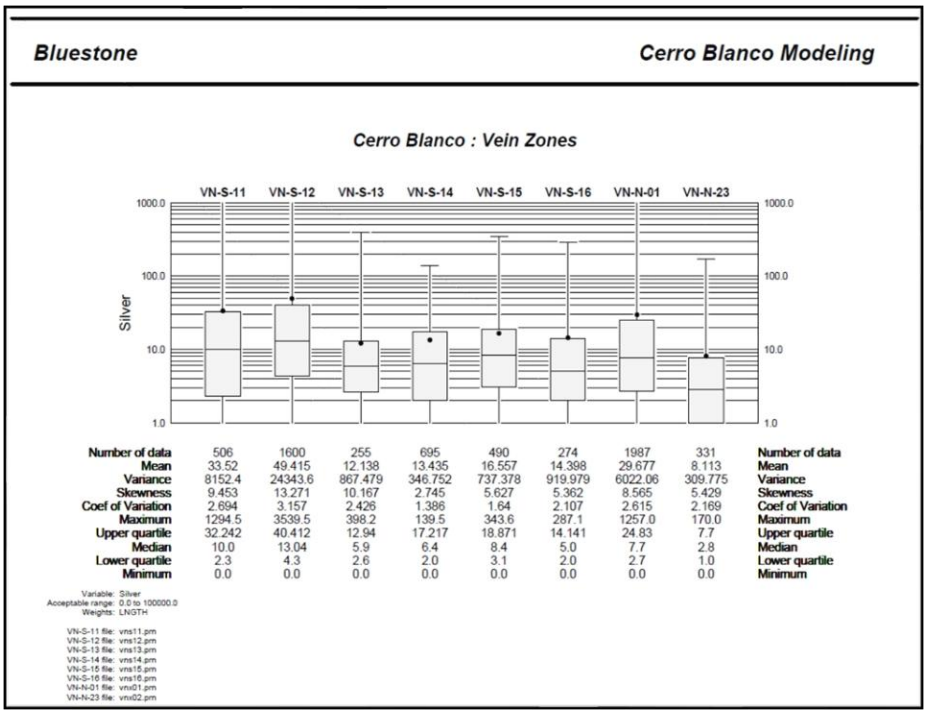
Source: Kirkham, 2025.

Table 11-8: Au Composite Statistics Weighted by Length for Veins

Gold (g/t) Composites	South	North
Valid	3,791	2,316
Length	5,536	3,249.2
Minimum	0	0
Maximum	798.64	276.90
Mean	7.97	7.28
1st Quartile	0.70	0.35
Median	2.30	2.33
3rd Quartile	6.48	7.20
Standard Deviation	25.32	15.54
Variance	641.32	241.43
Coefficient of Variation	3.2	2.1

Source: Kirkham, 2025.

Figure 11-16: Box Plot of Silver Composites for Veins



Source: Kirkham, 2025.

Table 11-9: Silver Composite Statistics Weighted by Length for Veins

Silver Composites	South (g/t)	North (g/t)
Valid	3,791	2,316
Length	5,536	3,249.2
Minimum	0	0
Maximum	3,539.5	1,257.0
Mean	31.57	26.77
1st Quartile	3.03	2.34
Median	8.96	6.71
3rd Quartile	25.36	21.99
Standard Deviation	108.70	72.83
Variance	11814.74	5303.76
Coefficient of Variation	3.4	2.7

Source: Kirkham, 2025.

11.5.2 Low-Grade Composite Analysis

Figure 11-17 and Figure 11-18 show the box plots and basic statistics for the grouped (Table 11-10) gold and silver composites, respectively, for the low-grade estimation domains. Table 11-11 and Table 11-12 show the basic statistics for the 1.5 m gold and silver composite grades within the low-grade domains, respectively.

Table 11-10: Numeric Codes for Lithologies

CODE	Litho Unit
60	Salinas (SVC)
61	Sinter (SS)
62	MAT
70	MBT
71	MCV
72	MVO
73	MAT
74	MSS
75	MLS
99	Outside

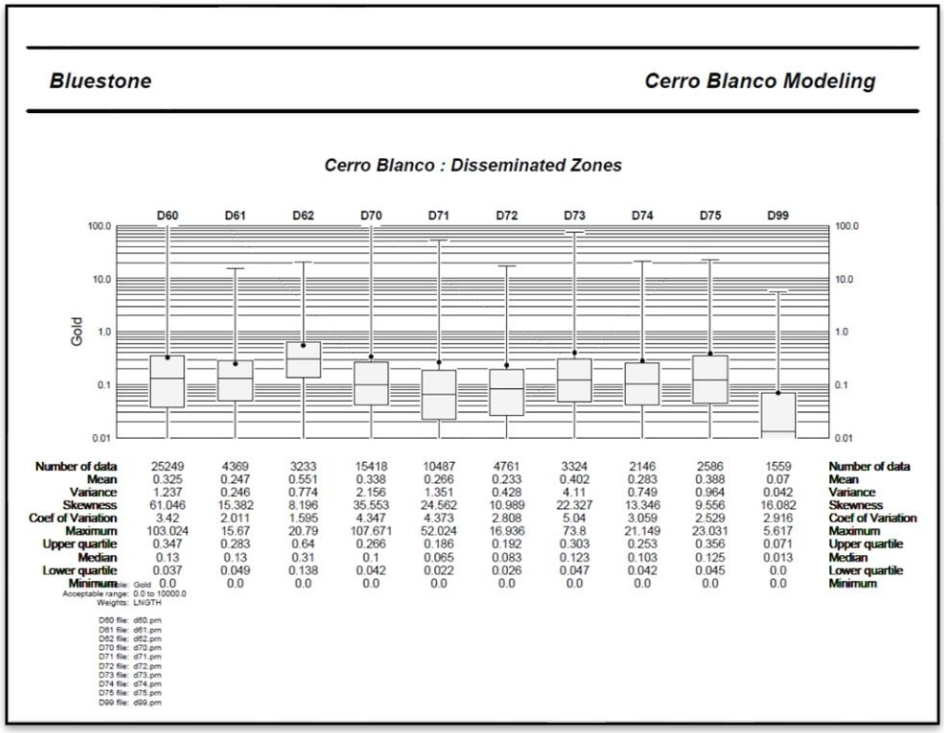
Source: Kirkham, 2025.

The low-grade envelopes show weighted average gold grades of between 0.23 and 0.55 g/t, whilst CVs between 1.6 and 5.0 show moderate to very high variability which are addressed by a conservative grade limiting and cutting strategy. It is interesting to note that the Salinas (Svc)are markedly higher grade than grade than those analysed previously which have increased from 0.19 g/t to 0.32 g/t. This may be primarily attributable updated and revised modeling of the Salinas and Sinter units which was guided by the 2021 drilling program that focussed on delineating and defining the surface resources. In addition, the Salinas Group Basal Conglomerate (Scgl) is a significantly higher-grade unit which has mean gold grade 0.55 g/t, has been defined by the updated modeling.

The mean Silver grades range from 1.7 to 3.4 g/t which is also lower than the 3.6 to 6.9 g/t ranges for the low-grade envelopes previously, with the CVs ranging the spectrum from low (1.2) to extreme (maximum of 39.0). As with the gold, grade limiting or cutting will further reduce the CVs. Again, it is clear that the low-grade domain composites require aggressive cutting.

In addition, the silver and gold grades have similar distribution characteristics, with their being an approximately a 7:1 ratio Ag:Au.

Figure 11-17: Box Plot of Gold Composites for Low-Grade Domains



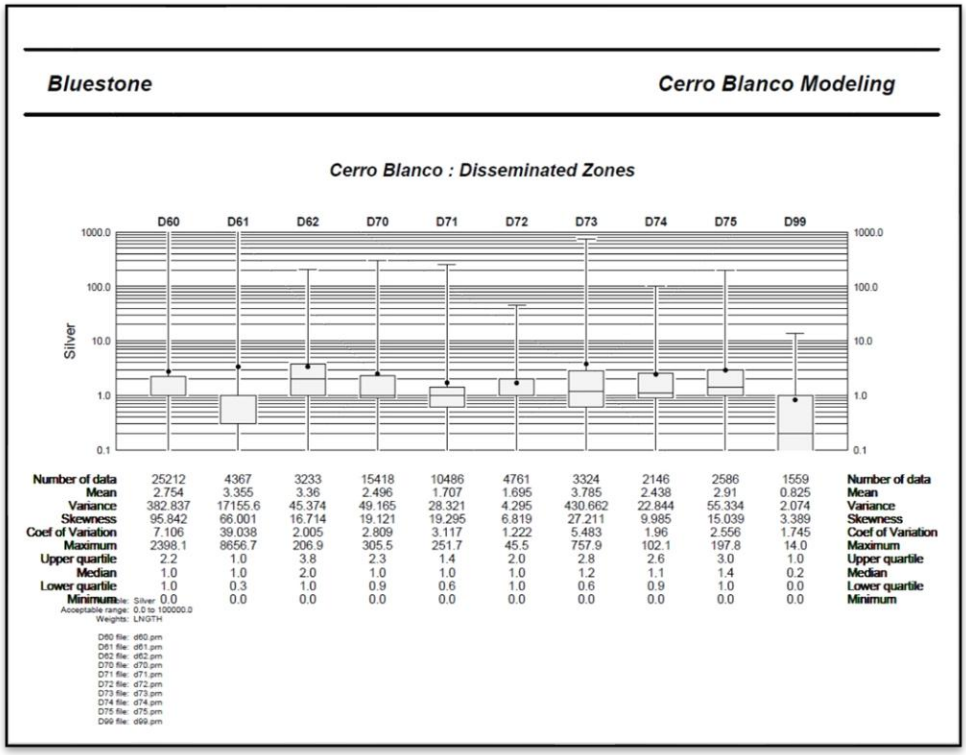
Source: Kirkham, 2025.

Table 11-11: Gold Composite Statistics Weighted by Length for Low-Grade Domains

Domain Code	Domain Name	#	Length (m)	Maximum (g/t)	Mean (g/t)	CV
60	Svc	25,248	37,832.51	103.02	0.32	3.4
61	Ss	4,369	6,556.73	15.67	0.25	2.0
62	Scgl	3,233	4,848.43	20.79	0.55	1.6
70	Mbt	15,418	23,098.32	107.67	0.34	4.3
71	Mcv	10,487	15,718.36	52.02	0.27	4.4
72	Mvo	4,761	7,125.94	16.94	0.23	2.8
73	Mat	3,324	4,934.78	73.80	0.40	5.0
74	Mss	2,146	3,217.06	21.15	0.28	3.1
75	Mls	2,586	3,871.43	23.03	0.39	2.5
99	Outside	1,559	2,336.16	5.62	0.07	2.9

Source: Kirkham, 2025.

Figure 11-18: Box Plot of Silver Composites for Low-Grade Domains



Source: Kirkham, 2025.

Table 11-12: Silver Composite Statistics Weighted by Length for Low-Grade Domains

Domain Code	Domain Name	#	Length (m)	Maximum (g/t)	Mean (g/t)	CV
60	Svc	25,211	37,777.01	2,398.10	2.75	7.1
61	Ss	4,367	6,553.73	8,656.70	3.36	39.0
62	Scgl	3,233	4,848.43	206.9	3.36	2.0
70	Mbt	15,418	23,098.32	305.5	2.5	2.8
71	Mcv	10,486	15,717.11	251.7	1.71	3.1
72	Mvo	4,761	7,125.94	45.5	1.7	1.2
73	Mat	3,324	4,934.78	757.9	3.78	5.5
74	Mss	2,146	3,217.06	102.1	2.44	2.0
75	Mls	2,586	3,871.43	197.8	2.91	2.6
99	Outside	1,559	2,336.16	14	0.83	1.8

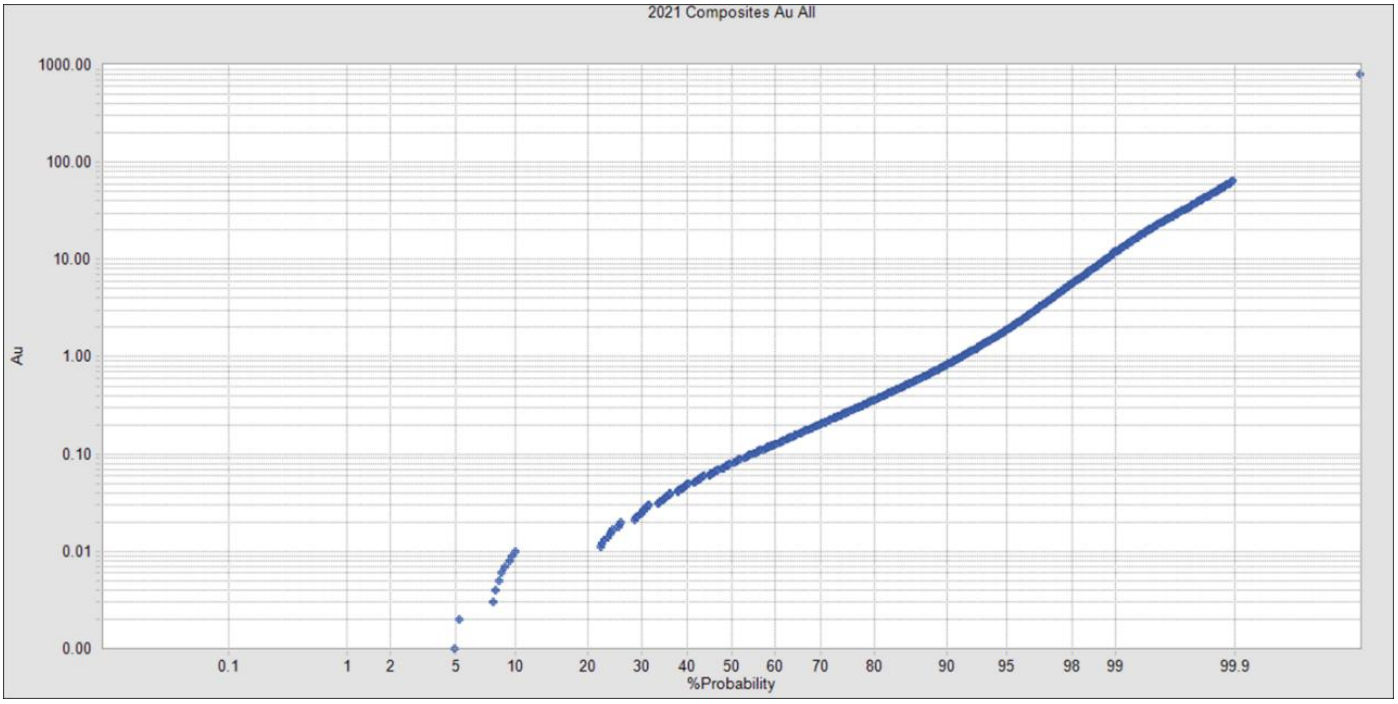
Source: Kirkham, 2025.

11.6 Evaluation of Outlier Assay Values

During the estimation process, the influence of outlier composites is controlled to limit their influence and to ensure against over-estimation of metal content. The high-grade outlier thresholds were chosen by domain and are based on an analysis of the breaks in the cumulative frequency plots for each of the vein groupings and the individual low-grade domains. Figure 11-19 and Figure 11-20 show examples of the gold and silver cumulative frequency plots for all composites, respectively.

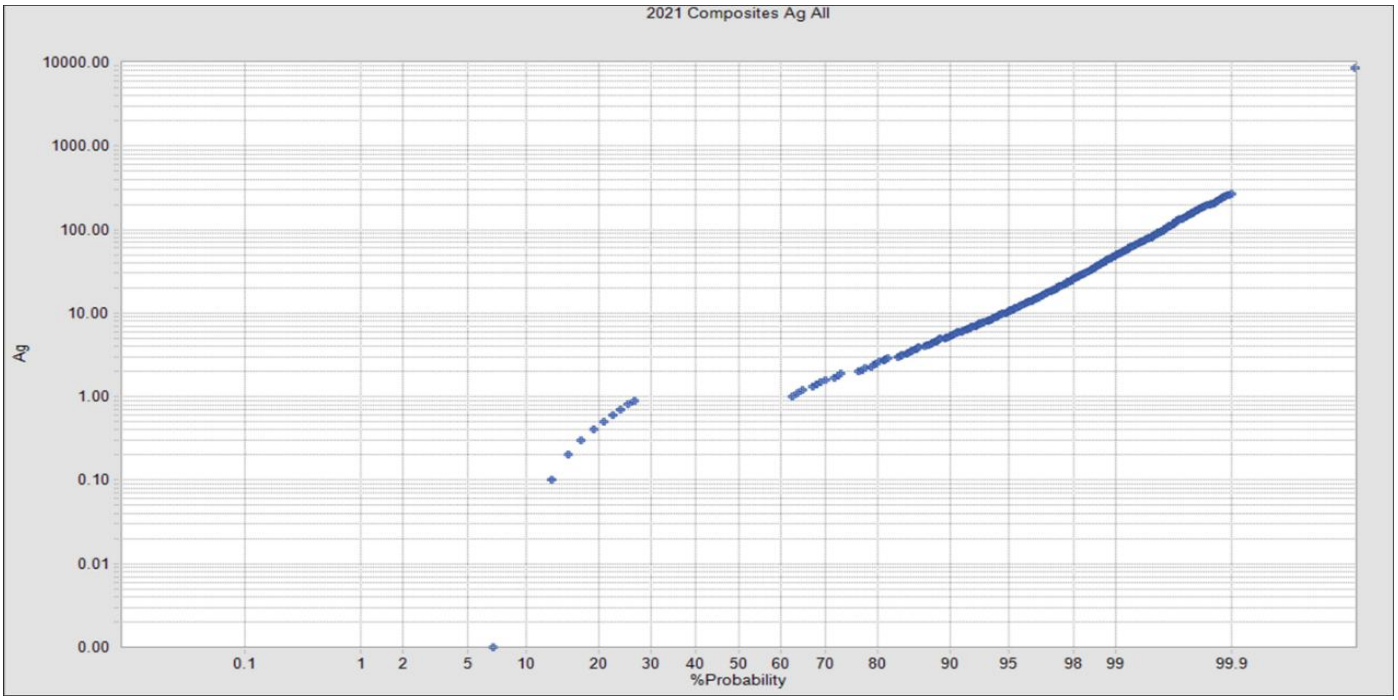
In the case of the gold composites, within the high-grade vein domains, values as high as 110 g/t were cut, with those as high as 500 g/t for silver cut. Table 11-13 shows the various cut thresholds for the vein groupings and Table 11-14 shows those for the low-grade domains.

Figure 11-19: Au Cumulative Frequency Plot



Source: Kirkham, 2025.

Figure 11-20: Ag Cumulative Frequency Plot



Source: Kirkham, 2025.

Table 11-13: Cut Grades for Au & Ag within Vein Domains

Vein Domains Group	Domains	Au Cut Threshold (g/t)	Ag Cut Threshold (g/t)
VN Group 1	VN1-VN16, VN21-VN23, VN25	80	280
VN Group 2	VN17, VN18-VN20, VN24, VN26-VN30	15	40
VS Group 11	VS101 - VS103, VS121	110	180
VS Group 12	VS105-VS118	110	500
VS Group 13	VS119-VS120	10	110
VS Group 14	VS122-VS128	22	90
VS Group 15	VS132-VS138	20	95
VS Group 16	VS130-VS131, VS139	50	110

Source: Kirkham, 2025.

Table 11-14: Cut Grades for Au & Ag within Low-Grade Domains

Low-Grade Domain Name	Domain Code	Au Cut Threshold (g/t)	Ag Cut Threshold (g/t)
Salinas	60	11	110
Sinter	61	6	50
SCGL	62	10	50
MBT	70	15	50
MCV	71	4.5	15
MVO	72	4.5	45.5
MAT	73	4	50
MSS	74	7	35
MLS	75	5	50
Outside	99	0.6	10

Source: Kirkham, 2025.

Table 11-15 and Table 11-16 shows the effects of cutting the outlier grades within the high-grade vein domain groupings and the low-grade Salinas and Mita units, respectively. The conclusion is that the cutting strategy is highly successful in addressing the outlier grade populations, both within the high-grade veins and the lower grade Salinas and Mita units.

Table 11-15: Cut vs. Uncut Comparisons for Gold and Silver Composites within the High-grade Vein Domain Groupings

Au	Maximum (g/t)	Mean (g/t)	CV	Cut Threshold (g/t)	Mean (g/t)	CV	Mean (g/t)	CV
1	276.90	7.90	2.1	80	7.53	1.7	-5%	-16%
2	66.38	3.27	1.9	15	2.87	1.4	-12%	-26%
11	798.64	15.39	3.4	110	11.91	1.8	-23%	-48%
12	424.15	9.95	2.2	110	9.38	1.8	-6%	-19%
13	99.93	2.57	2.8	10	2.03	1.3	-21%	-54%
14	95.82	3.36	2.1	22	2.99	1.4	-11%	-31%
15	118.74	4.65	2.2	20	3.80	1.2	-18%	-45%
16	219.40	5.09	3.4	50	3.89	1.9	-24%	-43%
Total	798.64	7.70	2.9	110	6.93	2.0	-10%	-32%

Ag	Maximum (g/t)	Mean (g/t)	CV	Cut Threshold (g/t)	Mean (g/t)	CV	Mean (g/t)	CV
1	1,257.0	29.68	2.6	280	26.56	1.9	-11%	-28%
2	170.0	8.20	2.2	40	6.65	1.4	-19%	-33%
11	1,294.5	33.52	2.7	180	26.97	1.5	-20%	-44%
12	3,539.5	49.42	3.2	500	42.88	1.9	-13%	-40%
13	398.2	12.14	2.4	110	10.82	1.5	-11%	-40%
14	139.5	13.44	1.4	90	13.18	1.3	-2%	-5%
15	343.6	16.74	1.6	95	15.55	1.3	-7%	-22%
16	287.1	14.40	2.1	110	12.91	1.6	-10%	-22%
Total	3,539.5	29.75	3.3	500	26.16	2.1	-12%	-37%

Source: Kirkham, 2025.

Table 11-16: Cut vs. Uncut Comparisons for Gold and Silver Composites within the Salinas and Mita Domains

Au	Maximum (g/t)	Mean (g/t)	CV	Cut Grade (g/t)	Mean (g/t)	CV	Mean (%)	CV (%)
60	103.02	0.324	3.4	11	0.316	2.0	-2%	-41%
61	15.67	0.247	2.0	6	0.242	1.6	-2%	-21%
62	20.79	0.551	1.6	10	0.546	1.5	-1%	-9%
70	107.67	0.361	4.0	15	0.344	2.7	-5%	-33%
71	52.02	0.260	4.5	4.5	0.223	2.4	-14%	-46%
72	16.94	0.233	2.8	4.5	0.221	2.2	-5%	-20%
73	73.80	0.403	5.0	4	0.306	1.8	-24%	-64%
74	21.15	0.284	3.1	7	0.268	2.3	-6%	-24%
75	23.03	0.388	2.5	5	0.360	1.9	-7%	-24%
Total	107.67	0.327	3.7	15	0.308	2.3	-6%	-38%
Ag	**Maximum (g/t)**	**Mean (g/t)**	**CV**	**Cut Grade (g/t)**	**Mean (g/t)**	**CV**	**Mean (%)**	**CV (%)**
60	2,398.1	2.75	7.1	110	2.55	2.1	-7%	-70%
61	8,656.7	3.36	39.0	50	1.35	1.9	-60%	-95%
62	206.9	3.36	2.0	50	3.24	1.4	-4%	-32%
70	305.5	2.61	2.8	50	2.46	1.8	-6%	-35%
71	251.7	1.69	3.1	15	1.43	1.4	-15%	-54%
72	45.5	1.70	1.2	45.5	1.70	1.2	0%	0%
73	757.9	3.79	5.5	50	2.97	2.0	-22%	-63%
74	102.1	2.44	2.0	35	2.35	1.5	-4%	-21%
75	197.8	2.91	2.6	50	2.73	1.7	-6%	-35%
Total	8,656.7	2.60	13.3	110	2.28	1.9	-12%	-85%

Source: Kirkham, 2025.

11.7 Specific Gravity Estimation

Table 11-17 shows the specific gravity (SG) assignment by zone using 1,308 individual measurements and standard water displacement methods. The SG assigned for the veins is determined to 2.52, which is derived from 534 measurements. There is an expanded ongoing program to increase the number and distribution of SG measurements. It is recommended that future work programs should continue to include SG measurements to expand the density distributions, particularly within the upper lithology units.

Table 11-17: SG Zone Assignments

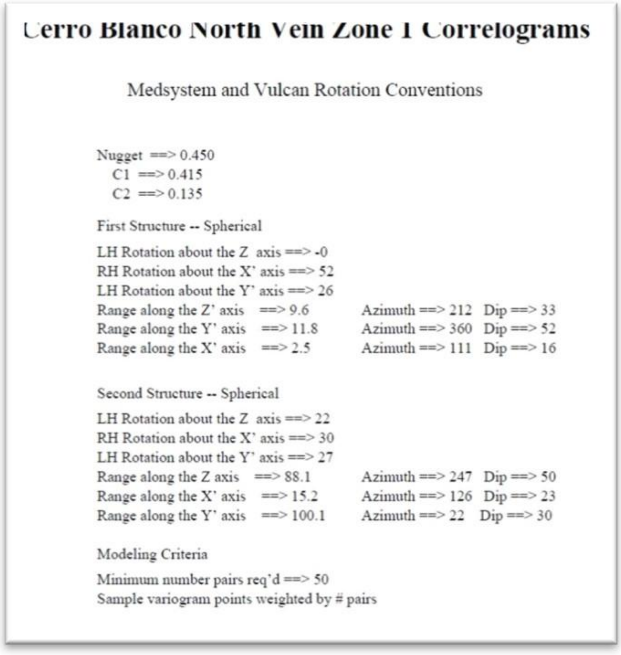
Lithology Group	Domain / Rock	#	Density (gm/mm^3)	Average Density (gm/mm^3)
SALINAS	Ss	27	2.49	
	Scgl	35	2.46	
GROUP	Svc	115	2.46	
	Rp	6	2.48	
	Total	183		2.47
	Mat	48	2.54	
	Mbt	272	2.58	
MITA GROUP	Mss	88	2.56	
	Mls	36	2.62	
	Mcv	102	2.59	
	Mvo	38	2.52	
	Silt	7	2.56	
	Total	591		2.57
VEIN	Vt	534	2.52	
	Total	**1308**		**2.54**

Source: Kirkham, 2025.

11.8 Variography

Experimental variograms and variogram models in the form of correlograms were generated for gold and silver grades. The definition of nugget value was derived from the downhole variograms. The correlograms for gold and silver within veins in the south and north zones are shown in Figure 11-21, Figure 11-22 and Figure 11-23 for gold and silver, respectively. These variogram models were used to estimate gold and silver grades using ordinary kriging as the interpolator used to estimate the high-grade veins.

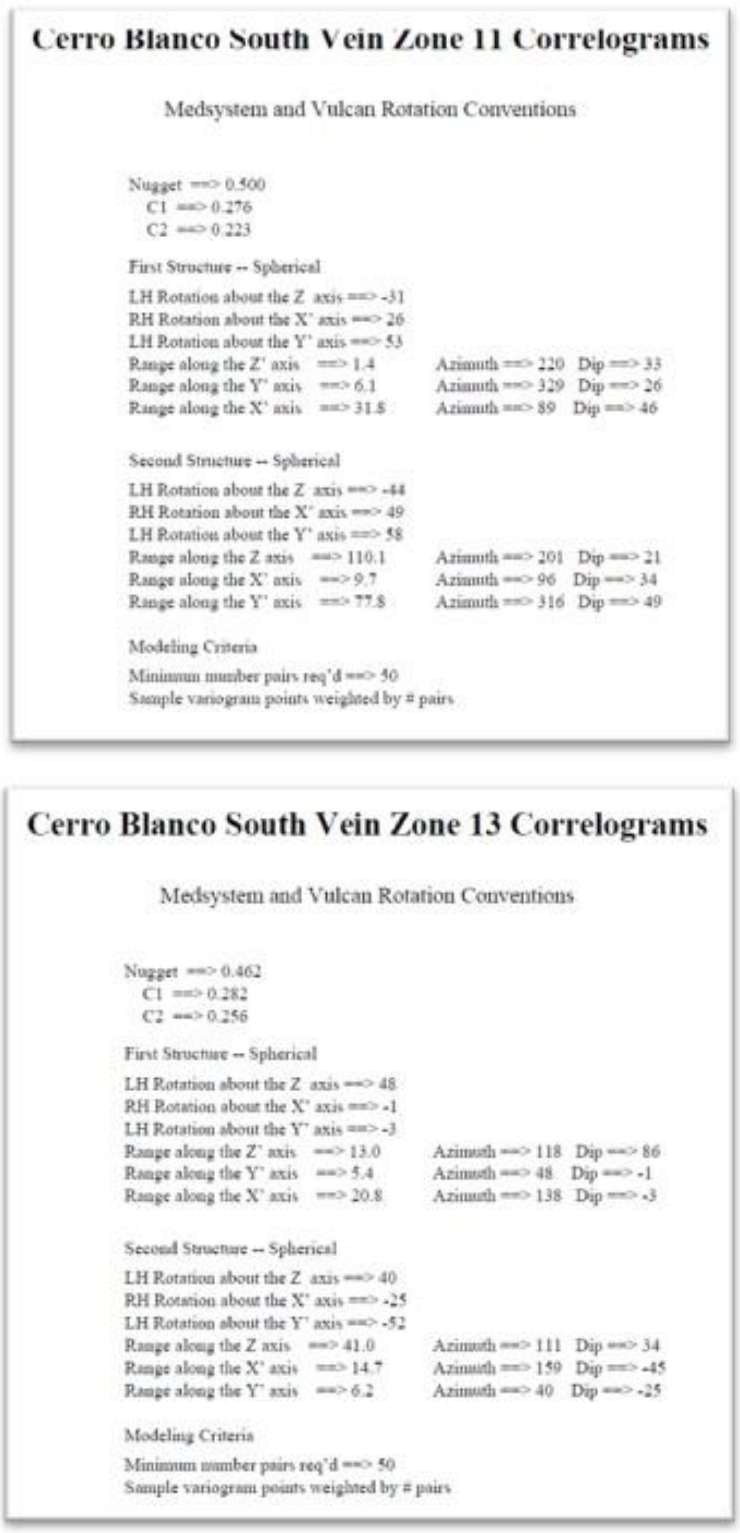
Figure 11-21: Au Corellogram Models





Source: Kirkham, 2021.

Figure 11-22: Ag Correlogram Models



Cerro Blanco South Vein Zone 11 Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.500
C1 ==> 0.276
C2 ==> 0.223

First Structure -- Spherical

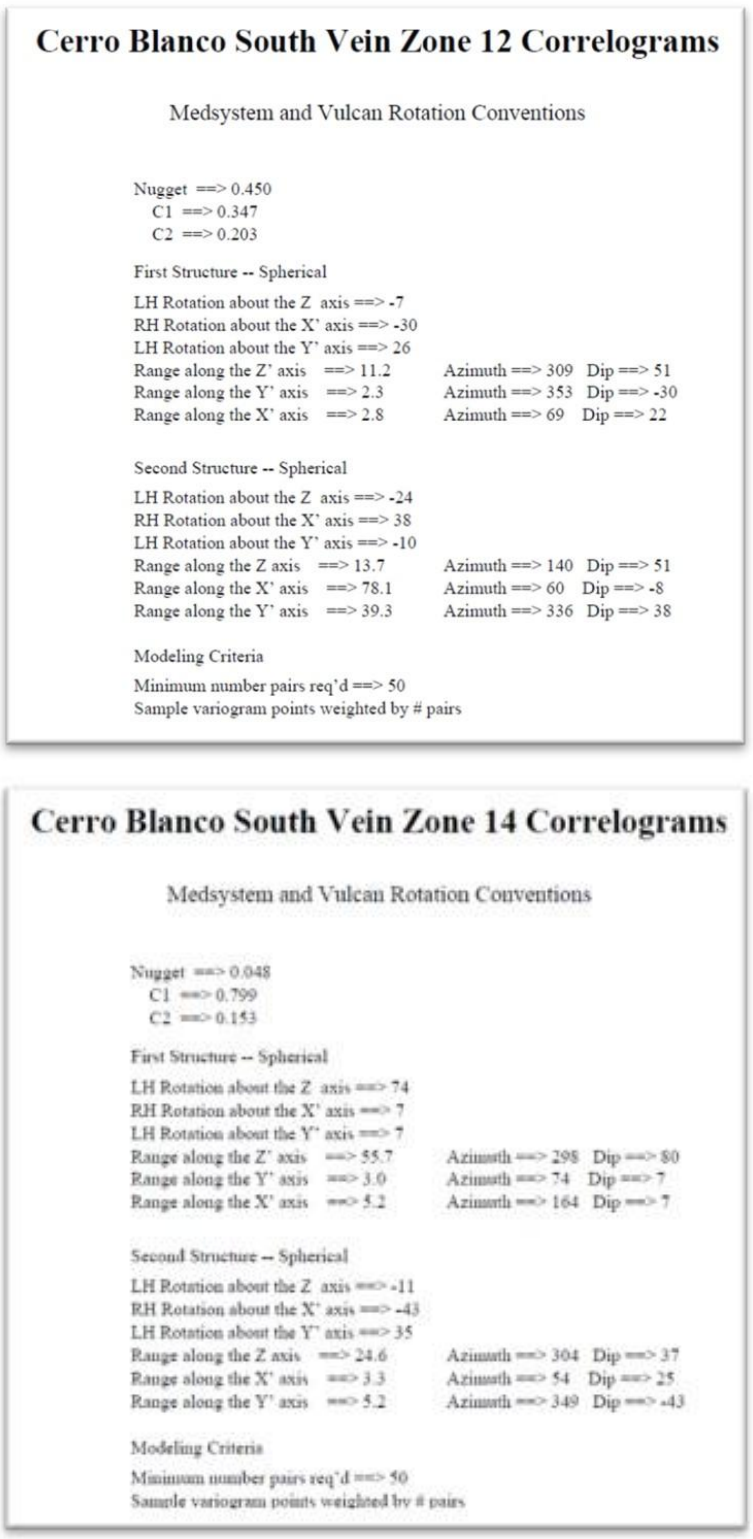
LH Rotation about the Z axis ==> -31
RH Rotation about the X' axis ==> 26
LH Rotation about the Y' axis ==> 53
Range along the Z' axis ==> 1.4 Azimuth ==> 220 Dip ==> 33
Range along the Y' axis ==> 6.1 Azimuth ==> 329 Dip ==> 26
Range along the X' axis ==> 31.8 Azimuth ==> 89 Dip ==> 46

Second Structure -- Spherical

LH Rotation about the Z axis ==> -44
RH Rotation about the X' axis ==> 49
LH Rotation about the Y' axis ==> 58
Range along the Z axis ==> 110.1 Azimuth ==> 201 Dip ==> 21
Range along the X' axis ==> 9.7 Azimuth ==> 96 Dip ==> 34
Range along the Y' axis ==> 77.8 Azimuth ==> 316 Dip ==> 49

Modeling Criteria

Minimum number pairs req'd ==> 50
Sample variogram points weighted by # pairs



Cerro Blanco South Vein Zone 12 Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.450
C1 ==> 0.347
C2 ==> 0.203

First Structure -- Spherical

LH Rotation about the Z axis ==> -7
RH Rotation about the X' axis ==> -30
LH Rotation about the Y' axis ==> 26
Range along the Z' axis ==> 11.2 Azimuth ==> 309 Dip ==> 51
Range along the Y' axis ==> 2.3 Azimuth ==> 353 Dip ==> -30
Range along the X' axis ==> 2.8 Azimuth ==> 69 Dip ==> 22

Second Structure -- Spherical

LH Rotation about the Z axis ==> -24
RH Rotation about the X' axis ==> 38
LH Rotation about the Y' axis ==> -10
Range along the Z axis ==> 13.7 Azimuth ==> 140 Dip ==> 51
Range along the X' axis ==> 78.1 Azimuth ==> 60 Dip ==> -8
Range along the Y' axis ==> 39.3 Azimuth ==> 336 Dip ==> 38

Modeling Criteria

Minimum number pairs req'd ==> 50
Sample variogram points weighted by # pairs

Cerro Blanco South Vein Zone 13 Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.462
C1 ==> 0.282
C2 ==> 0.256

First Structure -- Spherical

LH Rotation about the Z axis ==> 48
RH Rotation about the X' axis ==> -1
LH Rotation about the Y' axis ==> -3
Range along the Z' axis ==> 13.0 Azimuth ==> 118 Dip ==> 86
Range along the Y' axis ==> 5.4 Azimuth ==> 48 Dip ==> -1
Range along the X' axis ==> 20.8 Azimuth ==> 138 Dip ==> -3

Second Structure -- Spherical

LH Rotation about the Z axis ==> 40
RH Rotation about the X' axis ==> -25
LH Rotation about the Y' axis ==> -52
Range along the Z axis ==> 41.0 Azimuth ==> 111 Dip ==> 34
Range along the X' axis ==> 14.7 Azimuth ==> 159 Dip ==> -45
Range along the Y' axis ==> 6.2 Azimuth ==> 40 Dip ==> -25

Modeling Criteria

Minimum number pairs req'd ==> 50
Sample variogram points weighted by # pairs

Cerro Blanco South Vein Zone 14 Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.048
C1 ==> 0.799
C2 ==> 0.153

First Structure -- Spherical

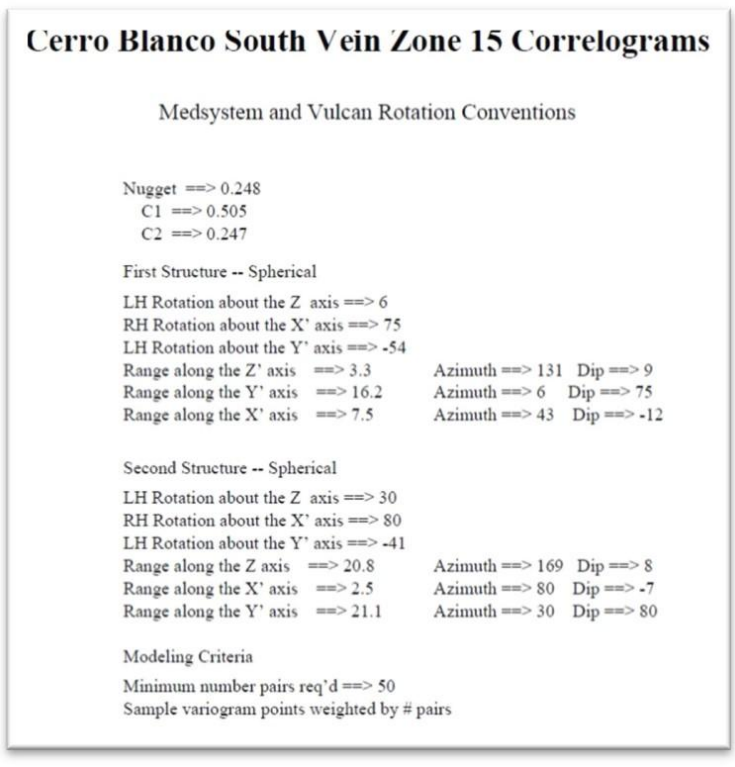
LH Rotation about the Z axis ==> 74
RH Rotation about the X' axis ==> 7
LH Rotation about the Y' axis ==> 7
Range along the Z' axis ==> 55.7 Azimuth ==> 298 Dip ==> 80
Range along the Y' axis ==> 3.0 Azimuth ==> 74 Dip ==> 7
Range along the X' axis ==> 5.2 Azimuth ==> 164 Dip ==> 7

Second Structure -- Spherical

LH Rotation about the Z axis ==> -11
RH Rotation about the X' axis ==> -43
LH Rotation about the Y' axis ==> 35
Range along the Z axis ==> 24.6 Azimuth ==> 304 Dip ==> 37
Range along the X' axis ==> 3.3 Azimuth ==> 54 Dip ==> 25
Range along the Y' axis ==> 5.2 Azimuth ==> 349 Dip ==> -43

Modeling Criteria

Minimum number pairs req'd ==> 50
Sample variogram points weighted by # pairs



Cerro Blanco South Vein Zone 15 Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.248
C1 ==> 0.505
C2 ==> 0.247

First Structure -- Spherical

LH Rotation about the Z axis ==> 6
RH Rotation about the X' axis ==> 75
LH Rotation about the Y' axis ==> -54
Range along the Z' axis ==> 3.3 Azimuth ==> 131 Dip ==> 9
Range along the Y' axis ==> 16.2 Azimuth ==> 6 Dip ==> 75
Range along the X' axis ==> 7.5 Azimuth ==> 43 Dip ==> -12

Second Structure -- Spherical

LH Rotation about the Z axis ==> 30
RH Rotation about the X' axis ==> 80
LH Rotation about the Y' axis ==> -41
Range along the Z axis ==> 20.8 Azimuth ==> 169 Dip ==> 8
Range along the X' axis ==> 2.5 Azimuth ==> 80 Dip ==> -7
Range along the Y' axis ==> 21.1 Azimuth ==> 30 Dip ==> 80

Modeling Criteria

Minimum number pairs req'd ==> 50
Sample variogram points weighted by # pairs



Cerro Blanco South Vein Zone 16 Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.020
C1 ==> 0.770
C2 ==> 0.210

First Structure -- Spherical

LH Rotation about the Z axis ==> -2
RH Rotation about the X' axis ==> 4
LH Rotation about the Y' axis ==> -19
Range along the Z' axis ==> 37.4 Azimuth ==> 100 Dip ==> 71
Range along the Y' axis ==> 16.8 Azimuth ==> 358 Dip ==> 4
Range along the X' axis ==> 5.0 Azimuth ==> 86 Dip ==> -19

Second Structure -- Spherical

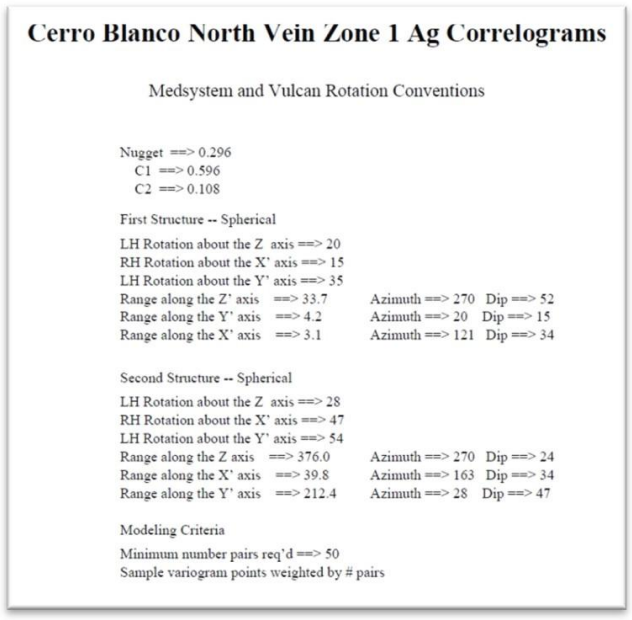
LH Rotation about the Z axis ==> -29
RH Rotation about the X' axis ==> 17
LH Rotation about the Y' axis ==> -8
Range along the Z axis ==> 27.1 Azimuth ==> 124 Dip ==> 71
Range along the X' axis ==> 49.3 Azimuth ==> 58 Dip ==> -8
Range along the Y' axis ==> 80.1 Azimuth ==> 331 Dip ==> 17

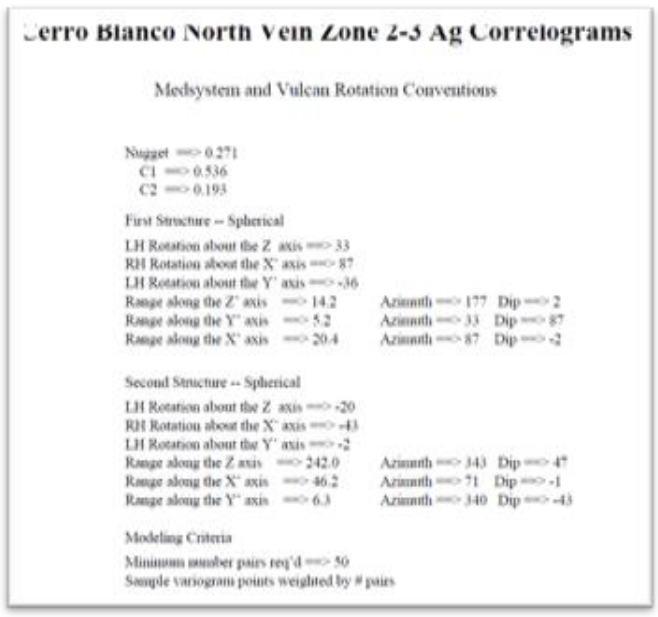
Modeling Criteria

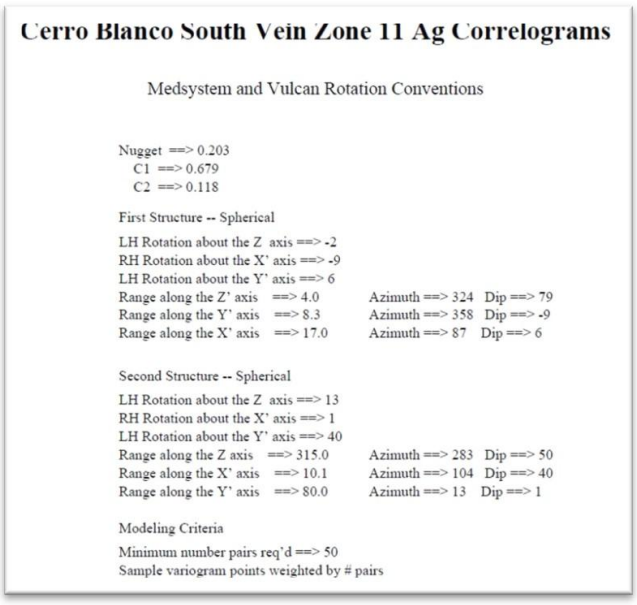
Minimum number pairs req'd ==> 50
Sample variogram points weighted by # pairs

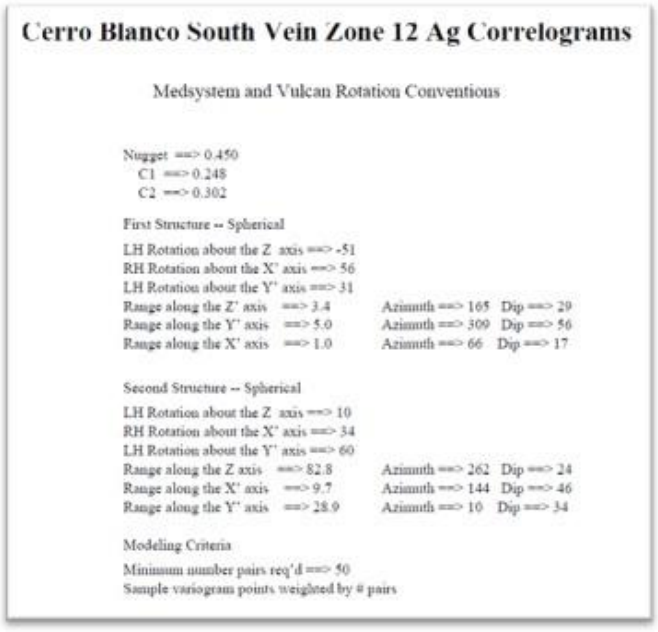
Source: Kirkham, 2021.

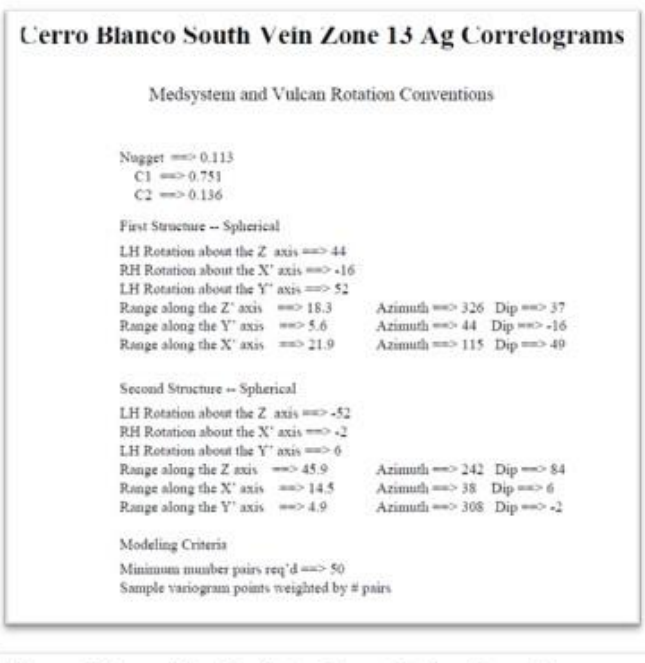
Figure 11-23: Ag Correlogram Models

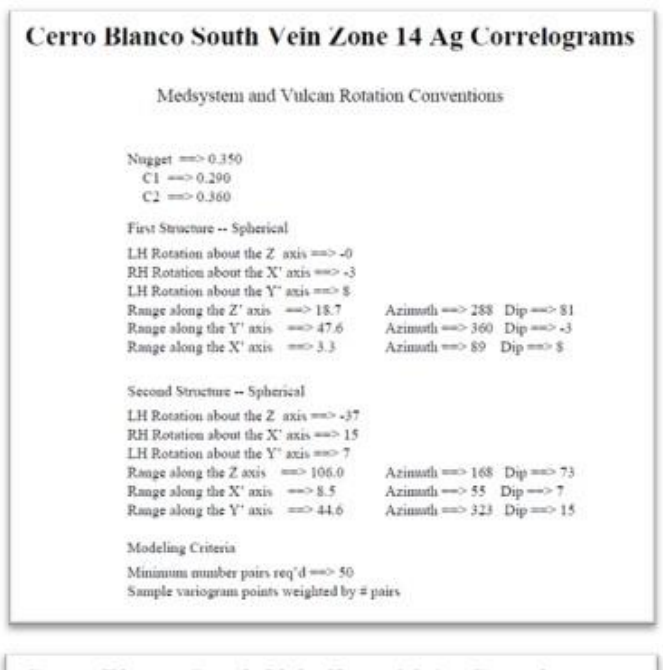


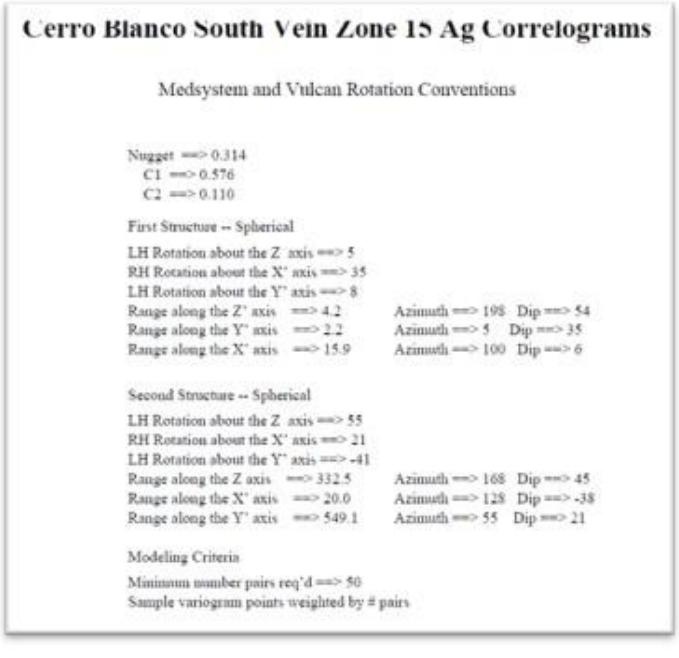


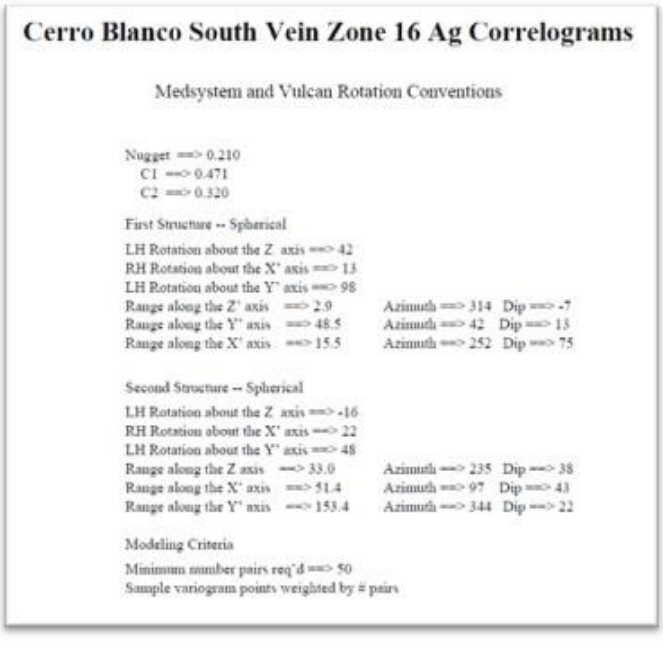












Source: Kirkham, 2021.

In addition, experimental variograms and variogram models in the form of correlograms were also generated for gold and silver grades within the low-grade domains namely, Salinas and Mita units. As above, the definition of nugget value was derived from the downhole variograms. The correlograms models for gold and silver are shown in Table 11-18 and Table 11-19, respectively. These variogram models were used to estimate gold and silver grades using ordinary kriging as the interpolator.

Table 11-18: Geostatistical Model Parameters for Gold by Lithology Unit

CODE	60	61	62	70	71	72	73	74	75
Domain Name	Salinas	Sinter	MAT	MBT	MCV	MVO	MAT	MSS	MLS
Nugget (C0)	0.45	0.1	0.384	0.475	0.5	0.597	0.184	0.588	0.6
First Sill (C1)	0.439	0.512	0.406	0.466	0.456	0.343	0.56	0.236	0.333
Second Sill (C2)	0.111	0.388	0.21	0.059	0.044	0.059	0.256	0.176	0.067
1st Structure									
Range along the Z'	18.1	3.6	9.7	7.2	7.8	7.9	26.9	8.9	2
Range along the X'	10.8	26.9	9.4	4.9	4.9	22.3	2.9	33.9	5.8
Range along the Y'	25.7	2.3	4.5	5.2	5.5	3.6	31.8	1.9	44.2
R1 about the Z	-151	-91	-7	4	-21	15	91	1	37
R2 about the X'	35	-52	8	-37	-50	57	-47	41	-2
R3 about the Y'	-4	2	-11	56	57	81	73	-42	-4
2nd Structure									
Range along the Z'	136.6	152.6	204.4	196.5	100.8	275	76.2	12	302.3
Range along the X'	103	56.1	94.3	63.6	55	67.5	13.6	82	126.8
Range along the Y'	402.9	105.6	49.8	134.6	289	332	26.5	246	1405.4
R1 about the Z	2	24	45	2	-73	34	32	19	-15
R2 about the X'	-10	56	1	24	58	171	14	41	37
R3 about the Y'	-4	-23	-14	30	54	-28	33	54	41

Source: Kirkham, 2025.

Table 11-19: Geostatistical Model Parameters for Silver by Lithology Unit

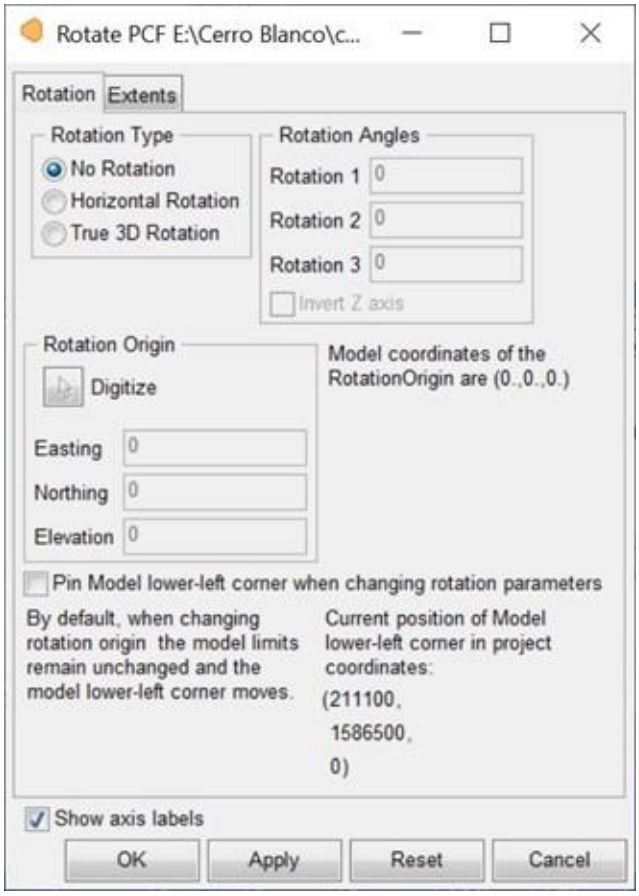
CODE	60	61	62	70	71	72	73	74	75
Domain Name	Salinas	Sinter	MAT	MBT	MCV	MVO	MAT	MSS	MLS
Nugget (C0)	0.4	0.231	0.3	0.425	0.167	0.462	0.35	0.279	0.274
First Sill (C1)	0.415	0.528	0.465	0.494	0.542	0.377	0.533	0.599	0.44
Second Sill (C2)	0.185	0.241	0.235	0.081	0.291	0.161	0.117	0.122	0.285
1st Structure									
Range along the Z'	20.2	3.8	8.2	6.2	17.3	6.8	4.9	5.1	20.1
Range along the X'	4	32	3.4	9.3	8.3	17.9	30.6	37.4	7.9
Range along the Y'	8.8	2.7	33.5	4.2	3.8	43.8	19.8	2.7	1.8
R1 about the Z	1	7	-67	-34	4	23	-14	-54	-18
R2 about the X'	-44	-13	87	23	-10	9	-31	-15	-1
R3 about the Y'	41	-24	20	52	-15	-22	33	-53	-20
2nd Structure									
Range along the Z'	278.7	133.2	265.1	153	157.8	132.8	77.6	70.3	108.3
Range along the X'	45.5	10	86.3	67.6	16.8	278.3	19	115.7	13.4
Range along the Y'	70.8	89.5	73.4	208.2	27.9	71	117.9	67.3	36.7
R1 about the Z	-16	8	49	42	15	7	61	-27	79
R2 about the X'	21	32	43	182	-30	35	10	90	15
R3 about the Y'	71	-8	21	-39	36	-44	-39	-5	-20

Source: Kirkham, 2025.

11.9 Block Model Definition

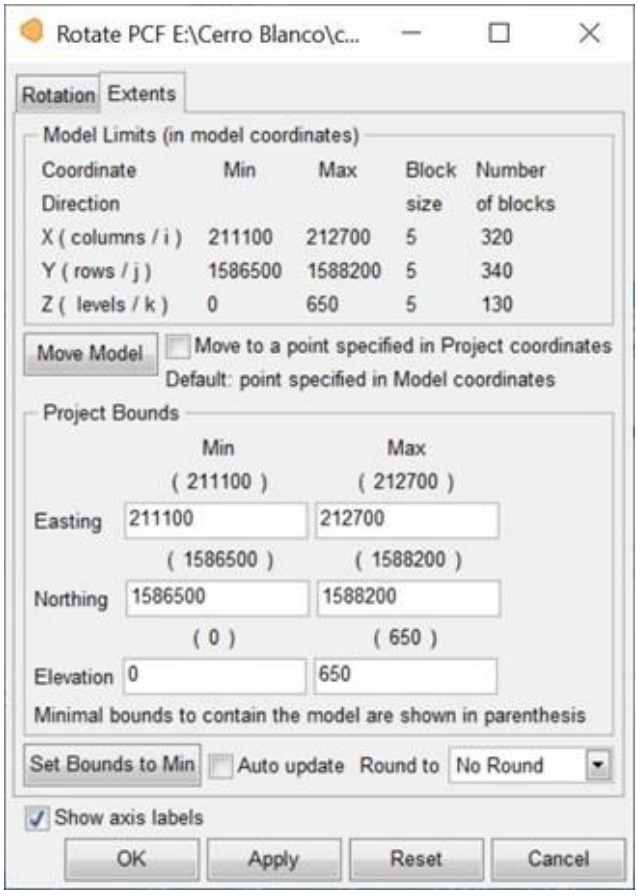
The block model used for estimating the resources was defined according to origin and orientation shown in Figure 11-24 and the limits specified in Figure 11-25.

Figure 11-24: Block Model Origin & Orientation



Source: Kirkham, 2025.

Figure 11-25: Block Model Extents & Dimensions



Source: Kirkham, 2025.

The block model employs whole blocking for ease of mine planning and is orthogonal and non-rotated, roughly reflecting the orientation of the north and the south vein sets within the deposit. The block size chosen was 5 m by 5 m by 5 m. Note that MineSight™ uses the centroid of the blocks as the origin.

11.10 Resource Estimation Methodology

The estimation strategy was a two-step process that entailed estimating the high-grade vein component of each block and then the low-grade mineralised host rock component. Once completed, the final whole block grades were created by determined by way of a weighted average calculation.

The estimation plan for the high-grade vein component was:

- vein code of modelled mineralization stored in each block along with partial percentage

- specific gravity estimation for the vein
- block gold and silver grade estimation by ordinary kriging
- one pass estimation for each individual vein using hard boundaries.

A minimum of three composites and maximum of nine composites, and a maximum of three composites per hole were used to estimate block grades. Following Herco analysis, it was determined there is an appropriate amount of smoothing.

For the vein domains that make up the Era Dorada deposit, the search ellipsoids are omni-directional to a maximum of 100 m; however, hard boundaries were used so that the domains are tightly constrained and grade is not smeared between veins.

The estimation plan for the low-grade mineralised host rock component included:

- domain code of modelled mineralization stored in each block
- specific gravity estimation based on rock type code
- block gold and silver grade estimation by ordinary kriging
- one pass estimation for each domain using hard boundaries.

A minimum of three composites and maximum of twelve composites, and a maximum of three composites per hole were informed to estimate block grades. Following Herco analysis, it was determined there is an appropriate amount of smoothing for the low-grade domains.

For the vein domain domains that make up the Era Dorada deposit, the search ellipsoids are omni- directional to a maximum of 100 m, and hard boundaries were used so that grade is not smeared between the units.

11.11 Mineral Resource Classification

Mineral resources were estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserve Best Practices” Guidelines (2019). Mineral resources are not mineral reserves and do not have demonstrated economic viability. Mineral resources for the Era Dorada deposit were classified according to the SEC Regulation S-K Subpart 1300 by Garth Kirkham, P.Geo., of Kirkham Geosystems Ltd. (KGL), an Independent Qualified Person.

The mineral resources may be impacted by further infill and exploration drilling that may result in an increase or decrease in future resource evaluations. The mineral resources may also be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors.

Mineral resource categories can be based on an estimate of uncertainty within a theoretical measure of confidence. The thresholds for the uncertainty and confidence are based on rules of thumb, however they can vary from project to project depending upon the risk tolerance that the project and the company is willing to bear. Indicated resources

may be estimated so the uncertainty of yearly production is approximately ±15% with 90% confidence and Measured resources may be estimated so the uncertainty of quarterly production is no greater than ±15% with 90% confidence. The results presented above indicate the reliability is around ±15% for the assumed production rate at roughly 50 m spacing.

It should also be noted that the confidence limits only consider the variability of grade within the deposit. There are other aspects of deposit geology and geometry such as geological contacts or the presence of faults or offsetting structures that may impact the drill spacing.

The spacing distances are intended to define contiguous volumes and they should allow for some irregularities due to actual drill hole placement. The final classification volume results typically must be adjusted manually to come to a coherent classification scheme. The thresholds should be used as a guide and boundaries interpreted and defined to ensure continuity.

Drill hole spacing is sufficient for preliminary geostatistical analysis and evaluating spatial grade variability. The classification of resources was based primarily upon distance to the nearest composite; however, the multiple quantitative measures, as listed below, were inspected and taken into consideration.

The estimated blocks were classified according to:

- confidence
- in interpretation of the mineralised zones
- number of composites used to estimate a block
- number of composites allowed per drill hole
- distance to nearest composite used to estimate a block
- average distance to the composites used to estimate a block.

Therefore, the following lists the spacing for each resource category to classify the resources assuming the current rate of metal production:

- Measured: Note that based on the CIM definitions, continuity must be demonstrated in the designation of measured (and indicated) resources. Therefore, measured resources were delineated from at least three drill holes spaced on a nominal 25 m pattern.
- Indicated: Resources in this category would be delineated from at least three drill holes spaced on a nominal 50 m pattern.
- Inferred: Any material not falling in the categories above and within a maximum 100 m of one hole.

To ensure continuity, the boundary between the indicated and inferred categories was contoured and smoothed, eliminating outliers and orphan blocks. The spacing distances are intended to define contiguous volumes and they

should allow for some irregularities due to actual drill hole placement. The final classification volume results typically must be adjusted manually to come to a coherent classification scheme.

Mineral resources are classified under the categories of measured, indicated and inferred according to SK-1300. Mineral resource classification for gold was based primarily on drill hole spacing and on continuity of mineralization. Measured resources were defined as blocks within a distance to nearest composite of 25 m. Indicated resources were defined as those within a distance to three Drill holes of less than ~50 m. Inferred resources were defined as those with an average drill hole spacing of less than ~100 m and meeting additional requirements. All resources are constrained in the following manner: primarily, by the continuous vein solids, secondarily, the low-grade envelope, and thirdly, by the Salinas group tertiary member. Blocks outside the aforementioned were estimated in a last pass to determine waste grade and volumes. Final resource classification shells were manually constructed on plan and sections.

The suggested classification parameters are roughly consistent with the past classification scheme. Classification in future models may differ, but principal differences should be due to changes in the amount of drilling.

Mineral resource estimates for epithermal gold–silver deposits are inherently uncertain due to strong structural control on mineralization, variable vein thickness, and pronounced short-range grade variability. These characteristics directly affect confidence in geological interpretation and grade continuity and are the primary factors governing mineral resource classification.

The Qualified Person considered all material sources of uncertainty, including drilling density and orientation, sampling and analytical quality, geological and structural interpretation, and grade estimation methodology, and evaluated their cumulative impact when assigning Measured, Indicated, and Inferred mineral resources.

Measured mineral resources are restricted to areas of closely spaced drilling where vein geometry, structural controls, and grade continuity are well constrained. Indicated mineral resources are supported by sufficient drilling to reasonably interpret continuity, but with remaining local geological or grade uncertainty. Inferred mineral resources are based on limited drilling and interpreted continuity that cannot be verified with sufficient confidence.

Sampling and analytical uncertainty reflects core recovery, sample support relative to vein width, analytical variability, and nugget effects common to epithermal gold mineralization. Geological uncertainty arises from the interpretation of faults, veins, and breccia zones that may change orientation or continuity over short distances. Grade estimation uncertainty is elevated due to localized high-grade zones and sharp grade boundaries.

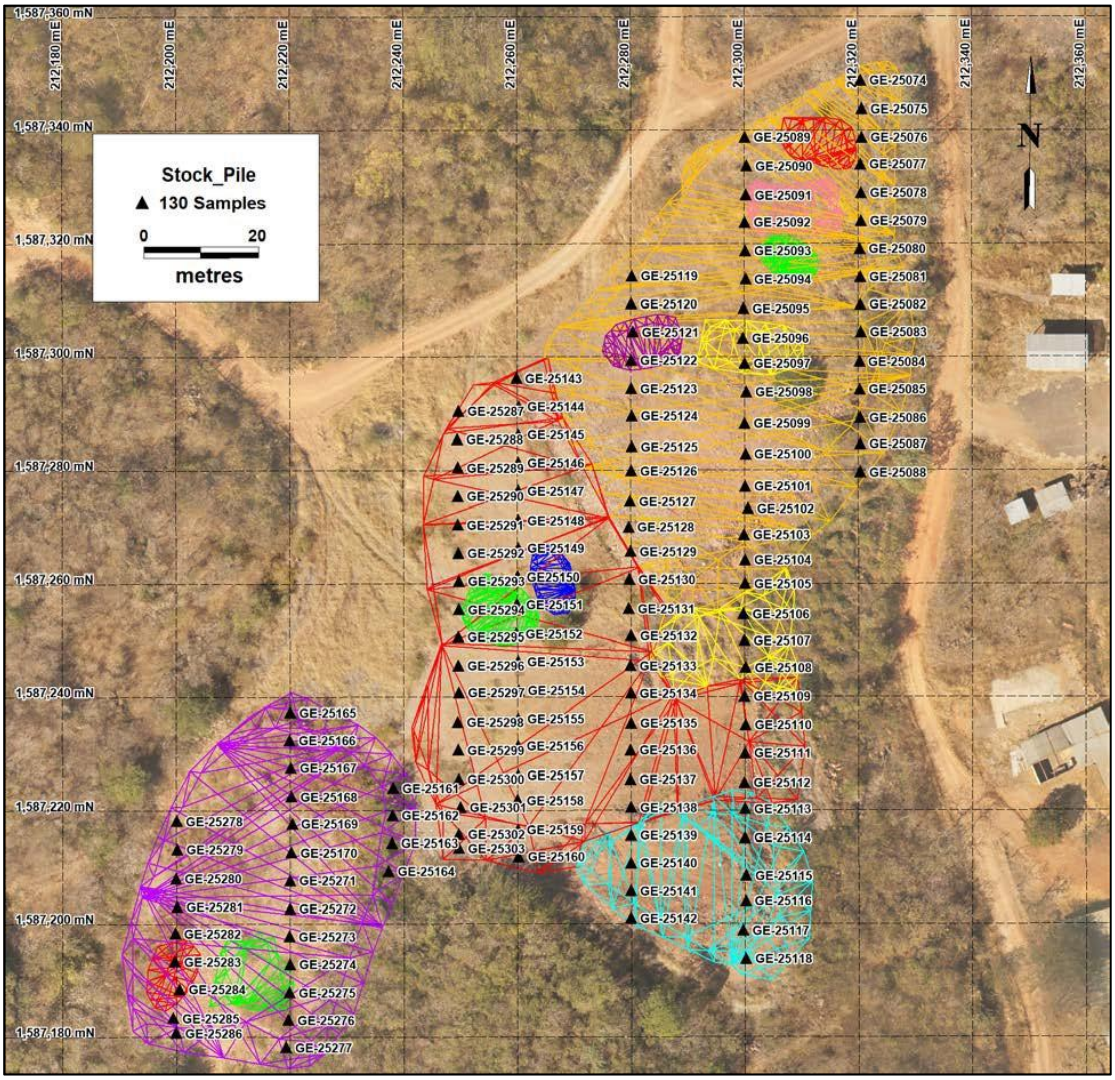
Geostatistical methods were used to support estimation and assess spatial continuity; however, numerical confidence measures were not relied upon as the sole basis for classification. The Qualified Person applied professional judgment to integrate geological understanding with quantitative analysis, recognizing that geostatistical outputs may not fully capture structural and grade variability typical of epithermal deposits.

The Qualified Person concludes that the mineral resource classifications appropriately reflect the level of uncertainty inherent in an epithermal gold–silver deposit and are consistent with the requirements of SEC Regulation S-K Subpart 1300.

11.12 Stockpile Resources

Mineralised material from mining activities undertaken to date at Era Dorada, including ramp development and access, has been stockpiled on site and segregated for future processing. This material may be considered for inclusion within the initial years of mine production or within the ramp-up phase. However, this requires an accurate representation of the volumes and grades so a comprehensive sampling program was designed and implemented. The stockpile surfaces were surveyed to accurately determine volumes and the sampling program entailed excavating trenches on 20 m grid lines and 2 m sample intervals as shown in Figure 11-26.

Figure 11-26: Plan View of Stockpile, Sample Locations & Domain Solids



Source: Kirkham, 2019.

Correlograms for gold and silver were created and employed to estimate the stockpile resources using ordinary kriging. The estimate was validated using nearest neighbour and inverse distance methods, illustrating good agreement of results.

Table 11-20 shows the volume and tonnage based on an unconsolidated specific gravity of 2.0 along with gold and silver grades and metal content. These resources are classified as measured.

Table 11-20: Stockpile Resource Estimate (Measured Resource)

Volume (BCM)	Mine (t)	Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)
14,863	29,726	5.35	22.59	5,108	21,590

Source: Kirkham, 2019.

11.13 Mineral Resource Estimate

This estimate is based upon the reasonable prospect of eventual economic extraction based on continuity an underground mining shapes, using estimates of operating costs and price assumptions. The “reasonable prospects for eventual economic extraction” were tested using stope optimizations performed using Datamine Studio UG v.2.57TM based on reasonable prospects of eventual economic assumptions, as shown in Table 11-21.

Metal prices are based on long-term three-year forecast consensus financial institution estimates published by CIBC (Canadian Imperial Bank of Commerce). The time frame contemplated for metal pricing assumptions is 2–3 years due to the current advanced study stage and project advancement. The pricing used is viewed as conservative in comparison to current spot market pricing which is experiencing a highly volitile period. Metal pricing poses both risks and opportunities in relation to the economic feasibility of the project. The reference point of the mineral resource estimate was seleceted to be the effective date of November 30, 2025.

Table 11-21: Parameters Used for Stope Optimization and Cut-off Grade

Parameter	Unit	RPEEE UG Mining Method	
		LH	MCF
Gold price	US$/oz Au	2500	
Silver price	US$/oz Ag	28	
Project Parameters			
Process Recovery	%	96.00%	
Payable metal	%	99.92%	
TC/RC	US$/oz Au	2.21	
Royalty			
Royalty NSR	% of NSR	1.05%	
Guatemalan Gov't Royalty (Gross)	% total payable metals revenue	1.00%	
OPEX Estimates			
Mining	US$/t milled	100	115
Processing	US$/t milled	32	32
Site Services	US$/t milled	18	18
G&A	US$/t milled	20	20
Total OPEX estimate	US$/t milled	170	185
Cut-off Grade			
In-situ cut-off Au grade	g/t	2.25	2.45

Source: Snowden, 2025.

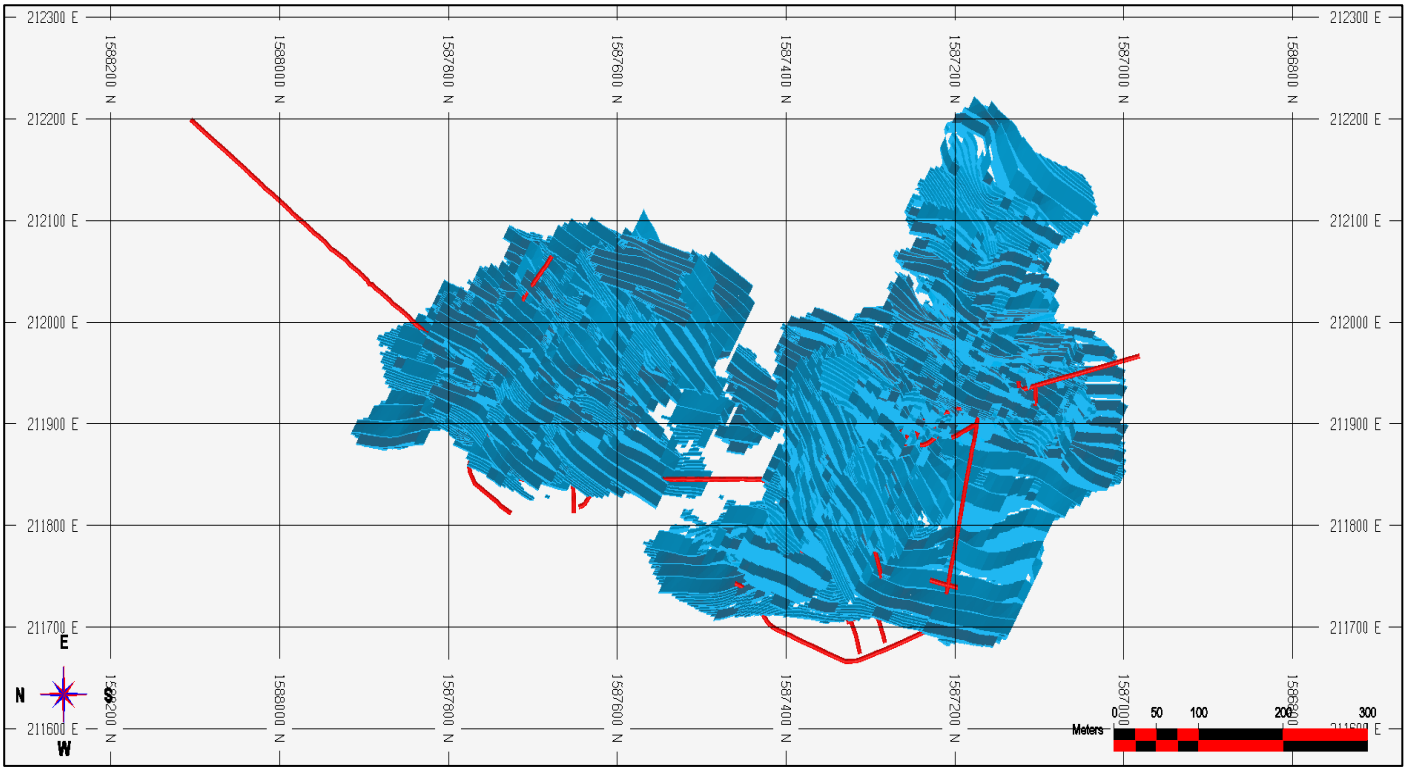
Figure 11-27 illustrates the gold block model along with the “reasonable prospects of eventual economic extraction” underground mining shapes.

The stope optimization results are used solely for testing the “reasonable prospects for eventual economic extraction” and do not represent an attempt to estimate mineral reserves. The point of reference is the effective date of November 30, 2025.

There is a reasonable expectation that the majority of inferred mineral resources could be upgraded to indicated or measured mineral resources with continued exploration.

Table 11-22 and Table 11-23 show tonnage and grade in the Era Dorada deposit and include all domains at a 2.25 g/t Au cut-off grade.

Figure 11-27: Plan View of Gold Block Model with Reasonable Prospects Optimized Mine Shapes with Existing Underground Ramps



Source: Kirkham, 2025.

Table 11-22: Resource Estimate using 2.25 g/t Au Cut-off Inclusive of Reserves

Resource Category	Tonnes (kt)	Au Grade (g/t)	Ag Grade (g/t)	AuEq Grade (g/t)	Contained Gold (koz)	Contained Silver (koz)	Contained AuEq (koz)
Measured							
Indicated	7,059	9.03	30.66	9.36	2,049	6,958	2,125
Measured & Indicated	7,059	9.03	30.66	9.36	2,049	6,958	2,125
Inferred	736	5.94	19.22	6.16	141	455	146

Table 11-23: Resource Estimate using 2.25 g/t Au Cut-off Exclusive of Reserves

Resource Category	Tonnes (kt)	Au Grade (g/t)	Ag Grade (g/t)	AuEq Grade (g/t)	Contained Gold (koz)	Contained Silver (koz)	Contained AuEq (koz)
Measured							
Indicated	2,460	6.36	22.76	6.61	503	1,801	523
Measured & Indicated	2,460	6.36	22.76	6.61	503	1,801	523
Inferred	736	5.94	19.22	6.16	141	455	146

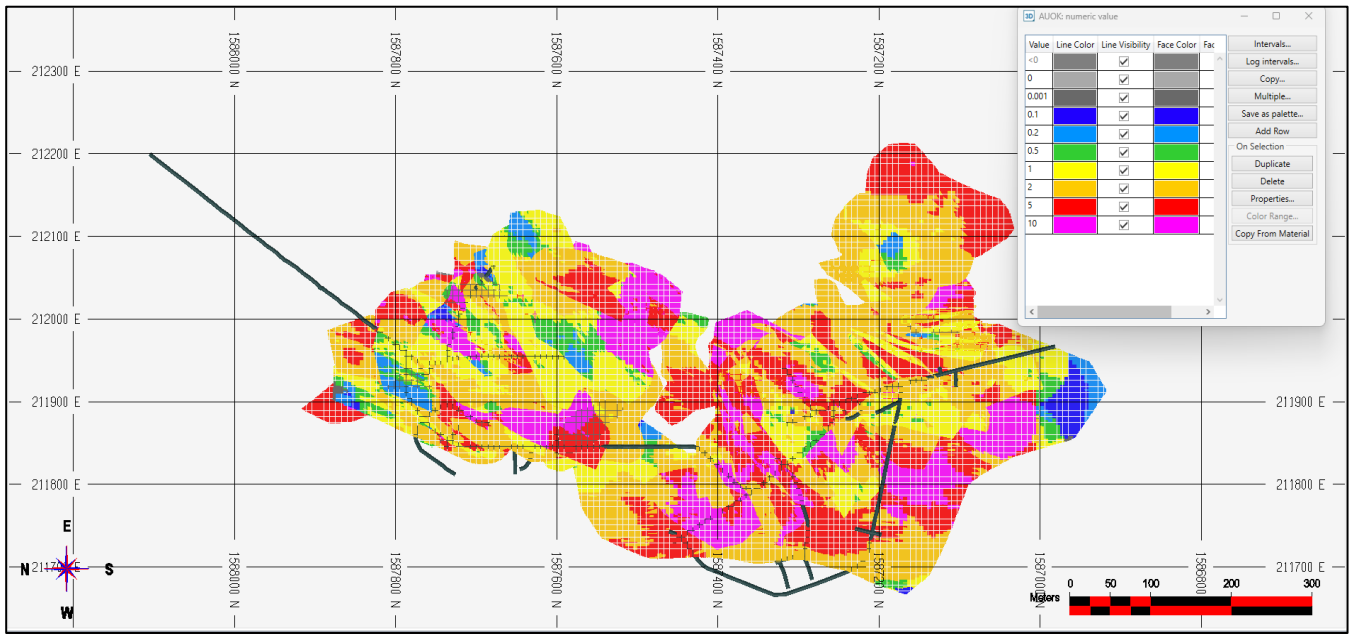
Notes: The mineral resource statement is subject to the following:

1. Mineral Resources are reported in in accordance with S-K 1300.
2. Mineral resource estimates have been prepared by Garth Kirkham, P.Geo., a Qualified Person as defined by SK-1300.
3. The Mineral Resource estimate is reported on a 100% ownership basis.
4. Underground mineral resources are reported at a cut-off grade of 2.25 g/t Au. Cut-off grades are based on a assumed metal prices of US$2,500/oz gold and US$28/oz silver, and assumed metallurgical recovery, mining, processing, and G&A costs.
5. Mineral Resources are reported without applying mining dilution, mining losses, or process losses.
6. Resources are constrained within underground shapes based on reasonable prospects of economic extraction, in accordance with SK-1300. Reasonable prospects for economic extraction were met by applying mining shapes with a minimum mining width of 2.0 m, ensuring grade continuity above the cut-off value, and by excluding non-mineable material prior to reporting.
7. Metallurgical recoveries reported as the average over the life of mine and are assumed to be 96% Au and 85% Ag, respectively.
8. Bulk density is estimated by lithology and averages 2.47, 2.57 and 2.54 g/cm^3 for the Salinas, Mita and mineralized vein domains, respectively.
9. Mineral resources are classified as Indicated, and Inferred based on geological confidence and continuity, spacing of drill holes, and data quality.
10. Effective date of the Mineral Resource Estimate is November 30, 2025.
11. Tonnage, grade, and contained metal values have been rounded. Totals may not sum due to rounding.
12. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

Source: Kirkham, 2025.

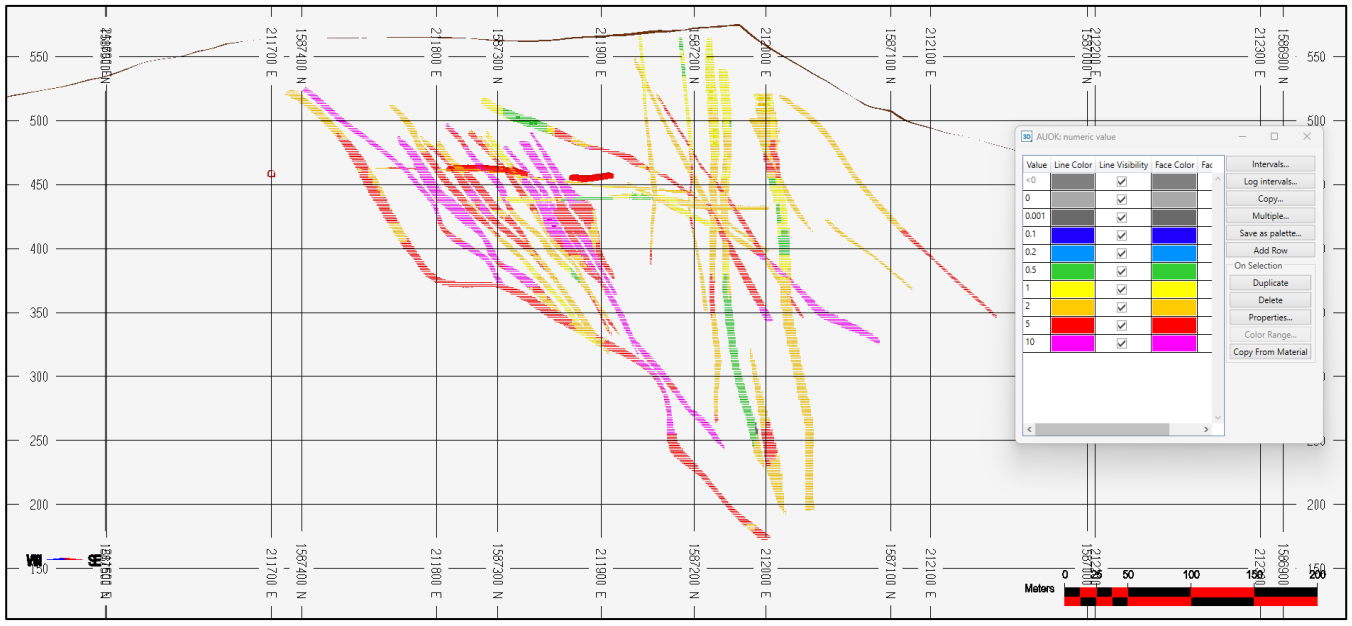
Figure 11-28 illustrates a plan view of the 3-dimentional block model for the resources within the mineralized veins. Figure 11-29 through Figure 11-32 show sectional views of the high-grade veins for gold and silver in the north and south, respectively. Figure 12-10 through Figure 11-36 show sectional views of the total block model with the high-grade vein and low-grade host rock components resulting in the whole block grade for gold and silver in the north and south, respectively.

Figure 11-28: Plan View of Au within Veins along with Existing Ramp Development



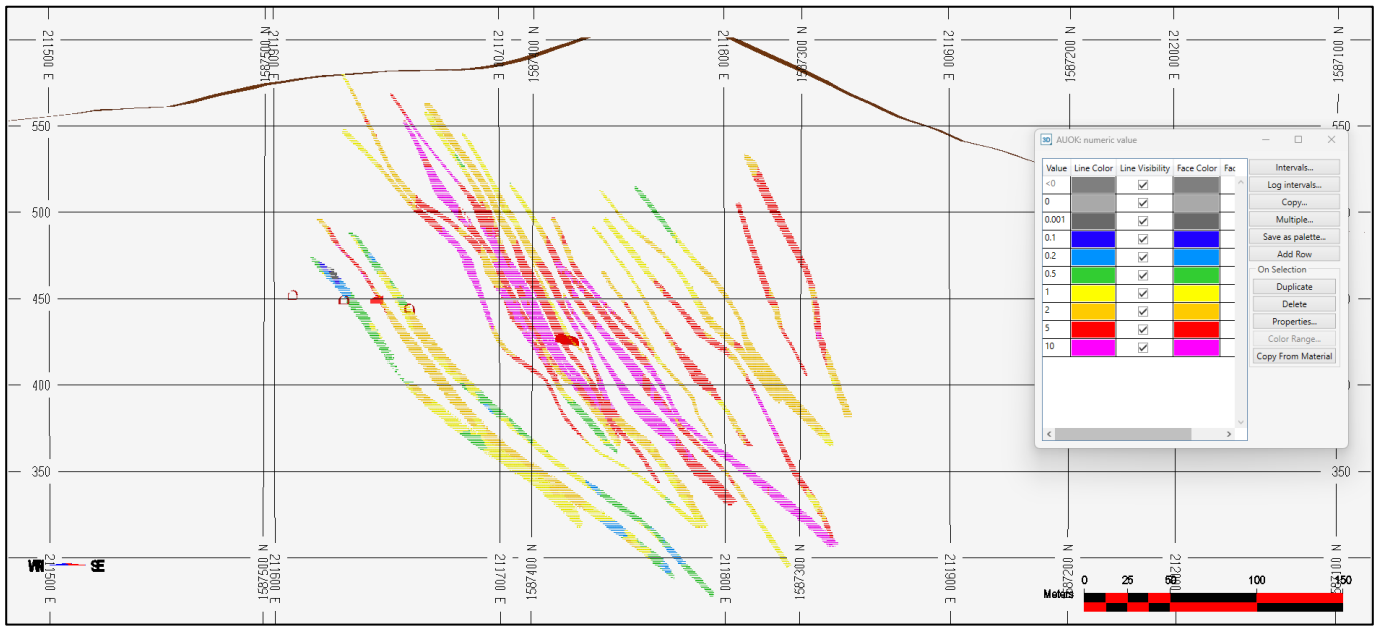
Source: Kirkham, 2025.

Figure 11-29: Section View of Au South Zone Veins



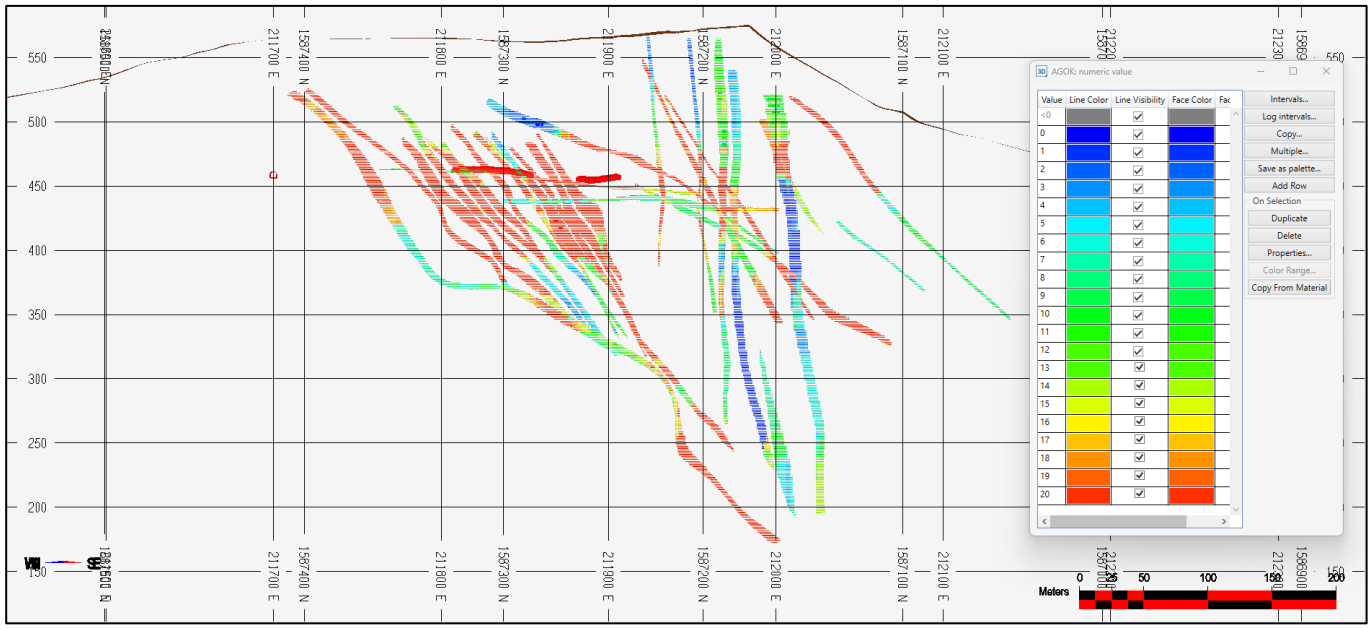
Source: Kirkham, 2025.

Figure 11-30: Section View of Au Block Model North Zone Veins



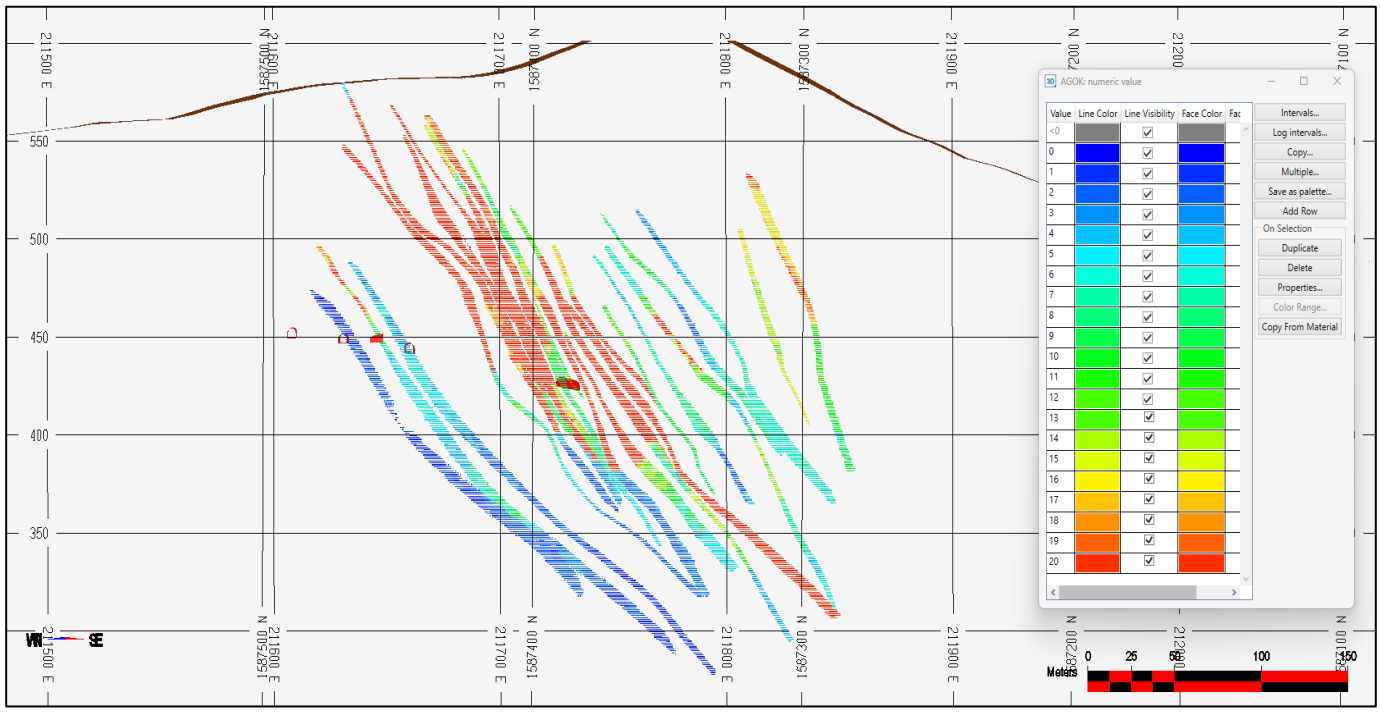
Source: Kirkham, 2025.

Figure 11-31: Section View of Ag Block Model South Zone Veins



Source: Kirkham, 2025.

Figure 11-32: Section View of Ag Block Model North Zone Veins



Source: Kirkham, 2025.

11.14 Sensitivity of the Block Model to Selection Cut-off Grade

The mineral resources are sensitive to the selection of cut-off grade. Table 11-24 shows tonnage and grade in the Era Dorada deposit at different gold cut-off grades. Note that the base case is highlighted in **bold**.

The reader is cautioned that these values should not be misconstrued as a mineral reserve. The reported quantities and grades are only presented as a sensitivity of the resource model to the selection of cut-off grade.

Table 11-24: Sensitivity Analyses of Tonnage along with Au & Ag Grades at Various Au Cut-off Grades

Resource Category	Cut-off	Tonnes (kt)	Grade Au (g/t)	Grade Ag (g/t)	Contained Gold (koz)	Contained Silver (koz)
Indicated	2	7,137	8.95	30.43	2,054	6,982
	2.25	**7,059**	**9.03**	**30.66**	**2,049**	**6,958**
	2.45	6,981	9.10	30.88	2,043	6,931
	2.5	6,950	9.13	30.97	2,041	6,921
	3	6,593	9.48	32.02	2,010	6,787
	3.5	6,137	9.96	33.42	1,965	6,594
	4	5,604	10.56	35.21	1,903	6,344
Inferred	2	762	5.81	18.93	142	464
	2.25	**736**	**5.94**	**19.22**	**141**	**455**
	2.45	714	6.06	19.40	139	445
	2.5	708	6.09	19.46	139	443
	3	621	6.58	20.19	131	403
	3.5	534	7.15	21.06	123	361
	4	465	7.68	21.74	115	325

Notes: The mineral resource statement is subject to the following:

1. All mineral resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum (CIM) definitions, with an effective date of November 30, 2025.
2. Mineral Resources reported demonstrate reasonable prospect of eventual economic extraction; mineral resources are not mineral reserves and do not have demonstrated economic viability.
3. Cut-off grades are based on a price of US$ 2,500/oz gold, US$ 28/oz silver and a number of operating cost and recovery assumptions, plus a contingency.
4. Numbers are rounded.
5. The mineral resources may be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors.
6. An inferred mineral resource has a lower level of confidence than that applying to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration.

Source: Kirkham, 2025.

11.15 Resource Validation

A graphical validation was done on the block model. The purpose of this graphical validation is to:

- check the reasonableness of the estimated grades, based on the estimation plan and the near by composites.
- check the general drift and the local grade trends, compared to the drift and local grade trends of the composites.
- ensure that all blocks in the core of the deposit have been estimated.

- check that topography has been properly accounted for.
- check against partial model to determine reasonableness.
- check against manual approximate estimates of tonnage to determine reasonableness.
- inspect and explain potentially high-grade block estimates in the neighbourhood of extremely high assays.

A full set of cross-sections, long sections and plans were used to check the block model on the computer screen, showing the block grades and the composites. No evidence of any block being wrongly estimated was found; it appears that every block grade could be explained as a function of the surrounding composites and the estimation plan applied.

These validation techniques included the following:

- Visual inspections on a section-by-section and plan-by-plan basis.
- The use of grade-tonnage curves.
- Swath plots comparing kriged estimated block grades with inverse distance and nearest neighbour estimates.
- An inspection of histograms of distance of the first composite to the nearest block, and the average distance to blocks for all composites used, which gives a quantitative measure of confidence that blocks are adequately informed in addition to assisting in the classification of resources.
- Validation of the block models was also performed by estimating the resources within the vein domains using partial block where the vein solids were coded as a percentage within the blocks.
- Discussion with respect to potential material risks to the Resources.
- There are no known environmental, permitting, legal, taxation, title, socio-economic, political or other relevant factors that materially affect the resources.
- It is the opinion of the qualified person that all issues relating to all relevant technical and economic factors likely to influence the prospect of economic extraction can be resolved with further work with the exception of commodity prices. Price volitility poses both opportunity and risk which are difficult to predict under the current market conditions and geopolitical factors.

12 MINERAL RESERVE ESTIMATES

12.1 Introduction

This section presents the basis for the Mineral Reserve estimate for the Era Dorada deposit.

The estimate was completed using industry-standard methodologies and software, and the resulting Mineral Reserveis reported in accordance with S-K1300 requirements.

The Mineral Reserve estimate was subject to Legal and Permitting constraints and other modifying factors such as the plant and infrastructure timing and capacities, the metallurgical recoveries for gold and silver, economic factors as gold and silver prices, costs and exchange rates as well as technical modifying factors derived from geotechnical constraints, mining methods and productivity.

Ruy Lacourt, BSc. Mining Engineering, MSc., Registered Member of the SME, Associate Executive Consultant of Snowden Optiro and Qualified Person visited the project from July 7 to July 9 2025. The site visit included inspection of the property, underground works and associated surface infrastructure, core storage facilities, workshops and offices.

12.2 Economic Parameters and Cutoff Grades

The economic assumptions which underpin the conversion of Indicated Resources to Probable Reserve were defined in the begining of the Feasibility Study. The gold and silver prices are consistent with Aura guidance and were deemed adequate at that stage by the QP – they were later reviewed according to more recently market forecasts for the financal analysis. Method-specific gold equivalent cutoff grades were used to define economically mineable shapes. The assumptions to define the cutoff grades are shown in Table 12-1.

Table 12-1: Mineral Reserve Cut-off Grade

Parameter		Unit	Value	
			LH	MCF
Au price		US$/oz	2,000	2,000
Ag price		US$/oz	25	25
Project Parameters				
Au Process Recovery		%	96.00	96.00
Ag Process Recovery		%	85.00	85.00
Au Payable metal		%	99.92	99.92
Ag Process Recovery		%	99.50	99.50
TC/RC		US$/oz Au	2.21	2.21
Royalty				
Royalty NSR		% of NSR	1.05%	1.05%
Guatemalan Gov't Royalty (Gross)		% total payable metals revenue	1.00%	1.00%
OPEX Estimates	Mining (Underground)	US$/tonnes milled	100	115
	Processing	US$/tonnes milled	32	32
	Site Services	US$/tonnes milled	18	18
	G&A	US$/tonnes milled	20	20
	Total OPEX estimate	US$/tonnes milled	170	185
In-situ cutoff Au grade		g/t Au eq	2.82	3.07

Assumptions for gold and silver recoveries as well as mining costs are based on previous studies for the deposit and are assessed as adequate, within industry standards, by the QP.

A gold equivalent grade was used, based on the metal prices and metallurgical recoveries above defined as:

- Au eq = Au grade + [(Ag Rec * Ag Price) / (Au Rec *Au Price)] * Ag grade
- Au eq = Au grade + 21/1918 * Ag grade or
- Au eq = Au grade + 0.011 * Ag grade

12.3 Stope Optimization

The Mineable Shape Optimiser (MSO) module within Datamine Studio UG software was used to generate optimized mathematical stope shapes, based on a set of design constraints including minimum dip angle, stope width, and gold equivalent cutoff grade (COG). The optimization was run according to the parameters shown in Table 12-2.

Table 12-2: Stope Optimization Parameters

Parameter	Unit	Long Hole		Cut-and-Fill
		Geotech Domain 1	Geotech Domains 2 and 3	
Block model	June 30 2025 bm v9.csv (original) june302025_bm_export_v9_bm_reserve.dm (treated)			
Variable	AUEQ			
Length	m	10	10	5
Height	m	20	20	4
COG Au Eq.	g/t	2.82	2.82	3.07
Minimum Width	m	1.5	1.5	1.0
Maximum Width	m	50	50	50
Parallel Pillar Distance	m	6	6	6
Wall Dilution Foot	m	0.4	0.4	0.25
Wall Dilution Hang	m	0.7	0.7	0.25
Minimum Dip Foot Near	°	60	45	45
Maximum Dip Foot Near	°	150	135	135
Minimum Dip Foot Far	°	60	45	45
Maximum Dip Foot Far	°	150	135	135
Minimum Dip Hang Near	°	60	45	45
Maximum Dip Hang Near	°	150	135	135
Minimum Dip Hang Far	°	60	45	45
Maximum Dip Hang Far	°	150	135	135

Material classified as Inferred Minera Resources had its grades set to zero for both optimization and reporting.

Long hole stope optimization was discretized according to the geotechnical domains previously coded in the block model, with stopes dipping between 45° and 60° restricted to Domains 2 and 3. Cut-and-Fill stopes were optimized across the entire mine to identify potential mining areas where Long hole stopping was not suitable.

12.4 Mine Design

After optimization, the stope solids were grouped and selected such that Long hole mining (transverse or longitudinal) was preferred over Cut-and-fill, given its advantages as for the safety of the operation, higher productivity and better economics.

Very isolated stopes, particularly cut-and-fill stopes, were excluded from the optimized solids whenever extra development costs did not justify their mining.

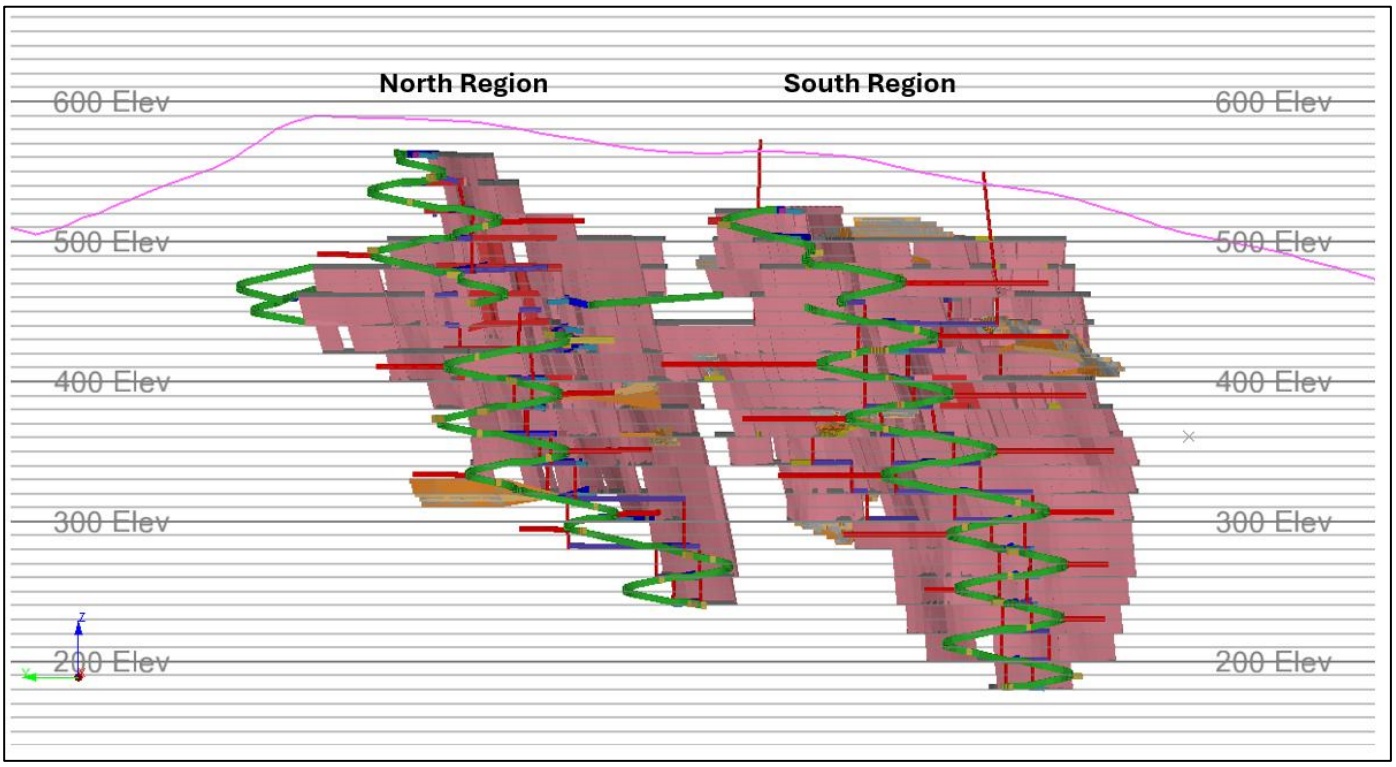
Transverse Long hole stopping was adopted for stopes with thickness greater than 20 m. The transverse stopes represent 8% of the tonnage for Long hole mining.

The following geometric parameters were adopted for the mine design:

- Panel Height: 100m (4 sublevel of 20m each + sill pillar 20m)
- Crown Pillar: 10m from the base of Saprolite
- Minimum Ramp to Stope Distance: 25 m
- Minimum Spacing Between Drifts: 10 m
- Minimum Spacing Between Footwall Drive and Transverse Stopes: 10 m
- Minimum Spacing Between Raises and Drifts: 10m
- Minimum Spacing Between Raises and Stopes: 10 m
- ‘Zig-zag’ (switchback) layout for the ramp
- Minimum radius of curvature for the ramp: 25 m
- Ramp Gradient: 13.5% with smoothing to 0% (over 20 m) for sublevel accesses
- Pivot Access to CAF method: approximately ±15%

Figure 12-1 shows the Mine Design.

Figure 12-1: East View of the Mine



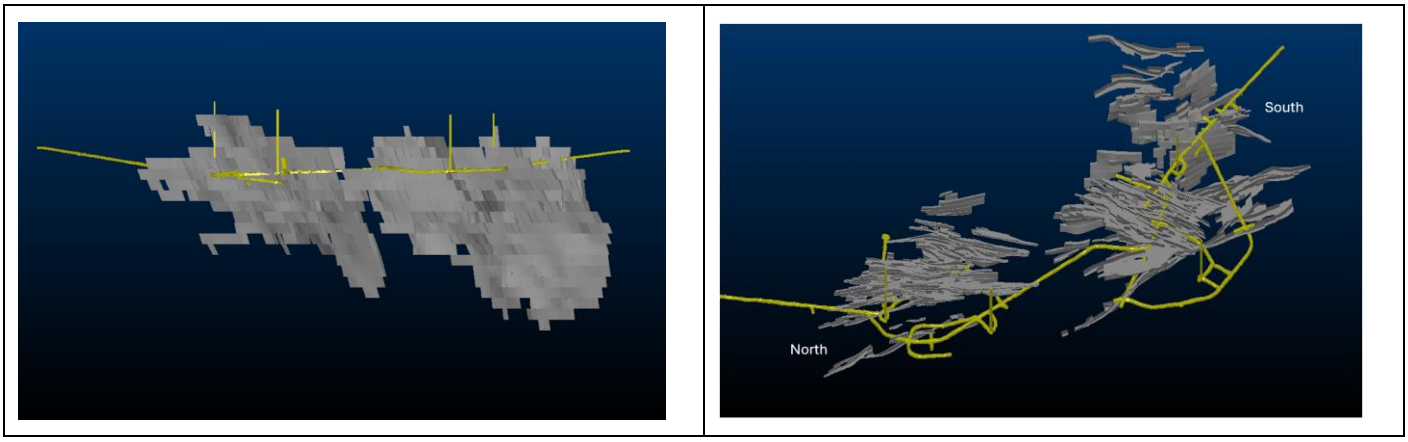
Source: Snowden Optiro, 2025.

The mine will be accessed via two existing main ramps: one servicing the South Zone and another for the North Zone. The ramps serve as the primary access to the mine, provide haulage routes for ore and waste, and act as fresh-air intakes for mine ventilation. The declines will also be the primary escapeways, providing redundancy and ensuring personnel can be evacuated even if one route becomes partially obstructed.

Previous exploration campaigns resulted in the development of more than 2,700 m of underground workings, including two portals, ramps, crosscuts (X-cuts), and ore drifts within the veins. These openings were developed at dimensions of 4.5 m wide by 5.0 m high and are equipped with electrical power supply, ventilation infrastructure and water services.

These existing galleries, shown in yellow in Figure 12-2 were integrated into the mine plan, providing initial access, production levels and ventilation airways.

Figure 12-2: As-built (Yellow)



Source: Snowden Optiro, 2025.

The existing portals were constructed on hillsides using steel sets, corrugated steel plates and shotcrete. Surface infrastructure at each portal includes process water tanks, and electrical substations.

This dual-ramp configuration increases the reliability of the escape system, reduces evacuation time, and provides operational flexibility to direct people along the safest route based on the emergency location. The declines will be kept clearly marked and free of obstructions, and will be integrated into the Emergency Response Plan, including provisions for lighting, communications, and refuge/breathing stations where applicable, to support safe travel to surface.

The underground ventilation system has been planned around four existing exhaust raises of 3.0 m diameter, each approximately 100 m long, fitted with square concrete collars extending 1.5 m above surface.

Additional ramps will be developed at a maximum gradient of 15%, with typical dimensions as of 5.0 m by 5.0 m to accommodate 30-t haul trucks and temporary 1.4 m diameter ventilation ducting. Separate ramps will be constructed to access deeper levels in both the South and North Zones, as well as the upper levels.

The switchback ramps will provide access to each production sublevel, with vertical spacing of 20 m. The ramps will be developed at a gradient of −13.5%, with a 5.0 m × 5.0 m profile and a minimum curvature radius of 25 m. At each operating sublevel, the ramp will flatten to 0% gradient over a 20 m length (10 m before and 10 m after the X-cut), providing improved maneuverability and visibility for equipment entering and leaving the haulage ramp.

Each sublevel will be serviced by:

- X-cuts (5.0 m × 5.0 m) linking the ramps to each sublevel;
- Footwall drives (5.0 m × 5.0 m) to support transverse Long Hole stoping where necessary;
- Sublevel drives (4.0 m × 4.0 m), developed longitudinally along the orebody where Long Hole Longitudinal stoping is applied, and transversely to the orebody where Transverse Long Hole stoping is applied;

- Ventilation drives (4.0 m × 4.0 m) to allow for the ventilation of the working areas, ensuring connections between fresh-air and return (exhaust) raises and the stope levels.

Mechanized cut-and-fill (MCF) zones will be accessed via attack ramps developed from nearby drives. The attack ramps will be developed at a maximum gradient of 15% and developed as a vertically offset ramp stack to provide access to multiple production levels from a single access point, as illustrated in Figure . MCF drives will have a 3.5 m × 3.5 m section to maintain structural integrity in less favorable ground conditions where cut-and-fill will be applied.

Remucks (mucking bays) 15 m long (5.0 m × 5.0 m) will be excavated along the main ramp and footwall drives to reduce development haulage cycle times. The remucks will be spaced at a maximum of 150 m.

Water collection sumps (4.0 m × 4.0 m) will be established for the sublevels adjacent to the X-cuts. Additional niches will be developed beside each level sump to accommodate a portable pump, which will collect water from the level sumps and pump it directly to the main dewatering sumps.

Electrical distribution stations (4.0 m × 4.0 m) will be positioned in niches off the access drives on each sublevel. Additional substations will be installed near major demand points, such as the main dewatering sumps.

Refuge station niches (4.0 m × 4.0 m) will be established on all levels adjacent to the fresh air raises. Portable refuge chambers will be moved between these niches as required, depending on the activities in the mine. The chambers will have sufficient capacity for all personnel working within their area of influence.

Dedicated diamond-drilling drives (4.0 m × 4.0 m) will be developed from the ramp, extending parallel to the orebody strike.

Drop raises will be developed to connect the access X-cuts on each sublevel. These raises will be excavated using Long Hole equipment and sequenced in a staggered sequence, allowing fresh air to be directed towards the advancing ramps. Some return-air raises will also be fitted with drainage lines and paste distribution lines as required to service each level of the mine.

In general, long-term development will incorporate an arched back with a 1.0 m radius, whereas all temporary drives will be developed with a flat back. In less favorable ground areas, it may be necessary to develop production sublevels with an arched, supported back.

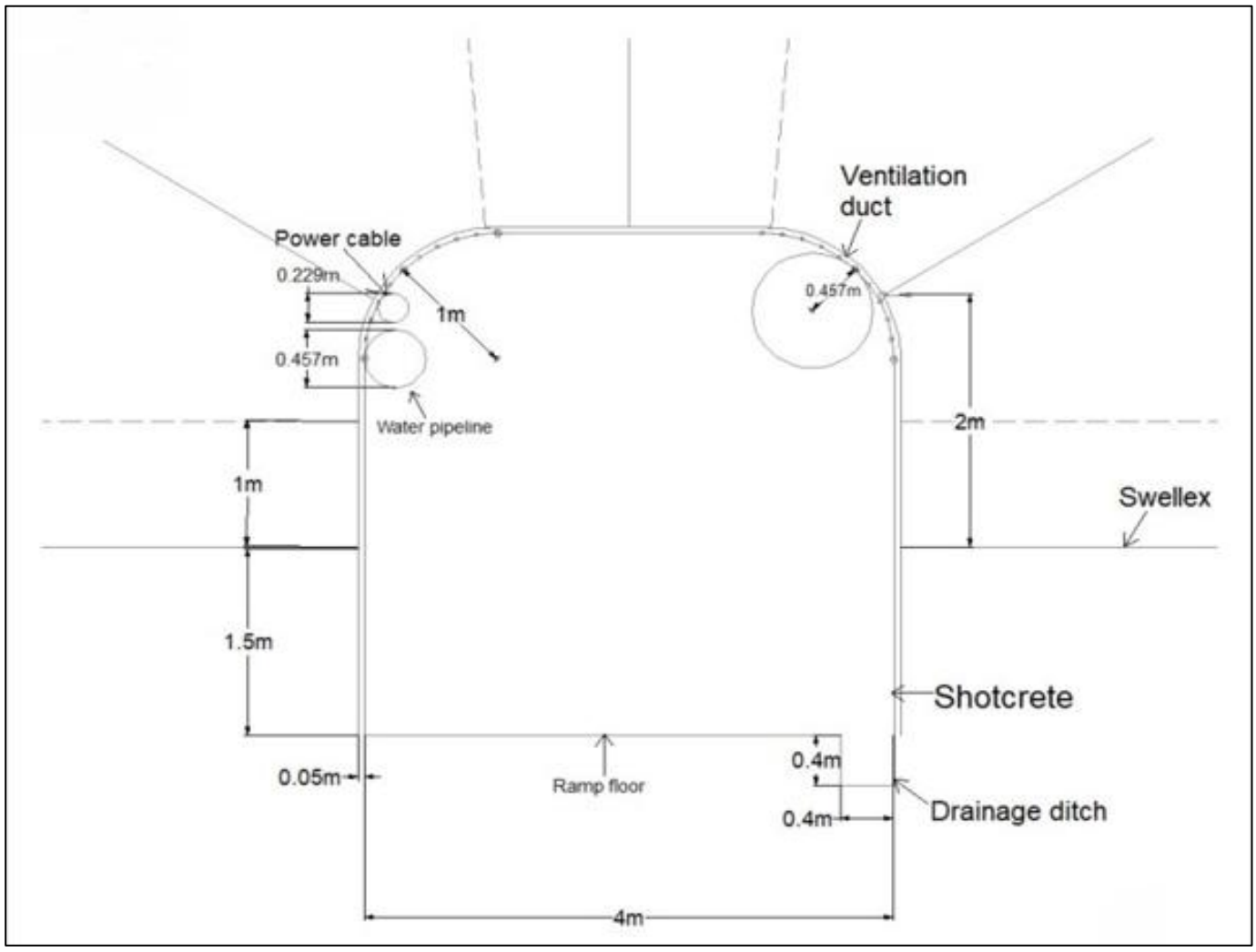
A summary of the underground development cross-sections used is provided in Table 12-3.

Table 12-3: Drift Sections

Development Type	Width (m)	Heigth (m)
Ramps	5.0	5.0
Access Drives		
Footwall Drives		
Remucks		
Sumps	4.0	4.0
Secondary Access Drives		
Ventilation Access		
Electrical Subestation		
Sublevel Drives		
Exploration Drives		
C&F Drives	3.5	3.5
C&F Drives (Top of 20m Sublevel)	3.5	3.0
Ventilation Raises	Diam 3.6	—

Figure 12-3 shows the development standards for the 4 x 4 m^2 sections as an example.

Figure 12-3: Design of 4 m x 4 m Drifts



Source: Snowden Optiro, 2025.

12.5 Mine Schedule

Following the completion of the mine design, a detailed production schedule was prepared to ensure the proposed extraction sequence is operationally practical for both Long hole stoping and mechanized Cut-and-fill, while meeting the Project production targets. The sequencing was developed in Datamine Task Scheduler (DTS) and prioritizes early access to higher-grade zones, integrating the two existing main declines and historical development as initial access and production levels.

The schedule enforces key operational constraints, including maximum annual development of 8,500 m, sublevel spacing at 20 m, method-specific stope dimensions and cutoff grades, and the planned ramp-up to meet plant requirements.

The geotechnical assessment supports independent extraction of individual stopes for the Long hole method, without imposing a mandatory hangingwall-to-footwall mining sequence, which provides additional scheduling flexibility. MCF stopes were inserted where required by ground conditions and unfavorable Long hole geometry.

The production ramp-up has been structured to ensure a progressive integration between the underground mining capacity and the processing plant throughput. During the initial months of operation, mine production is primarily constrained by the development advance rate, limited stockpile availability (which restricts ore development), the initial availability of stoping areas, and the operational learning curve. As development progresses and additional mining areas are brought into the plan, production rates are increased in a controlled manner, supported by gradual improvements in productivity.

In addition, the schedule is tightly coupled to enabling infrastructure and services: stoping below the 420 level water table is contingent on effective dewatering and hot-water management, paste fill placement rates and minimum curing times control the availability of adjacent stopes, and ventilation/cooling capacity governs the number of active faces.

Stockpiling of early ore is also incorporated to manage the plant start-up timing and grade smoothing.

Collectively, these controls ensure the sequence reflects mining operability and supports the production of approximately 100 koz recovered Au per year, with higher production in the early years.

The development rates assumed for scheduling are summarized in Table 12-4.

Table 12-4: Development Rates

Drift	Value	Rate
Access Drive	50	m/ mo
Access Drive2	60	m/ mo
CAF Drive 1	40	m/ mo
CAF Drive 2	40	m/ mo
Electrical cut-out	50	m/ mo
Exploration Drive	80	m/ mo
Footwall Drive	80	m/ mo
Ramp	80	m/ mo
Remucks	80	m/ mo
Sublevel Drive	80	m/ mo
Sumps	50	m/ mo
Vent Drive	80	m/ mo

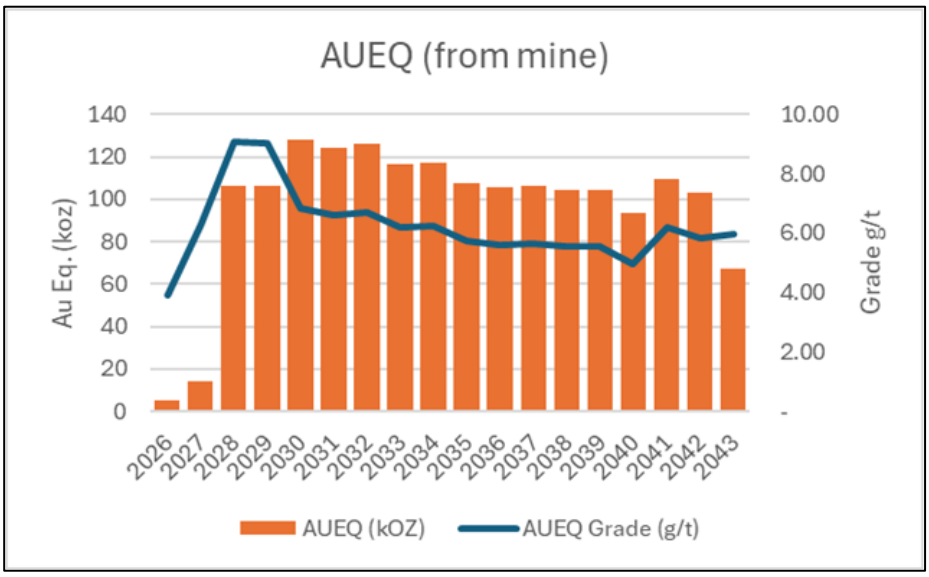
Drift	Value	Rate
Ventilation raise	60	m/ mo
X-Cut	60	m/ mo
Mining		
Cut and Fill	30,000	tonnes/ mo
LH_Long	17,000	tonnes/ mo
LH_Transv	44,000	tonnes/ mo
Room and Pillar	30,000	tonnes/ mo

The following assumptions were adopted for production scheduling:

- Development of drives and raises commencing in Year 1;
- Start of mine production (productive development and/or stoping) according to plant start-up;
- Ore extracted prior to plant start-up to be sent to stockpile, aiming to minimize both stockpiled tonnage and grade;
- Processing plant ramp-up: Commissioning is planned to reach 20% of nameplate capacity in October 2027, 40% in November, 80% in December 2027, and 100% in January 2028. This corresponds to approximately 6 kt, 12 kt, 24 kt, and 30 kt of plant feed, respectively.
- Mine production ramp-up: Underground production is scheduled to start in September 2027, with planned ROM delivery of 4 kt in September, 8 kt in October, 12 kt in November, 18 kt in December 2027, and ramping to 30 kt in January 2028, matching the plant's full throughput requirement.
- Independent extraction sequence of Long hole stopes at the same areas;
- Metallurgical recoveries of 96% for gold and 85% for silver;
- Recovery of sill pillars at the end of the Life of Mine/ LoM;
- Reclaiming of the stockpile during the last three years of the LoM;
- Target of 100 koz recovered per year, seeking to maintain above this level in the early years;
- Maximum mine development of 8,500 m/a;
- Mine production of 365 kt/year (1,000 t/d) up to 2029 and 584 kt/year (1,600 t/d) for the remainder of the LoM.

Figure 12-4 shows the schedule results, maintaining a high gold production in the early years, with ramp-up by the end of 2027, production capacity of 1,000 kt/d in 2028 and 2029, and production capacity of 1,600 kt/d from 2028 onward.

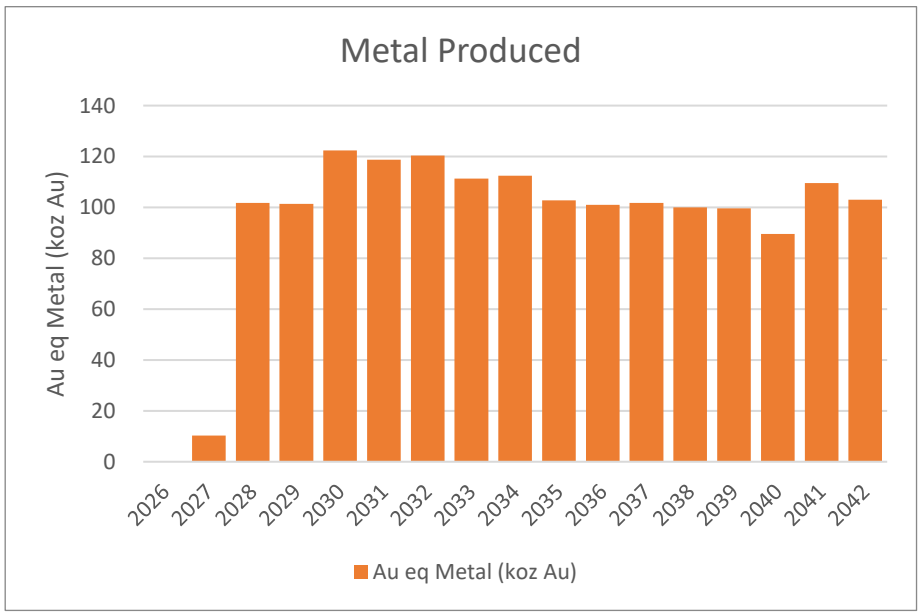
Figure 12-4: Mine Schedule – Au Equivalent/ROM



Source: Snowden Optiro, 2025.

Gold production is shown in Figure 12-5 in Au eq units.

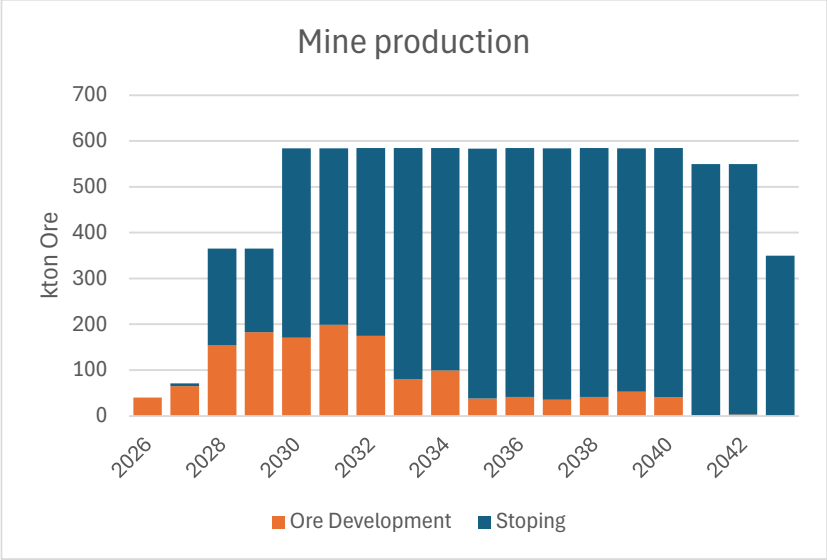
Figure 12-5: Mine Schedule – Plant Production



Source: Snowden Optiro, 2025.

Figure 12-6 shows the ROM production between ore development and stoping.

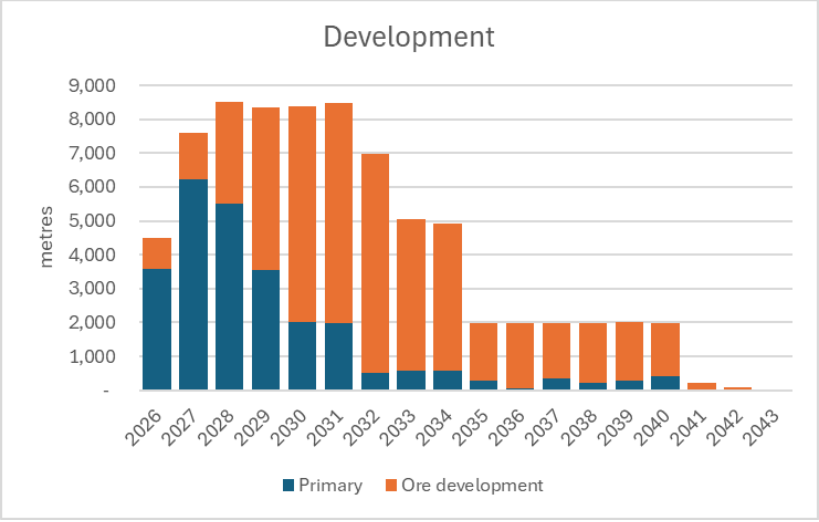
Figure 12-6: Mine Schedule – ROM Tonnage



Source: Snowden Optiro, 2025.

Figure 12-7shows the mine development profile.

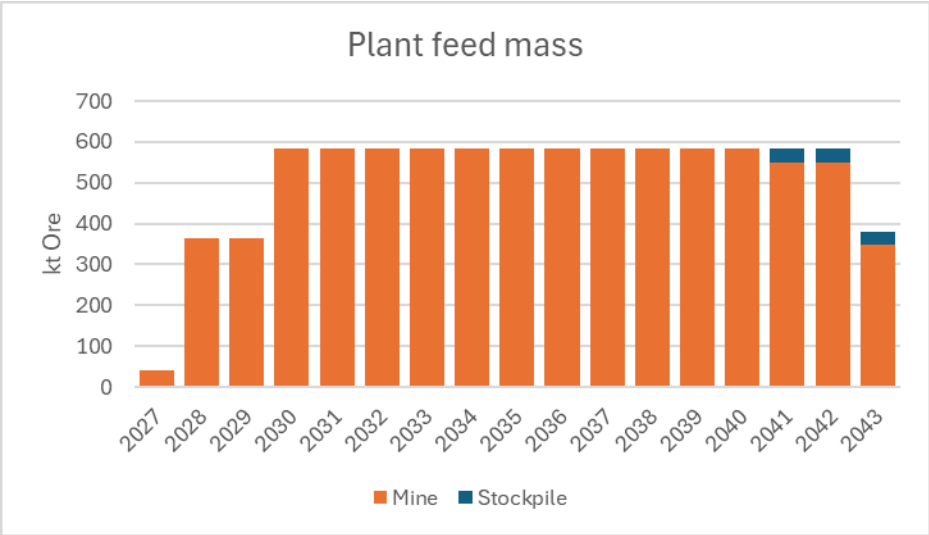
Figure 12-7: Mine Schedule – Mine Development



Source: Snowden Optiro, 2025.

Figure 12-8 shows the plant feed tonnage.

Figure 12-8: Mine Schedule – Plant Feed Tonnage



Source: Snowden Optiro, 2025.

Table 12-5, Table 12-6, Table 12-7 and Table 12-8 show schedule details.

Table 12-5: Mine Schedule - Production

Mine Production	Yr-1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17
Ore tonnage - Mining (kt)	0	6	212	183	413	385	410	504	485	545	543	549	543	531	544	548	547	350
Ore tonnage - Develop (kt)	40	65	154	183	171	199	175	80	99	38	41	35	42	53	41	1	3	0
Ore tonnage - Total (kt)	40	71	365	365	584	584	584	584	584	583	584	584	584	584	584	549	549	350
Ore volume- Total (x1000 m^3)	16	28	144	144	230	231	231	230	230	226	231	231	231	231	231	217	217	138
Au eq grade (g/t Au)	3.92	6.30	9.07	9.02	6.82	6.61	6.70	6.19	6.25	5.73	5.62	5.67	5.57	5.55	4.98	6.21	5.83	5.97
Au eq metal (koz Au)	5	14	107	106	128	124	126	116	117	107	106	106	105	104	94	110	103	67
Au grade (g/t Au)	3.79	6.04	8.68	8.68	6.53	6.38	6.48	6.01	6.09	5.56	5.43	5.45	5.34	5.32	4.80	6.04	5.62	5.64
Au metal (oz Au)	5	14	102	102	123	120	122	113	115	104	102	102	100	100	90	107	99	64
Ag grade (g/t Ag)	12.30	24.14	34.82	31.33	26.25	20.88	20.11	16.46	14.28	16.12	16.94	20.24	20.83	20.46	17.05	14.74	18.72	30.04
Ag metal (oz Ag)	16	55	409	368	493	392	378	309	268	302	318	380	391	384	320	260	331	338

Table 12-6: Mine Schedule - Development

Development	Yr-1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17
Primary (m)	3,593	6,238	5,508	3,559	2,018	1,969	521	575	593	275	40	358	208	275	415	3	-	-
Ore development (m)	896	1,366	3,001	4,803	6,353	6,522	6,459	4,463	4,324	1,701	1,947	1,633	1,772	1,731	1,582	209	94	-
Total (m)	4,489	7,604	8,509	8,363	8,371	8,491	6,979	5,039	4,916	1,977	1,987	1,990	1,980	2,006	1,997	212	94	-
Raise Boring (m)	94	-	136	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Drop Raises (m)	143	290	388	192	92	81	206	-	-	-	-	-	-	-	-	-	-	-

Table 12-7: Mine Schedule – Stock Evolution

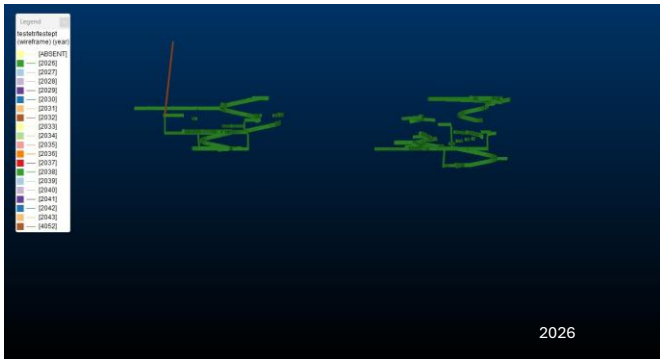
Stock	Yr-1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17
Ore mass - Total (kt)	70	197	99	99	99	99	99	99	99	99	99	99	99	99	99	64	30	0
Au eq grade (g/t Au)	4.63	4.43	4.43	4.43	4.43	4.43	4.43	4.43	4.43	4.43	4.43	4.43	4.43	4.43	4.43	4.43	4.43	4.43
Au eq metal (koz Au)	10	28	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14
Au grade (g/t Au)	4.35	4.17	4.17	4.17	4.17	4.17	4.17	4.17	4.17	4.17	4.17	4.17	4.17	4.17	4.17	4.17	4.17	4.17
Au metal (oz Au)	10	26	13	13	13	13	13	13	13	13	13	13	13	13	13	13	13	13
Ag grade (g/t Ag)	17.07	16.93	16.93	16.93	16.93	16.93	16.93	16.93	16.93	16.93	16.93	16.93	16.93	16.93	16.93	16.93	16.93	16.93
Ag metal (oz Ag)	38	107	54	54	54	54	54	54	54	54	54	54	54	54	54	54	54	54

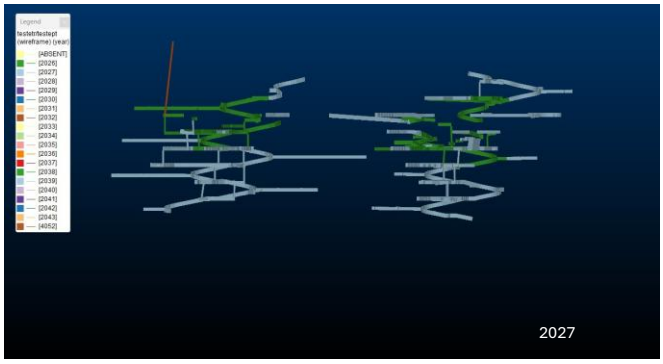
Table 12-8: Mine Schedule – Material to Plant

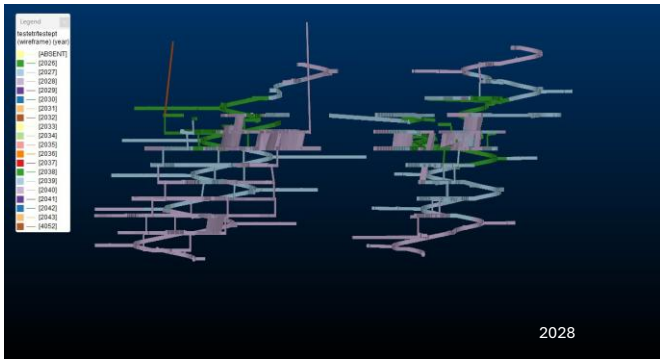
Plant	Yr-1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17
Ore from mine (kt)	0	42	365	365	584	584	584	584	584	583	584	584	584	584	584	549	549	350
Au eq grade from mine (g/t Au)	0	0	9.07	9.02	6.82	6.61	6.70	6.19	6.25	5.73	5.62	5.67	5.57	5.55	4.98	6.21	5.83	5.97
Au grade from mine (g/t Au)	0	7.60	8.68	8.68	6.53	6.38	6.48	6.01	6.09	5.56	5.43	5.45	5.34	5.32	4.80	6.04	5.62	5.64
Ag grade from mine (g/t Ag)	0	29.15	34.82	31.33	26.25	20.88	20.11	16.46	14.28	16.12	16.94	20.24	20.83	20.46	17.05	14.74	18.72	30.04
Ore from stock (kt)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	34	34	30
Au eq grade from stock (g/t Au eq)	0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	4.43	4.43	4.43
Au grade from stock (g/t Au)	0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	4.17	4.17	4.17
Ag grade from stock (g/t Ag)	0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	16.93	16.93	16.93
Ore tonnage - Total (kt)	0	42	365	365	584	584	584	584	584	583	584	584	584	584	584	584	584	380
Au eq grade (g/t Au eq)	0	0	9.07	9.02	6.82	6.61	6.70	6.19	6.25	5.73	5.62	5.67	5.57	5.55	4.98	6.10	5.75	5.85
Au grade (g/t Au)	0	7.60	8.68	8.68	6.53	6.38	6.48	6.01	6.09	5.56	5.43	5.45	5.34	5.32	4.80	5.93	5.54	5.53
Ag grade (g/t Ag)	0	29.15	34.82	31.33	26.25	20.88	20.11	16.46	14.28	16.12	16.94	20.24	20.83	20.46	17.05	14.87	18.62	29.01
Au eq metal (koz Au eq) - plant output	0	10	102	101	122	119	120	111	112	103	101	102	100	100	89	109	103	68
Au metal (koz Au) - plant output	0	10	98	98	118	115	117	108	110	100	98	98	96	96	86	107	100	65
Ag metal (koz Ag) - plant output	0	34	348	313	419	333	321	263	228	257	271	323	333	327	272	237	297	301

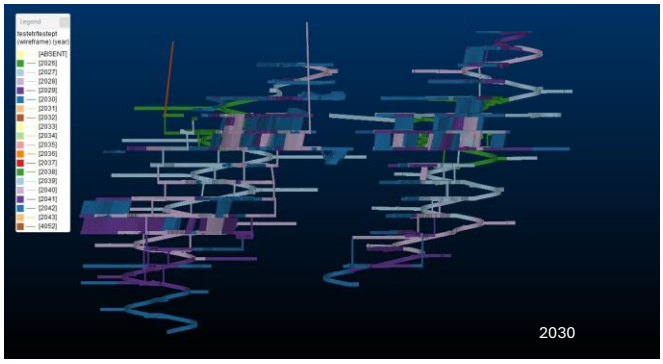
Figure 12-9 show the mine evolution for the life-of-mine:

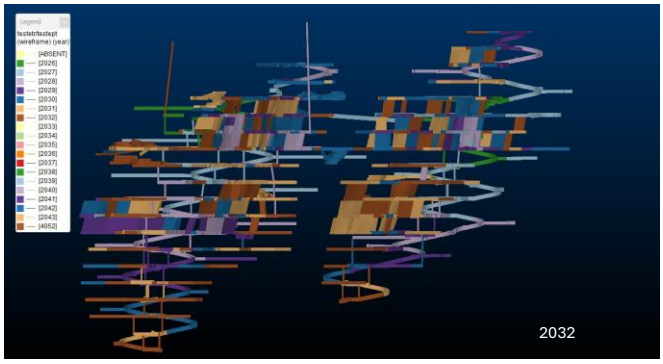
Figure 12-9: Mine Schedule LoM

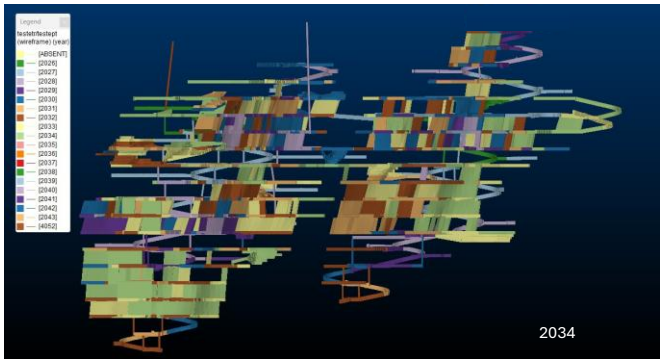


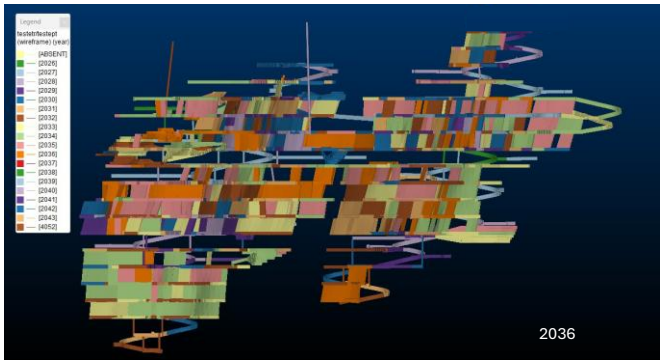


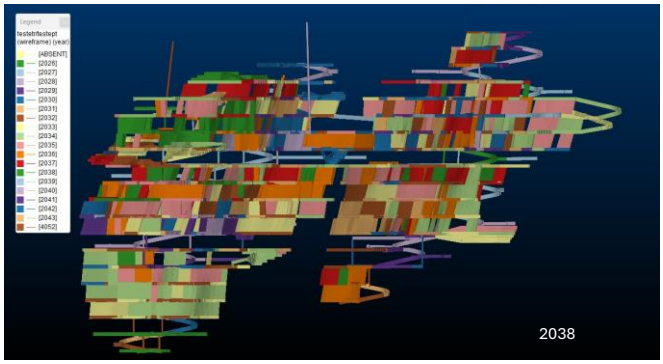


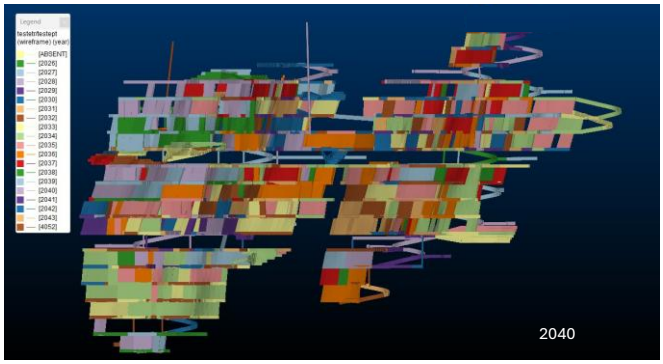


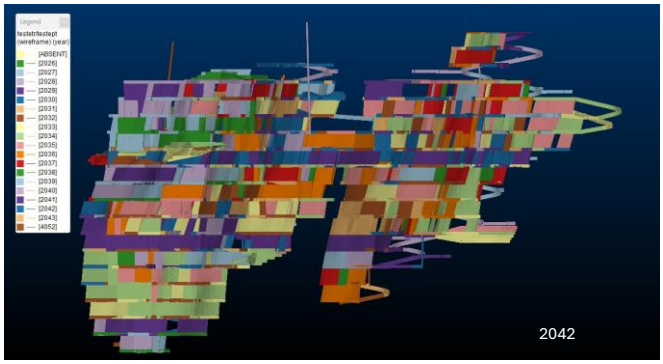


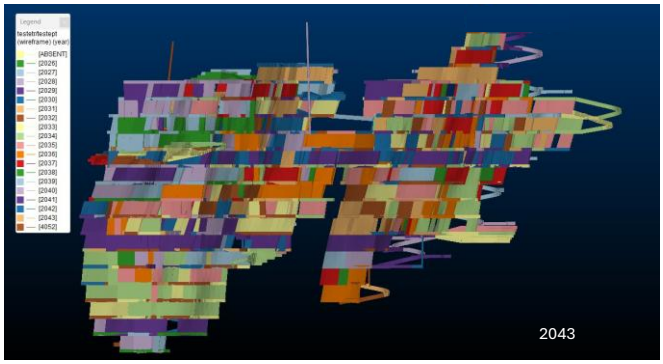












Source: Snowden Optiro, 2025.

12.6 Mineral Reserve Statement

The Mineral Reserve computed from Mineral Resources after the application of modifying factors as geotechnical and hydrogeological constraints, cutoff grades, optimization, design, infrastructure constraints and mine scheduling is shown in Table 12-9.

Table 12-9: Mineral Reserves

	Tonnage (kt)	Au grade (g/t)	Au metal (koz)	Ag grade (g/t)	Ag metal (koz)	Au Equiv grade (g/t)	Au Equiv metal (koz)
Proven	30	5.35	5	22.59	22	5.60	5
Probable	8,717	6.01	1,684	20.39	5,715	6.23	1,746
Proven + Probable	8,747	6.01	1,689	20.40	5,736	6.23	1,751

Mineral Reserve Notes:

1. The Mineral Reserve was estimated and classified in accordance with the USA S-K 1300 standards.
2. Mineral Reserve have an effective date of December 5, 2025. The Qualified Person for the estimate is Ruy Lacourt, B.Sc. Mining Engineering, MSc., Registered Member of the SME, an Associate of Snowden Optiro.
4. The Mineral Reserve was estimated using metal prices of US$2,000/oz Au and US$25/oz Ag, and metallurgical recoveries of 96% Au and 85% Ag. Underground mining costs were assumed as US$100/t (Long Hole mining) and US$115/t (Cut-and-Fill mining), with processing, site services and G&A costs as of US$32/t, US$18/t and US$20/t, respectively. Royalties comprise 1.05% NSR to the previous owner plus a 1.0% gross government royalty. Cut-off grades in gold equivalent are 2.82 g/t for underground Long Hole mining and 3.07 g/t for cut-and-fill.
5. The formula for gold equivalent: AuEq = Au grade + 0.011 * Ag grade.
6. The Mineral Reserve is presented on a 100% ownership basis fully attributable to Aura Minerals.
7. Tonnages and grades have been rounded in accordance with reporting guidelines. Tonnages are rounded to the nearest 1,000 t, metal grades are rounded to two decimal places. Tonnage and grade are in metric units; contained gold and silver are reported as thousands of troy ounces. Totals may not sum due to rounding.

The Mineral Reserve estimates were completed using industry-standard methodologies and software, and the resulting Mineral Reserve is reported in accordance with S-K 1300 requirements. The mine plan supports an economically viable underground operation, subject to the stated assumptions, modifying factors, and risk controls.

The Mineral Reserve was estimated from Measured and Indicated Mineral Resources which demonstrate economic viability, incorporating relevant dilution allowances and mining recovery factors.

Any Inferred Mineral Resources enclosed in feasible Indicated Mineral Resources envelopes either inside the stope shapes, dilution material or forced mine development was treated as waste, i.e., such material carried its excavation costs and no revenue was accounted for.

Gold and silver prices assumptions were defined in the beginning of the Feasibilty Study in line with Aura guidance and were deemed adequate at the time they were established by the QP.

Assumptions for gold and silver recoveries and mining costs are based on previous studies for the deposit and are assessed as adequate, within industry standards, by the QP.

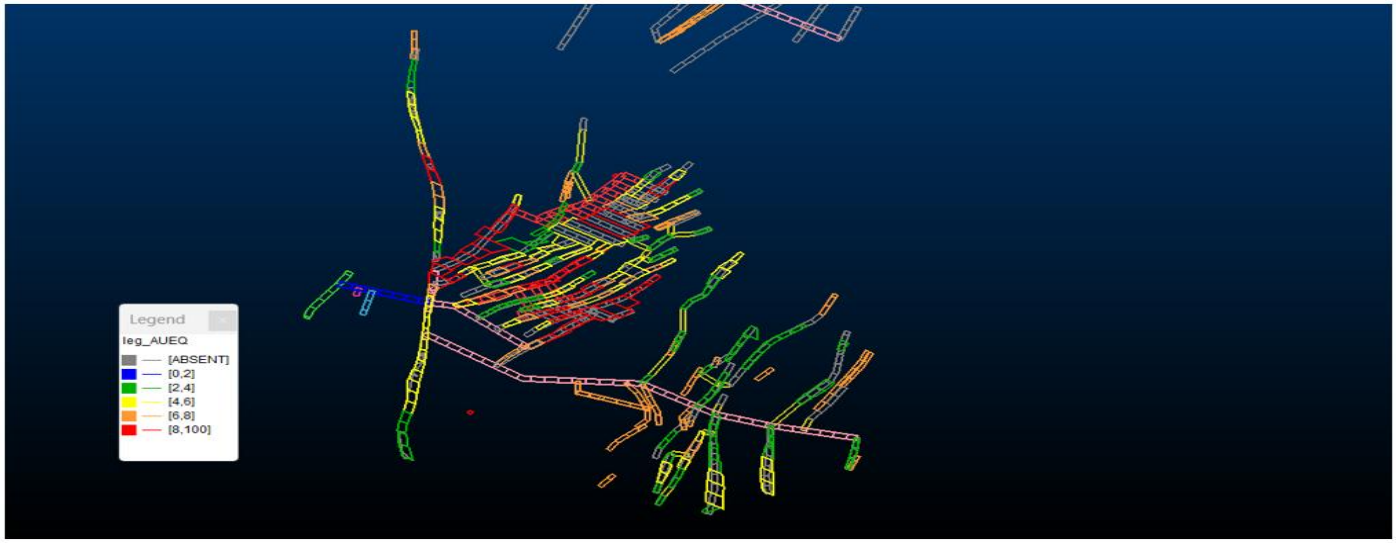
The Indicated Mineral Resources converted to Probable Reserve correspond to *in situ* ore to be excavated as there are no *in situ* Measured Resources at this stage and the Proven Reserve is composed by ore stockpiled at surface that was evaluated using the same criteria as those for the *in situ* Mineral Resources.

The mine plan delivers a metal equivalent production averaging 100 koz of gold equivalent for 15 years for a total mine life of 18 years.

12.7 Factors that may affect the Mineral Reserves

The key risks for this project include the complexity and uncertainty associated with vein positions. Incorrect interpretation or deviation in the estimation of vein geometry may lead to high dilution and/or an inability to mine adjacent stopes, particularly on sublevels where multiple sub-parallel veins occur, as illustrated in Figure 12-10.

Figure 12-10: Plan View of the 440 South Sublevel



Source: Snowden Optiro, 2025.

All Mineral Reserves are classified as Probable, as they have been derived exclusively from Indicated Mineral Resources. No Proven Mineral Reserves are reported because Measured Mineral Resources are not currently delineated within the areas contributing to reserve conversion. Accordingly, the Reserve estimate reflects the present level of geological confidence supporting the mine design and the application of modifying factors. Although the mine plan and associated modifying factors demonstrate reasonable prospects for economically viable extraction, a Reserve inventory comprised solely of Probable Reserves carries greater uncertainty than an inventory that includes a Proven component. Targeted infill drilling, together with ongoing grade and tonnage reconciliation, is expected to improve confidence in the supporting Resources and may provide a basis for future conversion of portions of the Probable Reserves to Proven status, subject to the results of such work.

Similarly to vein geometry, geotechnical or geometric assumptions that are either imprecise—often arising from limited sampling and characterisation—may lead to a reduction in the recovery of the Mineral Reserves. For example, underestimating the required pillar thickness between sub-parallel stopes could materially constrain extraction and leave parts of the orebody unmined. The current production schedule assumes that the Long Hole stopes within the same sublevel can be mined independently. Should geotechnical conditions ultimately require a mandatory extraction order or additional sequencing constraints exist between adjacent stopes, the planned mine sequence would be disrupted, with potential impacts on development priorities and production along the LoM.

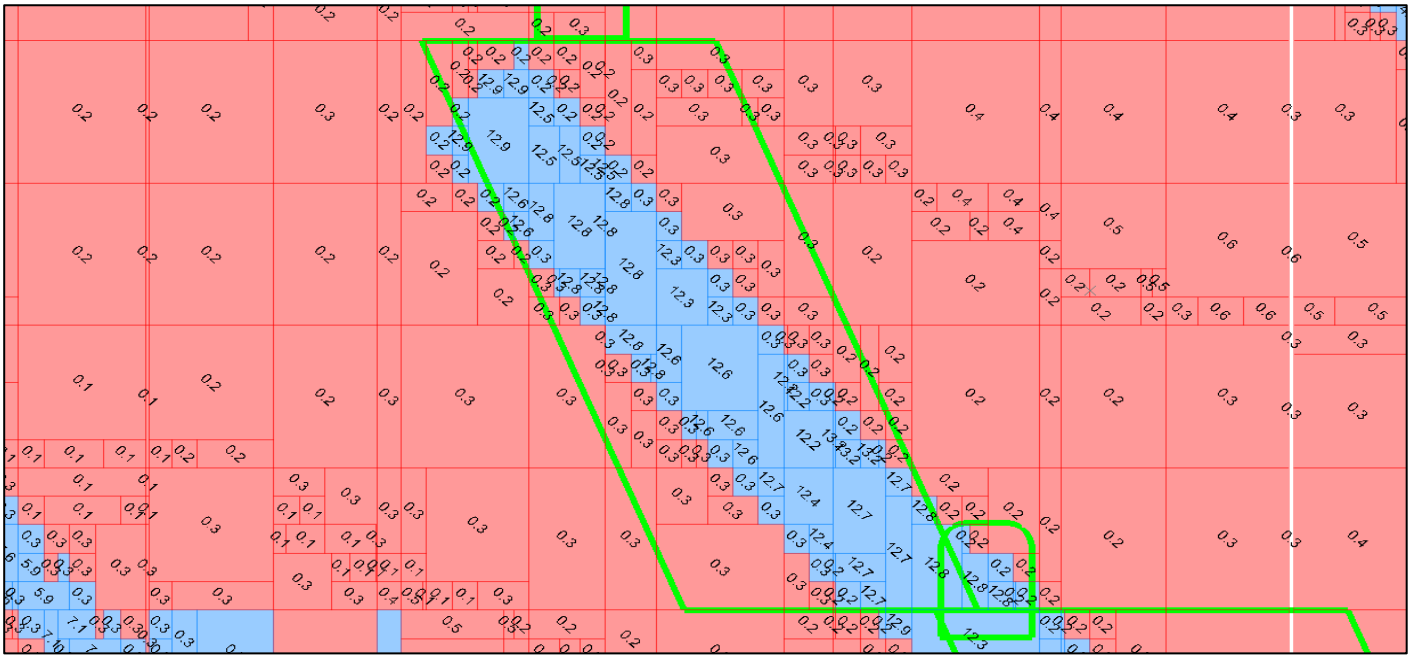
The geotechnical definition of the domains is highly relevant to ore recovery. Long hole stopes with dips less than 60 degrees are not recommended by the geotechnical assessment for Domain 1, which has fair geomechanical quality and intense fracturing. This limits the recovery of material by the Long Hole method in this domain, leading to higher dilution and a potential migration to cut-and-fill, which is less safe, less productive and more costly than Long Hole stoping.

Schedule of levels below the 420 level (water table) is high dependent on the effective operation of the dewatering system (via dewatering wells and pumping). This introduces operational safety risks due to the high temperatures involved, slower development rates (particularly for the ramps), reduced equipment availability, and increased cooling/ventilation requirements. The proper functioning of the dewatering wells also reduces the likelihood of encountering trapped water pockets in fractured zones.

The production sequence is also highly dependent on paste fill/CRF efficiency and performance. Any underperformance relative to the assumptions will delay stoping, adversely affecting operational safety and ore recovery.

There is potential for an increase in average stope grades, given that some diluent material within stopes is currently classified as Inferred and typically carries lower grades than Indicated material. For the purposes of this study, this dilution was treated as having zero grade. With conversion of this material to Indicated, the average grade and ounces within stopes are expected to increase. Figure 12-11 shows, within the green stope, gold grades associated with Measured/Indicated material in blue and Inferred material in red.

Figure 12-11: Detail of Dilution (Inferred Material in red) in a Sublevel Stope



Source: Snowden Optiro, 2025.

To achieve the 100 koz target as early as 2028, as stipulated, an accelerated development metreage was required. Any compromise leading to a lower production rate in 2028 would reduce this development requirement, while also allowing more time to build operational knowledge of the mine, improving the likelihood of meeting the target in 2029.

Given the uncertainties in the geomechanical characterization and rock mass response, there is a risk that areas currently planned for LH mining may need to be mined using C&F instead. This could result in additional requirements for C&F equipment and personnel.

To some extent, this risk is mitigated in the mine schedule, per se, as C&F mining is due to begin only in 2029, when the rock mass conditions and their geomechanical response should be better understood.

There is also a risk of higher-than-planned dilution, which could compromise the ability to meet production targets.

Ore stockpiling was limited to material mined prior to September 2027, preferably low grade. Assuming a larger stockpile volume including higher grades would provide greater flexibility to undertake productive development early on the existing levels, thereby giving more mining flexibility in the initial years. A temporary stockpile, together with the existing stockpile inventory (29,726 t grading 5.35 g/t Au and 22.59 g/t Ag), and limited to the first three years of operation, would be sufficient to provide this benefit. It is recommended not to blend high-grade and low-grade ore within the same stockpile.

There is potential to expand the Mineral Reserve inventory should a future evaluation support the development of an open pit to extract near-surface Mineral Resources that are not currently scheduled for recovery by the underground operation. Any such evaluation would be required to demonstrate technical and economic viability under S-K 1300, including confirmation of pit slope design criteria, stripping ratio, mining selectivity, surface infrastructure requirements, the metallurgical performance of these materials, environmental and social considerations, and the applicable permitting framework. Subject to positive study outcomes and corresponding updates to the modifying factors, portions of the presently excluded Mineral Resources could be reclassified as Mineral Reserves and incorporated into an integrated life-of-mine plan.

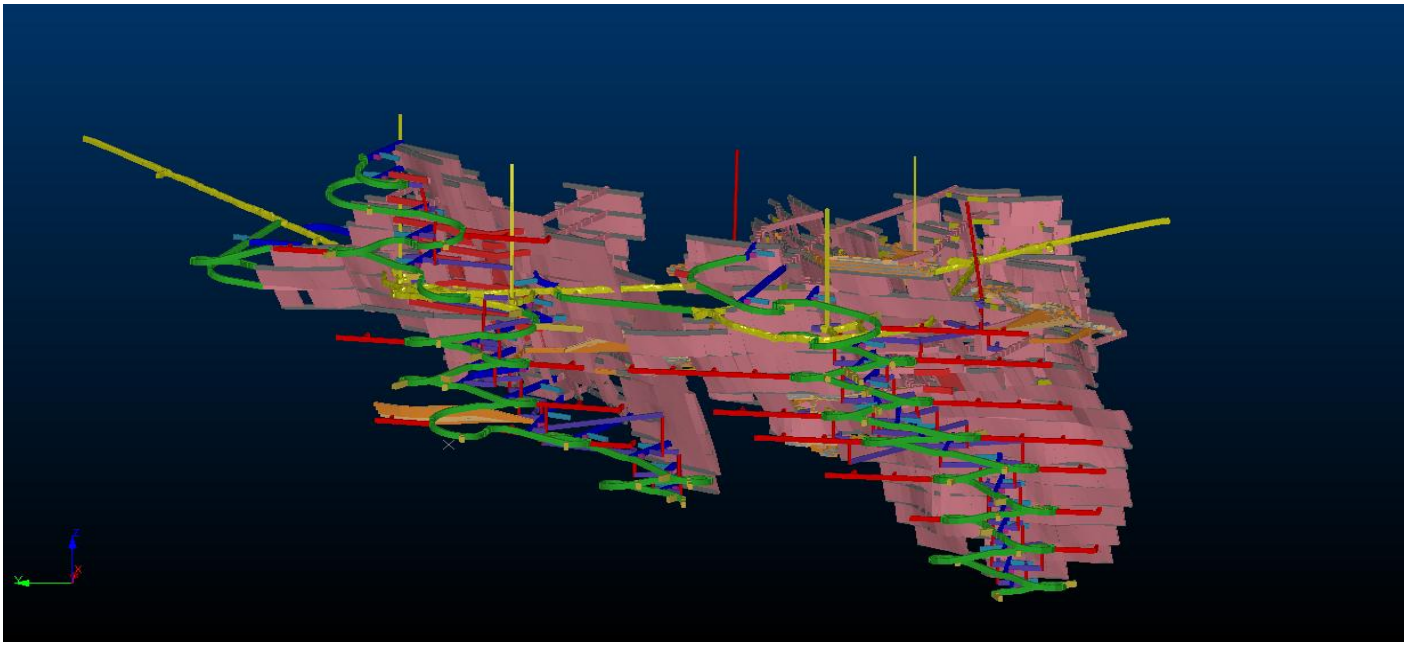
13 MINING METHODS

Mine optimization, design and production schedule were completed by Snowden Optiro using Datamine Studio UG. Stope shapes were generated with the Mineable Shape Optimiser (MSO) module and subsequently grouped and screened for economic value prior to integration with the mine development.

The resulting mining inventory, defined as the run-of-mine (ROM), includes Indicated mineralized material plus internal waste captured within the stope shapes as planned dilution. Production scheduling was then optimized using Datamine Task Scheduler (DTS), prioritizing early access to higher-grade zones while respecting operational constraints such as maximum development advance rates, nominal plant capacity, cemented rock fill (CRF) and paste fill placement and curing limitations as well as dewatering requirements.

Two underground mining methods were selected as the preferred alternatives for the Project based on orebody geometry, geotechnical conditions and productivity considerations. Long hole stoping (longitudinal and transverse) is the dominant method where the vein continuity and rock mass quality allow for larger spans across the stopes and efficient bulk extraction. Overhand mechanized cut-and-fill (MCF) was applied in areas of less favorable ground and/or less favorable stope geometry for long hole stoping, providing a selective method suited to narrow, high-grade veins under less favorable rock mass conditions. Together, these methods form the basis of the proposed mine plan, development layout (see Figure 13-1) and production schedule presented in this section.

Figure 13-1: Era Dorada Mine



Source: Snowden Optiro, 2025.

13.1 Mine Geotechnical

13.1.1 Mine Geotechnical Model Review

A detailed review of the available geological and geotechnical data was conducted to define the geotechnical parameters, ensuring the reliability of the inputs used for rock-mass characterization. The previous model classified domains only by lithology, which limited its ability to capture rock-mass variability. In the updated methodology, all RMR components were reassessed individually, with validated RQD values serving as the primary rock quality control parameter. Core-logging information was verified, reinterpreted from photographs, and incorporated into an updated distribution model used in subsequent geomechanical assessment.

With the recalculated parameters, a three-dimensional RMR model was produced to characterize the spatial distribution of rock-mass quality across the deposit.

Although a full 3D interpolation grid was not generated, the recalculated RMR values were projected onto the planned stopes using an inverse distance weighting scheme, effectively ‘stamping’ the RMR information on the mine design geometry. This approach enabled a direct visualization of geotechnical quality variations within the operational envelopes. This approach was selected as a practical solution for the current project stage, maximizing the use of existing borehole data while avoiding the artificial continuity that could result from full 3D interpolation.

The previous model had assigned RMR averages to lithological units only, resulting in overly simplified and homogeneous domains. The new dataset, however, revealed significant within- unit variability — demonstrating that lithology alone could not reliably predict rock-mass quality.

Initially, the project used three RMR-based domains defined as follows:

- Domain 1: RMR < 52
- Domain 2: RMR = 53–61
- Domain 3: RMR > 62

Snowden Optiro’s assessment highlighted that these intervals were too narrow, corresponding to less than one full RMR class (approximately 20 points). Consequently, the classification did not reflect meaningful mechanical distinctions and could not independently justify different mining approaches between domains. The narrow separation between these domains implies that any design differentiation based solely on domain boundaries would not be supported by statistically significant contrasts in rock-mass quality as depicted in Figure 13-2.

After the development of the RMR-based model, the same numerical domain structure (1–3) was kept for project consistency, but, instead, the domaining was defined from RQD-derived RMR as shown in Figure 13-3.

Figure 13-2: Original Domain Intervals (<52, 53–61, >62)

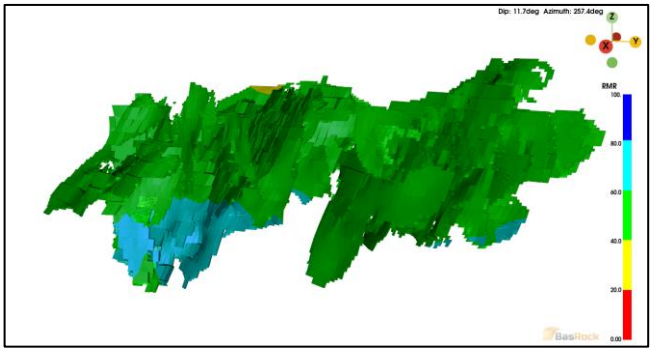
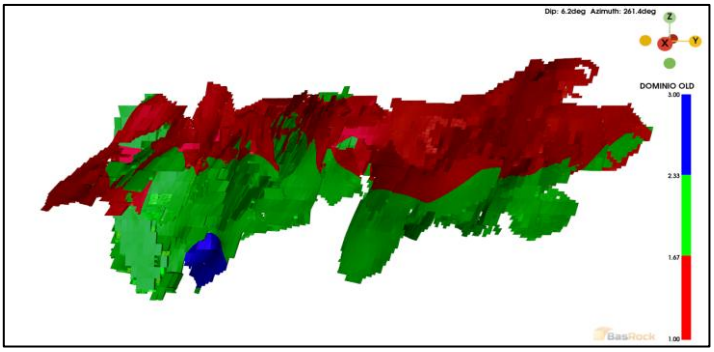


Figure 13-3: Revised Geotechnical Model with RQD-derived RMR Values



Source: Snowden Optiro, 2025.

The revised approach preserved the established domain nomenclature while ensuring that the parameters underpinning the model were technically valid. The updated RMR distribution provides a realistic framework for mine planning, allowing engineers to identify ground conditions more precisely and adapt excavation and support strategies to actual rock-mass behavior rather than to lithological assumptions.

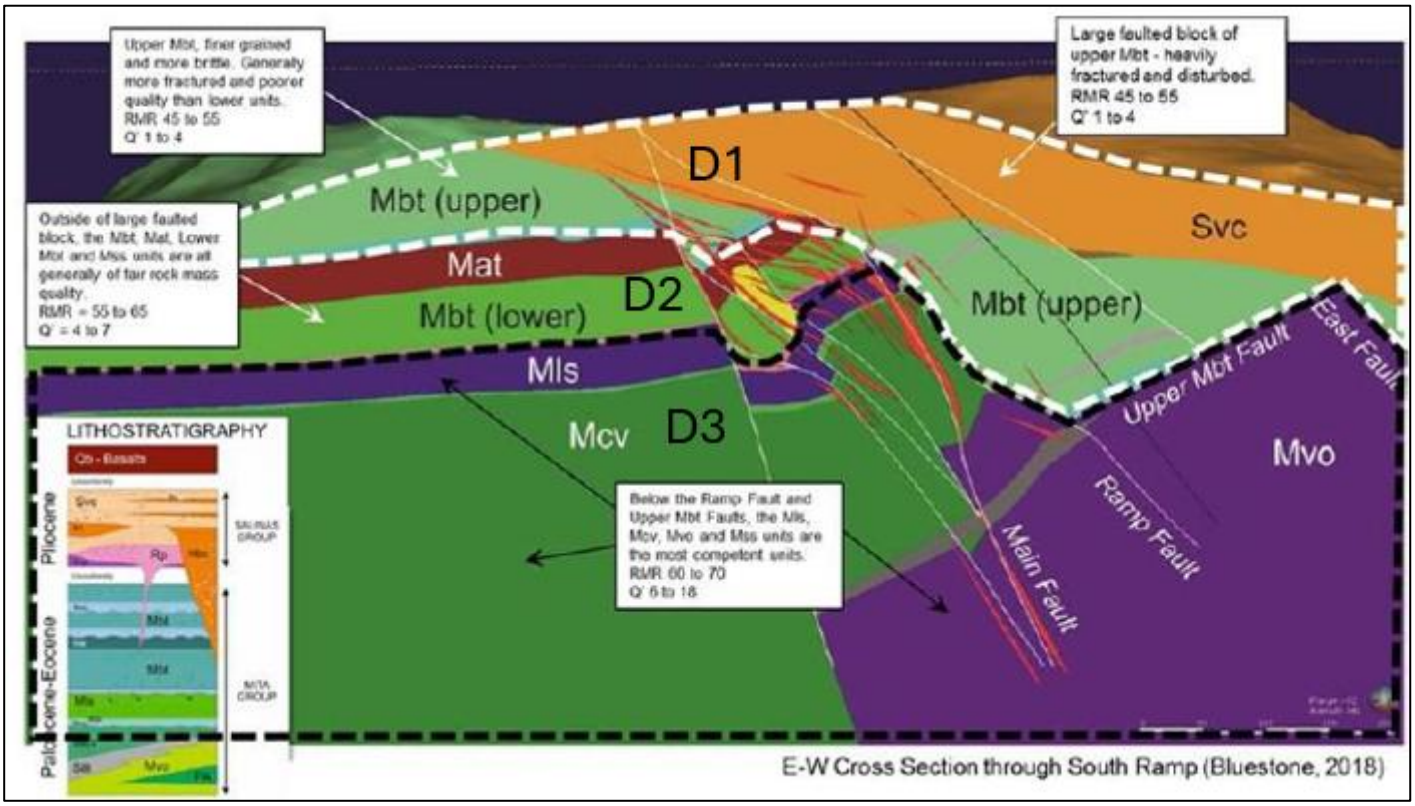
13.1.2 Material Properties

Material properties for both rock masses and backfill were assessed based on laboratory tests, previous reports and empirical correlations. The results are summarized in the following sections.

13.1.2.1 Rock Mass Properties

Based on previous work (GE21 Consultoria Mineral Ltda, 2024), three geotechnical domains were established, as shown in Figure 13-4.

Figure 13-4: Cross-Section of Geotechnical Boundaries



Source: GE21 Consultoria Mineral Ltda, 2024.

Geotechnical core logging, including RMR76 and Q' classifications for the three domains was previously documented (GE21 Consultoria Mineral Ltda, 2024). That study used sigci values estimated from point-load tests. However, uniaxial and triaxial laboratory tests performed in 2021 provide more reliable strength parameters for the main lithologies.

Table 13-1 compares laboratory UCS values with point-load estimates, and Table 13-2 presents the intact compressive strength (σci) and Hoek–Brown constant (mi) calculated from triaxial tests.

Table 13-1: UCS Values from Laboratory Tests and PLT.

Domain	Lithology	Laboratory				Plt
		UCS (Mpa)	E (Gpa)	N	AVERAGE	UCS (Mpa)
1 and 2	MBT	75.5	35.4	0.24	-	71 and 78
3	MLS	66.2	43.8	0.28	-	93
3	MVO	42.5	25.5	0.28	-	
1	SVC	104.1	55.6	0.24	182.7	71
1	SVC	154.4	57.6	0.2		
1	SVC	210.1	53.9	0.21		
1	SVC	297.8	76.5	0.17		
1	SVC	147.3	54.3	0.25		

Source: FF GEOMECHANICS ING. LTDA, 2021.

Table 13-2: Sigci and mi Calculation (Hoek-Brown Failure Criteria) from Trixial Results

Domain	Lithology	σci (MPa)	mi	RES
1 and 2	MBT	36	50	0.296
3	MCV	136	1	3.791-e32
3	MLS	-	-	-
3	MVO	81	6.6	1.018
3	SVC	-	-	-

Source: FF GEOMECHANICS ING. LTDA, 2021.

A comparison of laboratory and point-load results indicates that UCS values for SVC are considerably higher than those for MBT. To adopt a conservative approach, MBT strength parameters were used to represent Domain 1.

The MBT, MLS and MVO datasets contain fewer than five valid tests each and therefore do not meet ISRM requirements for statistical reliability.

The mi value of 50 obtained for MBT is not representative of this lithology; a value of 10 was adopted based on typical ranges and triaxial behaviour.

13.1.2.2 Fill Properties

Considering fill type variations of cemented rock fill (CRF) and pastefill (PF) in different proportions, it was decided to adopt backfill (BF) properties for stability studies. Any application of CRF and PF in any proportion will have higher resistance than BF alone. It is a conservative assumption to facilitate numerical modelling sequence.

Backfill was previously studied for Era Dorada Project by Paterson & Cooke (Paterson & Cooke, 2018). That report indicated an average UCS for backfill of 160kPa for 4.5 m thick stopes and it characterized the material with a density of 2.651±2 t/m3 for tailings and 2.644±2 t/m3 for aggregate.

A reference for friction angle (ϕ) value came from bibliography (Hatami & Bathurst, 2014) which suggests ϕ varying from 20 to 30° for granular soils with large fines content. For the Era Dorada Project, an average value of 25° was adopted.

Cohesion (c) as a function of ϕ and UCS can be obtained by the relation between compressive stress and the stresses related to the shear plane (shear (τ) and normal (n) stresses) created when the compressive stress is equal to UCS.

- $\tau = UCS\cos\phi$ (1)
- $n = UCS\,\mathrm{sen}\phi$ (2)
- $\tau = c + n\tan\phi$ (3)

Considering UCS = 160kPa and ϕ = 25° in equations above, the value for c obtained is 5kPa.

13.1.2.3 Properties Summary

A summary of rock mass and fill properties considered for the Era Dorada Geotechnical Studies, including information for finite element analysis model building is presented at Table 13-3.

Table 13-3: Rock Mass Properties for Era Dorada Underground Mine

Domain/ Material	Lithology	RMR76	GSI	Q'	UCS (MPa)	Tensile Strength (MPa)	E (GPa)	ν	mi	γ (g/cm^3)	D	mb	s	a	ϕ (°)	c (kPa)
1	MBT	50	45	2	75	-	35	0.24	10	2.59	0.65	0.544728	0.000409	0.508086	-	-
2	MBT	58	53	5	75	-	35	0.24	10	2.59	0.65	0.83178	0.001273	0.504656	-	-
3	MVO	63	58	18	48	-	26	0.28	5.6	2.59	0.65	0.606861	0.002587	0.503276	-	-
Backfill	-	-	-	-	-	0	6.3	0.17	-	2.6	-	-	-	-	25	5

13.1.3 Empirical Assessment for Stope and Design Guidance

Empirical assessment was made for stope stability, dilution, crown pillar stability and pillar stability between stopes, considering long hole mining method geometry. The objective is to set geometrical parameters of the stopes and pillars to start a new mine design.

13.1.3.1 Stope's Stability and Dilution

Stope stability and dilution were assessed by the relation between rock mass quality and THE stopes geometry IN stability charts. General hangingwall and footwall stability was assessed by THE updated stability chart published IN "Cablebolting in Underground Mines" (Hutchinson & Diederichs, 1996) and dilution was assessed by THE graph of Equivalent Linear Overbreak Sloughing (ELOS) (Clark & Pakalnis, 1997).

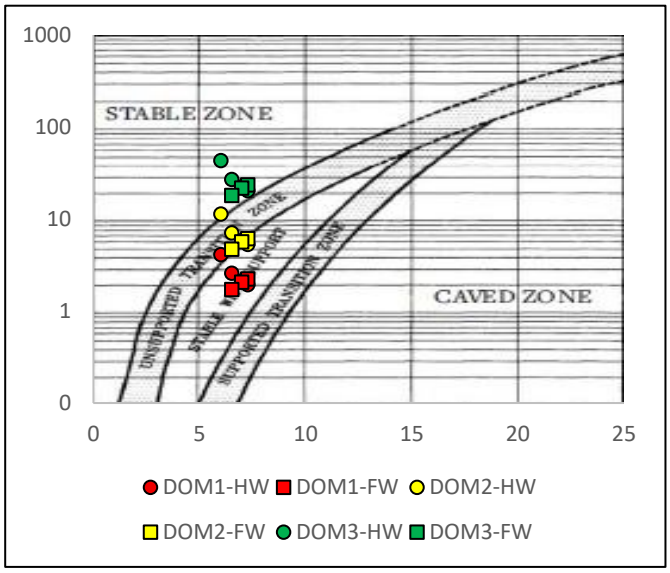
The charts are based on the relation N' vs. HR where N' is the "stability number" obtained from the rock mass classification index Q' modified by stress, joint orientation and gravity effect factors, named respectevely A, B and C. N' is the product of Q' and all factors as presented in equation (4).

N'=Q'xAxBxC (4)

A simplification of the stability chart process was made, considering the project level but in a conservative manner. The considerations for simplification are:

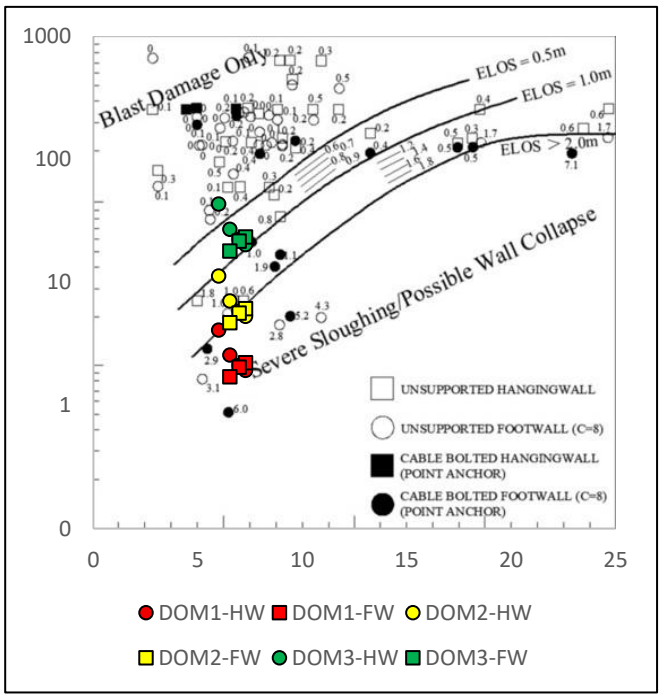
- All induced stress in all stopes was considered as the maximum vertical stress at 300m depth (8Mpa). For UCS values of 48 and 71MPa, factor A is 0.5 and 1.
- Main foliation was considered parallel to the orebodies (B = 0.3).
- Failure mode at the hangingwall was considered to be slabbing and sliding at the foootwall for factor C calculation.
- Dip values assessed were 45, 50, 60 and 90°
- Vertical heigth for each sublevel is 20m. Dimension h for HR calculation was considered 20/sin(dip).
- Strike long span is 30 m, a project requirement.

Figure 13-5: Modified Stability Chart



Source: Hutchinson & Diederichs, 1996.

Figure 13-6: ELOS Chart



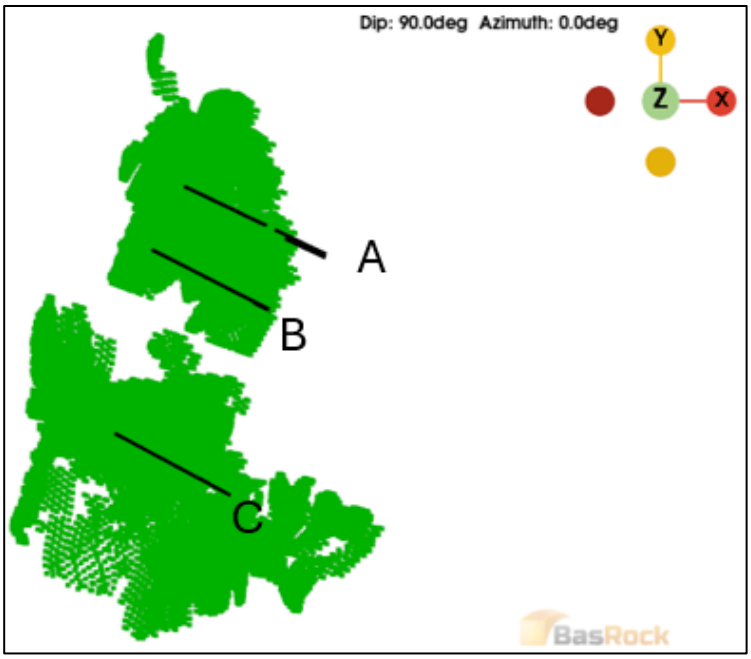
Source: Clark & Pakalnis, 1997.

Stability results (Figure 13-5) indicate domains 2 and 3 are adequate for the long hole mining method, considering the geometries assessed. Long hole for Domain 1 is feasible with systematic support for inclinations under 60°.

Dilution results (Figure 13-6) indicate an average dilution of 0.75m for long hole at Domain 3, but for Domains 1 and 2, values above 1.5 m are deemed not representative of the dilution at the mine. For such domains, an assessment by 2D stress x strain numerical model was made, considering the properties in Table 13-3, conceptual mine design geometry and totally open stopes. The materials were considered elastic. Initial element loading was considered by field stress and body force and k = 1.5.

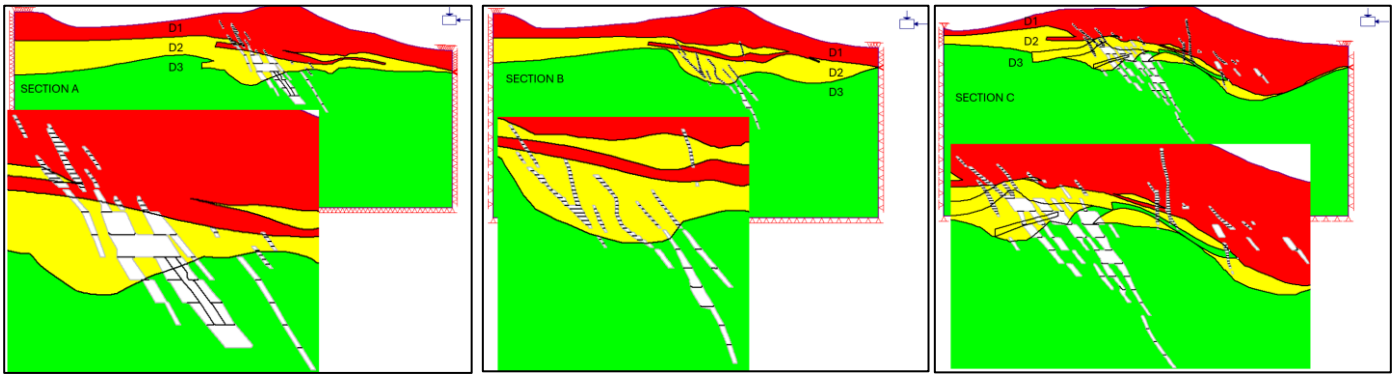
Three sections were selected for the assessment, two at the north ore bodies and one at the south (Figure 13-7, and Figure 13-8).

Figure 13-7: Sections A, B and C Plan View



Source: Snowden Optiro, 2025.

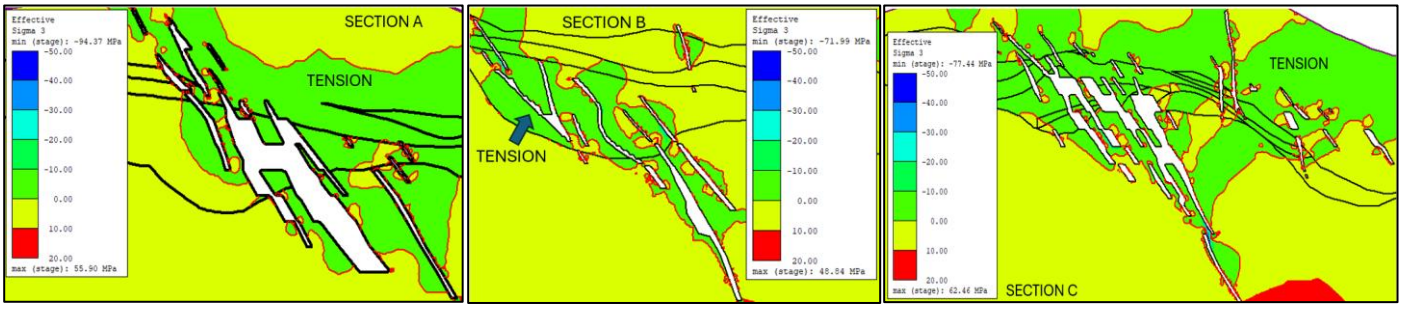
Figure 13-8: Geomechanical Sections A, B and C



Source: Snowden Optiro, 2025.

Dilution was considered to be directly related to the extension of tension stress (sigam 3 < 0) surrounding the surfaces. The results are presented in Figure 13-9.

Figure 13-9: Sigma 3 Results for Sections A, B and C



Source: Snowden Optiro, 2025.

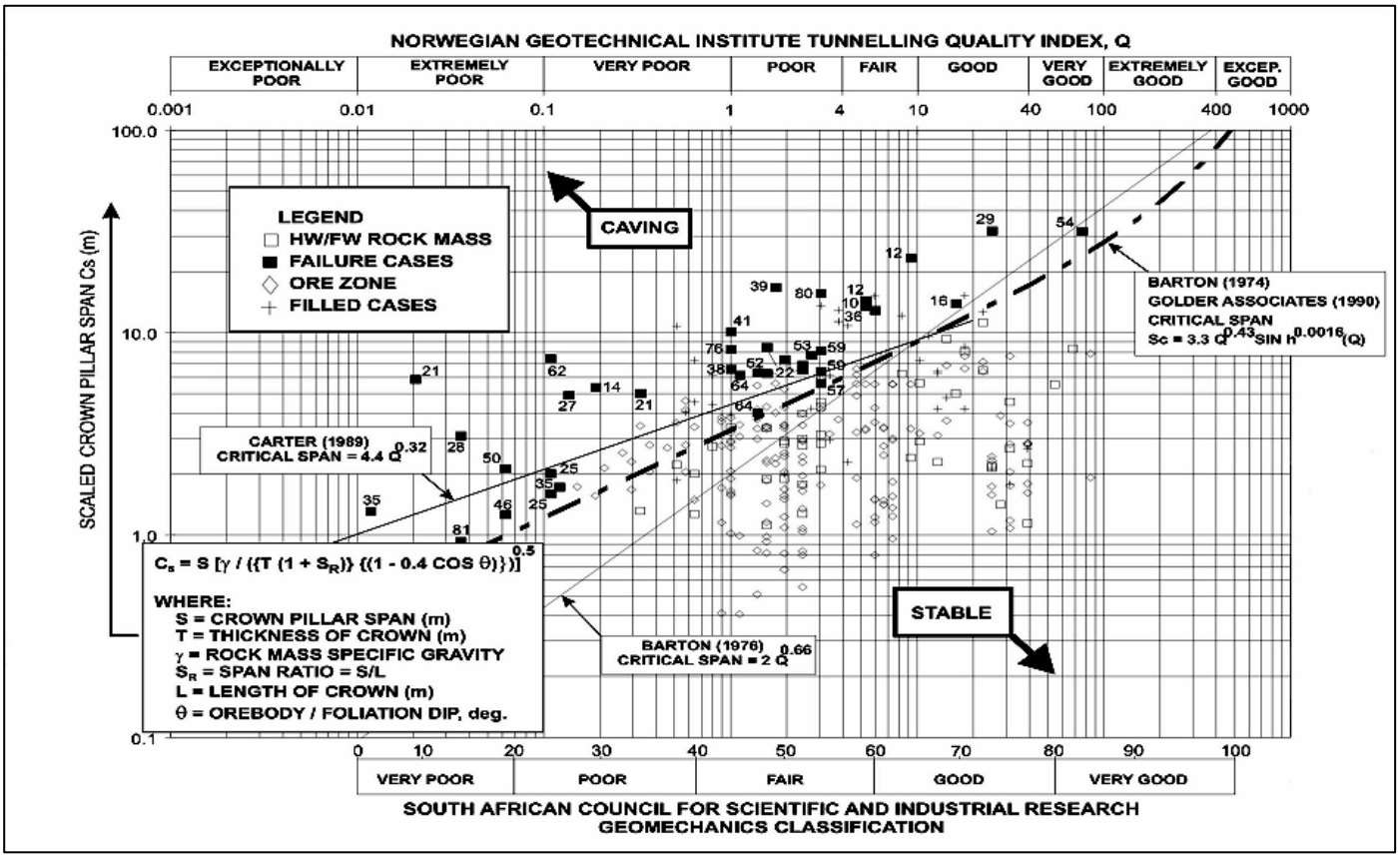
The results of Sigma 3 show general tension around the openings indicating impossible dilution geometries for mining budget. Nonetheless, the simulation considered totally open stopes irrespective of the level. This condition is extreme and will not happen during the mining operation which has a sequence of opening and fill each ore lens before starting the next. Thus, such extremely high distress observed at Figure 13-9 will not really happen in mining.

In general, the stopes have high dip and even in depth, with tension, the physical ways for rockfall are limited and this will be even more restricted and controlled by filling. Consequently, a planned dilution as of 0.75 m towards each wall rock was adopted for the mine optimization and initial stope design.

13.1.3.2 Crown Pillar Stability

The crown pillar stability was assessed by the empirical method "Guidelines for use of the 'Scaled Span (Cs) Method for Surface Crown Pillar Stability Assessment' (Carter, 2014). Carter's methodology correlates the scaled crown pillar's span with rock mass classification to assess its stability in a graphic output. The abacus and Cs formula are presented in Figure 13-10.

Figure 13-10: Crown Pillar's Stability Chart



Source: Carter, 2014.

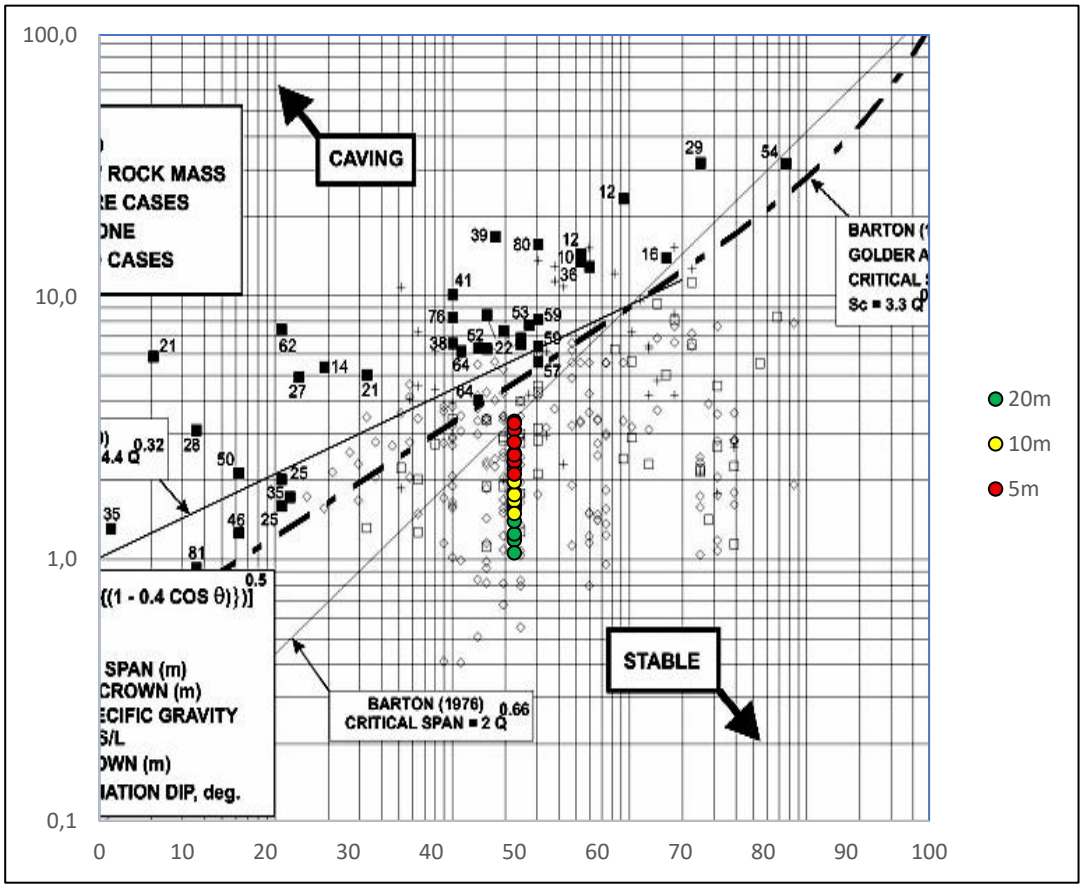
The RMR system was used for Era Dorada stability assessment, as presented in Table 13-3. The Crown Pillar geometry recommendations were then communicated to the mine planning technical team. Two different geometries were considered for the crown pillar assessment at the north and south orebodies. The spans adopted for the north and south orebodies are presented in Table 13-4.

Table 13-4: Span's Geometries for Era Dorada's Crown Pillar's Stability Assessment

Region	Depth (m)	Span (m)	Length (m)
North	552	2.7	58.3
South	512	4.0	57.1

The crown pillar thickness is a concern only for domain 1. Carter's Cs formula was applied for domain 1 at different dip inclinations, 45, 50 and 60° and γ = 2.59 g/cm^3. Three crown pillar thickness were assessed: 5, 10 and 20m. Results are presented in Figure 13-11.

Figure 13-11: Crown Pillar's Stability Results for 5, 10 and 20 m Thick Pillars



Source: Snowden Optiro, 2025.

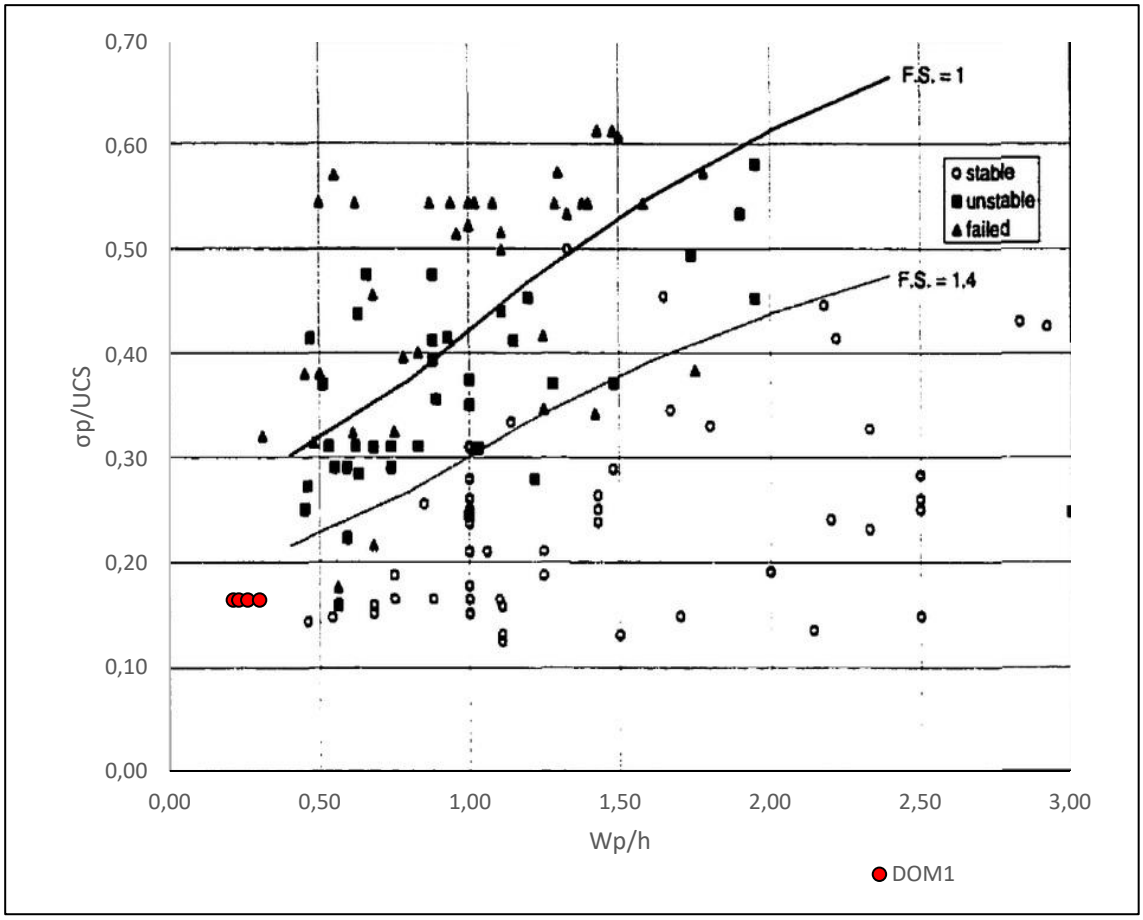
The crown pillar is stable for the thickness assessed although 5m thick pillar are approching Barton's stability limit (Figure 13-11). It is plausible to consider 10m thick pillars for the mine design.

13.1.3.3 Pillar Between Stopes

Parallel lenses with slender waste pillars between them are frequent in Era Dorada (Figure 13-8). The stability of the pillars between stopes was assessed by the empirical method purposed in 'Determination of the strength of hard-rock mine pillars' (Lunder & Pakalnis, 1997)(Lunder & Pakalnis, 1997). It is based on a correlation between the ratios of induced pillar stress (σ_p)/UCS and pillar width (Wp)/pillar height (h) to find the pillar factor of safety.

Only domain 1 (UCS = 71 MPa) pillars were simulated by the methodology. Vertical stress (σ_z) was calculated only for the maximum mine depth, 300 m, as the assessment for stope stability. σ_z is 8 MPa. Induced pillar stress (σ_p) was obtained by the tributary area theory (TAT) and the dimensions are explained in Figure 13-12.

Figure 13-12: Pillar Stress and Strength Relation Due to Geometry



Source: Snowden Optiro, 2025.

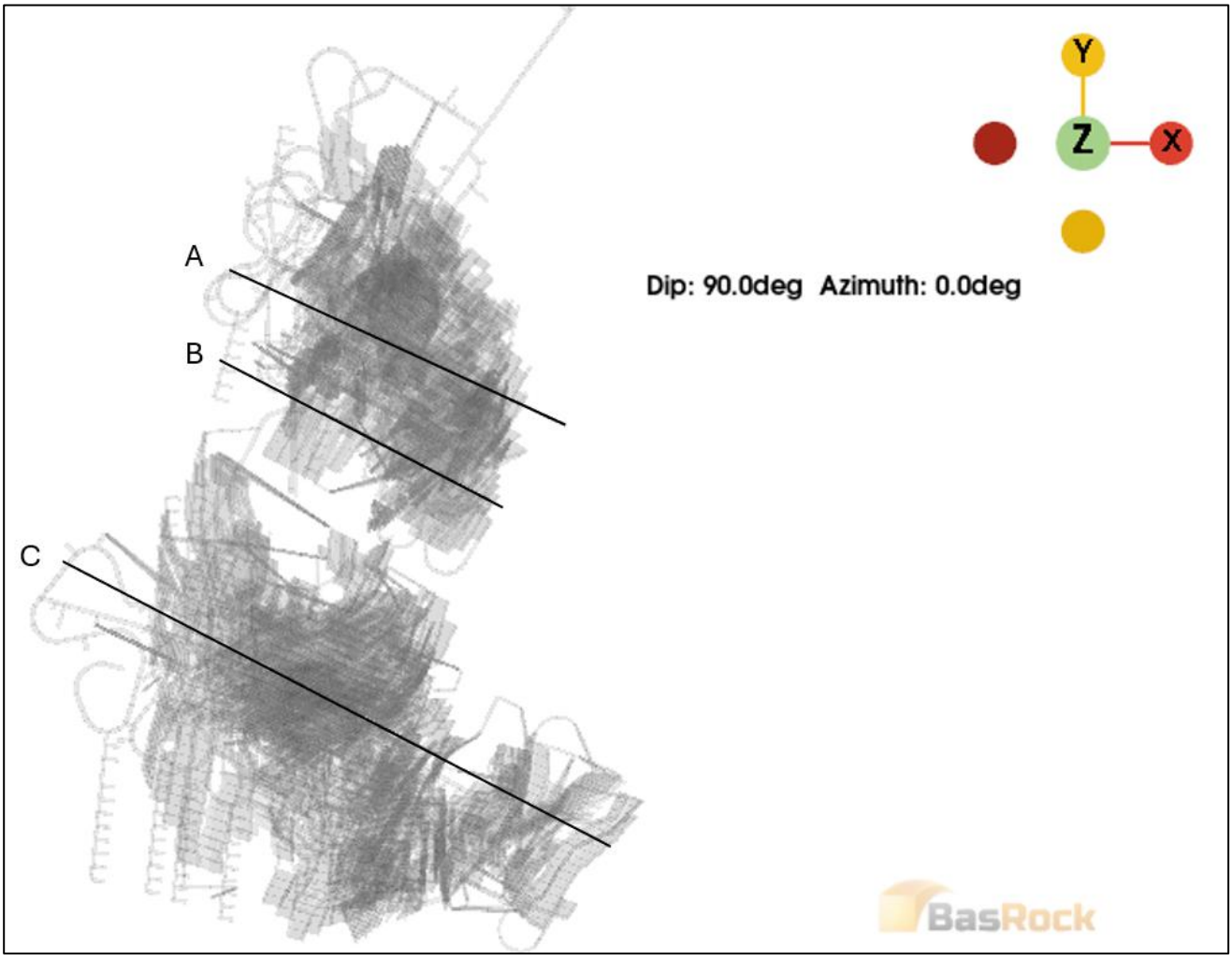
Results indicate slender pillars with a factor of safety values around 1.4. Stress may not produce pillar failures, but slender pillars are easily damaged during the mining process. Intact pillars between stopes aren't a requirement for

mining, as even plasticized pillars will contribute to control dilution. Anyway, filling will be determinant for successful mining.

13.1.4 2D Numerical Model Assessment for the Final Mine Design

The geometry assessed in the empirical studies formed the basis for the mine design, which was subsequently evaluated using a 2D finite element stress–strain numerical model. Three sections were selected for the analysis at the same positions used in the dilution assessment.

Figure 13-13: Sections A, B and C at Reviewed Mine Design



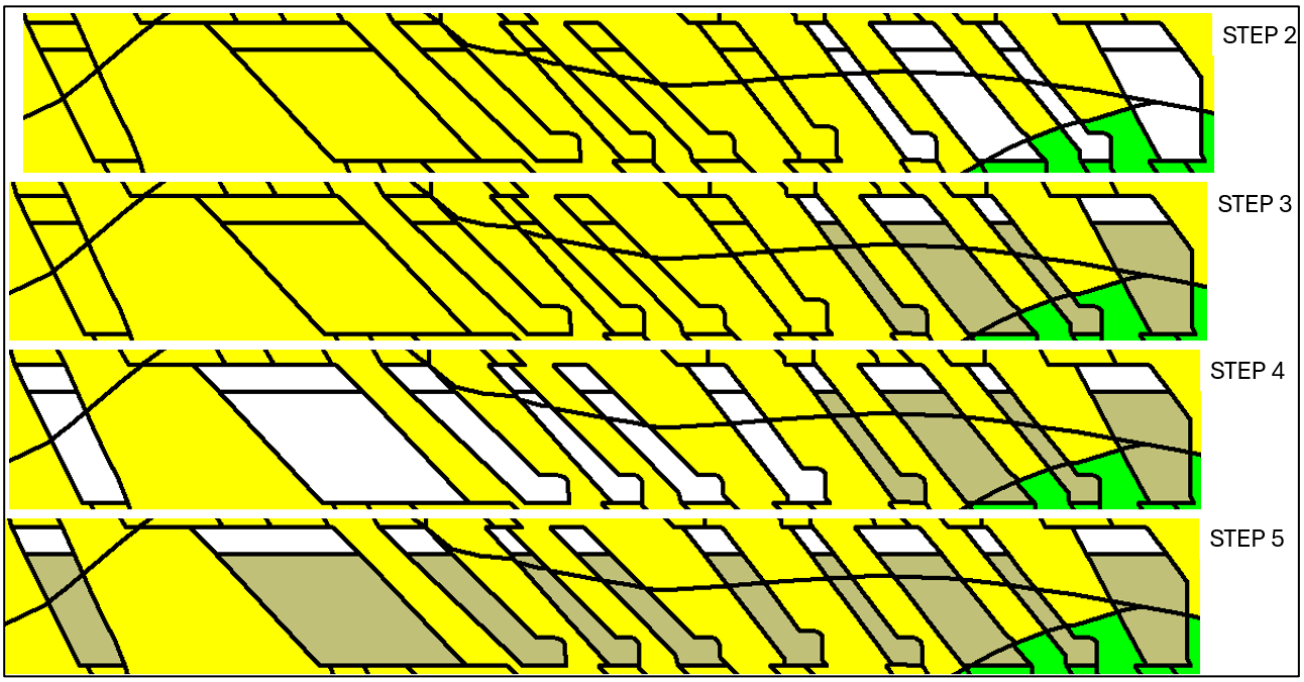
Source: Snowden Optiro, 2025.

The assessment of the final mine design included the evaluation of fill influence on stability, implying in the assessment of a sequence of mining steps. For practical reasons, it is deemed necessary a full sequencing in which each lens is mined, filled and then the next footwall lens is mined – such simulation would take anywhere between 100-200 steps. Hence, a simplified sequence was simulated for mining/filling steps from the hangingwall to the footwall. Thus, six steps were defined for such assessment:

1. Unmined;
2. Hangingwall lenses mined (about half of the lenses), simulated all the way to the level just above the first sill pillar;
3. Hangingwall lenses filled;
4. Footwall lenses mined;
5. Footwall lenses filled;
6. Totally mined and filled section.

Steps from 2 to 5 are presented for Section B in Figure 13-14 as an example.

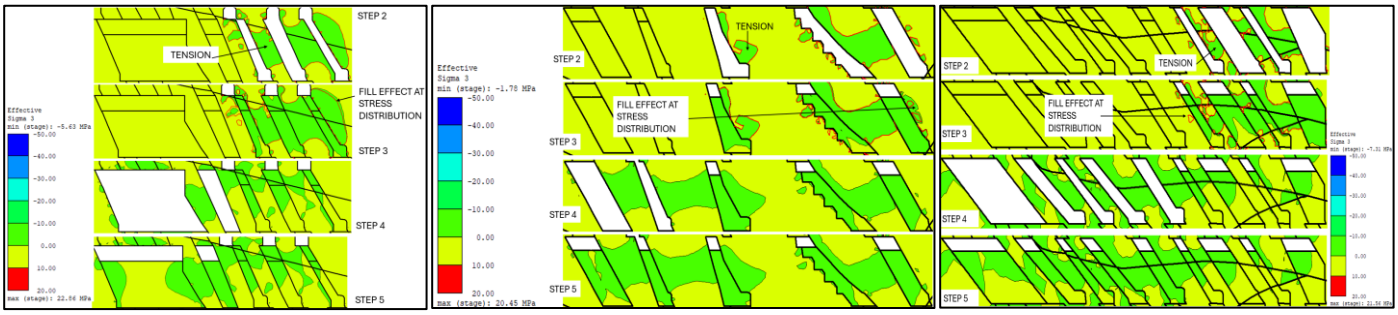
Figure 13-14: Steps from 2 to 5 for Section B



Source: Snowden Optiro, 2025.

The same material and stress properties used in the dilution assessment were applied here. Only geometry was changed. As the fill was included into the analysis, its initial element loading was considered only by body forces to avoid overestimating its stress-distribution capacity. The key analysis parameter was sigma 3, considering tension regions as an indication of ready to fall areas (the same as in the assessment for dilution). Results are presented in Figure 13-15.

Figure 13-15: Sigma 3 Results for Sections A, B and C



Source: Snowden Optiro, 2025.

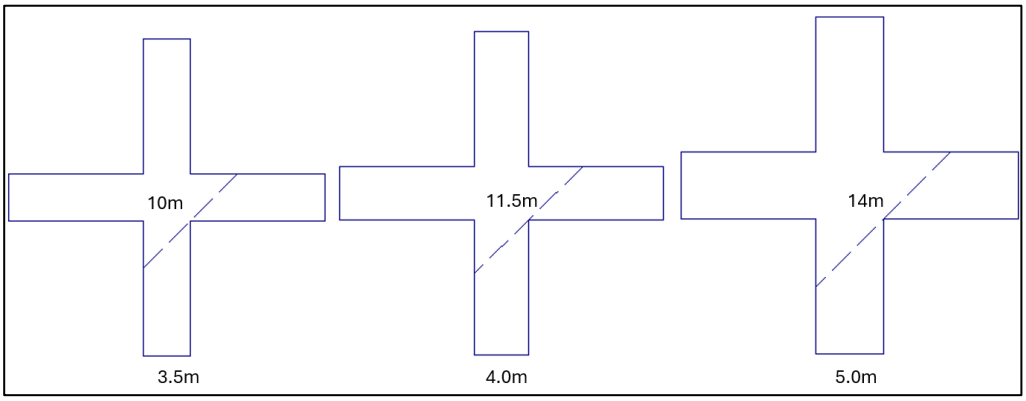
Results indicate overall tension in all stope's lenses. Failure is highly possible but considering general high dip angles, filling will have a dominant effect in avoiding fall of ground, maintaining pillar's mass. The fill showed stress distributions capabilities even with the body force loading. It is evidence that fill will be highly effective in walls confinement. Following the correct mining sequence without missing any of the filling steps will be a key fundamental procedure for stability control in the Era Dorada Mine.

13.1.5 Mine Development Reinforcement and Support Requirements

Development support recommendations were addressed using empirical assessment. Barton's chart was used. The formulas originally created in 1974 were updated in 'Using the Q-system Rock mass classification support and design NGI (NGI, 2015). The methodology is complete in providing support descriptions according to rock-mass quality and opening span. For this purpose, the method relates Span/ESR vs. Q in a chart for reinforcement and support types, mesh and shotcrete thickness.

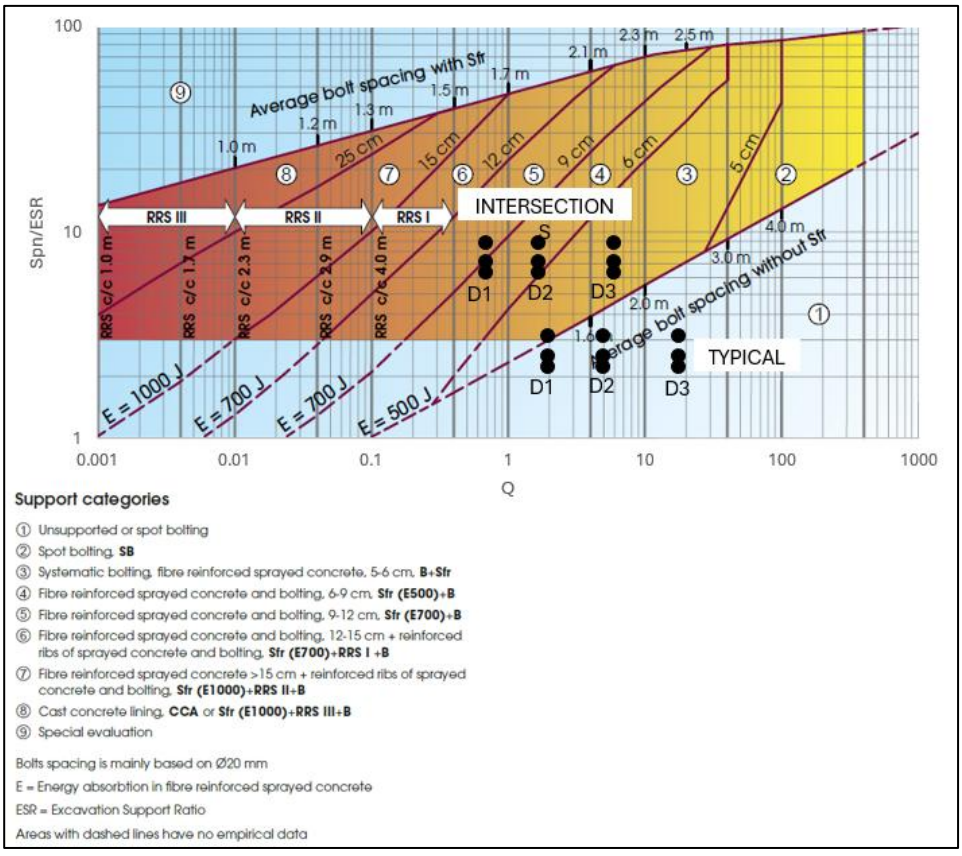
Q scores for all domains are presented in Table 13-3. ESR stands for excavation support ratio a value determined by type of excavation. Most of Era Dorada openings are classified as 'permanent mine openings' and its ESR score is 1.6. The mine design considered three different spans for galleries, including its intersections. Such spans are presented in Figure 13-16. The Barton's Support recommendation chart is presented in Figure 13-17.

Figure 13-16: Plan View of Main Galleries Spans and its Intersections



Source: Snowden Optiro, 2025.

Figure 13-17: Barton's Support Recommendation Chart



Source: NGI, 2015.

Due to unpredictable safety conditions related to underground fall of ground, any ground must be supported to secure safe conditions underground, hence, even the drifts in the 'no support' region in Figure 13-17 must be supported.

Considering visual observation at the mine it is good practice to apply rockbolts in all openings, irrespective of the domain and span. Surface support is also recommended but it can vary with domain and span.

The highest rockbolt load required is 3.9t. Operational tasks require hydrabolts ('swellex type') for development, as resin would not perform to desired standards at high rock temperatures to be encountered in the mine. 'Swellex Mn12' (or one of its similar) is a commercially known swellex type bolt with 11t of breaking load and 9t of yield load. Thus, it meets the required load and operational requirements, but as the site will have relevant subsurface water, the swellex bolts must be galvanized.

Some intersections require more than 2.4m long bolts which is the best suited commercial length for swellex vs. the size of the underground openings. Furthermore, there are many operational complexities in stopping the application of usual development support and start applying one specific for intersections. The application of cablebolts after finishing intersection development is then recommended to achieve larger anchoring lenghts.

Shotcrete thickness varied from 5 to 12 cm at Figure 13-19 and the recommendation is to use a thickness of 5cm thick and adjust for higher thickness based on observed shotcrete performance at the mine.

Weld mesh is an option of surface support in places where no support was required as typical galleries of domain 2 and domain 3. Aperture of 5cm x 5cm is sufficient to avoid potential falls of ground. The mesh must also be galvanized.

The final support recommendations are presented in Table 13-5.

Table 13-5: Era Dorada Underground Mine Development Support Recommendation

Gallery	Domain	Span (m)	Swellex		Cablebolt		Shotcrete (5cm)	Weld Mesh (5cmX5cm)
			Mesh (mxm)	Length (m)	Mesh (mxm)	Length (m)		
Typical	1	3, 4 and 5	2.0x2.0	2.4	-	-	Y	N
	2 and 3	3, 4 and 5	2.0x2.0	2.4	-	-	N	Y
Intersection	1 and 2	10 and 11.5	-	-	1.5x1.5	3.0	Y	N
		14	-	-	1.5x1.5	3.5	Y	N
	3	10 and 11.5	-	-	1.5x1.5	3.0	N	Y
		14	-	-	1.5x1.5	3.5	N	Y

The recommendation in Table 13-5 is a product of empirical methods and perception of the rock mass behavior in the site visit. It is important to consider the influence of joint sets, planar and wedge failures for the further project development.

13.2 Hydrogeology Analysis and Dewatering

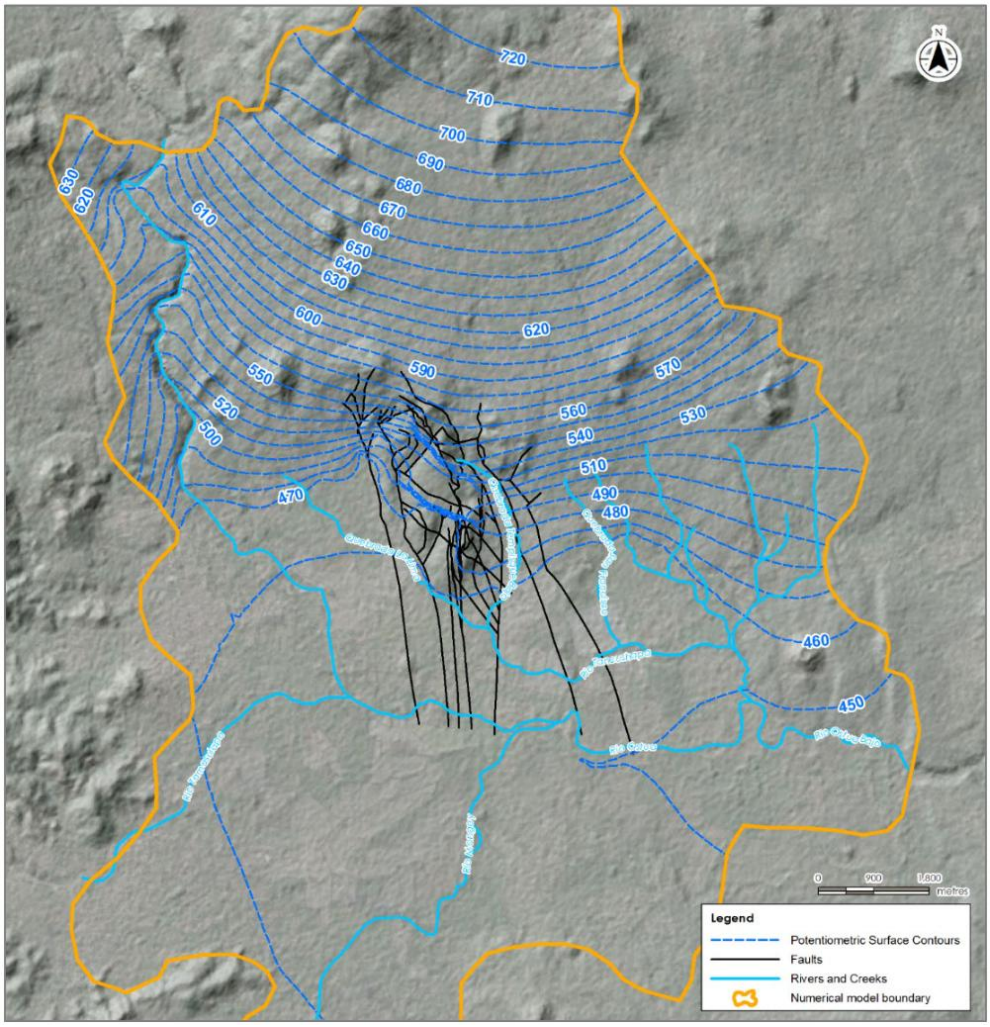
13.2.1 Hydrogeologic Setting

The interpretations presented in this section are derived from hydrogeologic assessments conducted by Stantec in 2018 and 2025, including the regional groundwater model and the updated hydrogeologic analysis developed for the Era Dorada project. Dewatering of the underground mine represents a key technical and economic constraint, as groundwater occurs under steep thermal gradients and locally very high temperatures, approaching 190 °C. These conditions affect water viscosity, pumping efficiency, and the potential for steam flashing in wells and mine workings, and therefore must be explicitly incorporated into mine planning and water-management strategies.

The Era Dorada hydrogeologic system comprises five principal hydrostratigraphic units with distinct hydraulic behavior. Alluvial and colluvial deposits form a shallow unconfined aquifer with high permeability and storage, particularly along the Río Ostua valley, where they act as the main discharge zone for the regional flow system, with reported hydraulic conductivity ranging from 0.54 to 0.87 m/d. The overlying Salinas Tuff and associated volcaniclastic sequence display heterogeneous permeability controlled by fracturing and hydrothermal alteration, with typical K values on the order of 0.04 to 0.3 m/d, while siliceous sinters within this unit form low-permeability zones that locally restrict groundwater flow. The Mita Group volcanic–sedimentary sequence transmits groundwater mainly through secondary porosity in fractures and faults, with hydraulic conductivity varying widely from 4 ×10⁻5 to 0.2 m/d, generally decreasing with depth, which limits vertical drainage and influences the interaction between underground workings and the surrounding rock mass. To the north, a basaltic unit is interpreted to have relatively high permeability, with reported K values spanning several orders of magnitude (0.01 to 1,728 m/d). It contributes to regional recharge, providing longer flow paths that connect upland recharge areas to the mine area and the Río Ostua catchment. At greater depth, the Tempisque Volcanic Complex presents very low matrix permeability, typically between 8×10^{-9} and 2×10^{-6} m/d, but fault zones within this unit act as localized conduits for geothermal upflow, supplying hot water to the overlying units and to the mine area. Faults and fractures in the area exhibit hydraulic conductivities ranging from 0.66 to 6.95 m/d, with most structures showing values near 1.1 m/d.

Groundwater flow is mainly controlled by topography and structure. Regional gradients drive flow from upland recharge zones toward the Río Ostua and other valleys, where baseflow sustains perennial streamflow, the potentiometric surface of local groundwater is shown in Figure 13-18. Recharge is predominantly seasonal and is concentrated where the alluvial aquifer is thicker and more permeable. Conceptual and numerical interpretations indicate upward geothermal flow along major faults beneath the project area, and mixing between deep hot fluids and cooler shallow groundwater generates thermal and chemical anomalies that are directly relevant for dewatering design and operational controls.

Figure 13-18: Map of Calibrated Potentiometric Surface, Showing Local Rivers, Creeks and Faults

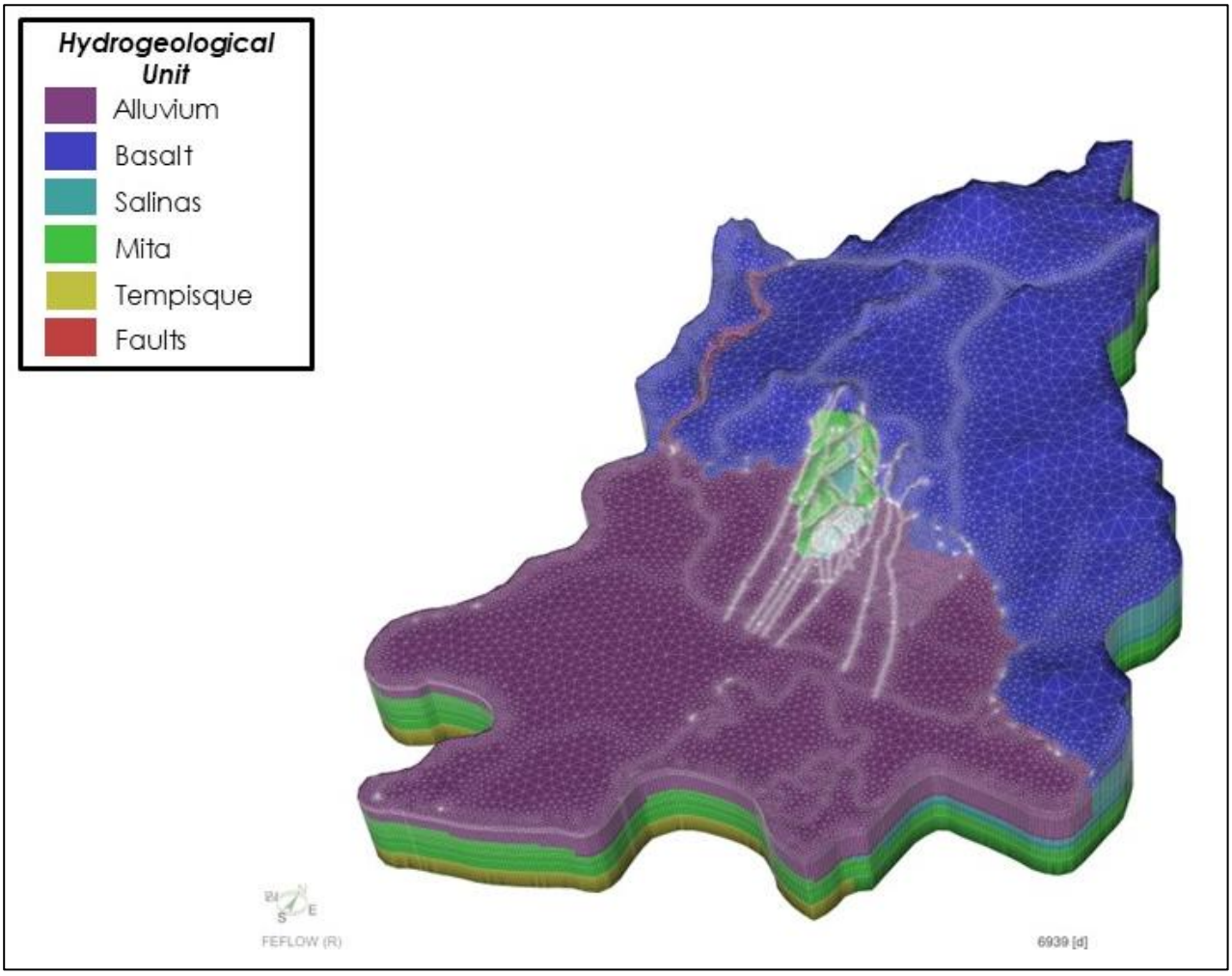


Source: Stantec, 2025.

13.2.2 Numerical Groundwater Model

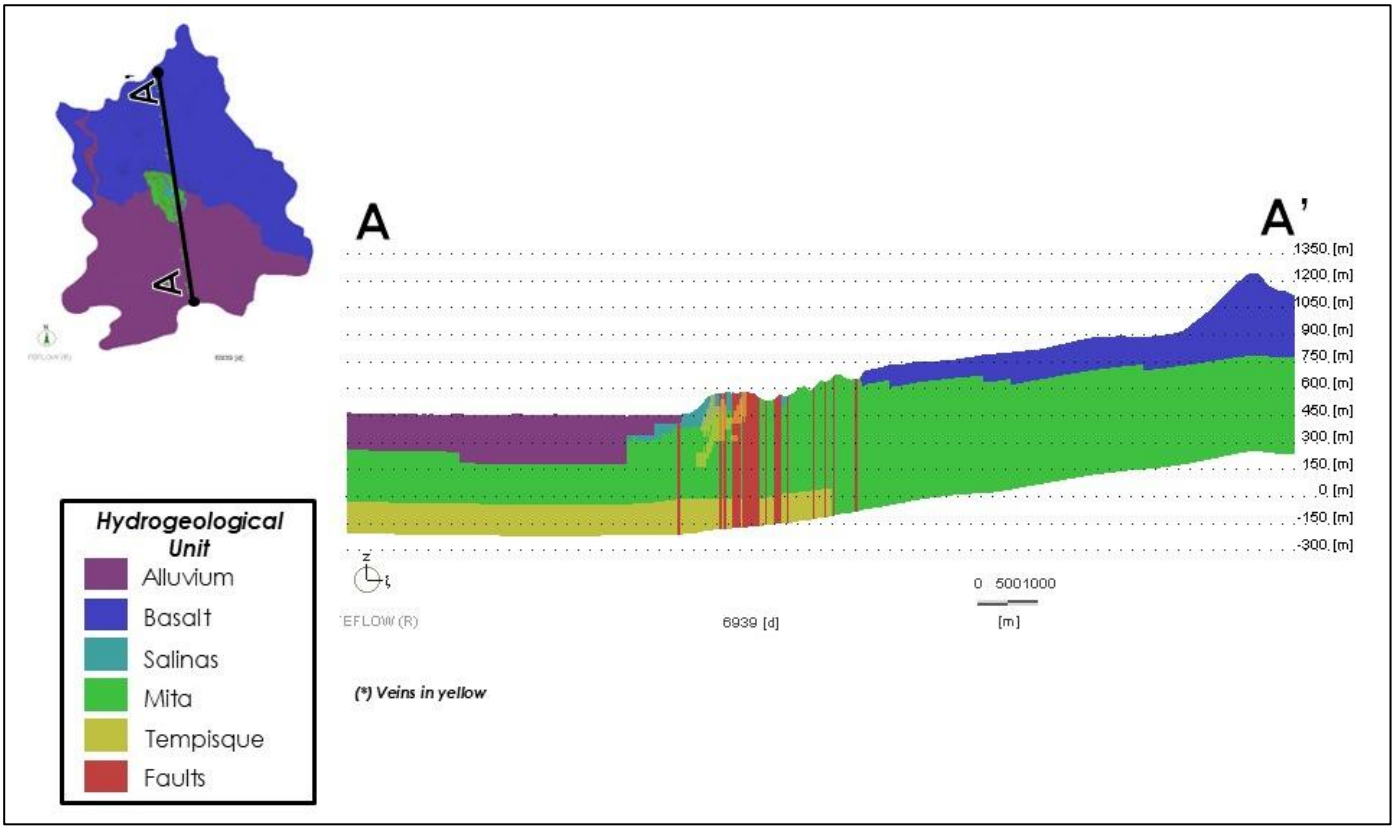
To support mine design and water-management planning, Stantec (2025) developed an updated FEFLOW groundwater-flow model that refines and expands upon the original Stantec (2018) model. The calibrated model domain covers approximately 174 km² and includes 25 layers representing the five principal hydrostratigraphic units, allowing the model to capture the influence of vein systems and major fault zones on groundwater flow. These hydrostratigraphic units and fault structures were explicitly incorporated into the model grid (Figure 13-19). A representative north–south cross-section (Section A) illustrates the vertical arrangement and relative thickness of the units (Figure 13-20).

Figure 13-19: Spatial Distribution of the Five Hydrostratigraphic Units and Major Faults Represented in the Numerical Model Grid (Stantec)



Source: Stantec, 2025.

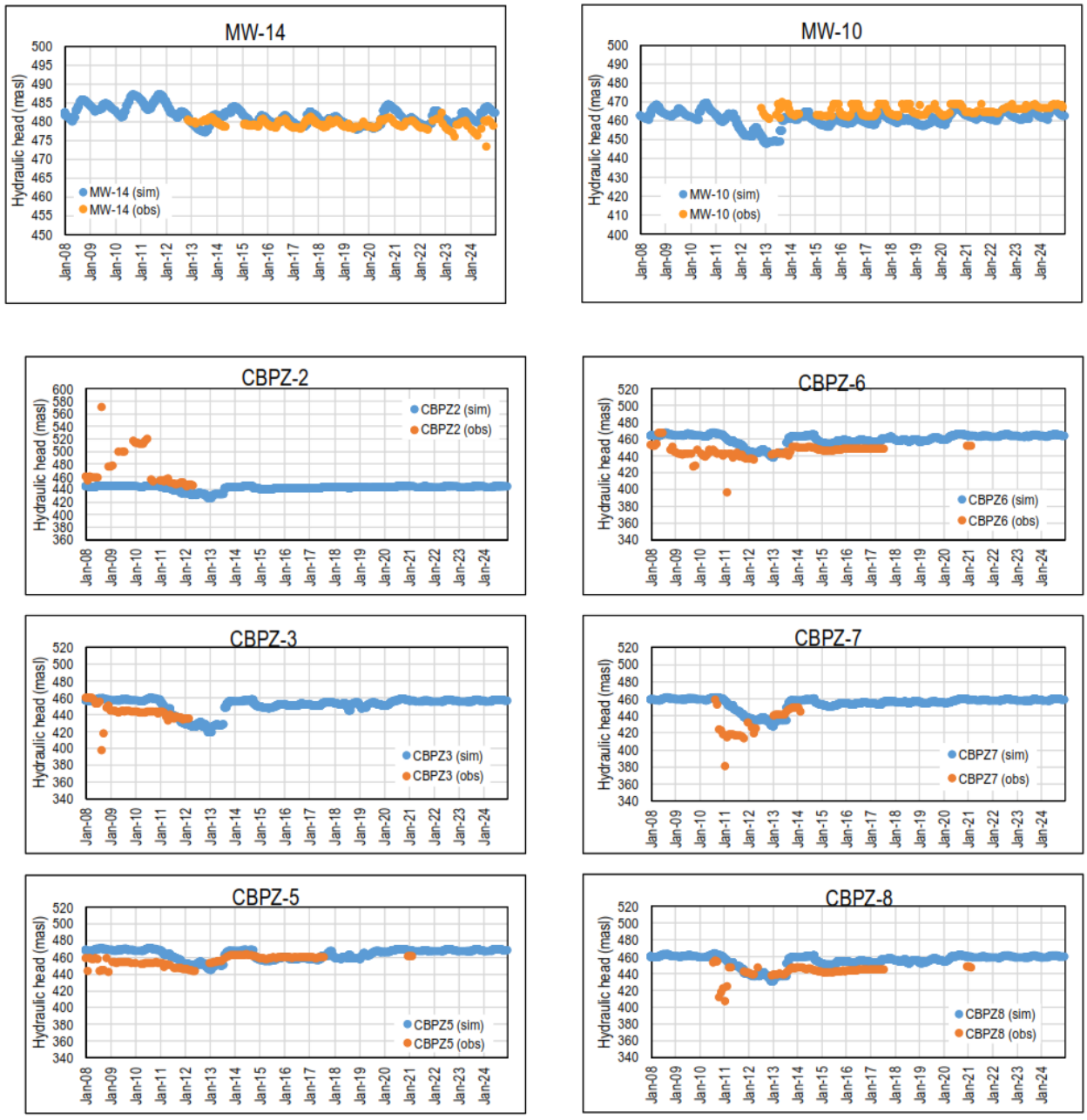
Figure 13-20: Representative cross-section (Section A, approximately north–south) showing the vertical succession and thickness of the hydrostratigraphic units (Stantec)



Source: Stantec, 2025.

A steady-state simulation was first developed to represent pre-mining conditions and was calibrated against 45 observation points by comparing simulated groundwater levels with measured field groundwater levels. Calibration achieved a Normalized Root Mean Square error of about 6.1%, which is considered acceptable for regional-scale modeling. A transient model for the period from 2008 to 2024 was then constructed using the calibrated steady-state model as a baseline. Historical pumping from tunnels and dewatering wells was incorporated, and model performance was validated against observed hydrographs, tunnel pumping flow rates and Río Ostua streamflow data. The calibrated model reproduces observed seasonal groundwater-level fluctuations in observation wells and long-term trends, as shown in Figure 13-21. The model matches field pumping data and confirms the Río Ostua as a gaining stream sustained by groundwater discharge throughout the year.

Figure 13-21: Observed vs Simulated Transient Hydrographs



Source: Stantec, 2025.

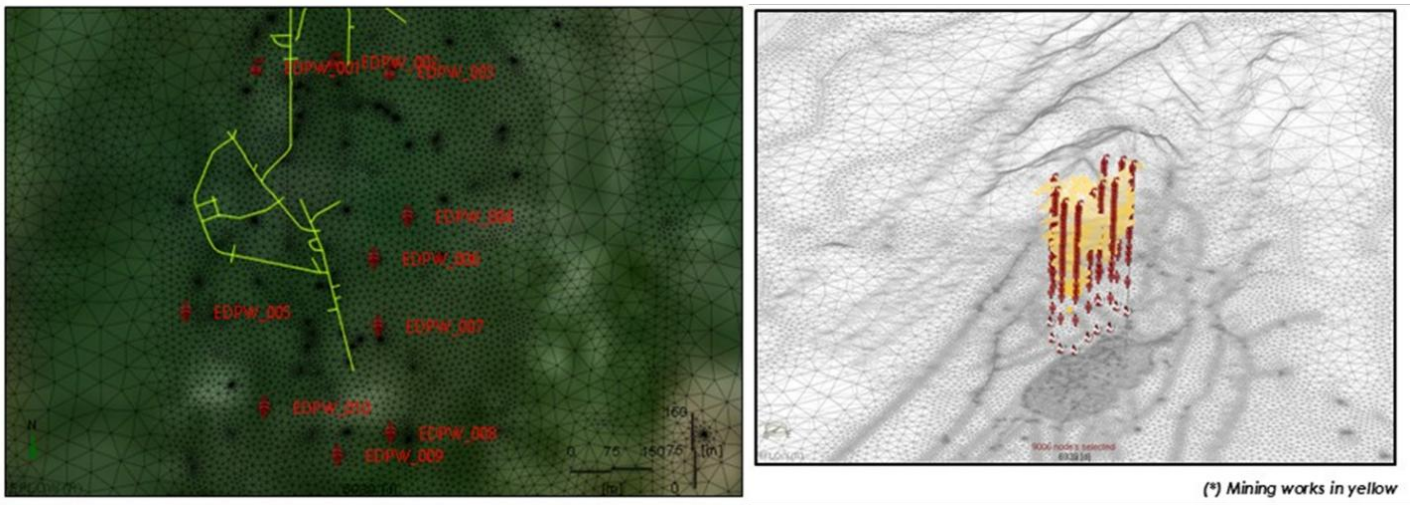
13.2.3 Dewatering System Objectives

The primary objective of the dewatering system is to reduce and maintain manageable groundwater inflow into mine workings throughout the life of the mine, such that planned underground operations can be conducted safely and economically. Consistent with the water-handling approach defined in Section 15.2 (Water Balance and Management), water extracted from dewatering activities will be conveyed to the site's Wastewater Treatment Plant (WTP) for treatment prior to any use. Treated water will then be reused within mine operations or allocated to other permitted uses in compliance with environmental regulations.

13.2.4 Projected Dewatering Requirements and Hydrologic Impacts

The numerical groundwater model was used to simulate future dewatering requirements under the planned mine development sequence. The spatial disposition of the planned dewatering wells within the model domain is shown in Figure 13-22, providing the framework for the simulated pumping schedule described below. The system will initiate in 2026 with two wells operating at approximately 1,220 gpm. By 2027–2028, four to five wells will be active, with combined rates increasing to roughly 3,000–3,800 gpm. Demand continues to rise through 2029, when eight wells are needed to achieve about 4,860 gpm. Full system build-out occurs in January 2031, when all ten wells become operational, yielding a sustained pumping capacity around 6,080 gpm (or up to 7,600 gpm when incorporating the 25% contingency applied by Stantec). This progression, shown in Table 13-6, highlights the increasing hydraulic pressures associated with deeper mine development and provides the basis for evaluating discharge capacity and supplemental disposal requirements.

Figure 13-22: Spatial Distribution of the Planned Dewatering Wells (red) in the Numerical Model, shown in 2D (left) and 3D (Right). Mining Works are Highlighted in Yellow.



Source: Stantec, 2025.

Model results indicate that the current permitted discharge capacity of approximately 5,250 gpm (about 330 L/s) will become insufficient as pumping requirements increase with mine development. Total dewatering demand is projected to reach about 6,080 gpm (roughly 384 L/s) by 2029, exceeding the current discharge limit. Under the maximum development scenario, up to ten dewatering wells are expected to be operating simultaneously by January 2031, with a combined pumping rate on the order of 7,600 gpm (approximately 480 L/s). These rates reflect the combined influence of highly permeable alluvial deposits, structurally controlled flow in the Salinas Tuff and Mita Group, and localized geothermal inflows from the Tempisque Volcanic Complex.

To accommodate projected excess flows and maintain operational flexibility, the modeling supports the implementation of two dedicated reinjection wells, each designed to handle about 1,000 gpm (approximately 63 L/s), starting in the fifth year of operation. These reinjection wells would operate in parallel with existing surface discharge infrastructure, providing additional capacity to manage peak dewatering rates beyond 2031 and reducing dependence on a single discharge pathway. Expansion of discharge permits and planning for additional disposal options are required to ensure that increased pumping rates can be managed without constraining mine production.

Table 13-6: Projected Dewatering Rate and Well Schedule

Year	Simulated Combined Flow Rates		Combined Flow Rates With 25% Contingency		Number Active Wells	Comments
	L/s	gpm	L/s	gpm		
2025	0	0	0	0	0	No active dewatering
2026	-77	-1222	-96	-1528	2	Two wells active starting on 06/01/26
2027	-153	-2432	-191	-3040	4	Four wells active on 01/01/27
2028	-192	-3048	-240	-3810	5	Five wells active on 01/01/27
2029	-307	-4864	-384	-6080	8	Eight wells active on 01/01/28
2030	-307	-4864	-384	-6080	8	
2031	-384	-6080	-480	-7600	10	Ten wells active on 01/01/31
2032	-385	-6097	-481	-7621	10	All ten wells active
2033	-384	-6080	-480	-7600	10	
2034	-384	-6080	-480	-7600	10	
2035	-384	-6080	-480	-7600	10	
2036	-385	-6097	-481	-7621	10	
2037	-384	-6080	-480	-7600	10	
2038	-384	-6080	-480	-7600	10	
2039	-384	-6080	-480	-7600	10	
2040	-385	-6097	-481	-7621	10	
2041	-384	-6080	-480	-7600	10	
2042	-384	-6080	-480	-7600	10	
2043	-384	-6080	-480	-7600	10	

Source: Stantec, 2025.

Simulated groundwater budget results show that sustained dewatering will have measurable, though generally moderate, impacts on surface-water flows at the catchment scale. For the Río Ostua, which functions as the primary gaining stream in the area, the model indicates that total annual surface-water flow decreases from approximately 1,564 L/s in 2025, prior to significant mine pumping, to about 1,376 L/s by 2043, corresponding to an overall reduction of roughly 12%. For the smaller Río Tacunshapa, model results indicate a reduction in simulated streamflow from about 6.5 L/s in 2025 to approximately 3.5 L/s by 2043, a decrease of around 46%. This larger relative change reflects the sensitivity of small catchments to regional water-table lowering and reduced hydraulic gradients feeding the stream.

Although these reductions are relevant in hydrogeologic terms, the current discharge permit capacity exceeds the magnitude of modeled baseflow depletion. In way that the treated effluent from the mine can generate a net surplus of water requiring disposal. As a result, the volumes of treated water discharged to receiving streams are expected to surpass the simulated reduction in natural baseflow, helping maintain downstream flows within regulatory and environmental criteria.

The geothermal nature of the system introduces specific design and operational risks. The presence of groundwater at temperatures up to 190 °C creates potential for steam flashing if pressures are not adequately controlled in wells, pipelines and underground workings. The dewatering system design therefore assumes the use of electrically submersible pumps (ESPs) with appropriate pressure control, backpressure management and materials selection suitable for high-temperature service.

13.2.5 Recommendations

Based on the current understanding of the hydrogeologic system and model results, the following actions are recommended to support mine planning and water management:

- **Conduct Sensitivity and Uncertainty Analyses:** Apply formal sensitivity and uncertainty methods to the groundwater numerical model to identify the key parameters and most sensitive zones controlling model behavior, and to quantify the plausible range of groundwater inflows. This will improve confidence in predictions used for mine planning and water management.
- **Refine Study Area Characterization:** Based on the sensitivity and uncertainty outcomes, define priority target areas for additional field investigation. Where feasible, existing wells and piezometers should be leveraged for enhanced monitoring and testing, complemented by focused in situ investigations (for example, geophysical surveys, packer testing, and long term well pumping to aquifer test). These activities should target major fault zones, high permeability or preferential flow pathways, and areas with geothermal upflow characteristics.**Develop an updated structural geological model and revise the hydrogeologic model accordingly:** Given that fault-controlled upward flows represent one of the largest sources of uncertainty, and potentially one of the greatest sensitivities, in the hydrothermal system, it is recommended to develop a dedicated structural geological model and integrate it into an updated hydrogeologic framework. This refinement will improve the representation of fault geometry, connectivity and transmissivity, particularly in zones where ascending geothermal flux is believed to occur, thereby reducing uncertainty and improving predictive reliability.

- **Integrate thermal–hydraulic analysis:** Incorporate heat-transport modeling to evaluate coupled thermal–hydraulic processes, including changes in water viscosity, heat exchange between groundwater and mine infrastructure, and temperature-dependent pump performance under geothermal conditions.
- **Optimize dewatering system design and operation:** Use additional scenario testing to assess alternative excavation sequences, dewatering well activation schedules and ramp-up strategies. Refine well spacing, consider deepening selected wells (particularly in the southern sector) and stage the activation of new wells to improve hydraulic control and minimize residual inflows as mining progresses.
- **Increase and diversify disposal capacity:** Expand existing discharge permits and evaluate potential additional surface discharge locations and operational reuse options to provide sufficient capacity for projected flows beyond 2031. Deep reinjection wells may be considered as a supplemental alternative; however, reinjection effectively acts as artificial recharge and introduces additional complexity. Planning must explicitly address (i) the risk of degrading groundwater quality in the receiving aquifer, (ii) the possibility that reinjected water may re-enter the dewatering capture zone and increase long-term pumping requirements, and (iii) potential pressure buildup) the possibility that reinjected water may re-enter the dewatering capture zone and increase long-term pumping requirements, and (iii) potential pressure buildup in the system that could require reassessment of pump sizing and operational setpoints. Any reinjection scheme should therefore be supported by dedicated hydrogeologic investigation, predictive modeling and regulatory review.
- **Manage environmental and social water impacts:** Recognize that sustained aquifer drawdown and streamflow depletion, particularly in the Río Ostua and smaller tributaries such as the Río Tacunshapa, may affect surface-water availability for downstream users and ecosystems. The surface-water and groundwater monitoring network should be maintained and, where necessary, expanded to track changes in water levels and flows at locations relevant to local communities, ecology and water-use points. Monitoring data should be routinely compared to pre-mining baseline conditions to identify material impacts on water supply, aquatic habitat and riparian vegetation. Where significant reductions in flow are confirmed, the operator should evaluate and implement appropriate mitigation or compensation measures, which may include augmentation or replacement of affected flows, provision of alternative water supplies to nearby communities, and adaptive adjustment of dewatering rates or infrastructure, in coordination with regulators and stakeholders.
- **Validate high-temperature equipment performance:** Carry out field testing of ESP systems, backpressure control equipment and well/liner configurations under representative thermal and hydraulic conditions to confirm reliable operation and to reduce the risk of steam flashing or thermal–mechanical failures. In addition, pilot wells should be constructed to intercept the principal modeled fault zones and evaluate the technical feasibility of pumping under expected geothermal conditions.
- **Develop and maintain a contingency plan:** Prepare a comprehensive contingency plan defining operational responses for excess inflows, temporary treatment plant outages, reinjection well underperformance and failures of high-temperature components. This plan should be supported by ongoing groundwater monitoring and periodic updates of the numerical model, ensuring that dewatering design, mining methods and water-management strategies remain aligned with observed field conditions, environmental commitments and regulatory requirements throughout the LOM.

- **Assess underground-based dewatering:** Evaluate the feasibility of initiating staged dewatering from advancing underground workings to provide a flexible, progressive approach to hydraulic control. The assessment should address expected inflows, high-temperature conditions, safety requirements and supporting infrastructure to determine whether this strategy can effectively complement or partially replace surface wells.

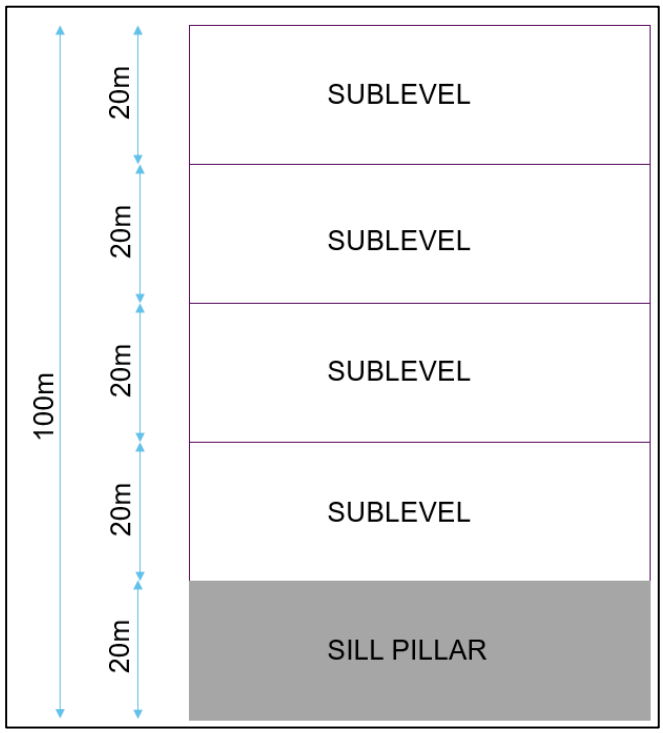
13.3 Mining Methods

The Mineral Resources at the Era Dorada deposit will be extracted using a combination of Long hole stoping (LH), Cut-and-fill (MCF) and minor Room-and-pillar, utilizing paste fill and cemented rockfill (CRF). Long hole stoping is the main method, expected to account for approximately 98.5% of total metal production, while cut-and-fill will contribute with 1.2% and room and pillar with only 0.1%.

The mining method selection was primarily guided by geotechnical rock quality, vein geometry, and orebody continuity. Long hole stoping was applied as the preferred mining method due to its safer working conditions, higher productivity and lower unit mining costs relative to MCF. Where geotechnical or geometric conditions are required, mechanized cut-and-fill (MCF) was otherwise applied. The proposed mine plan was designed to achieve a target production rate of 1,600 t/d, for a total mine life estimated as 18 years.

In order to establish the mining geometry, the mine was first divided into panels, each composed by four levels of 20m vertically, plus a sill pillar, also of 20m vertically that will be reclaimed at the end of the mine life, totaling 100m for each panel (Figure 13-23). For a given mine area in the South and North zones, each panel can be operated independently to allow for increased operational flexibility and secure production rates. Sill pillars will be established between the panels to ensure safe working conditions and support high recovery rates. Multiple stopes will be mined concurrently, enabling the Project to achieve the target production rate before the ramp reaches its full depth. For both the Long hole and Cut-and-fill areas, each mining block will be extracted using an overhand (bottom-up) sequence.

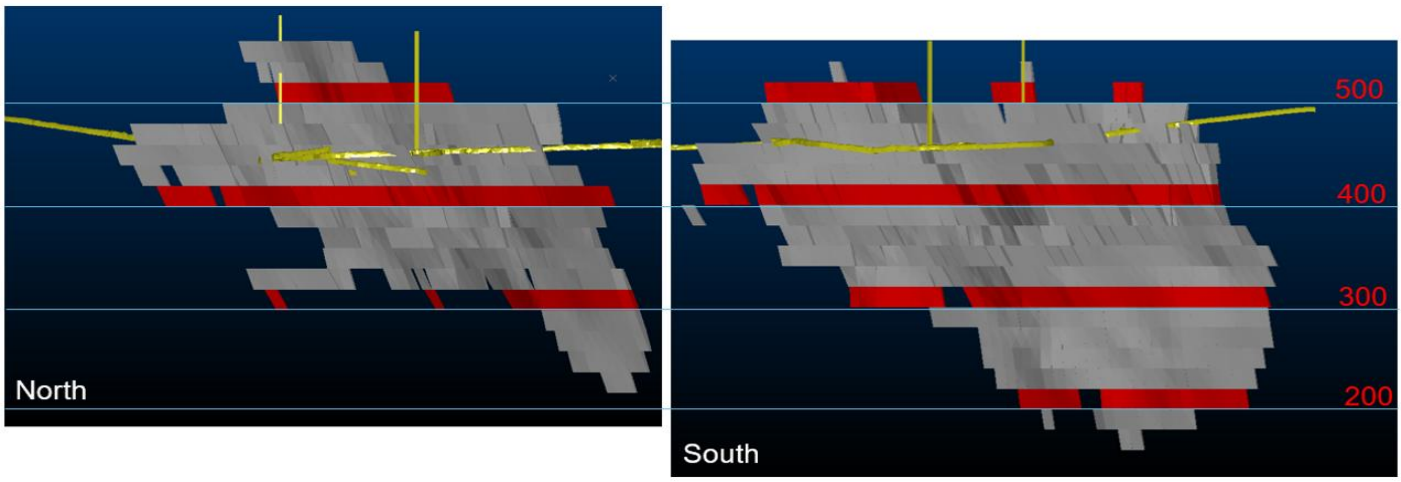
Figure 13-23: Layout of Typical Sizes of Panel



Source: Snowden Optiro, 2025.

Accordingly, temporary sill pillars were established on Levels 200, 300, 400, and 500 as illustrated in Figure 13-24. These pillars will be extracted in the final years of the LoM.

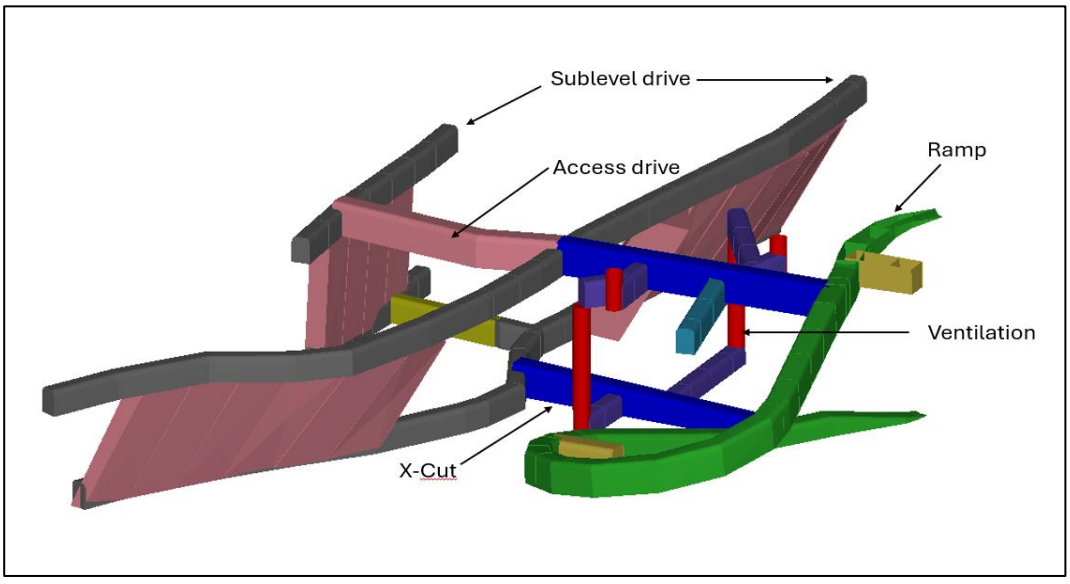
Figure 13-24: Sill Pillars



Source: Snowden Optiro, 2025.

A typical sublevel layout is illustrated in Figure 13-25.

Figure 13-25: Layout of Typical Sublevel



Source: Snowden Optiro, 2025.

13.3.1 Long Hole Mining

The preferred mining method for Era Dorada is sublevel Long Hole (LH) stoping, owing to its safer working conditions, higher productivity and lower operating costs. This method is well suited to continuous, steeply dipping veins hosted in high and to fair geotechnical quality rock masses. LH stoping will be applied wherever geotechnical and geometric conditions allow for efficient stope design and operation.

Two LH stoping configurations will be used: longitudinal and transverse. Transverse long hole stoping will be employed in the thicker and more continuous zones of the deposit (generally exceeding 20 m) either from individual lenses or when adjacent lenses coalesce into a single stope. Stopes are designed to be up to 10 m in strike length and 20 m in height, with thicknesses ranging from 2.6 m to 51.1 m (including wall dilution). The minimum stope width is 1.0 m prior to applying dilution allowances.

The stopes will be mined using an overhand sequence, retreating towards the level access. Each stope is accessed via 4 m × 4 m crosscuts developed above and below the stope (the ore drives).

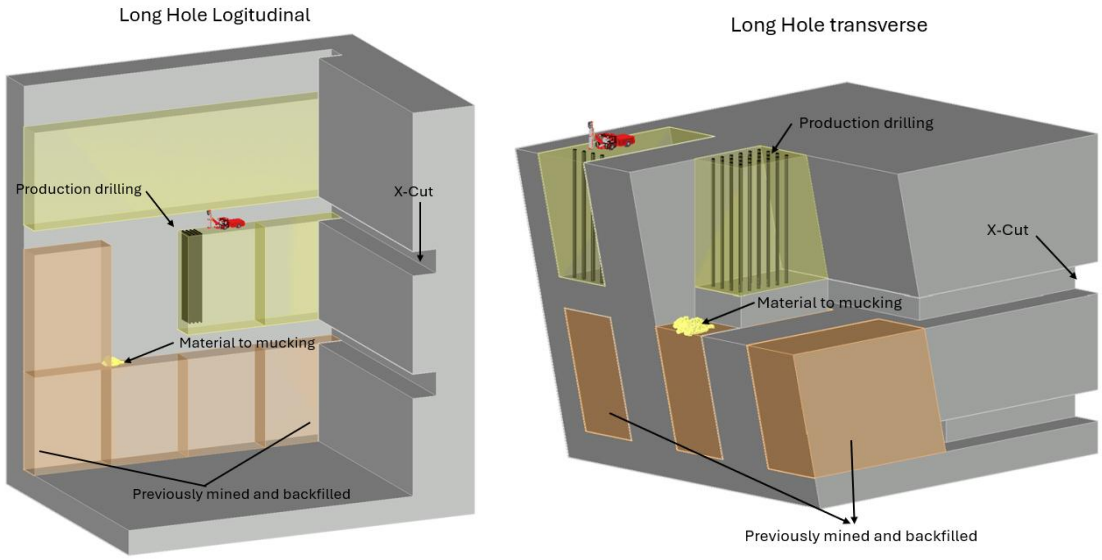
All long hole stopes will be backfilled with cemented paste fill or cemented rockfill to provide structural confinement where adjacent stopes are present or where the vein geometry requires support for subsequent extraction.

Arsenic-bearing process sludge generated by the plant will be managed through underground disposal by incorporation into the pastefill stream. The sludge will be blended with tailings, binder and additives at the pastefill plant and placed into mined-out stopes as cemented paste backfill, in accordance with the backfill design criteria.

In transverse stopes, a primary/secondary mining schedule will be implemented, with primary stopes backfilled to enable safe extraction of adjacent secondary stopes without the need for rib pillars. In longitudinal stopes, structural backfill will be placed in all mined stopes to ensure stability during extraction. Backfilling will be carried out from the upper sill using the paste fill lines or LHDs, depending on access and stope geometry.

Figure 13-26 shows a typical mining schedule for LH at Era Dorada.

Figure 13-26: Long Hole Mining



Source: Snowden Optiro, 2025.

13.3.2 Mechanized Cut-and-fill

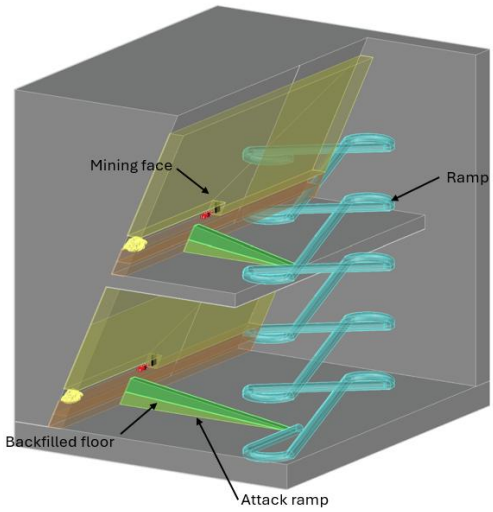
The overhand mechanized cut-and-fill (MCF) mining method is planned for areas with less favorable rock quality and/or where the mineralization geometry is not suitable for long hole (LH) stoping. Cut-and-fill is a highly selective underground mining method, well suited to narrow, high-grade veins with steep to shallow dips, usually applied under weak rock mass conditions.

Mining starts at the base of the ore block and advances upwards. Each stope lift is initially supported temporarily with rock bolts, followed by placement of cemented backfill to form a suitable working floor for the subsequent lifts. The backfill is designed primarily to provide floor support rather than full structural confinement.

The access between successive MCF lifts is provided via attack ramps developed from the main level access, with gradients ranging between −15% and +15% (the pivot ramps).

A typical stope development scheme is presented in Figure 13-27.

Figure 13-27: Mechanized cut-and-Fill



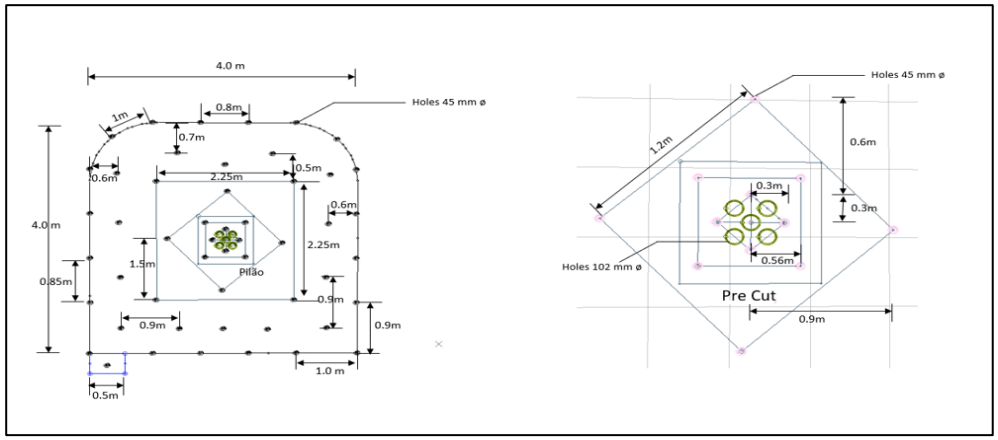
Source: Snowden Optiro, 2025.

13.4 Drill and Blast Patterns

Detailed drill and blast patterns were developed for development and stoping to support the productivity assessment and, later, fleet sizing and the requirements of supplies, equipment and personnel for the definition of the mining costs.

Figure 13-28 shows the drilling and blasting patterns for the 4x4 m^2 sections as an example.

Figure 13-28: Drilling Pattern for Development – 4x4 m^2 Section

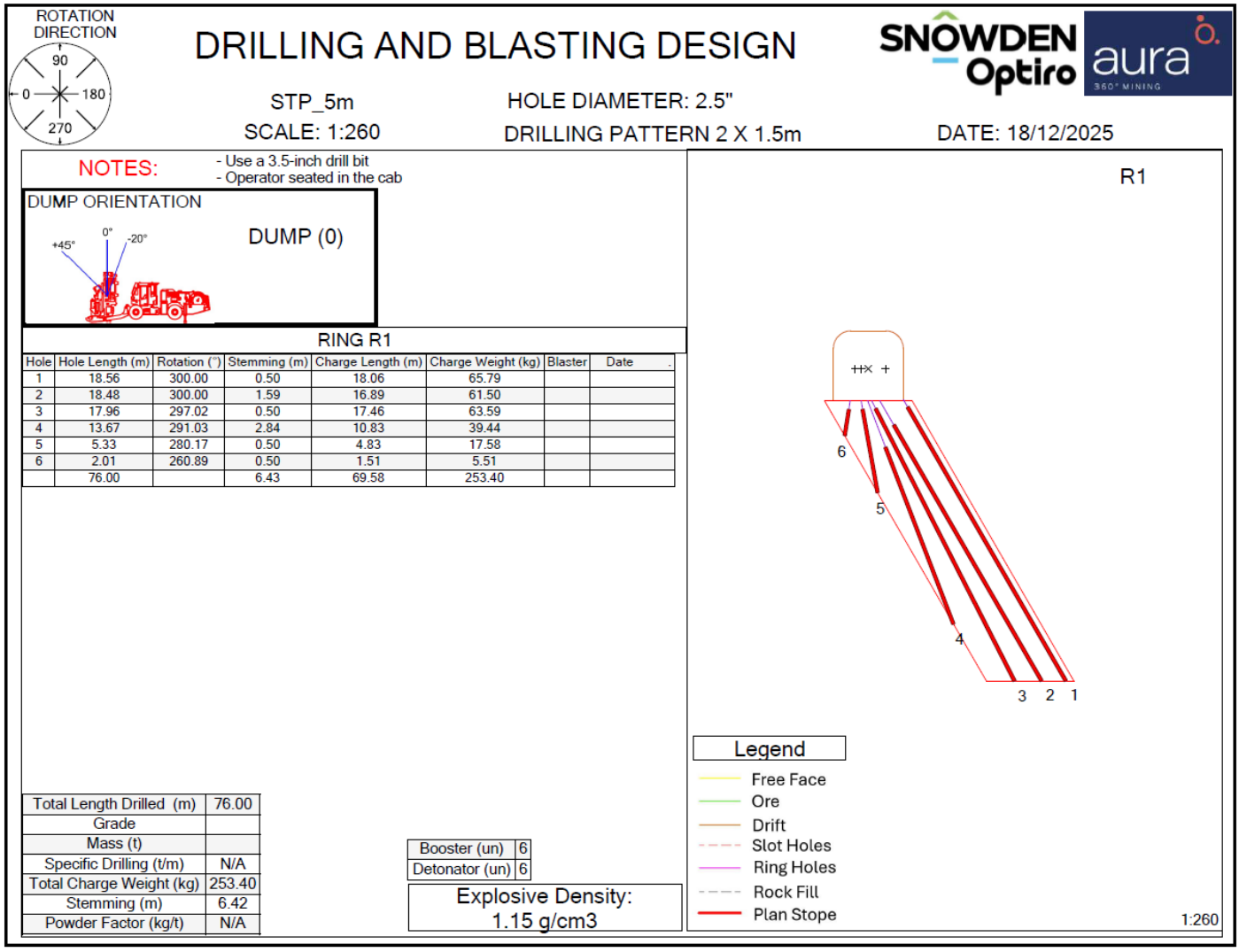


Source: Snowden Optiro, 2025.

All development drifts will be charged with cartridge emulsion while stoping will be charged with a combination of 50% bulk ANFO and 50% cartridge explosives (for wet or moist zones), priming will be with boosters for both development and stoping and primary initiation with non-electric fuses.

Detailed drill and blasting patterns were also developed for stoping for typical thicknesses as of 2 m, 3.5 m, 5 m, 10 m and 20 m (horizontally projected). Figure 13-29 shows the drilling and blasting patterns for stoping 5m wide stopes, for example.

Figure 13-29: Drill and Blasting Pattern for Stoping – 5 m (Horizontal) Section



Source: Snowden Optiro, 2025.

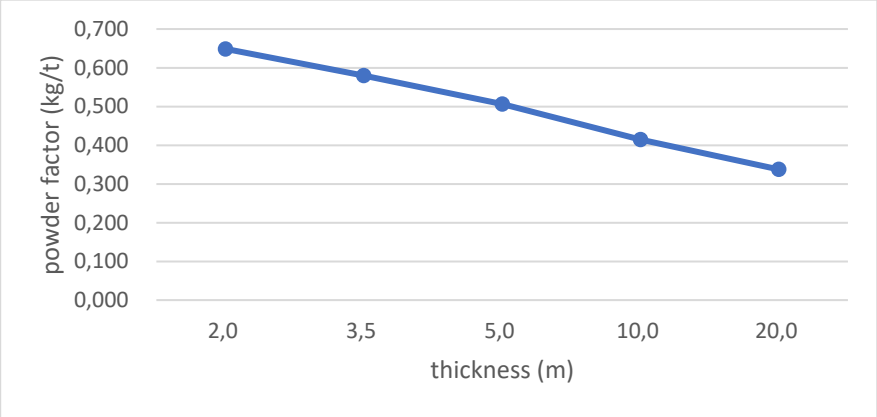
The drill and blast patterns were then combined to thickness distribution data for the stopes and the drill and blast parameters were defined as per Table 13-7.

Table 13-7: Drill and Blast Parameters for Stoping

Parameter	Unit	Thickness				
Thickness	**m**	**2.0**	**3.5**	**5.0**	**10.0**	**20.0**
Hole diameter	mm	64	64	64	76	76
Thickness	m	2	3.5	5	10	20
Stope area	m^2	30	53	75	150	300
Burden	m	2.50	2.50	2.50	3.00	3.00
Spacing	m	1.50	1.50	1.50	3.00	2.50
Volume	m^3/section	75	131	188	450	900
Density	t/m^3	2.54	2.54	2.54	2.54	2.54
Tonnage	t/section	190	333	476	1.142	2.283
Drilling	m drilled	37	58	76	114	173
Specific drilling	**t/mdrill**	**5.19**	**5.76**	**6.26**	**9.99**	**13.22**
% thickness (from histogram)		**8%**	**52%**	**19%**	**15%**	**7%**
average t/mdrill		Total	6.97			
		2 1/2"	5.82	78%		
		3"	11.01	22%		
Blasting						
Emulsion/ANFO density	(g/cm^3)	1.08	1.08	1.08	1.08	1.08
	mcharged/section	36	56	70	97	158
Charge per section	kg/section	123	193	241	473	770
Powder factor	kg/m^3	1.646	1.470	1.283	1.051	0.856
Powder factor	**kg/t**	**0.649**	**0.579**	**0.506**	**0.415**	**0.337**

Figure 13-30 shows the powder factors for stoping according to the thicknesses of the stopes (horizontal projection).

Figure 13-30: Powder Factor - Stoping



Source: Snowden Optiro, 2025.

13.5 Mine Mobile Equipment Fleet Sizing and Personnel Requirements

Aura will contract all mine development and will owner operate stoping.

The fleet-sizing requirements for both the contracted development and owner operation for stoping were defined from the productivities of the various types of equipment considering the parameters shown in Table 13-8.

Table 13-8: Mobile Equipment Productivity

Fleet/Parameter	Availability (%)	Use of Availability (%)	Productivity
Drilling fleet (Jumbos and Fandrills)	80%	60%	2 boom jumbos: 13,400 m drilled/month (150m/jumbo.month 1 boom jumbo: 8,000-10,000 m drilled/month (150m/jumbo.month) Fandrills: 5,300 m drilled/month for 64mm holes 4,000 m drilled/month for 76mm holes
Load and Transport fleet (LHDs and Trucks)	75%	70%	LHD 10t: 86t/h LHD 7t: 59t/h Truck 30t: according to tramming distance, average 39t/h
Aux Equipment	70%	60%	

The fleet productivities were then matched to the scheduling requirements to establish the fleet sizing requirements for the contracted development and for the owner fleet as shown in Table 13-9 and Table 13-10.

Table 13-9: Development Mobile Equipment Fleet - Contractor

Equipment/Year	Yr-1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17
2 boom jumbo	4	5	5	5	5	5	4	2	2	1	1	1	1	1	1	1	1	0
Production drill	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Cable bolter	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
LHD 10t	2	3	3	3	3	4	3	2	2	1	1	1	1	1	1	1	1	0
Truck 30 t/road trucks	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Explosives truck	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Shotcrete sprayer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shotcrete transporter/ mixer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Scaler	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Scissor lift	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Personnel carrier	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Fuel/lube truck	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Boom truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Grader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Backhoe	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Telehadler	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Utility vehicle	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Loader/general use	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Heavy equipment	30	32	32	32	32	33	31	28	28	26	26	26	26	26	26	26	26	24

Table 13-10: Stoping Mobile Equipment Fleet - Owner

Equipment / Year	Yr-1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17
1 boom jumbo	0	0	0	0	1	1	1	1	1	1	1	1	1	1	1	0	0	0
Production drill	0	0	1	1	1	1	1	1	1	2	2	2	2	2	2	2	2	2
LHD 10t	0	1	1	1	2	2	2	2	2	3	3	3	3	3	3	4	4	3
LHD 7t	0	0	0	0	1	1	1	1	1	1	1	1	1	1	1	0	0	0
Exploration drill/ 200m holes	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Exploration drill/ 100m holes	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Explosives truck	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shotcrete transporter/ mixer	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Scissor lift	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Personnel carrier	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel/lube truck	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Backhoe	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Telehadler	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Refuge Chambers 16 people	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Refuge Chambers 20 people	3	4	4	4	5	5	5	4	4	4	4	5	5	5	5	5	5	5
Heavy equipment	1	10	11	11	14	14	14	14	14	16	16	16	16	16	16	15	15	14

Aura then developed detailed personnel requirements, according to its standards as summarized in Table 13-11.

Table 13-11: Personnel Requirements – Mine - Owner

Personnel	Yr-1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17
Mine operation	67	79	79	79	79	79	79	79	79	79	79	79	79	79	79	79	79	79
Technical Services	30	49	49	49	49	49	49	49	49	49	49	49	49	49	49	49	49	49
Infrastructure	22	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50
Maintenance/ Surface	0	40	40	40	40	40	40	40	40	40	40	40	40	40	40	40	40	40
Maintenance/ Underground	31	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60
Aura Personnel Mine	150	278	278	278	278	278	278	278	278	278	278	278	278	278	278	278	278	278

The underground shift personnel will work in 8 hours shifts covered by 4 crews in a 6:1, 6:2, 6:3 (work:relief) rotation schedule while surface personnel will work in 8-hour journeys in a 6:1 scheme.

13.6 Mine Infrastructure

The ventilation, cooling, and underground pumping systems were designed at a Feasibility Study level. Capital and operating costs have been estimated with FS accuracy and integrated into the mine plan and economic analysis. The implementation of the mine infrastructure systems for ventilation, thermal conditions, and underground water management will enable industry standards for mine safety, production rates, or economic performance over the life of the mine.

13.6.1 Mine Ventilation

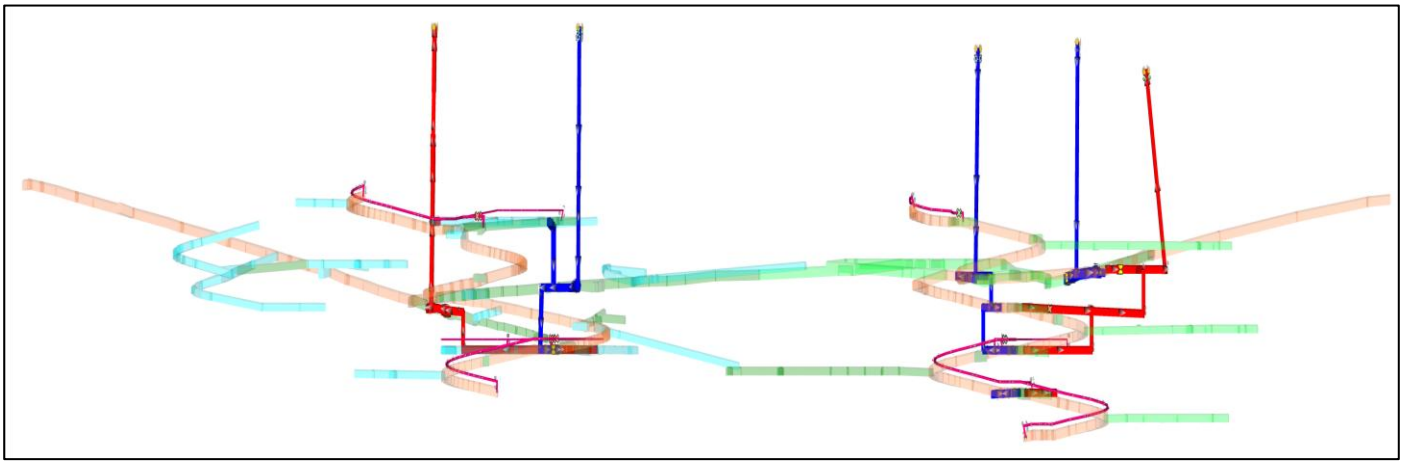
13.6.1.1 Design Criteria

Guatemala has not yet established safety regulations related to mine ventilation that specify criteria for calculating airflow requirements and air velocities among others. Hence, design criteria and standards from other jurisdictions, such as the USA and Brazil, were used, along with other criteria accepted in the mining industry.cThe airflow requirements were calculated based on diesel equipment and active/inactive levels (with the higher of the two usually being considered). Other requirements, such as those for the main pumping stations, were added to these values. Finally, 15% was added for leaks.cThe diesel requirement was calculated for the mine equipment fleet, considering MSHA-approved engines and 0.06 m^3/s/kW for those without approved engines. No other standards, such as CANMET,

were utilized in this instance. This is because the approved engines are based on diesel with sulfur content lower than the maximum permitted in Guatemala.

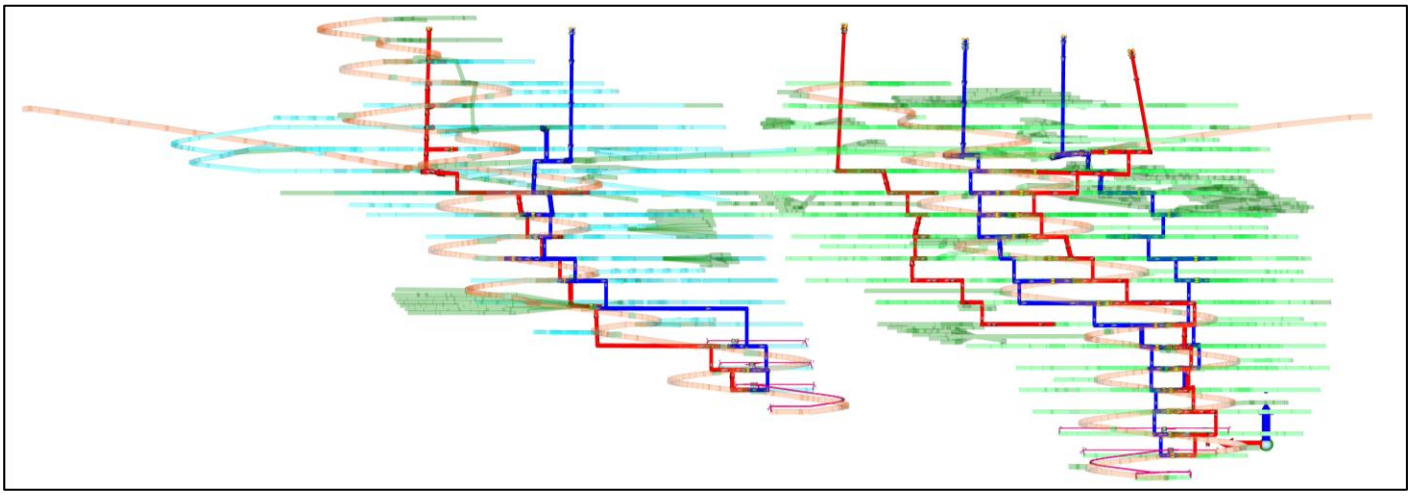
Also, the expected airflow and psychrometric parameters were calculated via Ventsim DESIGN software, simulations considering the short-, mid- and long-term planning were prepared (see Figure 13-31 and Figure 13-32) and used to determine the ventilation and cooling equipment main specifications such as pressure, airflow, shaft diameter, cooling capacity.

Figure 13-31: Short-term Ventsim Model



Source: Snowden Optiro, 2025.

Figure 13-32: Long-term Ventsim Model

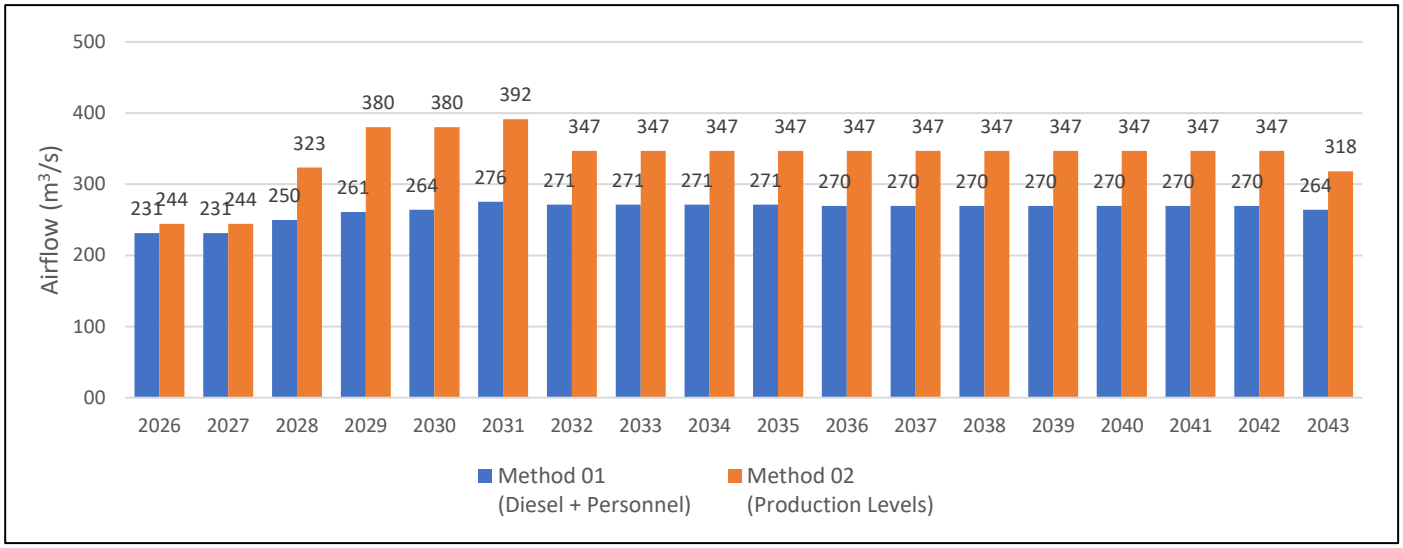


Source: Snowden Optiro, 2025.

13.6.1.2 Airflow Requirements

Airflow requirements are expected to peak at 392 m³/s in 2031, due to the number of active headings during that period (see Figure 13-33).

Figure 13-33: Airflow Requirements

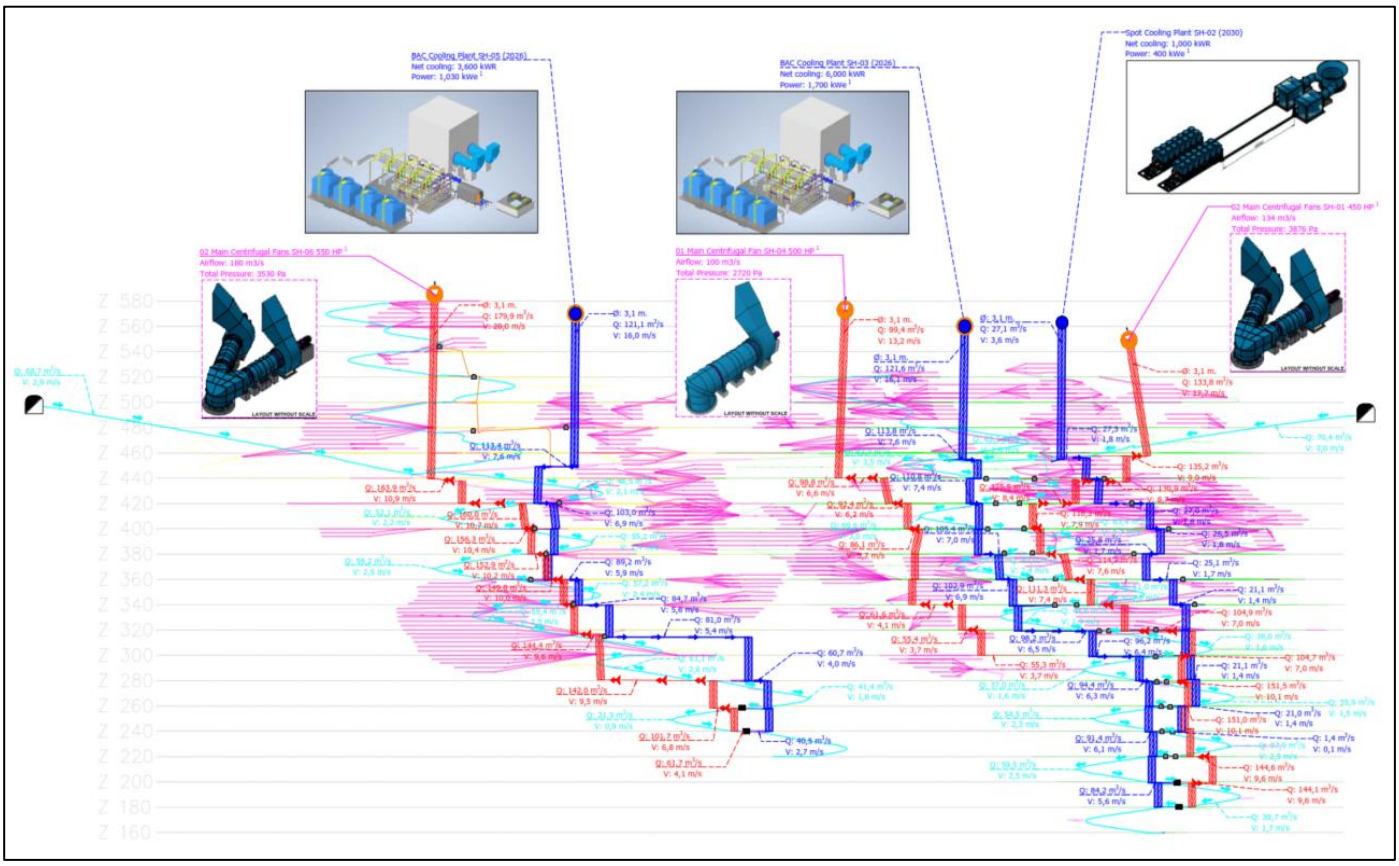


Source: Snowden Optiro, 2025.

13.6.1.3 Main Ventilation Circuit

The ventilation system for Era Dorada is a push-pull system, with all the main fans to be installed on surface (see Figure 13-34).

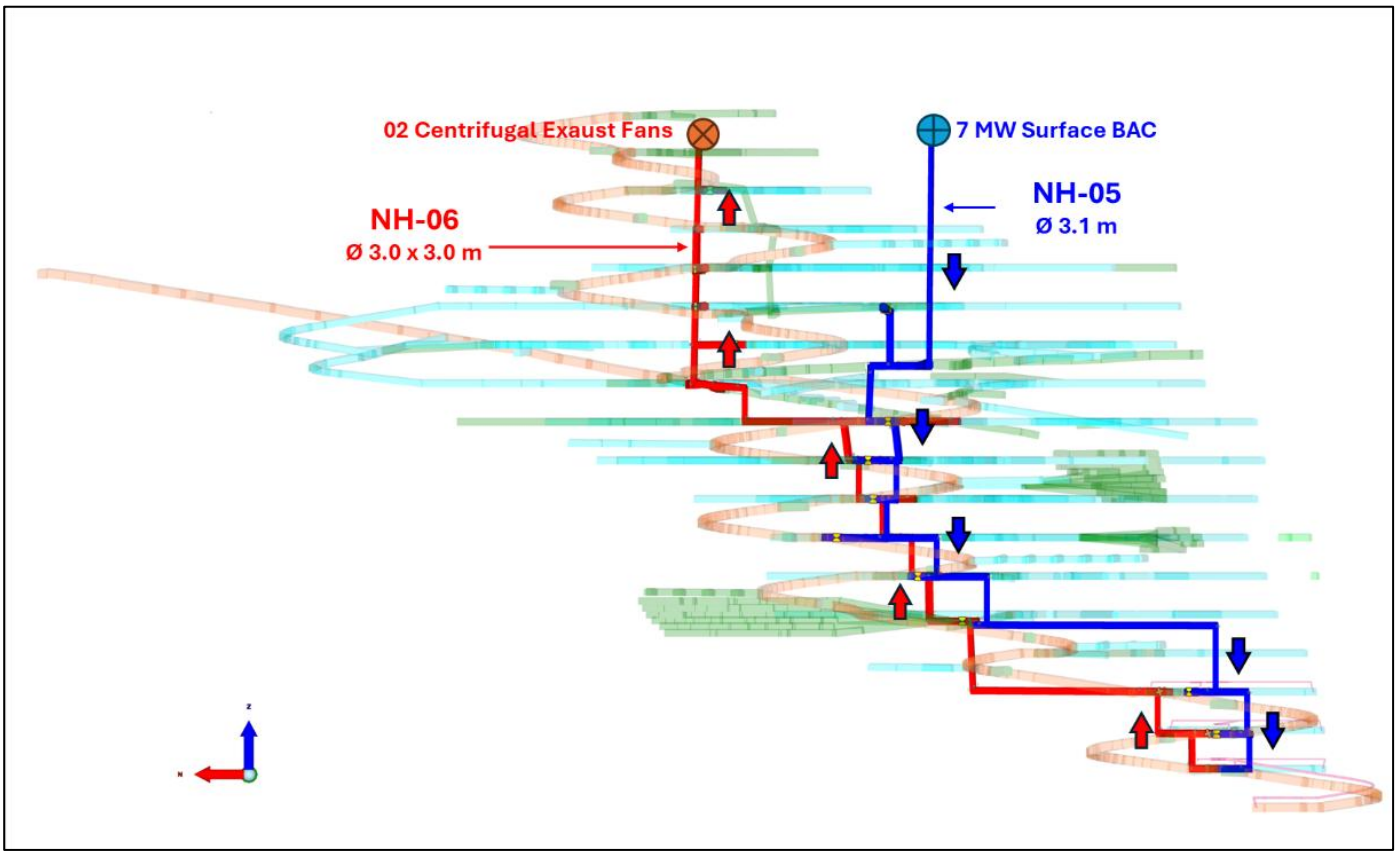
Figure 13-34: Ventilation System



Source: Snowden Optiro, 2025.

On the north side, there is a system with an intake shaft (NH05, built) that will be equipped with main fans in conjunction with a 3.6 MWR cooling plant and an exhaust shaft (NH06, built) to be equipped with two centrifugal fans (see Figure 13-35).

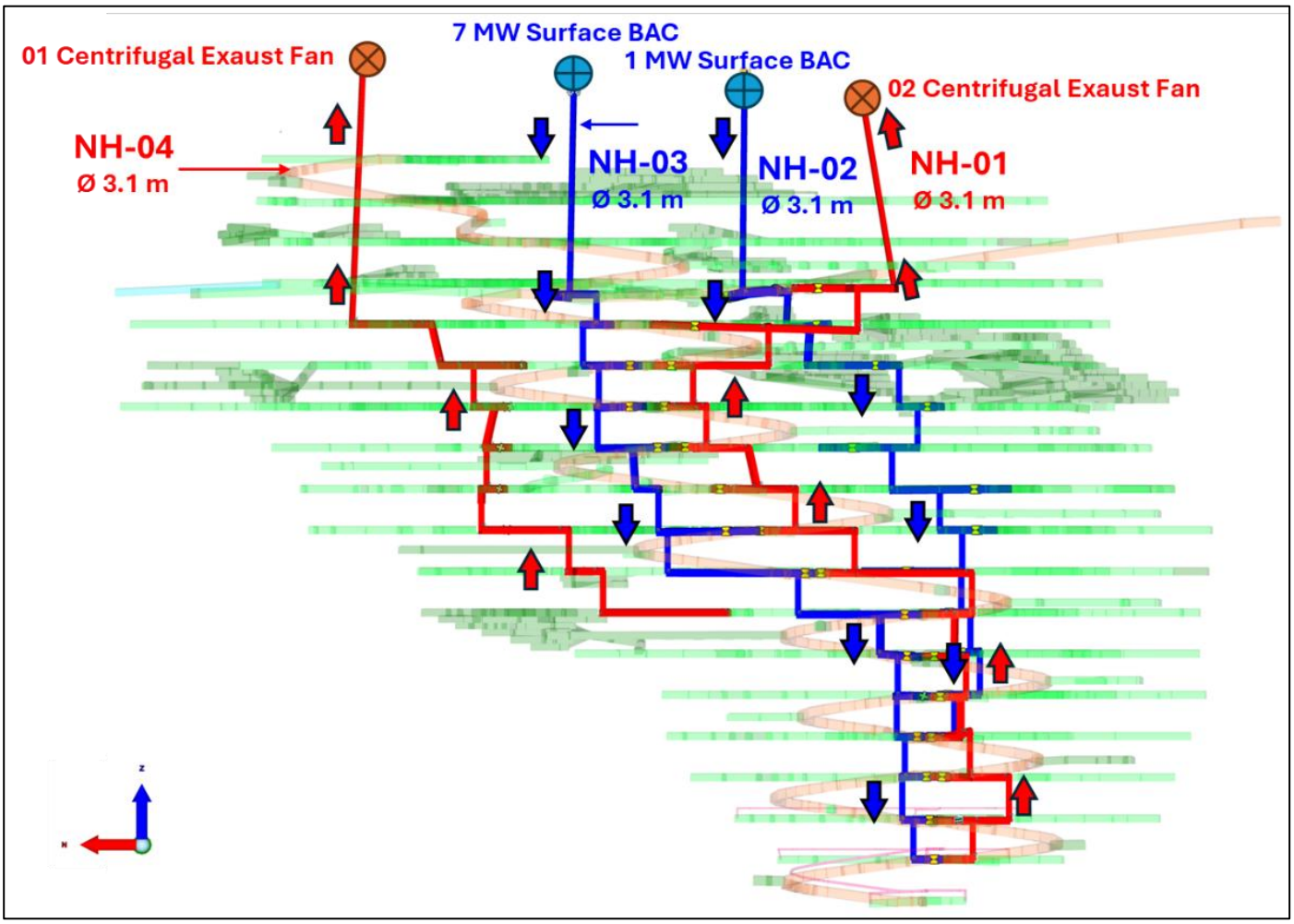
Figure 13-35: Ventilation System - North



Source: Snowden Optiro, 2025.

On the south side, the two existing raises (SH02 and SH03) will be used as intake raises. The SH02 model will be equipped with intake fans in conjunction with a 6 MWR plant, while the SH03 model will have a 1 MWR cooling plant (see Figure 13-36).

Figure 13-36: Ventilation system - South



Source: Snowden Optiro, 2025.

Due to its location and depth, the SH02 raise will be used to supply fresh air to the ramp development and production levels. Meanwhile, the SH03 raise will be used to supply fresh air to the areas located in the eastern part of the mine.

In 2026 and 2030, two raises (SH01 and SH04) with a diameter of 3100 mm will be developed for the exhaust circuit. Their location remains as licensed and stated in previous studies.

The SH01 exhaust raise will facilitate decline development and that of the central part of the production levels, while the SH04 will be used for the orebodies located further north of the south zone.

13.6.1.4 Main Fans

The duty point of the main fans was calculated conservatively, considering the circulating airflow at the deepest levels, as well as a safety margin of 5-10% in the system pressure. Table 13-12 shows the details of the technical characteristics of these fans.

Table 13-12: Fan Characteristics

System	Type	Commissioning year	Fan Qty	Airflow (per fan) m^3/s	Total Pressure (Pa)	Installed Power (total) kW	Fan type / Configuration
SH01	Exhaust	2026	2	67	3876	700	Centrifugal /Parallel
SH03	Intake	2026	2	70	2055	500	Axial / Parallel
SH04	Exhaust	2030	1	100	2650	400	Centrifugal / NA
NH05	Intake	2026	2	70	1000	300	Axial / Parallel
NH06	Exhaust	2026	2	90	3530	900	Centrifugal / Parallel

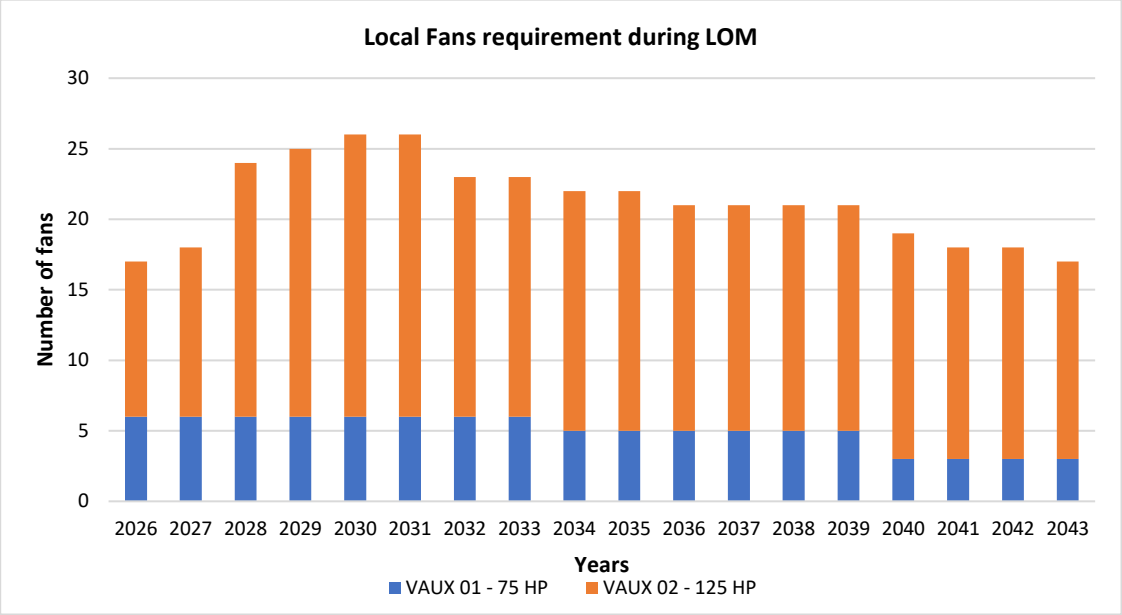
Centrifugal exhaust fans will be installed due to system resistance and air characteristics, which have high humidity and a high risk of corrosion. In this regard, the fans must have wear plates and paint schemes suitable for humid, abrasive, and corrosive environments.

13.6.1.5 Local Ventilation

Ventilation of the levels will be provided by auxiliary fans installed in the intake systems, which are to be cooled by the surface cooling systems (SH02, SH04, and NH05). To prevent air loss, it is essential that the intake system is isolated with walls and doors (that allow access to escape routes).

Furthermore, air exhaust will be carried out to the same levels through connections to the different exhaust systems (SH01, SH04 and NH06). In this regard, and due to the high temperatures, it is not possible to reuse the air at other levels. For this purpose, two types of local fans are required (Figure 13-37): 75 HP fans and 20 m^3/s for production, exploration and preparation workings and can ventilate two dead ends and up to 250 m, through 36 in. diameter flexible ducts. Also, 125 HP fans and 30 m^3/s for main development workings ventilation that can ventilate one dead end and up to 450 m, through 54 in. diameter flexible ducts.

Figure 13-37: Local Fans Requirement



Source: Snowden Optiro, 2025.

13.6.2 Mine Cooling

Due to its geographical location and geological conditions, the Era Dorada project will be exposed to high ambient temperatures. As a result, it will be necessary to implement mitigation measures at the mine design level and to install mechanized cooling systems to reduce temperatures to below 28°C WB, in accordance with the design criteria (Table 13-13).

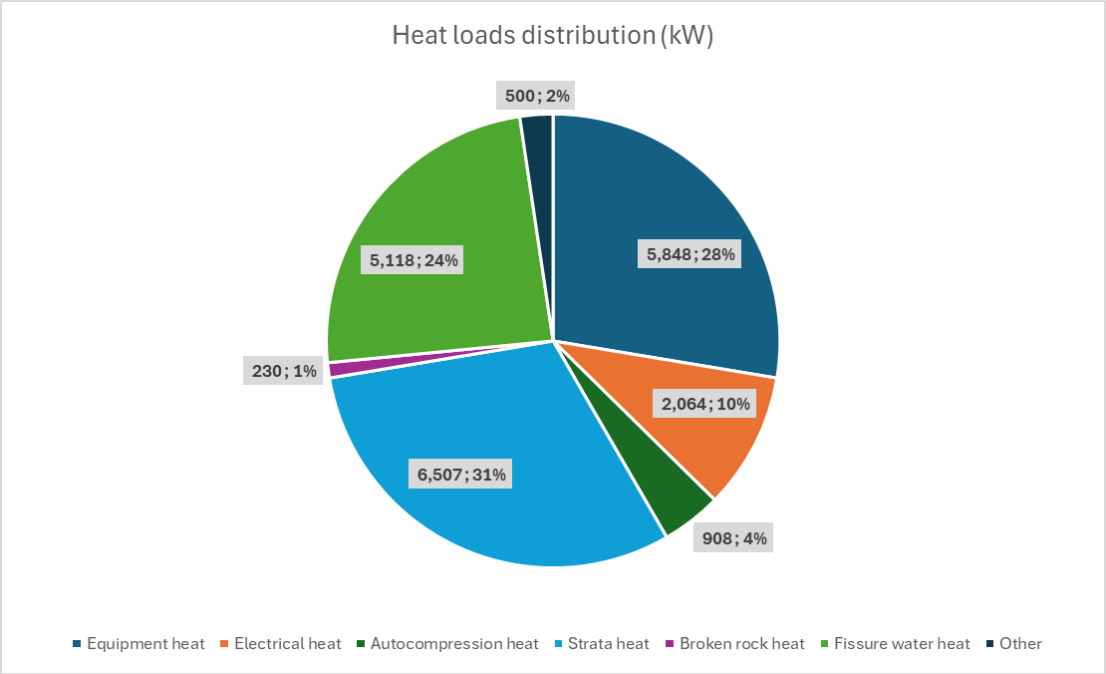
Table 13-13: Design Criteria

Item	Values	Comments
Surface Temperatures	24.52°C / 32.2°C 95 kPa	Percentil 98 Información por hora Periodo 2008 - 2025
Geothermal Gradient	0.1 °C/1 m 51°C @ 497 masl	JDS Report 2019 based on rock temperature data from main decline
Rock Properties	According to different lithologies	According to different lithologies and literature
Reject Temperature	28°C WB Level 32°C WB Decline	Industry accepted values based on productivity, health hazard and cost Decline temperature assuming short duration work and personnel will be in climatized cabins

13.6.2.1 Heat Loads

The heat loads total approximately 21 MW at peak production. The breakdown of these is shown in Figure 13-38 and reveals a significant component originating from the surrounding rock and thermal waters that infiltrate into the mine (representing approximately 55% of the total heat load). On the meantime, ambient air-cooling requirement will reach 7.4 MWR for the peak production conditions.

Figure 13-38: Heat Loads Distribution (kW)



Source: Snowden Optiro, 2025.

13.6.2.2 Cooling Plant Design

The cooling solution will consist of three cooling systems located in the intake raises (SH02, SH03, and NH05) and will be commissioned between 2026 and 2030. Table 13-14 shows the characteristics of the cooling plants.

Table 13-14: Cooling Plant

System	Commissioning year	Inlet Airflow m^3/s	Cooling Duty (MW)	Utilization/Annual basis
SH03	2026	140	7,000	100%
NH05	2026	140	7,000	93%
SH02	2030	30	1,000	97%

The cooling plants and their components were conceived to be established on surface, which minimizes investment (circa US$23.5 million) and operating costs (approximately US$1.8M/a), albeit, on the other hand, this compromises their efficiency as far as location to some extent. After all, heat modeling indicates that it is possible to achieve rejection temperatures below 28°C WB at these levels, to secure safe working conditions at the faces, considering the ventilation philosophy based on isolated intake systems.

13.6.3 Mine Pumping

The underground mine dewatering (pumping) system is designed to manage groundwater infiltration not captured by surface dewatering wells and rainwater infiltration as well as water used by the mine equipment.

FluidFlow 3.5 and the Pumpsim software were used for the pumping simulations and design of the system.

Groundwater infiltration was projected year by year for the northern and the southern zones. Table 13-15 shows the projected groundwater infiltration shared by Stantec for the Project.

Table 13-15: Projected Groundwater Infiltration – Underground Works

Year	North	South	Subtotal	Subtotal With 25% Contingency	North	South	Subtotal	Subtotal With 25% Contingency
	L/s				Gpm			
2025	-3.2	0.0	-3.2	-4.0	-50.7	0.0	-50.7	-63.4
2026	-2.8	0.0	-2.8	-3.5	-44.5	0.0	-44.5	-55.6
2027	-2.0	0.0	-2.0	-2.5	-32.3	0.0	-32.3	-40.4
2028	-1.8	-3.7	-5.4	-6.8	-28.1	-58.3	-86.4	-108.0
2029	-1.1	-4.1	-5.2	-6.5	-17.0	-64.8	-81.9	-102.4
2030	-1.0	-12.1	-13.0	-16.3	-15.2	-191.3	-206.5	-258.1
2031	-0.5	-11.1	-11.6	-14.5	-8.2	-175.2	-183.4	-229.3
2032	-0.5	-16.2	-16.7	-20.9	-8.3	-256.2	-264.4	-330.5
2033	-0.6	-19.4	-20.0	-25.0	-9.5	-307.3	-316.8	-396.0
2034	-0.7	-19.5	-20.1	-25.1	-10.6	-308.4	-318.9	-398.6
2035	-0.8	-21.6	-22.4	-28.0	-12.5	-341.9	-354.4	-443.0
2036	-0.9	-25.0	-25.9	-32.4	-13.8	-396.2	-410.0	-512.5
2037	-0.9	-25.1	-26.0	-32.5	-13.9	-398.5	-412.4	-515.5
2038	-0.9	-26.2	-27.1	-33.9	-14.4	-415.5	-429.9	-537.4
2039	-0.9	-29.8	-30.6	-38.3	-14.2	-471.6	-485.8	-607.3
2040	-0.9	-29.8	-30.8	-38.5	-15.0	-472.8	-487.8	-609.8
2041	-0.9	-29.8	-30.7	-38.4	-15.0	-471.8	-486.7	-608.4
2042	-0.9	-29.8	-30.7	-38.4	-15.0	-471.8	-486.8	-608.5
2043	0.0	-29.9	-29.9	-37.4	0.0	-473.6	-473.6	-592.0

The volume of the groundwater flowing into the mine tends to migrate toward the South Zone. In 2040, the maximum flow rate is reached, and this value is used as the basis for determining the number of pumping stations, the quantity and operating point of the pumps, system energy consumption, total piping length, and piping specifications.

The flow from mine mobile equipment as of 11.44 L/s (181 gpm) is detailed in Table 13-16.

Table 13-16: Equipment Fleet

Equipment	Peak Of Equipment in LOM	Equipment Of Reference	Model	Required Flow rate	
				L/s	gpm
1 boom jumbo	1	DD312i	DD312i	0.165	2.6
2 boom jumbo	4	DD322i	DD322i	6.68	105.9
Production drill	3	DL331, DL 321, DU412i ITH, SIMBA S40	DL321	3.2	50.7
Rock bolter	2	DS311	DS311	0.528	8.4
Cable bolter	1	DS422I	DS422I	0.8	12.7
Shotcrete sprayer	1	Spraymec SF 050 D	Spraymec SF 050 D	0.064	1.0

In addition, the fleet of equipment is divided into approximately 70% to the South Zone and 30% to the North Zone.

Table 13-17 shows details of the waterflows used for the sizing of the pumping system: the South Zone will have a peak of 41.64 L/s (660.8 gpm) reached in 2043 and a flow rate as of 5.84 L/s (92.6 gpm), reached in 2026, was used for the North Zone.

Table 13-17: Summary of Waterflow per Zone by Year

Waterflow per Zone	Waterflow (GPM)																	
	Yr-1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17
Northern Zone	93	78	79	65	73	65	48	49	50	53	47	47	47	47	48	48	48	29
Southern Zone	87	87	176	184	366	346	407	471	472	514	564	567	588	658	659	658	658	660
Total	179	164	255	250	439	411	455	520	523	567	610	613	635	705	707	706	706	690

In summary, capacities to meet water demand as of seven liters per second for the Northern zone, and 50 liters per second for the Southern zone were used as shown in Table 13-18.

Table 13-18: Summary of Infiltration

Zone	Requirement (l/s)
North	7
South	50
Total	57

13.6.3.1 Piping

The pumping system designed for the project conditions incorporates HDPE SDR 11 pipes. For initial or critical sections, metal pipes were selected to withstand the high pressures resulting from the physical phenomena associated with pumping water at high temperatures.

Table 13-19: Summary of Infiltration

Zone	Principal Line	Heading Line
Northern	4 inch, SDR- 11	2 inch, SDR- 11
Southern	10 inch, SDR- 11	6 inch, SDR- 11

13.6.3.2 Mine Pumping

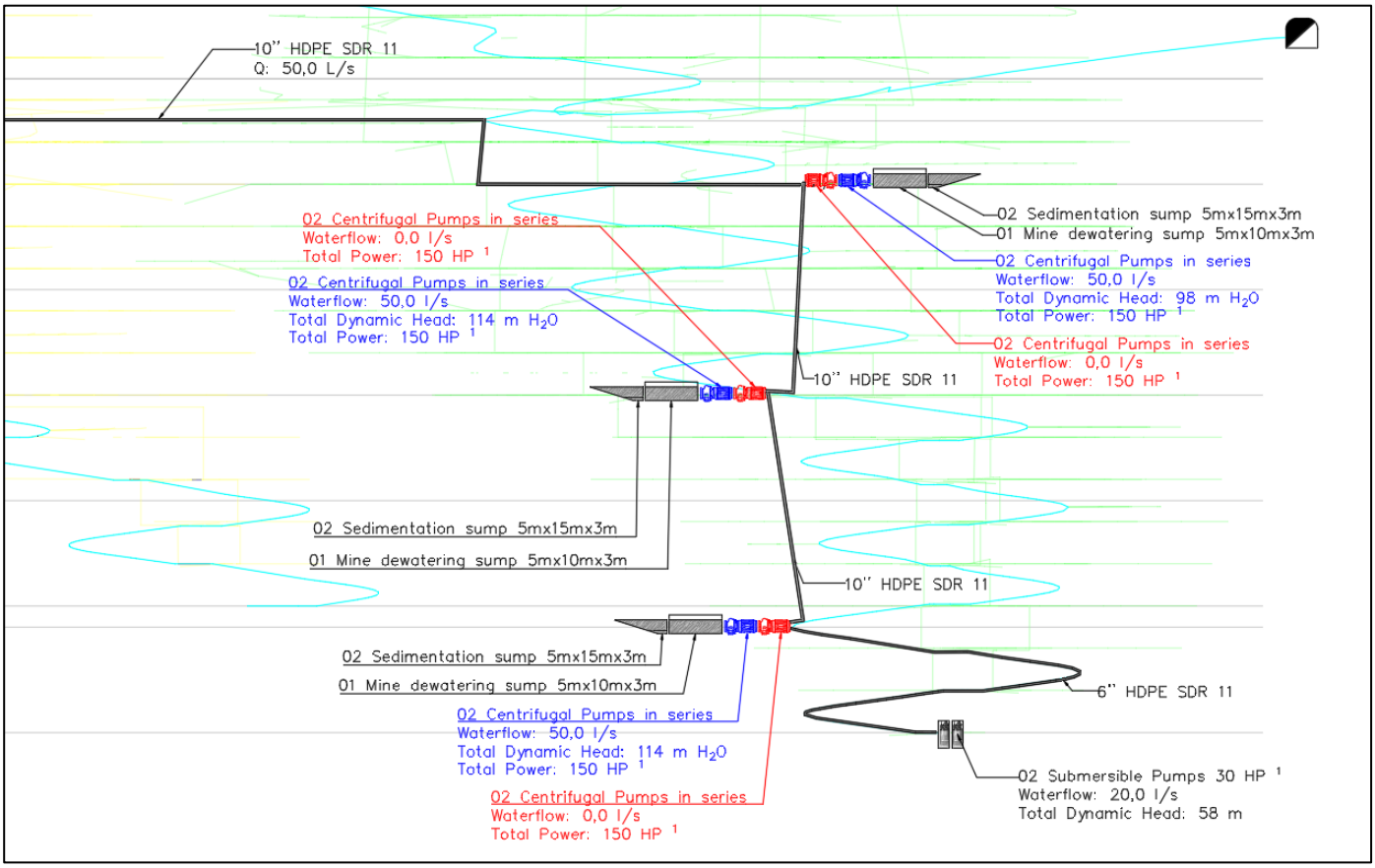
The underground pumping system to the surface includes main pumping stations for both the South and North zones of the mine.

The South Zone will have three main pumping stations on levels 210, 320 and 420. Six pumps with a designated duty point and power consumption of 285 kW will meet the required flow rate as of 50.0 liters per second.

Similarly, the North Zone will have two main pumping stations on levels 270 and 370. Four pumps operating at a single duty point and consuming 38 kW will handle a 7.0 liters per second flow.

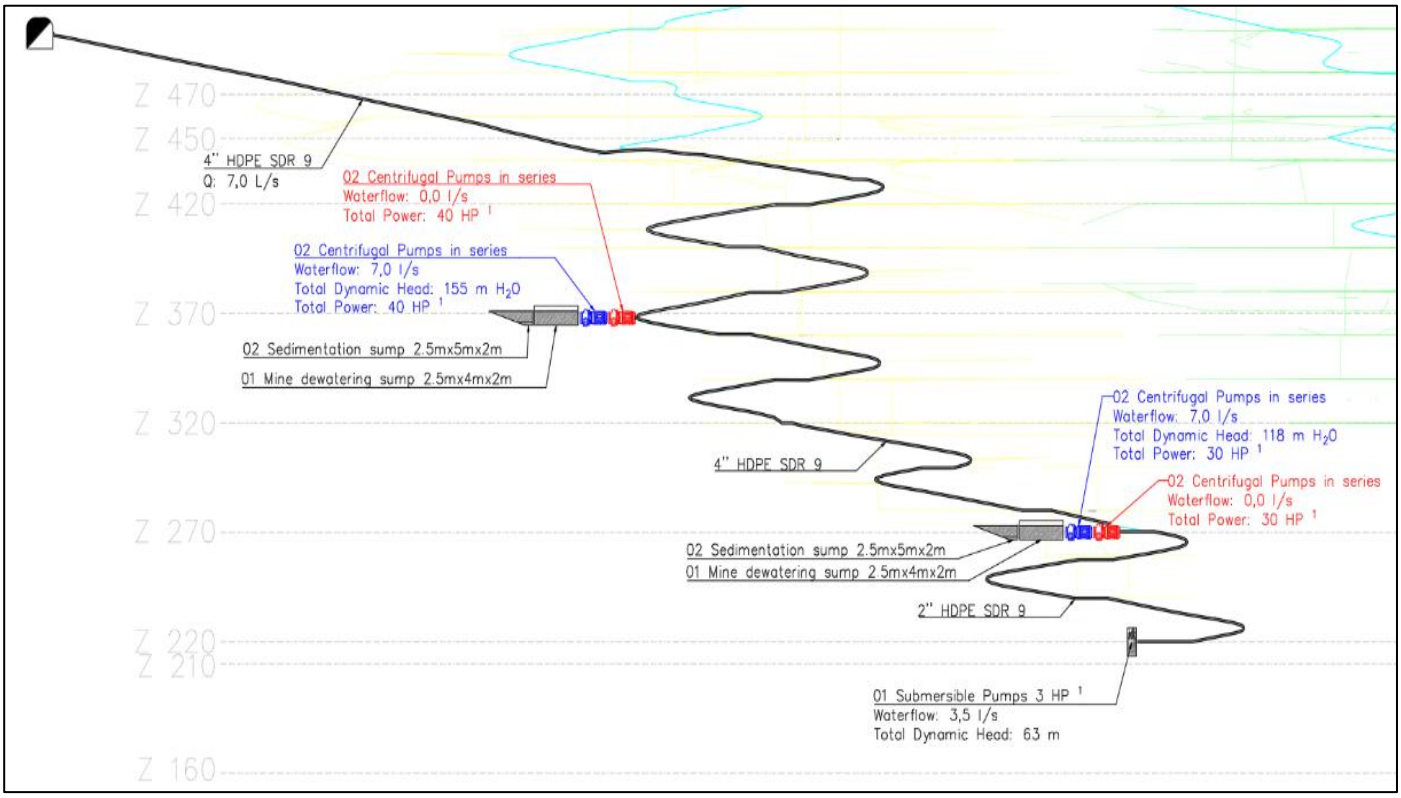
The general diagram of the main pumping system is presented in Figure 13-39 and Figure 13-40.

Figure 13-39: Main Pumping System Design for the South Zone



Source: Snowden Optiro, 2025.

Figure 13-40: Main Pumping System Design for the North Zone



Source: Snowden Optiro, 2025.

The Main Pumping Stations for each zone are presented as per Table 13-20.

Table 13-20: Summary of the Main Pumping Stations

Sump No	Zone	Elevation (masl)	Pump to	Pump (kW)
3	South	420	Water Treatment Plant	4x56 (2 operating in series)
4		320	S420	4x56 (2 operating in series)
5		210	S320	4x56 (2 operating in series)
1	North	370	Water Treatment Plant	4x15 (2 operating in series)
2		270	S370	4x11 (2 operating in series)

Additionally, submersible pumps will be deployed to drain water from the development headings. Six pumps with a designated operating point and a power consumption of 13.4 kW will be used.

14 PROCESSING AND RECOVERY METHODS

14.1 Overview

Based on the information and metallurgical test results summarized in Section 10, Era Dorada gold-silver mineralization is considered amenable to gravity concentration followed by cyanide leaching. The process plant will consist of a 1600 t/d, one stage crushing, SAG mill and pebble crusher, ball mill, leach, CIP, elution, electrowinning circuit, all of which are well-known, conventional, processing unit operations.

The process plant have the capacity to process 1,600 t/d or 72.5 t/h based on 8,059 hours per annum or 92% availability. The crushing section design is based on 75% availability, and the gold room availability is set one melt per week. The process plant is designed to operate with two shifts per day and 365 days per year and will produce doré bars.

Key design parameters derived from metallurgical testwork, as well as the resulting sizing parameters of major equipment, are shown in Table 14-1.

Table 14-1: Process Design Criteria

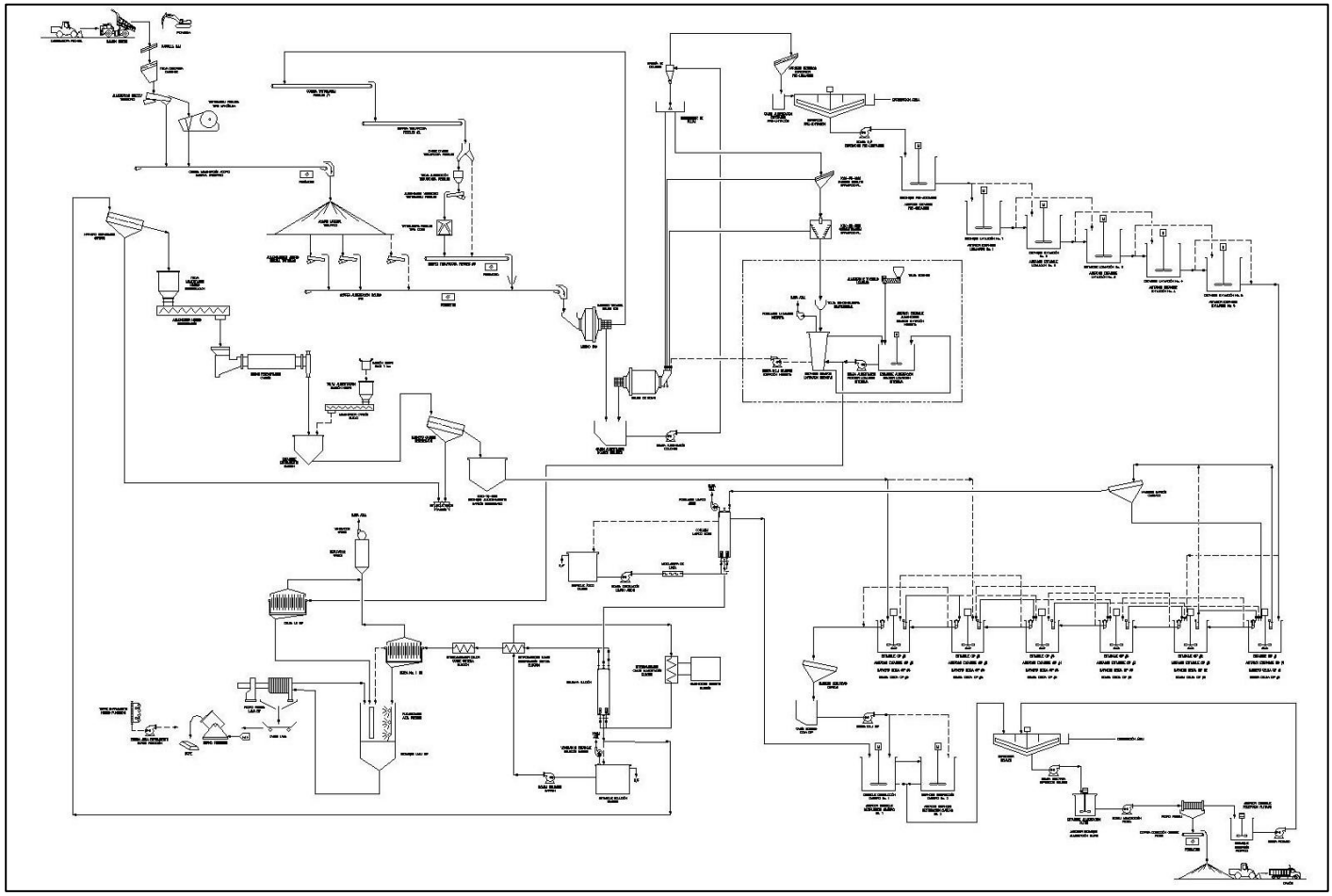
Description	Units	Value
General		
Ore throughput	t/d	1,600
Mine life	years	18
LOM average grade, Au	g/t	6.01
LOM average grade, Ag	g/t	20.39
Operating Schedule and Stockpile		
Crusher availability	%	75
Plant availability (milling and leach)	%	92
Crusher operating time	h/a	6,570
Plant operating time	h/a	8,059
Gold room operating days	d/a	104
Gold room smelting days	d/a	52
Stockpile type	-	Conical
Stockpile repose angle	degrees	37
Stockpile retention time	h	12
Ore Properties		
Specific gravity (average)	-	2.56
Jk Axb (25th percentile)	-	33.6
Bond rod work index (BRWi) (75th percentile)	kWh/t	20.9

Description	Units	Value
Bond ball work index (BBWi) (75th percentile)	kWh/t	23.0
Bond abrasion index (Ai) (average)	g	0.50
Crushing		
Throughput, nominal	t/h	88.9
Primary crusher type	-	Jaw
Primary crusher model	-	Metso C106 or similar
Closed size setting	mm	73
Feed size, F_{80}	mm	309
Crushing product, P_{80}	mm	88
Grinding		
Throughput, nominal	t/h	72.5
SAG mill diameter	m	4.9
SAG mill effective grinding length	M	4.9
Circuit configuration	-	Pebble recycle
Pebble recycle rate	%	18
SAG mill required power	kW	1,300
Primary Grinding Transfer Size (T_{80})	Mm	1.02
Ball mill diameter	m	4.6
Ball mill effective grinding length	m	7.3
Circuit Configuration	-	Closed
Ball mill required power	kW	2,600
Circulating load, max for design	%	250
Cyclone overflow solids	% solids by weight	31
Cyclone overflow grind size (P_{80})	µm	53
Gravity Concentration		
Concentrator Type	-	Semicontinuous Batch Centrifugal
Number of Units	#	1
Feed Source	-	Cyclone underflow
Recovery Method	-	Intensive leach reactor
Pre-Leach Thickening		
Pre-Leach Thickener Loading Rate	t/m²/h	0.40
Pre-Leach Thickener Underflow Density	% w/w solids	50
Leaching		
Pre-Oxidation	Y / N	Yes
Pre-Oxidation Retention Time	hours	2
Dissolved Oxygen Target (DO)	mg/L	20
Leach Retention Time	hours	36
Lead Nitrate Consumption	g/t	250

Description	Units	Value
Number of Leach Tanks	-	5
Sodium Cyanide Consumption	kg/t	0.70
Lime Consumption	kg/t	1.71
CIP		
CIP Retention Time	h	6
Number of CIP Tanks	-	6
Carbon Concentration	g/L	50
Carbon Loading	g/ Au	2,500
Carbon Processing		
Carbon Handling Capacity	t/d	4.0
Acid Wash Type	-	Hydrochloric Acid
Elution Type	-	Pressured Zadra
Elution Operating Temperature	ºC	140
Elution Operating Pressure	kPa	350 to 550
Smelting Furnace Type	-	GLP Furnace
Cyanide Destruction		
Feed Solution, CNWAD	mg/L	191
Discharge Solution, CNWAD	mg/L	<1.0
Design Retention Time	hours	3.0
Number of Tanks	#	2
SO_2 Consumption	g/g CN_{WAD}	4.00
Lime Consumption	g/g CN_{WAD}	0.80
Copper Sulphate Concentration	mg/L	25.0
Tailings Management		
Disposal Type	-	Dry stack/Paste
Tailings Filter Type	-	Pressure, plate and Frame
Filtration Rate	kg/h/m^2	174
Final Moisture Content	%	18.3

14.2 Process Flowsheet

Figure 14-1: Simplified Process Flowsheet



Source: Ausenco, 2025.

The process flowsheet was developed based on information from the metallurgical testwork as outlined in Section 10. The flowsheet developed previously was modified to a simpler, lower capital cost alternative from earlier studies comprising:

- Primary crushing circuit
- SABC grinding circuit
- Leach-CIP circuit with pre-oxidationuCyanide destructionuTailings filtration.uThe simplified overall flowsheet is shown in Figure 14-1. The plant site layout is shown in Figure 15-2.

14.3 Plant Design

The plant feed will be hauled from the underground mine to a crushing facility that will include a jaw crusher. The crushed ore will be ground by a SAG mill and a pebble crusher circuit, then sent to ball mill in closed circuit with a hydrocyclone cluster. The hydro-cyclone overflow with an 80% passing size (P_{80}) of 53 µm will flow to a leach–CIP recovery circuit via thickening and pre-aeration.

Gold and silver leached in the leach/CIP circuit will be recovered onto activated carbon and eluted in a pressure Zadra-style elution circuit and then precipitated by electrowinning in the gold room. The gold–silver sludge will be dried in an oven and then mixed with fluxes and smelted in a furnace to pour doré bars. Carbon will be re-activated in a carbon regeneration kiln before being returned to the CIP circuit.

CIP tails will be treated in cyanide destruction, filtered and disposed on a tailings storage facility (TSF) for disposal or used on the back fill plant, mixed with cement to return to the underground mine.

14.3.1 Crushing

Ore from the underground mining operations will feed a crushing plant that consists of primary crushing. The plant will process 88.9 t/h of ore, operate 18 hours per day and produce a final product with a P80 of 88 mm.

The major equipment and facilities at the ROM receiving and crushing areas will include:

- Ore stockpile ROM hopper
- Vibrating grizzly feeder
- Primary Jaw Crusher
- Product conveyor

Ore will be trucked from underground and dumped directly into the ROM hopper or onto the outdoor stockpileduring crushing circuit downtime. A front-end loader will reclaim ore from the stockpile and feed it to the ROM hopper as necessary.

The ROM hopper will continuously feed a vibrating grizzly feeder which will discharge the oversize into the primary jaw crusher. Jaw crusher discharge and vibrating grizzly fines discharge onto the stockpile feed conveyor and discharge onto a conical stockpile. The stockpile feed conveyor is equipment with a weightometer to measure tonnes crushed. A belt magnet and a metal detector are provided to detect and remove tramp metal.

Crushed Ore Stockpile: The crushed ore stockpile provides 735 t, or ten hours, of live storage capacity. The stockpile has a further 1,940 t, or 26 hours of capacity in the dead volume that can be recovered by a front-end-load to a hopper equipped with a vibrating pan feeder. Two vibrating pan feeders, located underneath the stockpile, are provied and are fitted with variable frequency drives (VFD) to control the reclaim rate feeding the SAG mill circuit. Each feeder is capable of providing full plant throughput of 72.4 t/h.

Grinding: The grinding circuit consists of a primary SAG mill, a pebble crusher, followed by a ball mill. A gravity concentration circuit will be installed in the ball mill circuit to recover any gravity recoverable gold. The SAG mill will operate in open circuit, the trommel oversized will be crushed with a cone crusher and returned to the SAG mill feed conveyor, while the ball mill will operate in reverse closed circuit with a cluster of hydrocyclones. Part of the cyclone underflow will be processed through the gravity circuit. The grinding circuit will be able to process a nominal throughput of 72.4 t/h (fresh feed), producing a final product size P_{80} of 53 µm.

The major equipment and facilities at the grinding and gravity concentration areas will include:t4.7 m diameter x 5.2m m EGL (effective grinding length) SAG mill

- 4.9 m diameter x 4.9 m SAG mill
- Pebble (cone) crusher
- 4.6 m diameter x 7.3 m ball mill
- 10 x 254 mm diameter hydrocyclones
- Gravity concentrator
- Intensive Leach Reactor.

14.3.2 SAG mill

Reclaimed ore from the crushed material stockpile will feed a 4.9 m diameter by 4.9 m EGL grate discharge SAG mill via the SAG mill feed conveyor. The mill will be installed with a 1,300 kW induction motor and a VFD to control the speed of the mill. A belt-scale on the feed conveyor will monitor feed rate. Process water will be added to the SAG mill to maintain the slurry density of 75% by weight (w/w). Ground slurry will discharge from the SAG mill over a trommel screen, with the undersize flowing into the cyclone feed pump box, combining with ball mill discharge and gravity concentrator tailings, while the oversize will be conveyed and crushed and crushed in a dedicated cone crusher. Pebble crusher discharge is retured to the SAG mill feed conveyor. The primary grinding circuit has been designed to produce a transfer size of 80% passing 1 mm.

14.3.3 Pebble Crusher

The oversize material from the SAG mill trommel will be conveyed via a conveyor belt equipped with a metal detector and metal extractor, in order to prevent uncrushable materials from being fed into the crusher. The conveyor will discharge into a 7 m^3 surge bin which provides 46 minutes retention time. A belt feeder will regulate the feed to the crusher, with the feed rate being monitored and recorded by a scale installed on the feeder.

The pebble cone crusher will discharge directly onto the SAG mill feed conveyor, returning the crushed pebbles to the grinding circuit.

14.3.4 Ball Mill

Slurry from the cyclone feed pump box will be pumped up to a cluster of ten (6 operating/4 standby) 250 mm hydrocyclones for size classification. The coarse cyclone underflow will be split into two streams, with 75% of the slurry flowing by gravity to the ball mill for additional grinding, and 25% feeding the gravity circuit. The cyclone overflow, at a final target product size P_{80} of 53 µm, will be gravitate to the preleach thickener through a trash screen. The hydrocyclones have been designed for a 250% circulating load.

Cyclone underflow will feed a 4.6 m diameter by 7.3 m long overflow ball mill with an installed power of 2,600 kW. Ground slurry will overflow from the ball mill onto a trommel screen attached to the discharge end ofthe mill. The trommel screen oversize, consisting mainly of scats, will discharge into a trash bin for removal and disposal, while the undersize will flow into the cyclone feed pump box.

14.3.5 Gravity Concentration

Approximately 25% of the cyclone underflow will flow by gravity to the gravity concentrator feed screen. With an aperture size of 1 mm, the feed screen will remove any oversize particles prior to gravity concentration. The screen undersize will feed a semi-continuous batch gravity concentrator. Using high gravitational forces, high density gravity recoverable gold will collect in the concentrate cone, while lower density material will flow out of the tailings discharge port and combine with the gravity feed screen oversize in the gravity tailings pump box. The material will then be pumped to the ball mill feed box.

The gravity concentrator will operate in forty-minute cycles. During a cycle, gravity recoverable gold will collect in the concentrate cone. At the end of the cycle, the gravity concentrator feed will be diverted to the gravity tailings stream, and the concentrate cone will be flushed with water, sending the concentrate to an intensive leach reactor for further concentration.

14.4 Pre-Leach Thickening

Cyclone overflow will flow onto a vibrating trash screen for removal of trash material. Oversize material will discharge into a trash bin, while screen undersize will flow by gravity to an 18 m diameter pre-leach thickener. Flocculant solution will be added to the thickener feed to promote the settling of fine solids. The high-rate thickener will thicken the slurry to 50% w/w solids. The thickener underflow will be pumped to the pre-aeration tank, while the thickener overflow will flow by gravity into the process water tank to be used as make-up water in the grinding circuit.

14.5 Leaching

Pre-leach thickener underflow will be pumped to a 6.5 m diameter x 7.3 m high pre-aeration tank prior to leaching. Oxygen will be sparged into the bottom of the agitated tank and slurry will be conditioned for two hours to oxidize sulphide minerals.

Based on metallurgical testing, pre-aeration will help reduce the consumption of dissolved oxygen during cyanidation, improving metallurgical recovery. It will also reduce sodium cyanide (NaCN) consumption by preventing the formation

of thiocyanate and complexing some of the heavy metals such as iron. This step will also reduce reagent consumptions in the cyanide destruction circuit. Lead nitrate is added to complex any soluble iron or sulphur species that would increase cyanide consumption.

The oxidized slurry will then flow to the first of five 11.1 m diameter x 10 m high agitated leach tanks. The leach circuit is designed to provide 36 hours of retention time. Lime slurry will be added to the first and second leach tanks at a rate of up to 1.71 kg/t to maintain protective alkalinity at a design pH of 11.0, preventing the creation of hydrogen cyanide gas (HCN). NaCN solution will be added to the circuit at a rate of up to 0.70 kg/t, while oxygen will be sparged in from the bottom of each tank to maintain dissolved oxygen (DO) above 20 mg/L. As the slurry progresses through the circuit, gold and silver will be leached into solution.

Slurry from the leach circuit will then flow by gravity to the CIP circuit for carbon adsorption.

14.6 Carbon in Pulp (CIP)

Leached slurry will flow into the first of eight 5.9 m diameter x 5.4 m high CIP tanks. Each tank will be installed with an agitator and an inter-stage screen for retaining activated carbon. As the slurry flows through the six CIP tanks, gold-cyanide and silver-cyanide complexes will be adsorbed onto the pores of the activated carbon. The average carbon concentration in the CIP circuit will be approximately 50 g/L to maximize adsorption. The high carbon concentration is to accomodate the high silver concentrations. Ausenco completed a trade off study compared leach/CIP to CCD (counter current decantation) with Merrill-Crowe precipitation to determine which was technically and economically best suited to accomdodate the Au and Ag concentrations. Leach/CIP provides lower capital and operating costs while maintaining required Au and Ag recoveries.

As the slurry proceeds through the circuit, dissolved metal values in the solution will progressively decrease. The carbon will be transferred counter current to the slurry flow to maximize precious metal recovery and minimize soluble losses. Regenerated carbon, with the highest adsorption potential, will be introduced into the last CIP tank, interacting with the lowest concentrations of gold and silver. Loaded carbon, with the lowest adsorption potential, is in the first CIP tank, interacting with the highest concentrations of gold and silver. Once per day, loaded carbon from the first CIP tank will be pumped to the loaded carbon screen where the slurry will be separated with the carbon transferred to the acid wash circuit. Loaded carbon screen undersize slurry will flow by gravity back into the first CIP tank.

The tailings stream from the last CIP tank will flow onto a vibrating safety screen to capture any carbon that may have escaped the CIP circuit. Captured carbon particles will be collected in bins and processed to recover gold and silver. Safety screen undersize will then be pumped to the cyanide destruction circuit.

14.7 Carbon Elution and Regeneration

Carbon elution and regeneration has been designed to handle 4 t/d of loaded carbon, producing gold and silver doré using the pressure Zadra process. On average one batch of carbon will be processed per day through the acid wash, elution and regeneration circuits.

14.7.1 Acid Wash

Loaded carbon from the CIP circuit will flow by gravity into a four-tonne capacity acid wash vessel constructed of fibre-reinforced plastic (FRP). The carbon will be treated by a circulating 3.5% hydrochloric acid (HCl) solution to remove calcium deposits, magnesium, sodium salts, and fine iron particles. Organic foulants, such as oils and fats, are unaffected by the acid and will be removed after the elution step in the thermal regeneration circuit using a horizontal electric kiln.

During the acid wash cycle, an HCl solution will be pumped from the dilute acid tank upward through the acid wash vessel, overflowing back into the dilute acid tank in the beginning of the process and, in the end, disposed to cyanide detox. The carbon will then be rinsed with a solution of fresh water and sodium hydroxide to remove the remaining acid.

A carbon eductor will transfer acid washed carbon from the acid wash vessel into the elution vessel using transport water. Carbon slurry will discharge directly into the top of the elution vessel. Under normal operations, one acid wash and elution cycle will take place per day.

14.7.2 Carbon Stripping (Elution)

The carbon stripping (elution) process will utilize fresh water, sodium cyanide and sodium hydroxide solution to strip the loaded carbon, creating a pregnant gold and silver solution which will be pumped through the electrowinning cells for precious metal recovery.

The strip vessel will be a carbon steel tank with a capacity to hold approximately 4 t of carbon. During the strip cycle, solution containing approximately 2% sodium hydroxide and 3.0% NaCN, at a temperature of 140°C, will be pumped up through the strip vessel at a pressure of 450 kPa. Solution exiting the top of the vessel will be cooled below its boiling point by the heat recovery heat exchanger. Heat from the outgoing solution will be transferred to the incoming cold barren solution prior to passing through the solution heater. An electric boiler will be used as the primary heating source. The strip will be complete in approximately 12 hours allowing additional strips to accommodate higher feed grade material.

14.7.3 Carbon Regeneration

The carbon regeneration circuit will thermally regenerate the stripped carbon, re-activating the pores and removing any organic foulants, such as oils and fats. Fresh activated carbon will be added to account for any carbon lost during the adsorption and desorption processes.

An eductor will transfer the stripped carbon from the elution vessel to the carbon dewatering screen. Oversize carbon from the screen will discharge by gravity into the regeneration kiln feed hopper. Screen undersize carbon, containing carbon fines and water, will drain by gravity into the carbon fines tank. Periodically, the carbon fines will be collected in bags and sent to a refinery to recover gold and silver.

A horizontal propane fired kiln will be utilized to treat 4 t of carbon per day, equivalent to 100% regeneration of stripped carbon. The regenerated carbon from the kiln will flow by gravity into the carbon quench tank, cooled by fresh water.

Fresh carbon is added to the quench tank. Regenerated and fresh carbon are pumped from the quench tank to the carbon sizing screen. Sizing screen undersize flows to the carbon fines tank while oversize is sent to the CIP circuit.

14.7.4 Electrowinning and Refining

Pregnant solution from the strip circuit will be pumped to the refinery for electrowinning, producing a gold and silver sludge. The sludge will then be filtered, dried and refined in a propane fired furnace, producing gold and silver doré bars.

Pregnant solution will be pumped through two electrowinning cells, one for gravity intensive leach and one for elution. Gold and silver will plate on 36 stainless steel cathodes in each cell, while the barren solution will flow into the barren return tank and be pumped back to the elution.

Gold and silver rich sludge will periodically be washed off the stainless-steel cathodes into the electrowinning sludge tank using high pressure water. Once the tank is filled, the sludge will be drained, filtered, dried, mixed with fluxes, and smelted in a propane fired furnace, producing gold and silver doré. This process will take place within a secure and supervised area, and the precious metal dore will be stored in a vault until shipped off site.

14.8 Cyanide Destruction

The cyanide destruction circuit will consist of two 9.2 m diameter x 8.5 m high mechanically agitated tanks, each with a capacity to handle the full slurry flow for the required residence time of three hours. Cyanide will be destroyed using the SO_2/air process. Treated slurry from the circuit will then be pumped to the final tailings thickener. The cyanide destruction circuit will treat CIP tailings slurry, and process bleed streams.

Oxygen will be sparged from near the bottom of the tanks. Lime slurry will be added, as needed, to maintain the optimum pH of 8.0 – 8.5 and copper sulphate will be added as a catalyst, maintaining a 25 mg/L copper concentration in solution. A sodium metabisulphite (SMBS) solution will be dosed into the system as the source of SO_2, at a mass ratio of 4.0:1 SO_2:CN_{WAD}, (weak acid dissociable cyanide). This system has been designed to reduce the CN_{WAD} concentration to less than 1.0 mg/L.

14.9 Final Tailings Thickener

Treated slurry from the cyanide destruction circuit will be directed by gravity to an 18 m diameter final tailings thickener. Flocculant solution will be added to the thickener feed to promote the settling of fine solids. The high-rate thickener will thicken the slurry to 57% w/w solids. The thickener underflow will be pumped to the filter feed tank.

The overflow will flow by gravity into de process water tank to be used as make up water in the plant.

14.10 Tailings Management

The final tailings thicker underflow will be directed to a 12.8 m diameter x 9.8 m high agitated filter feed tank, with eight hours of capacity, and then pumped to three 2,000 mm x 2,000 mm plate and frame pressure filters, where the tailings will be dewatered to a moisture content of 18.6%. The tailings will then be transported by truck to the dry stack

tailings facility (DSTF) or to the paste fill plant. Filtrate and wash down water will be circulated back to the final tailings thickener feed to prevent any solids contamination in the process water tank.

14.11 Product/Materials Handling

14.11.1 Reagents

Reagents consumed within the plant will be prepared on-site and distributed via the reagent handling systems. These reagents include sodium cyanide (NaCN), lime, lead nitrate ($Pb(NO_3)_2$), hydrochloric acid (HCl), caustic soda (NaOH), copper sulphate, sodium metabisulphite, antiscalant, flocculant and activated carbon. All reagent areas will be bunded and fitted with sump pumps which will transfer any spills to their respective storage tanks. The reagents will be mixed, stored and then delivered to the thickener, leach, CIP, acid wash, elution, and cyanide destruction circuits. Dosages will be controlled by flow meters and automatic control valves. The capacity of the storage tanks will be sized to handle one day of production. The reagents will be delivered in dry form, except HCl, Sodium Hydroxide and antiscalant, which are delivered as solutions.

Table 14-2 summarizes the reagents used in the process plant and their estimated daily consumption rates. The table also includes other major process consumables.

Table 14-2: Reagents and Consumables Daily Consumption Rates

Description	Units	Usage
Sodium Cyanide	t/d	2.71
Lime	t/d	3.2
Lead Nitrate	kg/d	408
Hydrochloric Acid	L/d	667
Caustic	m^3/d	1.87
Copper Sulphate	kg/d	270
SMBS	t/d	3.32
Antiscalant	L/d	87
Flocculant	kg/d	168
Activated Carbon	kg/d	120
SAG Mill Grinding Media – 125 mm chrome steel	t/d	1.42
Ball Mill Grinding Media – 50 mm chrome steel	t/d	1.32

14.12 Process Plant Labour

The process labour organization chart has been divided into two categories, operations, maintenance. Staff roles and numbers have been benchmarked against similar operations from Aura, with staff drawn from local communities. The organization chart is based on two 12-hour operating shifts per day with a four-panel rotating roster for shift personnel.

The labour roster will vary over the life of the operation. Labour rates include provision for labour overheads such as health plans and medical examinations, life insurance, holidays, overtime, redundancy benefits, etc. Wage and salary levels were determined by Aura, and it is believed they reflect community expectations for an operation of this size and location.

Table 14-3: Plant Operations and Maintenance Personnel

Personnel	Total Number of Employees
Maintenance	
Industrial Maintenance Coordinator	1
MP&S Supervisor	1
Mechanical/Electrical Provisioner	1
Maintenance Planner	2
Senior Mechanical Reliability Engineer	2
Senior Electrical Reliability Engineer	1
Lubrication Technician II	1
Maintenance/Planning Inspector	2
Electrical Maintenance/Planning Inspector	1
Electrician II	4
Administrative Electrician II	3
Electrician Technician III	4
Administrative Electrician Technician	1
Electrical Maintenance Supervisor	1
Automation Specialist Technician	1
Industrial Autom./Automation/Instrumentation Supervisor	1
MechanicalMechanic Shift II	8
Administrative Mechanic II	15
Specialized Mechanical Technician	2
Mechanical Maintenance Supervisor	1
Operation	
Beneficiation Manager	1
Plant Coordinator	1
Senior Process Engineer	1
Senior Operations Analyst	1
Senior Process and Performance Analyst	1
Plant Supervisor	4

Personnel	Total Number of Employees
Process Technician II	1
Control Room Operator II	4
Process Operator III – Comminution	8
Process Operator III – Hydrometallurgy	8
Process Operator III – Comminution	8
Process Operator III – Hydrometallurgy	8
Process Operator III – Reagents	4
Process Operator III – Filtration	8
Process Operator III – WWTP (ETE)	4
Foundry Operator II	1
Personnel Total	**117**

14.13 Energy, Water, and Process Materials Requirements

14.13.1 Energy

The process plant will draw electrical power from the local grid and from site generated power. The maximum total process electrical power consumption is 4.9 kWh/t and 63,342MWh/a.

14.13.2 Air Supply

An instrument and plant air system with four compressors and associated dryers, filters, and receivers will be provided and located in a compressor room inside the plant building. Two extra compressors will be installed for pressure filters operation.

Oxygen will be used in the pre-aeration, leach, CIP and cyanide destruction circuits and will be supplied by two oxygen generation systems.

14.13.3 Water Supply and Consumption

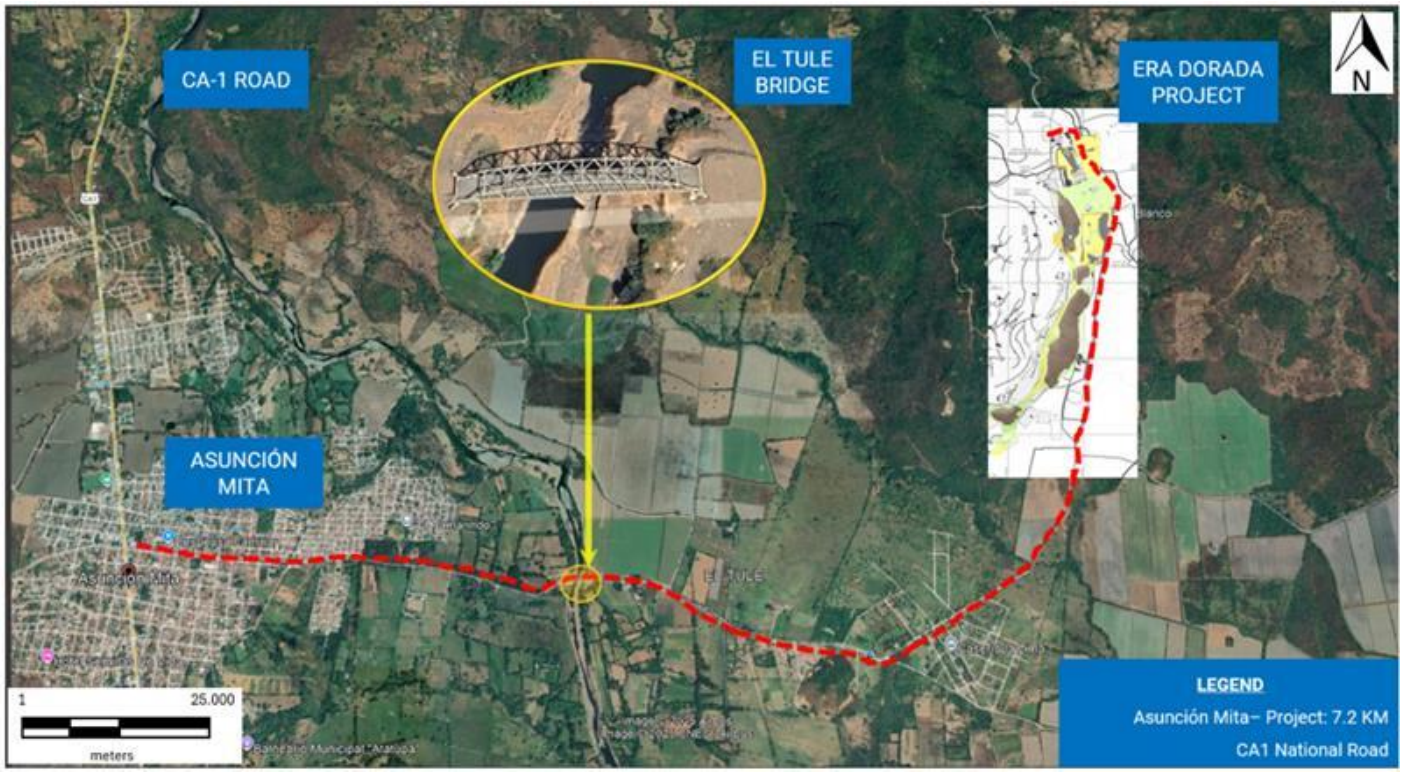
Overflow water from the pre-leach and tailings thickeners will be used as process water. This water will have a low precious metals concentration and will be used in the grinding circuits to dilute slurry into the required densities. Treated water will supply process make-up water, gland water, reagent make-up water and cooling water services in the strip circuit. A bleed of 25% of the thickener overflow water will be sent to the water treatment plant to reduce the buildup of chemicals and metals in the circuit.

15 INFRASTRUCTURE

15.1 Introduction

The Era Dorada Project is a gold project located in southeastern Guatemala, approximately 160 km from Guatemala City by road. It is located in the municipality of Asunción Mita, department of Jutiapa, approximately 9 km from the western border with El Salvador. The closest town to the project is Asunción Mita, a community with a population of approximately 17,500 inhabitants, located about 7 km from the site. The project covers a concession area of 15.25 km² and is located entirely in the municipality of Asunción Mita, district of Jutiapa as indicated in Figure 15-1.

Figure 15-1: Mine Era Dorada – Location

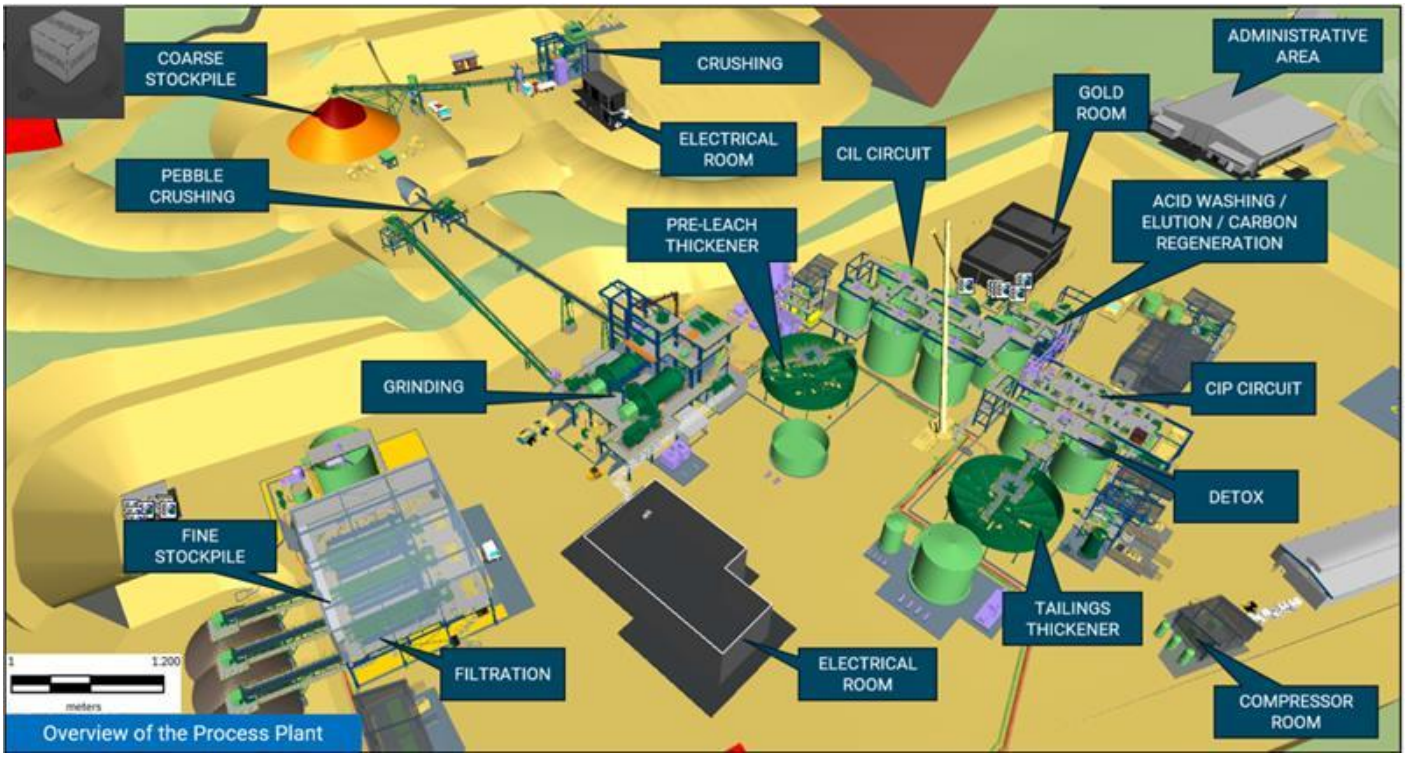


Source : Ausenco 2025.

The scope of the project is to design, build and put into production the Era Dorada mining project, which consists of an underground operation, which includes waste dumps, tailing storage facilities, stockpiles, processing plant and infrastructure and services within the mining operation Figure 15-2).

The Process Plant composed of coarse and fine stockpile, pebble crushing, grinding, filtration, electrical room, pre-leach thickener, CIL and CIP circuit, gold room, tailings thickener, detox, compressor room, acid washing, elution, carbon regeneration and administrative area, as shown in Figure 15-2.

Figure 15-2: Overview of the Process Plant



Source: Ausenco, 2025.

15.2 Site Access

Road access to the site is currently through Asunción Mita, which includes several narrow streets. The project site is accessed by gravel road from the eastern edge of Asuncion Mita, crossing the river Grande de Mita using a bridge (El Achotal) that has a 27 t capacity. The route is not considered suitable for year round delivery of heavy equipment and materials during construction and operations. A base case design including a road and river crossing over the Río Grande de Mita has been prepared to support the feasibility study cost estimate. The new access road will be able to support the heavy equipment loads anticipated during construction and operations. In addition, it is expected that the road will originate from the CA-1 Pan American highway to avoid the residential development around Asunción Mita. The road is designed for a maximum speed of 50 km to accommodate two way traffic. The development of the new road will include upgrading some sections of existing farm access roads. The main site roads that will be developed include the access road (from the access control entrance to the plant and infrastructure site) and the portal connector road (between the North and South portals). Additional ancillary roads will be developed for mine dewatering wells

and reinjection wells access and maintenance. Various roads have already been developed for accessing wells and drilling locations, and these will continue to be used wherever practical. The access road stays within the property boundary and security fencing from the access point to the plant site. It will continue on the lower elevations around the waste rock dump and DSTF and enter the plant site from the south. Ore will be transported on the portal connector road from the North portal to the crusher. Waste rock will be transported from the South portal to the waste rock dump from the portal connector road. Various temporary construction access roads will be made or modified from existing roads for temporary.

15.3 Built Infrastructure

The definition of construction types was based on construction feasibility, cost-benefit ratio, functionality, construction time, and plant operating time, in addition to the use of existing buildings.

The building construction types must be able to withstand seismic activity, governed by standards NSE-2-2024 and NSE-3-2024, which establish the criteria for the analysis and dimensioning of structures subject to seismic stress in Guatemala, since the project is located in Seismic Zone 4.1, with a PGA (Peak Ground Acceleration) of 0.40 g being adopted for seismic stability analyses.

Thus, the following types were defined:

- Prefabricated Modular Buildings

  All new administrative buildings, including extensions - main gate, changerooms, new administrative office + control room, industrial kitchen and support buildings for paste fill, filtration and pile - will be prefabricated thermo-acoustic modular constructions supplied complete with structures, closures, frames, ceilings, electrical, plumbing, data, telephone, and air conditioning installations, sanitary ware, and internal partitions, fully finished and ready for use after installation of furniture and IT/TELECOM equipment.

  The supplier of the modular structures must strictly follow the dimensions, specifications, and recommendations contained in the architectural design, comply with the relevant standards and specifications, and monitor the entire manufacturing and assembly process of the structures.

- Steel Structure buildings

  The Mine Workshop/Vehicle Wash, Reagent Storage, and Cyanide Storage will be steel structures enclosed with concrete blocks up to 2.0 m high and metal roofing above the top of masonry. The warehouses shall be supplied complete, with structures, closures, frames, electrical, plumbing, data, telephone, and air conditioning installations, fully finished and ready for use after installation of the furniture.

- Masonry Buildings

  The new electrical substations (Filtering, Crushing, and Process Plant) will be constructed using concrete cast on site and enclosed in concrete block masonry, with a metal roof.

- Existing Buildings

  Buildings to be renovated must follow the existing typology, as ambulatory and warehouse - made of masonry, and the dining room – a steel roof that will be enclosed with thermoacoustic panels and glass.

15.3.1 Accommodation

During construction. Contractors, including the EPCM contractor(s), will be responsible for providing their own accommodation, transportation, and meals. No construction camp is planned.

An operations camp will be constructed in proximity to the project site to ensure adequate housing infrastructure.

15.4 Mine Waste Facilities

In Era Dorada project, mining operations will generate considerable volumes for waste rock and tailings. This section presents the geotechnical structures that will be implemented for the disposal of these materials. Descriptive aspects of the areas that will contain the topsoil excavated from the foundation and the low-grade ore material are also presented, as well as the mass and volume balance for the waste rock and tailings generated.

In summary, two streams of tailings deposition will be used: one stream of tailings will be filtered and deposited back underground as paste backfill, while the other one will be filtered and deposited in engineered Dry Stack Tailings Facilities (DSTFs) on the surface. Most of the waste rock will be deposited underground as either cemented rock fill (CRF) or loose rock fill (LRF). Above the ground, engineered Waste Rock Dumps (WRDs) will be implemented for the disposal of the remaining volume.

1.2.1 Site Characterization

The Cerro Blanco mine is a planned underground gold and silver mine located in southeast Guatemala approximately 160 km by road from the capital, Guatemala City. It is located in the Jutiapa District, approximately 9 km west of the border with El Salvador. Elevation in the region ranges from 450 masl to 600 masl. The coordinates (JDS, 2017) of the proposed DSTF will range from 1,589,500 N, 210,500 E (northwest corner) to 1,585,000 N, 214,000 E (southeast corner), an approximate area of 15.25 km^2.

The climate and vegetation in the mine property are typical of a tropical dry forest environment (Golder, 2012; JDS, 2017). The project site is classified as Zona Oriental and its principal characteristics are a deficiency of rain for much of the year with high ambient daytime temperatures. Most of the vegetation in the area loses its foliage because of a lack of precipitation to support growth during the dry season.

Generally, the project occurs within a south-southwest trending bedrock ridge that extends from higher ground to the north, outward into the basin and flood plain deposits of the Rio Ostua. The elevation of the upper part of the ridge is over 600 masl. The elevation of the basin and flood plain deposits range from about 460 masl to 490 masl.

The west side of the ridge is flanked by a south-southeast trending perennial drainage called Rio Tancushapa. The east side of the ridge is flanked by a seasonal drainage called Quebrada El Tempisque which also trends to the south-southeast. These drainages join to the south-southeast and flow into the Rio Ostua.

The regional area is generally hilly to mountainous with broad flood plains formed by some of the larger streams and rivers. Three volcanoes are within sight of the project, as follows:

- Suchitan to the northwest.
- Ixtepeque to the north.
- Las Viboras to the southwest.

Climate records for Cerro Blanco are based on records from the Asunción Mita and Cerro Blanco Mine weather stations. Daily weather records for both stations were provided by Blueste. The records were reviewed to develop rainfall regressions, which in turn were used to develop design storms as part of stormwater management design and to evaluate typical day-to-day conditions which are presented in the Stormwater Management and Water Balance Report (Stantec, 2018). Using both weather stations, the climate record spans 48 years. The average daily temperature is 26°C, dry season is from November to April and the rainy season throughout the rest of the year (May to October). The site receives on average 1342 mm of annual precipitation, and it is also characterized by relatively an average pan evaporation of 2533 mm per annum. The annual average air humidity is 62%.

The major tectic features in the region, with Guatemala at the boundary of the North American and Caribbean Plates. Seismic and volcanic activity in the region is caused by plate movement, specifically subduction of the Cocos Plate beneath the Caribbean and North American Plates. The Jocotan-Chamelecon Fault System extends from the Caribbean Sea to the Mexican border and forms a left-lateral strike-slip boundary between the Caribbean and North American Plates (White, 1985; Villagran, et al., 1997) and is the closest known subduction zone fault to the project site. The Jocotan Fault Zone, the major fault zone closest to the mine, is postulated to be another major seismic source along with subduction zone events for development of seismic hazard analysis.

The results of probabilistic hazard assessment (PSHA) for the referent site conditions (very dense soil/ soft rock site with Vs=760 m/s, Class C) indicate peak ground accelerations (PGA) of 0.45g, 0.61g, 0.75g and 0.88g for the return periods of 1/475, 1/2,475, 1/5,000 and 1/10,000, respectively. Considering that the sites of the surface facilities are on top of 30m of stiff clays (Site Class D), the correction factor of 1.6/1.3 or 1.23 should be applied to the above values in the seismic-resistant design. Obviously, high seismicity is a major risk for the facility and for that reason, the conventional slurry facility is unfeasible for the site.

Other risks include subsidence and differential settlement in clay-rich zones with shallow groundwater, possible collapse of loose granular soils under saturation, and landslide potential due to hillside terrain and colluvial deposits. Additionally, localized faulting near the Ipala Graben demands conservative design measures and ongoing monitoring to ensure long-term stability.

The site was investigated by eight (No 8) drill holes and twenty-one (No 21) test pits. Most of the holes were drilled to 30 m depth and one borehole was drilled to 8m depth. All boreholes were terminated in clay, as bedrock was not encountered to 30m depth.

A total of 97 permeability tests, 97 Standard Penetration Tests (SPT) were performed, and several disturbed and undisturbed samples were taken for laboratory testing. Representative samples were subjected to a laboratory test program in the Fugro geotechnical lab in Houston, Texas, United States.

The executed drill holes revealed a subsurface profile comprised of alluvial, colluvial, and volcanic materials with varying degrees of cementation and plasticity. Underlying 0.3m to 0.5m thick topsoil is a heterogeneous material comprised of clayey and sandy gravels, and clay of medium and high plasticity. Based on SPT blow counts the clay was mostly in stiff (8-14) to very stiff (15-40) state of consistency with occasional layers of firm (4-8) and hard (>30) consistency. Groundwater was encountered approximately 10 meters below the surface.

Laboratory tests comprised the execution of the following tests for samples collected within the test pits and samples collected with Shelby samplers: Moisture Content, Granulometric Analysis, Atterberg Limits, Specific Gravity Gs, Standard Proctor, Triaxial CU, Bender Element, Cyclic Direct Shear, Permeability and 1-D Consolidation.

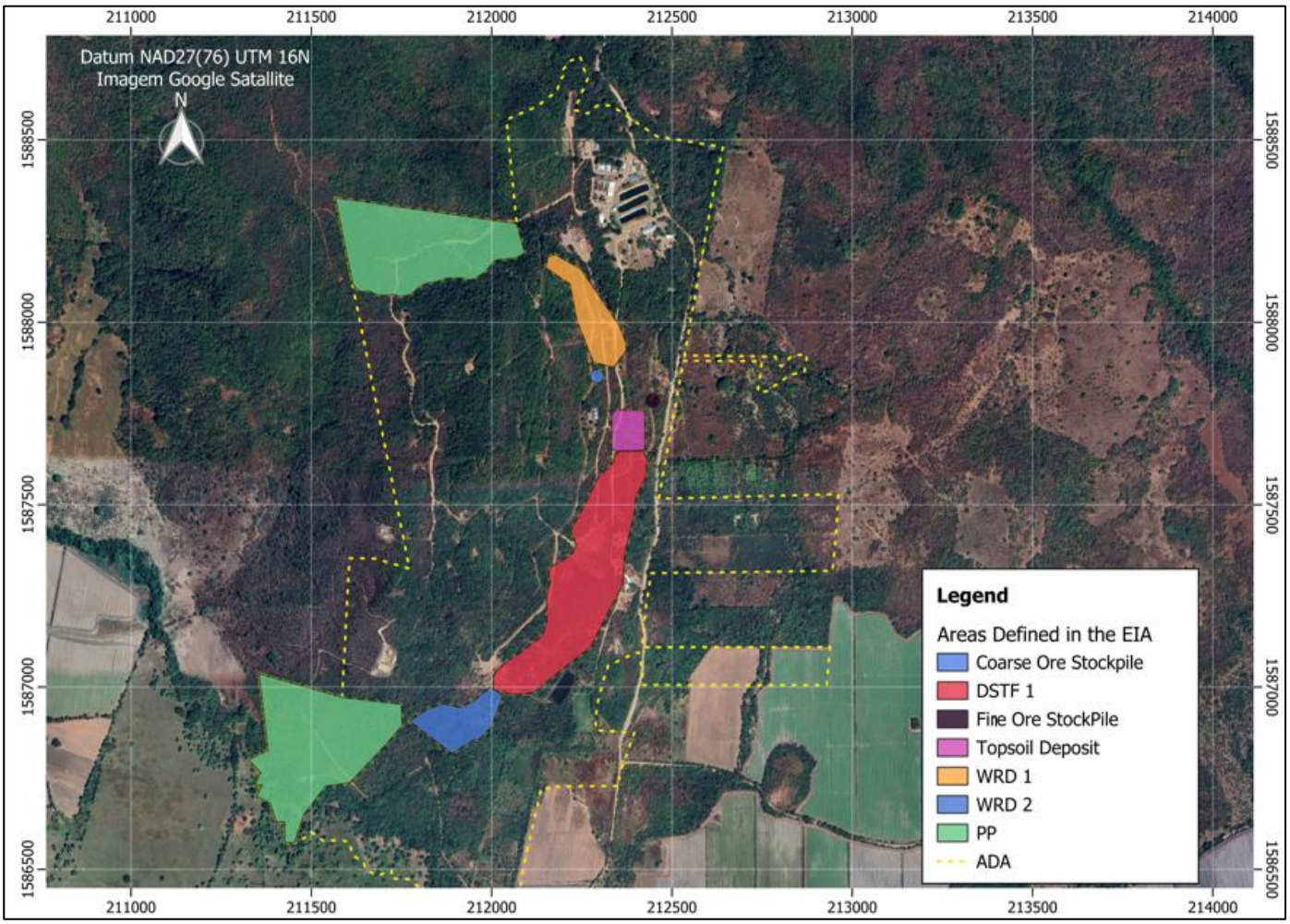
Based on the encountered heterogeneity of surface layers and environmental sensitivity of the project, Ausenco's assessment is that a campaign of supplementary geotechnical investigations would be necessary for proper geotechnical site characterization. Ausenco developed an investigation plan and a technical specification for that purpose. This campaign is currently being contracted by Aura.

15.4.1 Background

In 2007, an Environmental Impact Assessment Study (EIA 2007) for the Cerro Blanco Mining Project was published, and the company that owned the project at that time was Entre Mares de Guatemala. This study indicated specific areas for the construction of the following geotechnical structures (see Figure 15-3: Areas Present in EIA 2007**Erro! Fonte de referência não encontrada.**):

- "Escombrera Norte", named here as Waste Rock Dump 1 (WRD 1), with approximately 2.6 ha
- "Escombrera Sur", named here as Waste Rock Dump 2 (WRD 2), with approximately 2.3 ha
- "Depósito de Suelo", named here as Topsoil Stockpile, with approximately 1.0 ha
- "Pila de Mineral Grueso", named here as Coarse Ore Stockpile, with approximately 0.1 ha
- "Pila de Mineral Fino", named here as Fine Ore Stockpile, with approximately 0.1 ha
- "Depósito de Colas Secas": named here as Dry Stack Tailings Facility 1 (DSTF 1), with approximately 10.0 ha

Figure 15-3: Areas Present in EIA 2007



Notes: "PP" - Permanent Preservation Areas, "ADA" - Property Limits. Source: Ausenco, 2025.

In 2019, Stantec produced the Feasibility Study (FS) design of the facility for the Cerro Blanco Mining Project. The layout of the facility provided storage space for 2.4 Mt (with contingency to build for 3.0 Mt) of tailings, which corresponded with seven years of life-of-mine (LOM) at the annual production of 460,000 t. The mine waste facilities included:

- Dry stack tailings facility designed as a 26m high zoned centerline dry stack with subdrainage and underdrainage systems.
- Waste rock storage facility capable of storing 150,000 m^3 of waste rock material.
- Surface water management system for non-contact water.
- Ponds for contact water.

- Ore and top soil stockpiles.
- Water decantation system comprised of small diameter PVC penstocks draining into an underdrainage system.

The proposed general arrangement of the 2019 design was not in compliance with the areas licensed in EIA 2007 for the following pieces of infrastructure:

- Soil Stockpile
- Temporary Waste Rock Storage
- Dry Stack Tailings Facility (DSTF)
- Ore Stockpile.

In the period between 2019 and 2025 the project has been taken over by Aura Minerals, and the new Client produced an updated mine plan with 15% higher daily production (1,450 t/d) and an extended life of mine (from 7 to 16.5 years). The new plan increased the storage space requirement to 8.75 Mt of tailings: as 3.75 Mt of this material will be stored as mine backfill, the new, updated plans require the surface storage space for 5 Mt of tailings (an increase of ~67% from 2019 design).

Overall, under the current design plan, the facilities must include storage space for the materials and corresponding storage types presented in Table 15-1.

Table 15-1: Material that Requires Storage Space

Material	Quantity	Type of storage (permanent/temporary)
Tailings	5.00 Mt	Permanent
Topsoil	6,800 m^3	Temporary (varies)
Low-grade ore	100 kt	Temporary (18 years)
Waste rock	1.35 Mt	Permanent

For the updated Era Dorada project mine plan, the Client requested the development of a new geometric arrangement which would combine the limits already licensed by EIA 2007, with extensions required to provide additional storage, Figure 15-4. Based on the project's volume balance, the environmental permit for the new areas will need to be obtained

The introduction of the second facility significantly aids in achieving the required storage volume in a limited available space without introducing the additional cost due to self-supporting properties of the dry stack-filtered tailings facilities. The benefits from multiple impoundments can be considerable. In general, they are constructed sequentially, allowing for smaller initial capital expenditures and producing cash-flow benefits much the same as those realized for raised embankments. Multiple impoundments also offer considerable operational flexibility. Impoundment segments can be constructed either strictly on an as-needed basis or in advance of actual tailings storage requirements such as

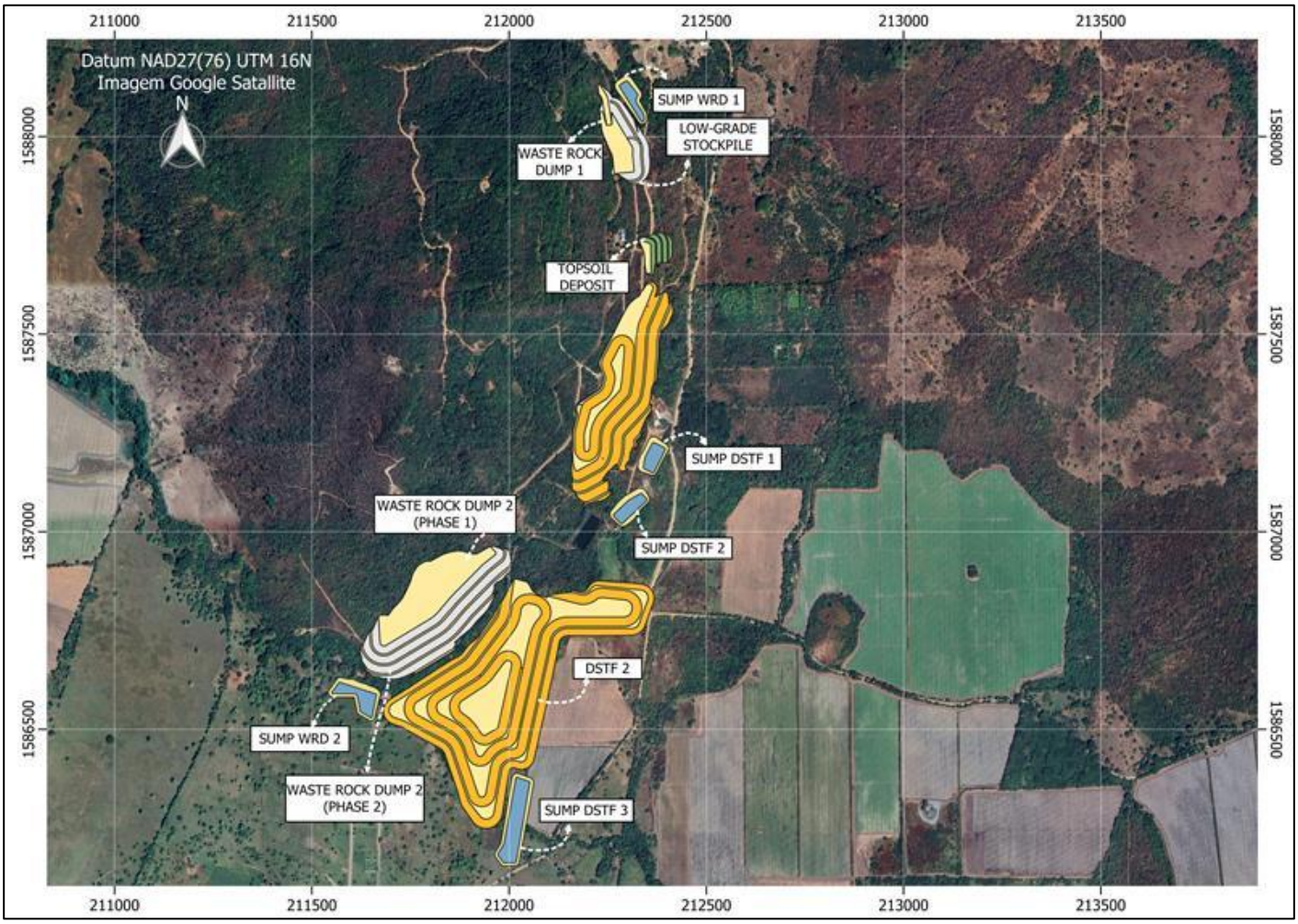
fill material or construction equipment become available. When more than one segment has been constructed, tailings deposition can be alternated between the impoundments to provide beneficial flexibility in impoundment operation.

Environmental benefits for multiple impoundments compared to single impoundments of equivalent capacity can also be major. Generally, multiple impoundments are constructed and filled sequentially. Thus, only a small portion of the eventual total impoundment area is covered with water at any given time. To the extent that seepage is directly proportional to the area over which flow occurs, seepage rates are considerably reduced. Also, the fact that reclamation can proceed concurrently with ongoing tailings disposal. Following filling of one multiple-impoundment segment, reclamation can begin as discharge is shifted to the next segment, thus minimizing the area disturbed at any one time and reducing problems related to blowing dust.

The updated general layout for all the mine waste structures proposed for this phase of the project now includes:

- Original Facilities
  - Topsoil Stockpile, with 1 ha footprint
  - Waste Rock Dump 1 (WRD 1), with 2.62 ha footprint
  - Waste Rock Dump 2 – Phase 1 (WRD 2 – Phase 1), with 2.34 ha footprint
  - Dry Stack Tailings Facility 1 (DSTF 1), with 10 ha footprint
- Additional Storage Facilities:
  - Waste Rock Dump 2 – Phase 2 (WRD 2 – Phase 2)
  - Dry Stack Tailings Facility 2 (DSTF 2)
  - Low-Grade Ore Stockpile.

Figure 15-4: New General Arrangement for The Proposed Geotechnical Structures



Source: Ausenco, 2025.

In summary, with the new arrangement WRD 2 (Phase 2), all the waste rock could be accommodated within the current property limits. For the tailings, however, even with the construction of DSTF 2, it will be necessary to construct a third DSTF (DSTF 3), and this demands new studies to access the necessity of land acquisition. This issue is further discussed and detailed in Section 15.4.7.

The topsoil resulting from planned earthworks will be disposed temporarily within a designated area previously authorized under EIA 2007. This area has an estimated capacity of 6,800 m³, according to the adopted geometry (slope height: 3 m; slope inclination: 4H:1V; berm width: 3 m).

The deposited topsoil must be used in restoration works. For the geotechnical facilities, topsoil will be used for the progressive construction of permanent caps while these stacks are operational and after completion of disposal.

The low-grade ore stockpile was not originally included in the 2007 EIA. For the current project, it has been incorporated as a co-disposal arrangement with waste rock within WRD 1. The facility is scheduled to receive low-grade ore during the initial years of mining operations (Year -1 and Year 0). The total storage capacity is 61,600 m³ (or 100 kt) for the adopted geometry (maximum slope height: 10 m; slope inclination: 2H:1V; berm width: 5 m), representing the accumulated ore volume to be temporarily stored prior to processing at the beneficiation plant.

The material placed in the stockpile is expected to remain stored for approximately 18 years (from Year --1 to Year 17). During this period, proper monitoring will be essential to minimize material losses. Other geotechnical requirements for this structure followed the same requirements for the waste rock facilities (see Section 15.4.3).

15.4.2 Tailings Disposal

Due to high seismicity, and relatively high risk of static and earthquake induced liquefaction, sidehill stacking of filtered tailings has been adopted in previous stages of the project and the prescriptive best available demonstrated control technology (BADCT) has been implemented. In addition to self-supporting property of the dry stacks, this mode of tailings disposal also minimizes the need for detailed consideration of site hydrogeology and vadose zone characteristics and other site factors. Therefore, design and operational components of Prescriptive BADCT are intended to be conservative.

Tailings generated by the gold extraction process will be dewatered by means of filter press equipment and then will be hauled to the two proposed DSTFs. According to the updated mine production schedule, the expected total mass of tailings to be produced is 8.75 Mt, between years Year 1 and Year 17. Considering it is planned to start on the second semester of Year 1, this period represents 16.5 years. Thus, the expected daily production of tailings is approximately 1450 t/d. The mining method of ore excavation requires 3.75 Mt of tailings as mine fill, which means that 5 Mt of the tailings will be stored in surface facilities (which over 16.5 years surmounts to 830 t/d for surface facilities only). For details about tailings production, see Section 15.4.4.

Filtered tailings will be placed in the DSTFs' areas by the haul trucks that will circulate over the previously compacted tailings. Fresh tailings will be hauled and spread to final location using dozers and spreaders. In order to increase the storage capacity, shear strength and reduce the compressibility, all of the material will be compacted to the minimum of 95% of maximum dry density (MDD) for Proctor Standard energy at Optimum Moisture Content (OMC).

For the compaction of filtered tailings, which typically behave like semi-cohesive or cohesive soils, a vibratory padfoot (or sheepsfoot) roller is generally considered the best choice

Bearing in mind the values of MDD (Section 15.4.2.1), the total expected volume of tailings should range from 3.1 Mm^3 to 3.2 Mm^3. The degree of compaction and the thickness of the layers to be compacted must be confirmed by the execution of field experimental embankments to guarantee dilative behavior for the material.

Considering that the moisture content of filtered tailings will be approximately 22.8% and that the OMC for tailings ranges from 16.8% to 20% in Proctor Standard energy the tailings will need to be dried prior to compaction in the DSTFs, which may present challenge during wet seasons. This drying process must be performed in specific areas, and this temporary stacking must be protected from direct rainfall incidence. Field density tests will be executed to evaluate

if compaction criteria were achieved. Moistening may be necessary during dry seasons. Whenever optimal moisture content is not achieved, tailings must be moistened or dried to achieve design criteria.

15.4.2.1 Tailings Characterization

15.4.2.1.1 Geomechanical characterization

In 2018, Stantec presented results of laboratory characterization tests for tailings' samples, as shown below:

- Specific gravity (Gs): 2.68 t/m³
- P_{80}: 50 µm
- Average fines content of 90.7% (passing #200 sieve)
- Maximum dry density to Standard Proctor: 1,696.4 kg/m³
- Liquidity limit: 27%
- Plasticity limit: 21%
- Optimum moisture content: 16.8%
- Permeability: 1.17 x 10-9 m/s
- Triaxial tests: ϕ' = 37.7°, c = 2.1 kPa (95% of the maximum dry density)

In 2022, NewFields Mining Design presented results of laboratory characterization tests for multiple tailings' samples, as described below.

- Specific gravity (Gs): 2.71 kN/m³
- P_{80}: 0.045 to 0.06 mm
- Maximum dry density to Standard Proctor: 1,625.0 kg/m³
- Optimum moisture content: 19% (18.0% to 20.0%)
- Permeability: 1 x 10-7 to 4 x 10-8 m/s
- Triaxial tests: ϕ' = 33.4°, c = 0 kPa

As can be seen, the tests performed on the tailings obtained reasonably different results, especially for the Atterberg limits, compaction (optimum moisture content), hydraulic conductivity, and triaxial tests. For this reason, it is considered essential to carry out a complementary campaign of characterization tests on the tailings to be disposed of in the DSTFs, especially after the commissioning of the beneficiation plant. The tests should be carried out in a systematic manner, at predetermined intervals and whenever there is a change in the characteristics of the tailings generated by the plant.

15.4.2.1.2 Geochemical Characterization

Before the DFS, samples of the tailings were subjected to geochemical testing and analysis. Two tests were performed prior to the DFS. WMC executed in 2006 a report summarizing Preliminary and Phase I material characterizations. Later, Maxxam Analytics submitted for analysis of Acid Base Accounting (ABA) for evaluation of potential for Acid Rock Drainage (ARD) and a Shake Flask Extraction (SFE), analysing potential of metal leaching. These datasets pre-date the DFS and should be confirmed as representative of the current mine plan and process flowsheet.

During the 2006 geochemistry investigation, waste rock and ore samples were selected from the five major ore veins: S1A, S1B, S2, S3, and North zone.

For the waste rock, based on ABA (using Sobek NP/AP and standard cut-off values) from the 315 waste rock samples, 35.7% of waste rock would be classified as PAG, 15.5% classified as uncertain, and 48.8% classified as NAG. The Sobek NP/AP ratios weighted to relative waste rock tonnage were 9.68 for NAG, 1.39 for uncertain, and 0.29 for PAG materials. Considering that ~51% of waste rock is PAG or uncertain, material segregation and placement controls are important inputs to the waste facility design.

Geochemical test results for ore samples, as reported by WMC, indicate that the ore body is not homogeneous regarding geochemical characteristics. There are portions of the ore that classify as PAG; however, on average, the ore samples tested were classified as NAG. If the samples analysed by WMC are representative of the entire ore body, the geochemical testing results indicate that the tailings would likely behave as NAG material with no generation of acidic drainage. Given the noted heterogeneity, confirmatory testing by ore domain and mining phase is warranted.

The ore samples tested by WMC would ultimately become tailings and the milling process would not likely change the original geochemical characteristics of the ore. WMC reported a range in NPR for ore with a low of 0.01 to a high of 1163 and a geometric mean of 3.4. Thus, on average for a skewed sample population, the tailings generated by processing the ore samples tested would most likely be NAG.

In the previous DSTF feasibility-level design and cost estimate report (Golder, 2012), there is description of geochemical testing for a single tailings sample (run in duplicate). The sample was evaluated using both static and kinetic test methods. Results indicate that the one sample of tailings classified as NAG with an abundance of carbonate (calcite) available to neutralize any acid generated by the limited amount of residual sulphide minerals in the tailings. Furthermore, no evidence was found of metal leaching either under aggressive conditions (NAG testing) or conditions more consistent with the ambient environment, as tested by the synthetic precipitation leaching procedure (SPLP). Thus, the single sample of tailings evaluated showed no potential for either development of acid drainage or leaching of metals. This result is based on one tailings sample and does not capture potential variability across the deposit or operating conditions.

Although the test results obtained so far indicate non-acid-generating (NAG) materials, it is considered essential to conduct new tests to characterize the materials that will be generated by the operation of the mine. To summarise the key ARD/ML risks and the corresponding controls and monitoring requirements for major facilities, a risk matrix based on probability and consequence is presented in Table 15-2. For this reason, specific foundation liner systems were planned for the mine waste piles, as well as specific drainage systems to collect contact and non-contact water for the waste rock and tailings piles.

Table 15-2: Risk Matrix (Risk Classified by Probability (P), Consequence (C), and Risk Level)

Contaminant Release Mechanism	Impacted Medium	P	C	Risk	Key Controls	Key Monitoring
Seepage from the waste rock pile	Groundwater	A	3	High	Base drain with a low-permeability layer; clay layer plus regrading (cover design); segregation of contact water; co-disposal with more alkaline material; treatment of base drain discharge	Upgradient/downgradient wells (pH, alkalinity, SO_4^{2-}, Al, Fe, Mn, Ni, NO_3^-); water level
Delayed seepage (at depth)	Groundwater	A	2	Moderate-High	Cover layers over reactive sectors; base drain with a low-permeability layer; surface drainage; water management; co-disposal with more alkaline material; treatment of base drain discharge	Downgradient wells (pH, SO_4^{2-}, NO_3^-, metals); piezometric monitoring
Runoff from storm events	Surface water	M	2	Moderate	Low-permeability layer at the base plus regarding (cover design); dedicated channels/ponds; surface protection (soil/vegetation)	Upgradient/downgradient points with flow; pH, SO_4^{2-}, Al, Fe, Mn, Ni
Seepage from the tailings dry stack	Groundwater	A	3	High	Low-permeability layer at the base plus regrading (cover design); base drain; segregation of contact water; treatment of base drain discharge	Upgradient/downgradient wells (pH, alkalinity, SO_4^{2-}, Al, Fe, Mn, Ni, NO_3^-); water level
Seepage (unlined base)	Groundwater	A	3	High	Low-permeability layer plus base drainage; stockpile effluent containment pond (treatment before discharge)	Nearby wells; pH, alkalinity/acidity, SO_4^{2-}, metals
Contact-water runoff	Surface water	M	2	Moderate	Segregation / dedicated collectors; low-permeability subgrade	Upgradient/downgradient points; pH, SO_4^{2-}, metals

Notes: Probability (P): Low (B), Medium (M), High (A); Consequence (C): Minor (1), Moderate (2), Major (3), considering magnitude/time of standard exceedance and spatial extent; Classification: B1/B2 = Low; M2 = Moderate; A2/M3 = Moderate-High; A3 = High. Source: Hidrogeo, 2025.

The updated program should include representative tailings composites aligned with the current flowsheet, and kinetic testing where needed to assess lag-to-acidity. For this reason, specific foundation liner systems were planned for the mine waste piles, as well as specific drainage systems to collect contact and non-contact water for the waste rock and tailings piles. These controls are consistent with a conservative design approach given geological variability and ARD uncertainty.

15.4.2.2 DSTFs Siting and Foundation Characterization

Runoff from the watershed upstream of the facilities will be managed using swales diversion channels or ditches. Alternatively, the facilities' storage capacity must account for potential run-on volumes. This requirement must be explicitly documented on a design basis.

15.4.2.3 Design Basis (Design Criteria)

The site characteristics and climatic considerations of the DSTFs design and the operational assumptions adopted for the DSTF 1 and DSTF 2 are presented in Table 15-3, Table 15-5, Table 15-6 and Table 15-6.

Table 15-3: Site Characterization

Site Characteristics - Climatic Considerations	
Climate and Vegetation Type	Tropical Dry Forest Environment
Average Daily Temperature	26 °C
Maximum Temperature**	41 °C
Minimum Temperature*	10 °C
Dry Season	November - April
Wet Season	May - October
Elevation	Approx. 500 masl
Annual Average Rainfall	1342 mm
Annual Average Pan Evaporation	2533 mm
Annual Average Humidity	62%

- Technical/legal requirements
  - DSTFs were designed in compliance with GISTM and CDA requirements, as well as Guatemalan regulatory requirements. The DSTFs will meet stability, water management and closure criteria that align with these regulations.
  - DSTFs are designed for a seismic event with a return period of 1 in 2,475 years during operation and 1 in 10,000 years at closure, with Peak Ground Accelerations (PGA) of 0.61 g and 0.88 g (MCE), respectively. The adopted PGA values were based on the seismic hazard assessment (SHA) for the Cerro Blanco mine project located in Jutiapa, Guatemala (Terrapro, 2020).
  - DSTFs are designed for a 1 in 2475 or 1 in 10,000 years return period for flood events.
  - Peripheral channels are designed for a 1 in 1,000 years flood event and will be implemented to divert surface runoff that would otherwise reach the piles.
  - According to GISTM criteria, the DSTFs have a high consequence classification.

Table 15-4: Stability Analysis Criteria

Condition	Criteria	Source
Seismicity/Earthquake Load	RP = 1/2,475 y (Operation) RP = 1/10,000 y/ MCE (Passive-Closure)	Global Industry Standard on Tailings Management, 2020 Canadian Dam Association, Dam Safety Guidelines, 2007
Static Factor of Safety (FoS)	1.5	Canadian Dam Association, Dam Safety Guidelines, 2007 US Army Corps of Engineers, 2003
Post-earthquake (FoS)	1.2	Canadian Dam Association, Dam Safety Guidelines, 2007

- Capacity/design life
  - DSTF 1 is designed to store 498,489 m^3 or approximately 780,000 t of tailings. DSTF 2 is designed to store 2,159,136 m^3 or approximately 3,390,000 t of tailings. A remaining quantity of tailings (456,032 m^3 or approximately 716,000 t) will possibly require a new area to be disposed of, outside the current limits of the property.
  - An average tailings dry density of 1.57 t/m^3 was considered, as a reference for 95% of maximum dry density and dilative behaviour (non-liquefiable). The in-place density of the tailings will need to be monitored continuously throughout the operation to adjust the LOM and the final capacity of the DSTFs.
  - The design life for DSTF 1 is about 4 years (between 2027 and 2030) and about fourteen years (between 2030 and 2043) for DSTF 2. The DSTFs operating life, annual tailings production and underground volume for backfill are presented in detail in Section 15.4.5.
- Construction Aspects
  - The construction of faciliteis will involve the controlled placement of the materials to form engineered embankments. The structures will be built in accordance with geotechnical design criteria to ensure long-term stability, proper drainage, and environmental compliance.
  - Foundation preparation: the foundation will be properly treated, with the removal of the surface layer of topsoil and any low-quality materials that may be found and are critical for DSTFs stability. A complementary geotechnical investigation campaign is being contracted.
  - Tailings will be hauled by trucks from the plant to the DSTFs facilities and properly spread (lift thickness of 30 cm during dry seasons, and 15 to 20 cm during wet seasons, to be confirmed with field experimental embankments).
  - Tailings will be mechanically compacted with sheepsfoot rollers for fine-grained tailings in 95% of maximum dry density. It must be compacted on the “dry side” of the critical state line to guarantee dilative behaviour. Technological field control for degree of compaction and moisture content will be required.
  - DSTFs are designed as 100% structurally zoned. A starter waste rock embankment is planned and will require 90,888 m3 of the ROM (run-of-mine) material. The starter embankment will serve as a structural outer shell. A protective filter between the waste rock and the tailings will be required, consisting of transition layers of gravel and sand, and geotextile.

- DSTFs' surfaces must maintain a minimum positive grade of 0.5% to ensure adequate surface runoff, thereby reducing water infiltration into the piles and minimizing slope erosion. This runoff will be collected by a properly dimensioned surface drainage system, which will carry the water flows downstream to ponds. Diversion ditches will intercept upstream runoff to collect the water and direct it into non-contact ponds.
- The final design surface of the piles will be compacted by compactor rollers (without compaction control) to minimize water infiltration and promote rapid surface runoff ("surface sealing").
- Construction quality assessment will be performed. Topographic surveys will be conducted to verify lift geometry and slope angles. Field density tests will be performed to assess compaction quality.

- Geochemistry/foundation lining
  - For conservative purposes, tailings have been classified as PAG. Therefore, the facilities will be lined and will have infiltration ponds to accommodate contact water stemming from the infiltration of direct precipitation.
  - The liner systems for the DSTFs will be composed of a 1.5 mm double-side textured HDPE geomembrane protected for a 275 g/m² geotextile, and a subsurface drainage system is predicted to indicate possible leakage through the geomembranes. An additional layer composed of sandy material will be place over the lines system to function as protection layer for the liner system.
- Physical Stability
  - Static / Long-term: normal loading conditions with effective friction angles assigned to all materials. Target Factor of Safety (FoS): 1.50.
  - Earthquake: pseudo-static, peak strength parameters, normal pore pressures, kh=0.5*PGA for return periods of 1 in 2,475 years during operation and 1 in 10,000 years at closure. Target FoS: 1.00.
  - Post-Earthquake: post-earthquake loading conditions using residual undrained shear strength for the tailings and foundation (overburden), effective friction angles for the waste rock and deep foundation, and no kh. Target FoS: 1.20. A deformation analysis may be appropriate for future design stages to confirm containment integrity under seismic loading.
- Internal drainage
  - Robust internal drainage systems, consisting of a rockfill drainage core and gravel and sand transitions, will intercept and convey seepage water on the foundation, preventing the formation of water table inside the geotechnical structures. Proper filter protection with geotextiles is planned.
- Monitoring and maintenance
  - Geotechnical monitoring of the structures will be carried out from the early stages of the construction and will last for the post closure phase of the project. Instrumentation like piezometers, inclinometers, settlement plates, and tiltmeters will be installed to monitor internal conditions of the structures and long-term performance.
  - Maintenance must be carried out whenever necessary, to guarantee that design requirements are met.

15.4.2.4 Design Features

Design features of the DSTFs include foundation preparation, foundation liner system, internal drainage system, waste rock starter confining embankment as a structural outer shell, and water management ponds to collect contact and non-contact water. Ditches will be constructed around the perimeters of the DSTFs to divert the non-contact water from upper catchment areas and minimize the direct contact water catchment area.

Tailings will be hauled by trucks from the plant to the DSTFs facilities, properly spread (lift thickness of 30 cm) and compacted in optimal moisture content regarding proctor standard energy, with sheep-foot rollers for fine-grained tailings. Technological field control for degree of compaction and moisture content will be required.

A waste rock compacted shell was projected for the DSTF. The final footprints of the DSTFs are shown in Figure 15-5 and Figure 15-6.

Figure 15-5: General Arrangement of the DSTF

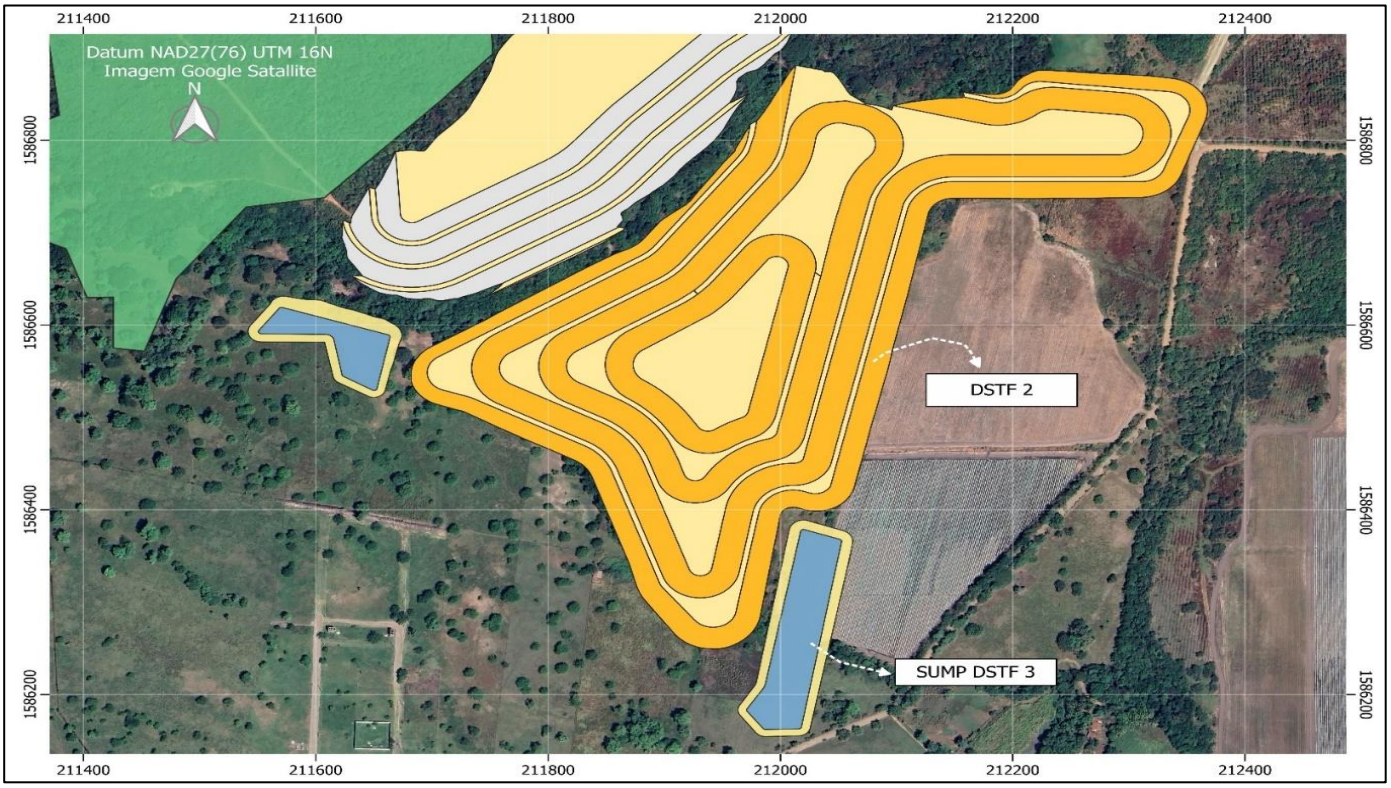


Source: Ausenco, 2025.

Table 15-5: Main Characteristics of the DSTF

DSTF	
Volume	498,489 m³
Tailings Deposition Methodology	Filtered, truck haul, spread by dozers, compacted by sheepsfoot rollers
Tailings Geochemistry	Non-acid generating (NAG)
Water Management Basis	Separated Non-Contact and Contact Water Management Systems
Maximum Height	30.4 m
Upstream Slope	3H:1V
Downstream Slope	3H:1V
Starter Embankment	
Volume	90,888 m³
Maximum Height	14.2 m
Upstream Slope	3H:1V
Downstream Slope	3H:1V

Figure 15-6: General Arrangement of the New DSTF



Source: Ausenco, 2025.

Table 15-6: Main Characteristics of the New DSTF

New DSTF	
Fill Volume	2,159,136 m³
Tailings Deposition Methodology	Filtered, truck haul, spread by dozers, compacted by sheepsfoot rollers
Tailings Geochemistry	Non-acid generating (NAG)
Water Management Basis	Separated Non-Contact and Contact Water Management Systems
Maximum Height	30.2 m
Upstream Slope	3H:1V
Downstream Slope	3H:1V
Volume of operational pond	51,270 m³

15.4.2.5 Mechanical Stability

Stability analyses were performed for the DSTFs considering static long term, earthquake and post-earthquake scenarios. Mechanical stability was analysed using the 2D limit-equilibrium analysis software Rocscience Slide2 (version 9.040) for the following scenarios and parameters presented in Table 15-7Table 15-7.

Table 15-7: Geotechnical parameters adopted for the stability analyses of Dry Stack Tailings Facilities (DSTFs)

Material	Unit Weight (kN/m³)	Static		Earthquake		Post Earthquake		
		ϕ (deg.)	c (kPa)	ϕ (deg.)	c (kPa)	ϕ (deg.)	c (kPa)	$Su/\sigma'_{v\ resid.}$
Tailings	18	30	0	23	0	18,2	0	-
Waste Rock	20	Leps (1970) lower bound						
Foundation (Overburden)	19	25	50	20	40	-	-	0,125
Foundation (Tuff)	20	Hoek Brown Envelope; UCS=1MPa; GSI=50; mi=14; D=0						

The following tables provide the results obtained from the analyses, and the corresponding figures can be found in Figure 15-7 through Figure 15-10. The target FoS was achieved for long-term and post-earthquake scenarios. However, scenarios considering earthquakes with return periods of 2,475 and 10,000 years did not meet the target FoS. Therefore, the permanent displacements caused by such seismic events were estimated (see Section 15.4.4).

Table 15-8: Stability Analyses Performed for Dry Stack Tailings Facility 1 (DSTF 1)

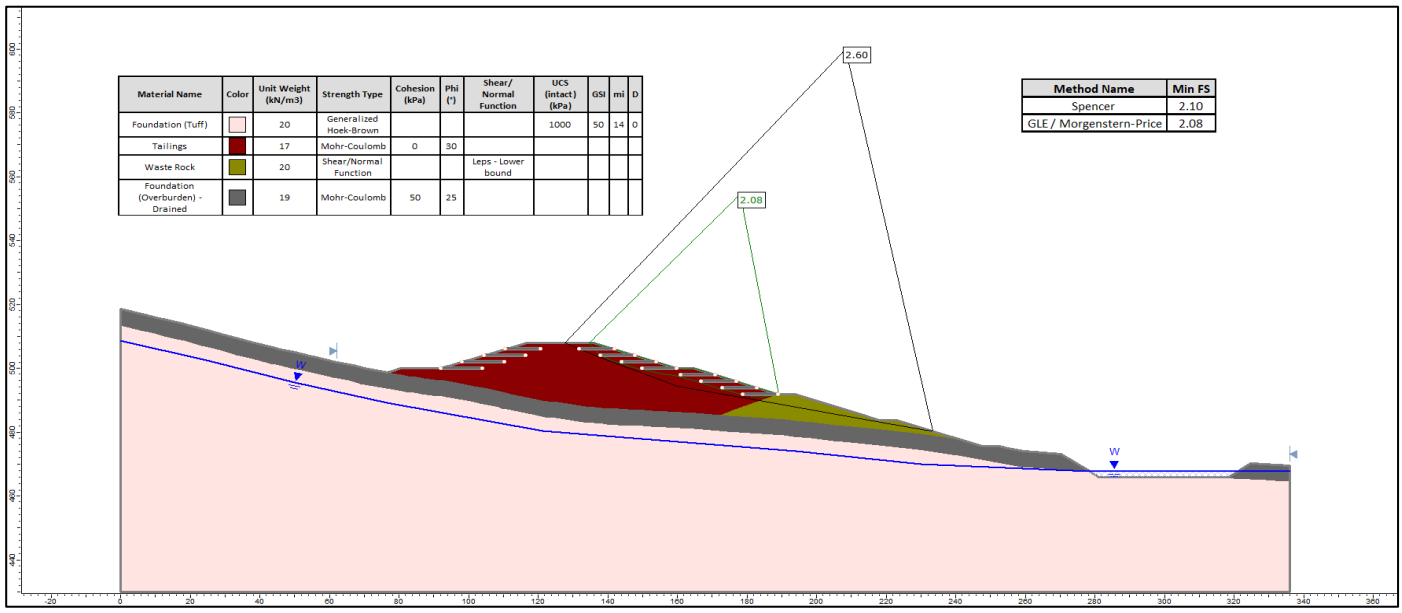
Scenario (Load Case)	Target FoS	Calculated FoS (local)	Calculated FoS (global)
Long term	1.50	2.08	2.60
Earthquake (2,475 years)	1.00	<1.00	<1.00
Earthquake (10,000 years)	1.00	<1.00	<1.00
Post-Earthquake	1.20	1.20	1.97

Table 15-9: Stability Analyses Performed for Dry Stack Tailings Facility 2 (DSTF 2)

Scenario (Load Case)	Target FoS	Calculated FoS (local)	Calculated FoS (global)
Long term	1.50	2.07	2.15
Earthquake (2,475 years)	1.00	<1.00	<1.00
Earthquake (10,000 years)	1.00	<1.00	<1.00
Post-Earthquake	1.20	1.20	1.25

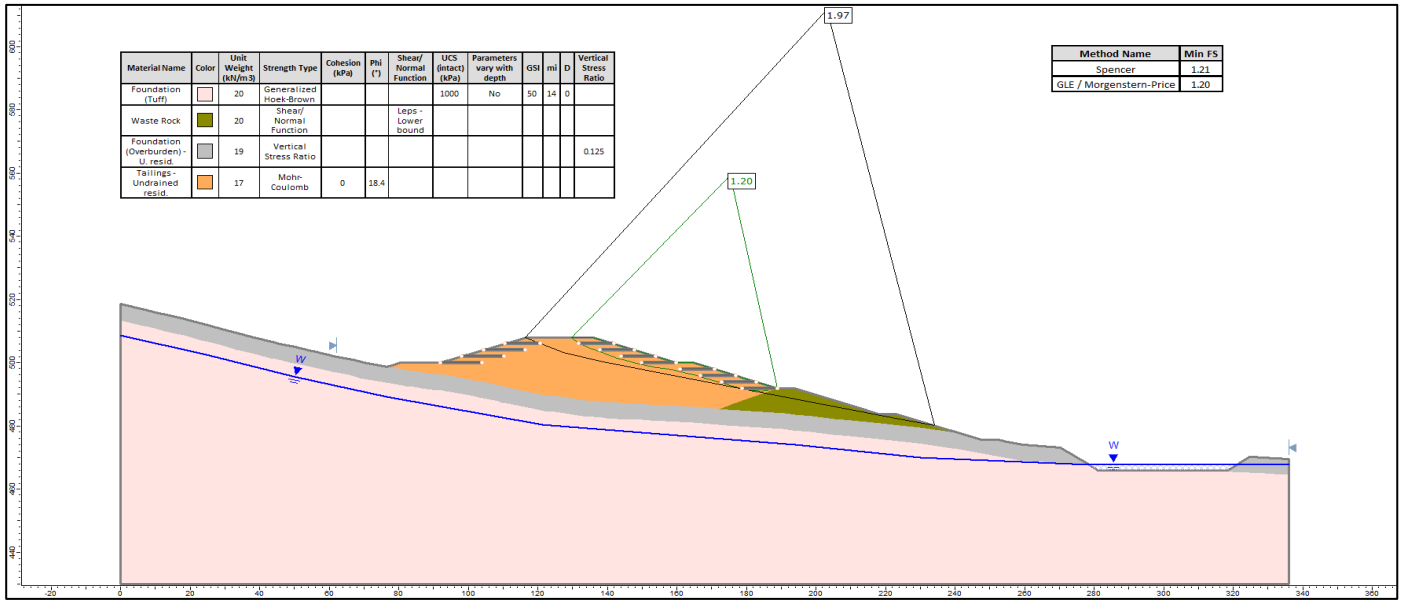
There is significant uncertainty regarding the behaviour and strength of the tailings, since the beneficiation plant is not constructed. Therefore, a 20% reduction in post-earthquake strength was applied, based on an assumed undrained shear strength (ϕ' = 18,2° and c=0 kPa). To achieve the minimum required FoS in the post-earthquake scenario under these assumptions, it would be necessary to implement a reinforcement solution for the tailings mass. The application of geogrid layers with an ultimate tensile strength of 110 kN/m and a length of 15.0 m, spaced every 2 m in height along the stack, is proposed. Consequently, it is recommended that the tailings investigation campaign be expanded in the next phase to confirm the assumptions adopted.

Figure 15-7: Stability Analysis for DSTF 1, Section A-A', Long Term Condition



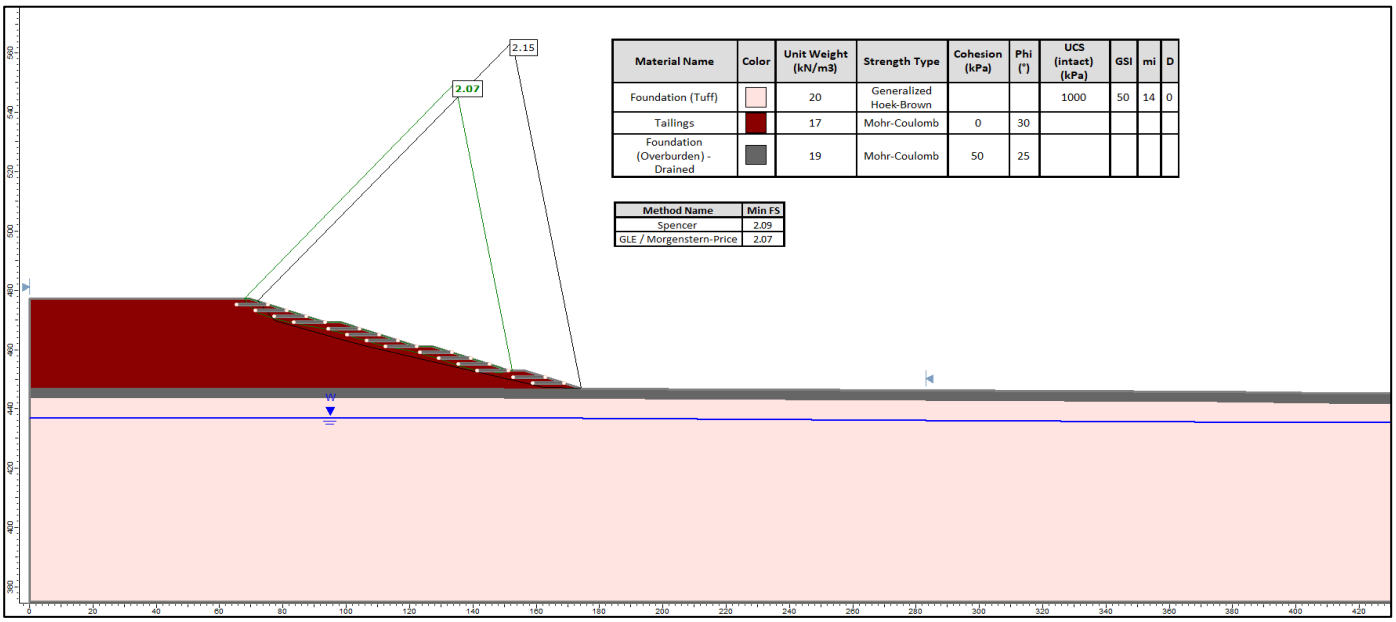
Source: Ausenco, 2025.

Figure 15-8: Stability Analysis for DSTF 1, Section A-A', Post-Earthquake Condition



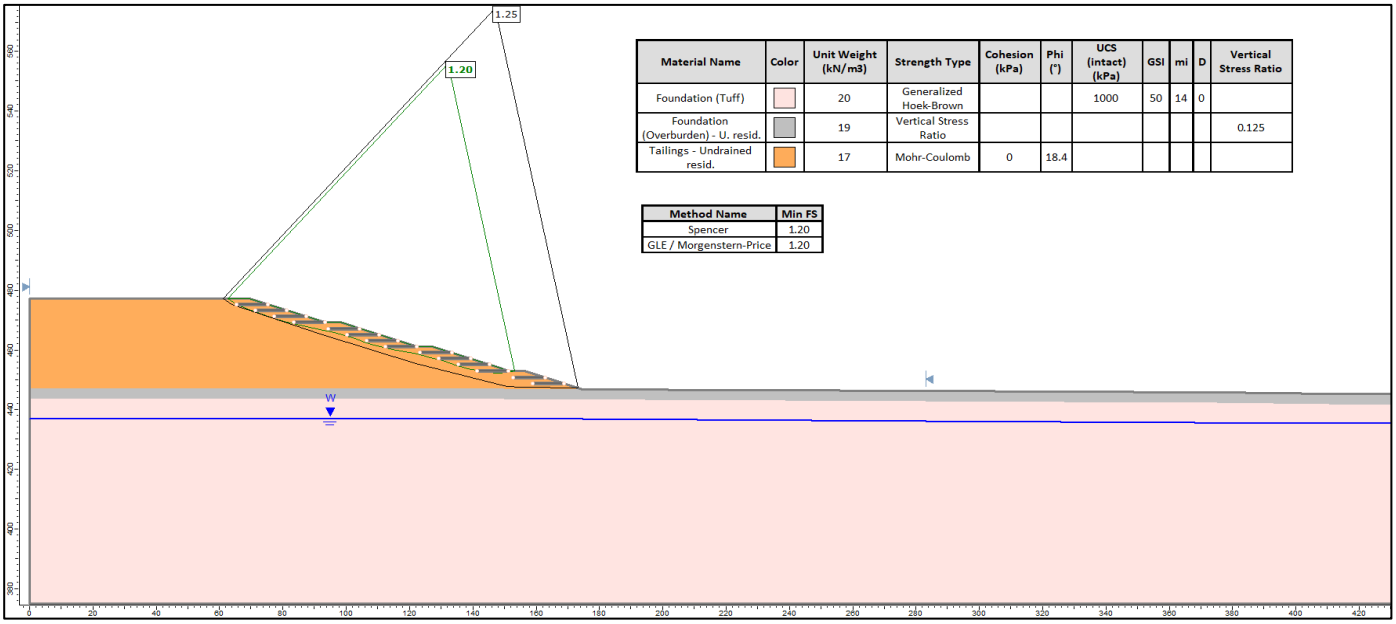
Source: Ausenco, 2025.

Figure 15-9: Stability Analysis for DSTF 2, Section I-I', Long Term Condition



Source: Ausenco, 2025.

Figure 15-10: Stability Analysis for DSTF 2, Section I-I', Post-Earthquake Condition



Source: Ausenco, 2025.

15.4.2.6 Water Management

The design of the surface drainage system for the DSTFs was developed using highly rigorous safety criteria, given the criticality of the structures and their location in an area characterized by high regional rainfall and recorded seismic activity. Accordingly, different return periods were adopted depending on the type of channel and the magnitude of the flow conveyed:

High-flow channels, responsible for conveying the largest discharges (such as down chutes and peripheral channels), were designed for a Return Period (RP) of 1000 years, ensuring robustness even under extreme hydrological scenarios.

Low-flow channels, including berm drains and crest channels, were designed for an RP of 500 years, ensuring adequate performance during severe events.

The surface water management infrastructure was developed with the objective of minimizing and, whenever possible, segregating precipitation and runoff that meet potentially contaminant-bearing materials from the natural surface waters of the watershed. Areas designated for filtered tailings storage were classified as contact water zones, whereas runoff from natural gullies and hillslopes was classified as non-contact water.

For the DSTFs, six (6) peripheral drainage channels were designed to intercept and convey contact water to the collection basins. Additionally, an upstream diversion (crown) channel was installed to segregate natural drainage (non-contact water) from runoff originating from the structure, directing clean water directly to the El Tempisque River without requiring treatment.

The drainage channels will be lined with geomembrane when constructed in riprap, and in sections with steep slopes, concrete lining will be used to mitigate erosion and provide hydraulic stability during the design event. Vehicle crossings include reinforced concrete slabs for light traffic and buried culverts at approximately 1 m depth for off-road haul trucks.

At the outfall locations, energy dissipation structures, lined with riprap or concrete, will be installed to prevent localized erosion at the base of the containment basins.

All accumulated water will be directed to the Effluent Treatment Plant (ETP) due to the potential for acid drainage generation and elevated arsenic concentrations, after which it will be discharged into the Óstua River in compliance with applicable environmental standards.

15.4.2.7 Closure Plan

The closure plan for the DSTFs starts concurrently with material disposal, once proposed geometries for the facilities are conservative. This design criteria reduces risks and allows for the minimization of erosion and for the reduction of costs associated with re-grading activities. As soon as the operational berm is completed (i.e., it reaches design geometry), a 15 cm thick layer of clay material must be placed over it, and then vegetation must be implemented for erosion minimization.

15.4.3 Waste Rock Facilities

15.4.3.1 Waste Rock Characterization

In the documents consulted, no characterization of the waste rock material was found. For this DFS phase of the project, waste rock was considered as a rockfill.

15.4.3.2 WRD Siting and Foundation Characterization

As previously mentioned, as part of the feasibility study at Cerro Blanco, Stantec (2018) supervised a geotechnical site investigation program with geotechnical Drill holes, permeability tests, Standard Penetration Tests (SPT), sampling, and excavation of test pits.

From this investigation program, most of the Drill holes were executed around the WRD 1 site (DH-08, DH-09 and DH-11). These boreholes revealed a subsurface profile composed of transported sediments (alluvial and colluvial deposits), and a pedological profile derived from the alteration/weathering of volcanic rocks (tuffs and breccias). There were no investigations executed nearby the WRD 2 site. However, given the geological context of the region, these structures are expected to exhibit similar stratigraphic arrangements.

15.4.3.3 Design Basis (Design Criteria)

The site characteristics and climatic considerations of the WRDs design and the operational assumptions adopted for the WRD 1, WRD 2 (Phase 1) and WRD 2 (Phase 2) are presented in Table 15-12, Table 15-13, Table 15-14Table 15-13.

Table 15-10: Site Characteristics

Site Characteristics - Climatic Considerations	
Climate and Vegetation Type	Tropical Dry Forest Environment
Average Daily Temperature	26 °C
Maximum Temperature**	41 °C
Minimum Temperature*	10 °C
Dry Season	November - April
Wet Season	May - October
Elevation	Approx. 500 masl
Annual Average Rainfall	1342 mm
Annual Average Pan Evaporation	2533 mm
Annual Average Humidity	62%

- Technical/legal requirements
  - WRDs were designed in compliance with CDA requirements, as well as Guatemalan regulatory requirements. The WRDs will meet stability, water management and closure criteria that align with these regulations.
  - WRDs are designed for a seismic event with a return period of 1 in 2,475 years during operation and 1 in 10,000 years at closure, with Peak Ground Accelerations (PGA) of 0.61g and 0.88g (MCE), respectively. The adopted PGA values were based on the seismic hazard assessment (SHA) for the Cerro Blanco mine project located in Jutiapa, Guatemala (Terrapro, 2020).
  - WRDs are designed for a 1 in 2475 or 1 in 10,000 years return period for flood events.
  - Peripheral channels are designed for a 1 in 500 years flood event and will be implemented to divert surface runoff that would otherwise reach the piles.
  - According to CDA criteria, the WRDs have a high consequence classification.

Table 15-11: Stability Analysis Criteria

Condition	Criteria	Source
Seismicity/Earthquake Load	RP = 1/2,475 y (Operation) RP = 1/10,000 y/ MCE (Passive -Closure)	Canadian Dam Association, Dam Safety Guidelines, 2007
Static Factor of Safety (FoS)	1.5	Canadian Dam Association, Dam Safety Guidelines, 2007 USACE - US Army Corps of Engineers, 2003
Post-earthquake FoS	1.2	Canadian Dam Association, Dam Safety Guidelines, 2007

- Capacity/design life
  - WRD 1 is designed to store 145,773 m^3 or approximately 233,000 t of waste rock. WRD 2 (Phase 1) is designed to store 67,107 m^3 or approximately 107,000 t of waste rock. A remaining quantity of waste rock (567,041 m^3 or approximately 907,000 t) can be disposed of in the proposed WRD 2 (Phase 2). This facility is located within the current property but will require a new license to be operated.
  - An average waste rock dry density of 1.60 t/m^3 was considered, and the waste rock was considered as a free draining material.
  - The design life for WRD 1 is about 2 years (Year -1 and Year 1), about 1,5 year (Year 1 and approximately first semester of Year 1) for WRD 2 (Phase 1), and about 6 years (Year 1 to Year 6) for WRD 2 (Phase 2). The WRDs operating life, annual waste rock production and underground volume for backfill are presented in detail in Section15.4.5.

- Construction aspects
  - The construction of the facilities will involve the controlled placement of the materials to form engineered embankments. The structures will be built in accordance with the geotechnical design criteria to ensure long term stability, proper drainage, and environmental compliance.
  - Foundation preparation: the foundation will be properly treated, with the removal of the surface layer of topsoil and any low-quality materials that may be found and are critical for WRDs stability. A complementary geotechnical investigation campaign is being contracted.
  - Waste rock will be hauled by trucks from the underground activities to the WRDs facilities and properly spread (lift thickness initially defined as 30 cm, but it depends on the maximum rockfill particle size).
  - Waste rock will be mechanically compacted by bulldozers used to spread the material. Technological field control for degree of compaction and moisture content will not be required for WRDs, but this criterion must be evaluated depending on the characteristics of the waste rock, since this type of material is usually highly heterogeneous.
  - WRDs' surfaces must maintain a minimum positive grade of 0.5% to ensure adequate surface runoff, thereby reducing water infiltration into the piles and minimizing slope erosion. This runoff will be collected by a properly dimensioned surface drainage system, which will carry the water flows downstream to ponds. Diversion ditches will intercept upstream runoff to collect the water and direct it into non-contact ponds.
  - The final design surface of the piles will be compacted by compactor rollers (without compaction control) to minimize water infiltration and promote rapid surface runoff ("surface sealing").
  - Construction quality assurance will be performed. Topographic surveys will be conducted to verify lift geometry and slope angles. In situ density tests may be required.
- Geochemistry / foundation lining
  - In previous studies, waste rock material was classified as PAG. Therefore, the facilities will be lined and will have infiltration ponds to accommodate contact water stemming from the infiltration of direct precipitation.
  - The liner systems for the WRDs will be composed of a 1.5 mm double-side textured HDPE geomembrane protected for a 275g/m² geotextile, and a subsurface drainage system is predicted to indicate possible leakage through the geomembranes.
- Physical stability
  - Static / Long-term: normal loading conditions with effective friction angles assigned to all materials. Target Factor of Safety (FoS): 1.50.
  - Earthquake: pseudo-static, peak strength parameters, normal pore pressures, kh=0.5*PGA for return periods of 1 in 2,475 years during operation and 1 in 10,000 years at closure. Target FoS: 1.00.
  - Post-Earthquake: post-earthquake loading conditions using residual undrained shear strength for the foundation (overburden), effective friction angles assigned to the other materials. Target FoS: 1.20.

- Internal drainage
    - Robust internal drainage systems, consisting of a rockfill drainage core and gravel and sand transitions, will intercept and convey seepage water on the foundation, preventing the formation of water table inside the geotechnical structures. Proper filter protection with geotextiles is planned.
- Monitoring and maintenance
    - Geotechnical monitoring of the structures will be carried out from the early stages of the construction and will last for the post closure phase of the project. Instrumentation like piezometers, inclinometers, settlement plates, and tiltmeters will be installed to monitor internal conditions of the structures and long-term performance.
    - Maintenance must be carried out whenever necessary, to guarantee that design requirements are met.

15.4.3.4 Design Features

Design features of the WRDs include foundation preparation, foundation liner system, and water management ponds to collect contact and non-contact water. Ditches will be constructed around the perimeters of the WRDs to divert the non-contact water from upper catchment areas and minimize the direct contact water catchment area.

Waste rock will be hauled by trucks from the underground mine to the WRDs facilities, properly spread (lift thickness initially defined as 30 cm, but it depends on the maximum diameter of rockfill) and compacted by passes of the dozer. Technological field control will be conducted using in-place density tests.

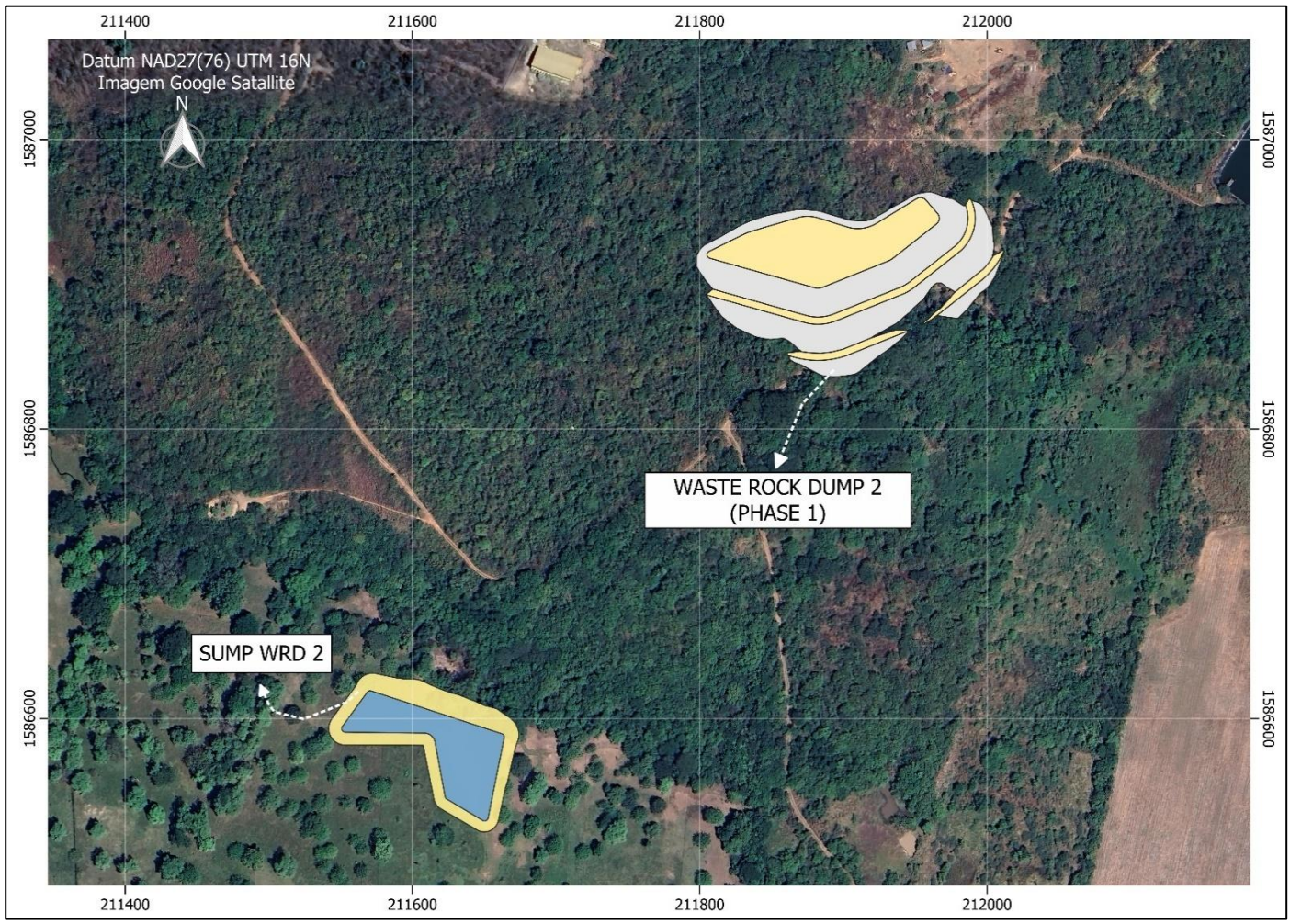
The final footprints of the WRDs are shown in Figure 15-11 through Figure 15-13.

Figure 15-11: General Arrangement of the WRD North



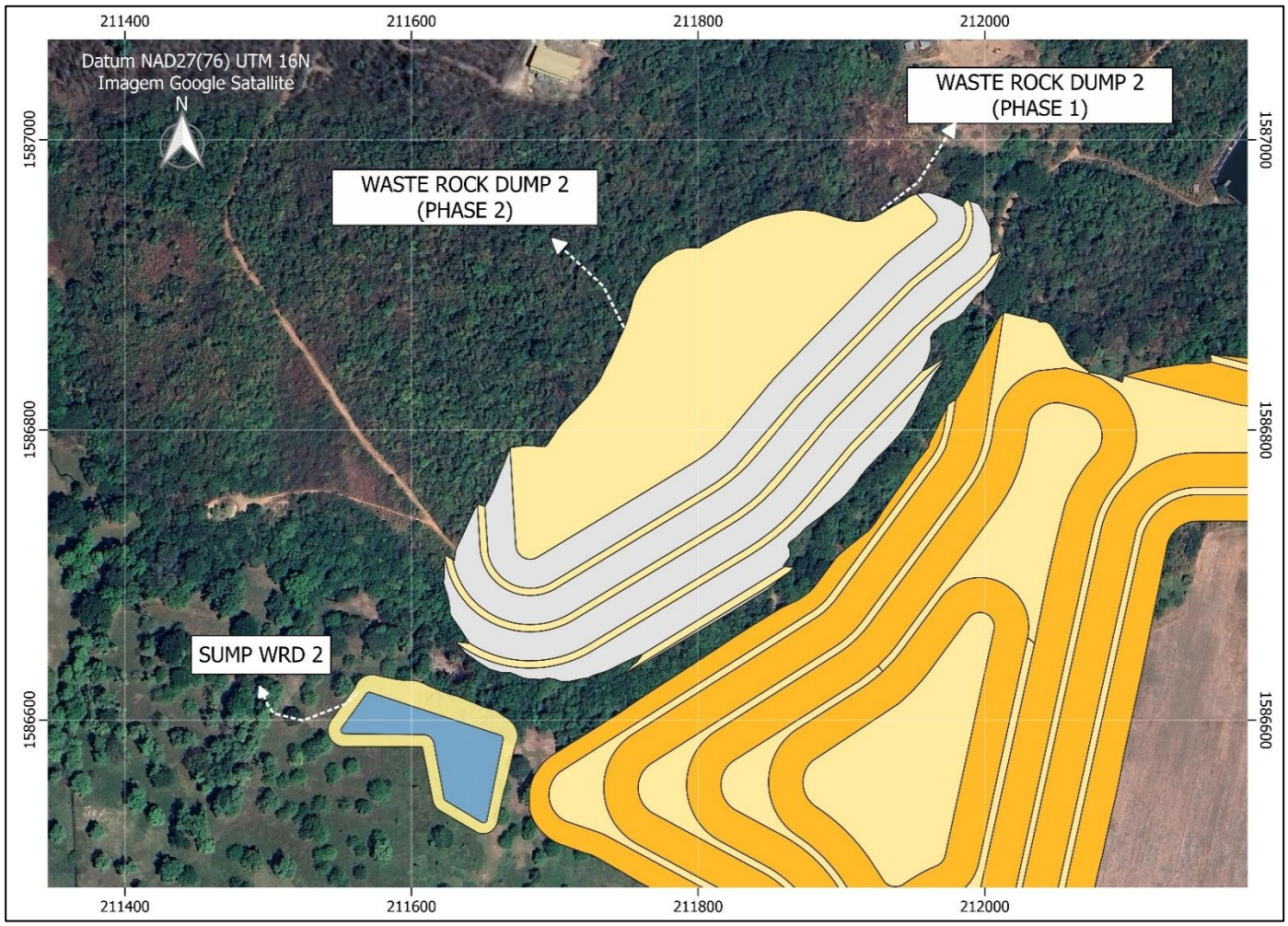
Source: Ausenco, 2025.

Figure 15-12: General Arrangement of the WRD South



Source: Ausenco, 2025.

Figure 15-13: General Arrangement of the New WRD



Source: Ausenco, 2025.

Table 15-12: Main Characteristics of WRD North

Waste Rock Dump North	
Fill Volume	145,773 m³
Waste Rock Deposition Methodology	Truck haul, spread by dozers, compacted by weight of dozer itself
Waste Rock Geochemistry	Potentially acid generating (PAG)
Water Management Basis	Separated Non-Contact and Contact Water Management Systems
Maximum Height	18.7 m
Upstream Slope	2H:1V
Downstream Slope	2H:1V

Table 15-13: Main Characteristics of WRD South

Waste Rock Dump South	
Fill Volume	67,107 m³
Waste Rock Deposition Methodology	Truck haul, spread by dozers, compacted by weight of dozer itself
Waste Rock Geochemistry	Potentially acid generating (PAG)
Water Management Basis	Separated Non-Contact and Contact Water Management Systems
Maximum Height	27.0 m
Upstream Slope	2H:1V
Downstream Slope	2H:1V

Table 15-14: Main Characteristics of the New WRD

New Waste Rock Dump	
Fill Volume	616,325 m³
Waste Rock Deposition Methodology	Truck haul, spread by dozers, compacted by weight of dozer itself
Waste Rock Geochemistry	Potentially acid generating (PAG)
Water Management Basis	Separated Non-Contact and Contact Water Management Systems
Maximum Height	41.3 m
Upstream Slope	2H:1V
Downstream Slope	2H:1V

15.4.3.5 Mechanical Stability

Stability analyses were performed for the WRDs considering static, earthquake and post-earthquake scenarios. Mechanical stability was analysed using the 2D limit-equilibrium analysis software Rocscience Slide2 (version 9.040) for the following scenarios and parameters presented in Table 15-15.

Table 15-15: Geotechnical Parameters Adopted for the Stability Analyses of the Waste Rock Dumps (WRDs)

Material	Unit Weight (kN/m^3)	Drained Shear Strength		Undrained Shear Strength		Post Liquefaction Strength
		ϕ' (deg.)	c' (kPa)	ϕ' (deg.)	c (kPa)	s_u/ σ'_v resid.
Waste Rock	20	Leps (1970) lower bound				
Foundation (Overburden)	19	25	50	20	40	0,125
Foundation (Tuff)	20	Hoek Brown Envelope; UCS=1MPa; GSI=50; mi=14; D=0				

The following tables provide the results obtained for the analyses, and the corresponding figures can be found in Figure 15-14 through Figure 15-22. The target FoS was achieved for all loading scenarios. The critical slip surfaces for both loading scenarios were shallow, passing through the embankment fill.

For the post-earthquake stability analysis of the WRD North, foundation treatment was necessary. This might be the condition for other parts of the WRDs and DSTFs, but only with the complementary investigation campaign this questions will be answered.

Table 15-16: Stability Analyses Performed for the Waste Rock Dump 1 (WRD 1)

Scenario (Load Case)	Target FoS	Calculated FoS (local)	Calculated FoS (global)
Static	1.50	1.89	2.21
Earthquake (2,475 years)	1.00	1.02	1.13
Earthquake (10,000 years)	1.00	<1.00	<1.00
Post-Earthquake	1.20	1.89	2.23

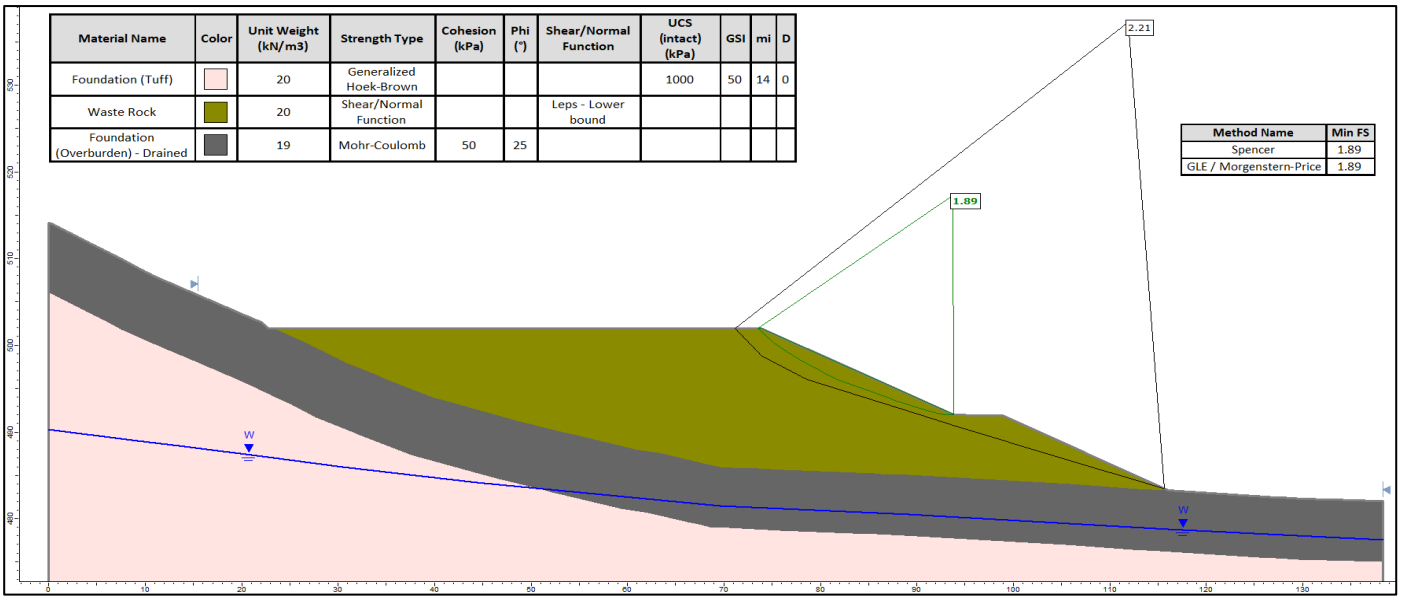
Table 15-17: Stability Analyses Performed for the Waste Rock Dump 2 (WRD 2)

Scenario (Load Case)	Target FoS	Calculated FoS (local)	Calculated FoS (global)
Static	1.50	1.85	2.44
Earthquake (2,475 years)	1.00	1.01	1.13
Earthquake (10,000 years)	1.00	<1.00	<1.00
Post-Earthquake	1.20	1.85	2.44

Table 15-18: Stability Analyses Performed for the New Waste Rock 2 (WRD 2 (Phase 2))

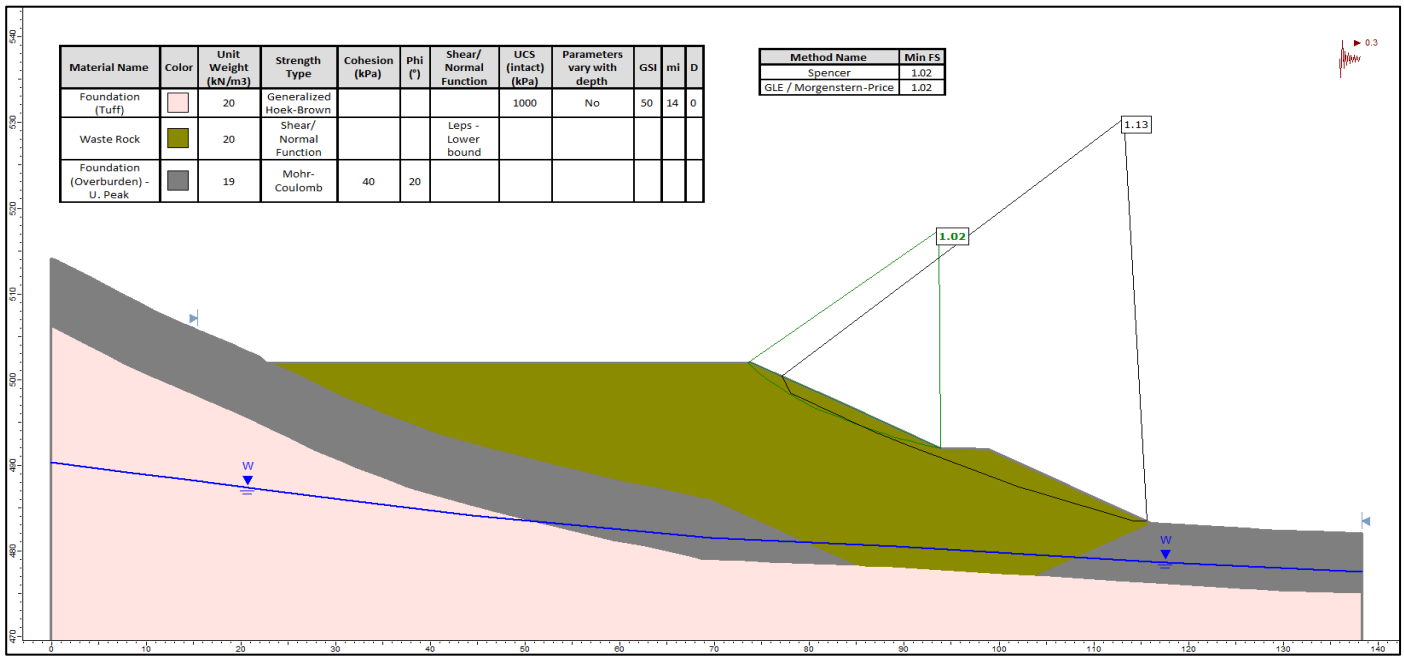
Scenario (Load Case)	Target FoS	Calculated FoS (local)	Calculated FoS (global)
Static	1.50	1.89	2.00
Earthquake (2,475 years)	1.00	1.02	1.07
Earthquake (10,000 years)	1.00	<1.00	<1.00
Post-Earthquake	1.20	1.89	2.02

Figure 15-14: Stability Analysis for WRD 1, Section D-D', Long Term Condition



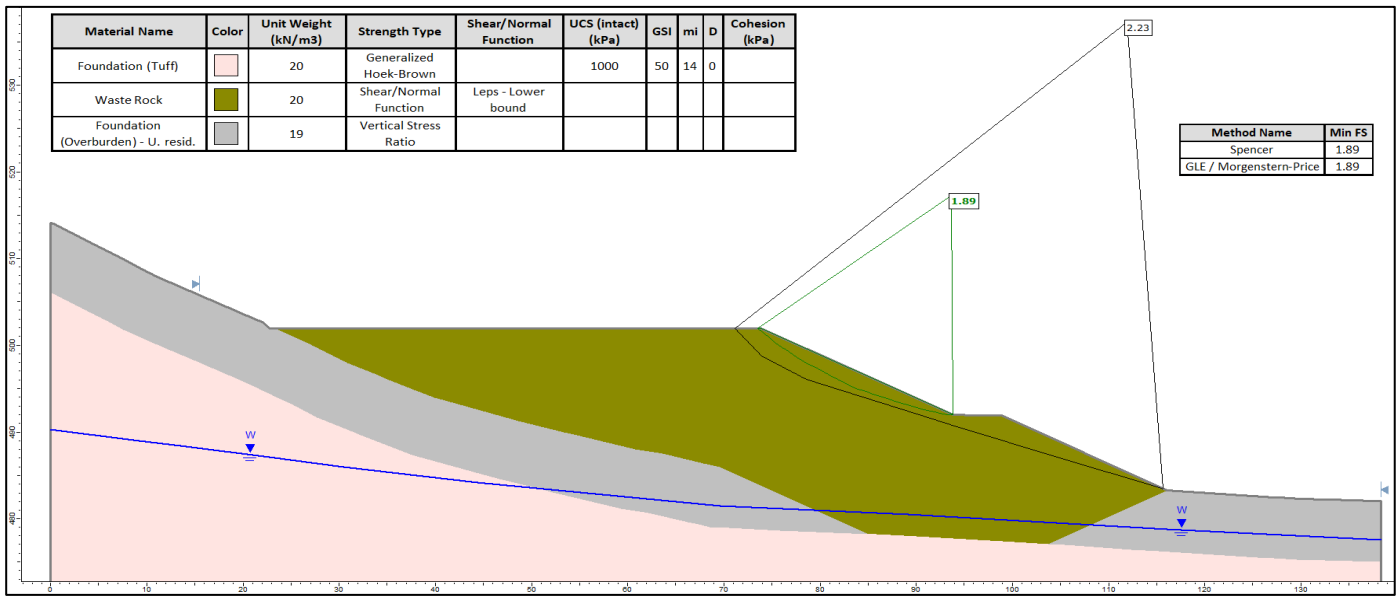
Source: Ausenco, 2025.

Figure 15-15: Stability Analysis for WRD 1, Section D-D', Earthquake 2,475 Years Condition



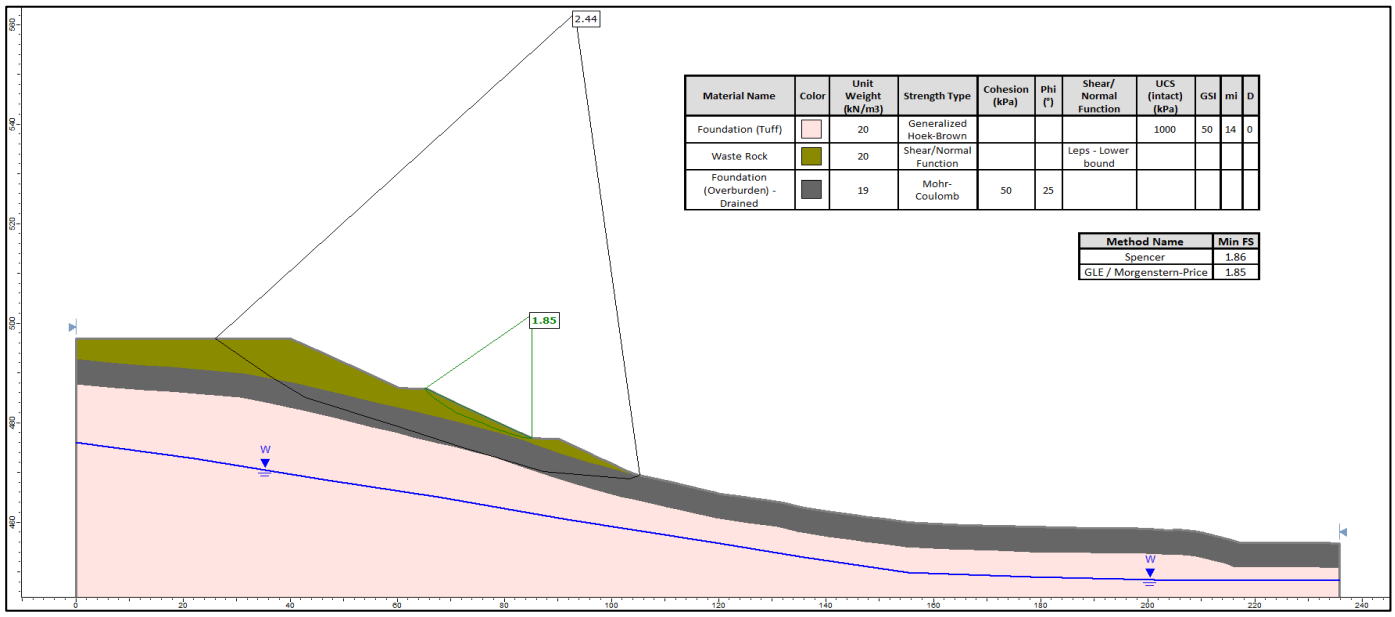
Source: Ausenco, 2025.

Figure 15-16: Stability Analysis for WRD 1, Section D-D', Post-Earthquake Condition



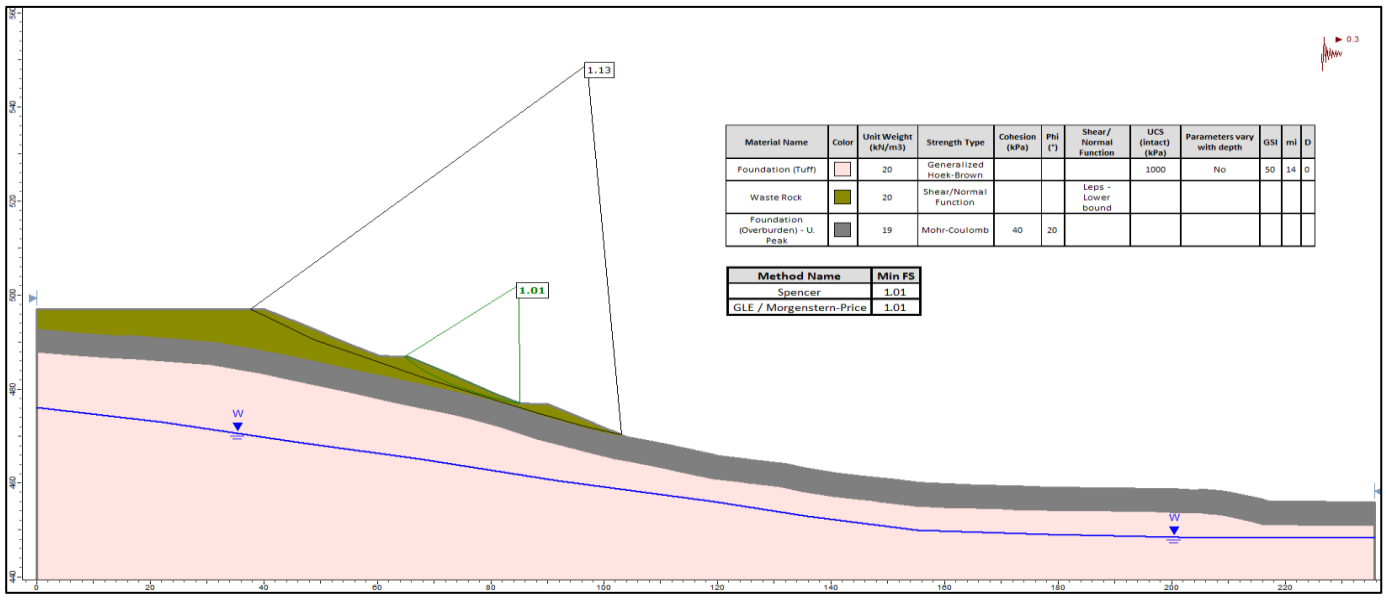
Source: Ausenco, 2025.

Figure 15-17: Stability Analysis for WRD 2, Section J-J', Long Term Condition



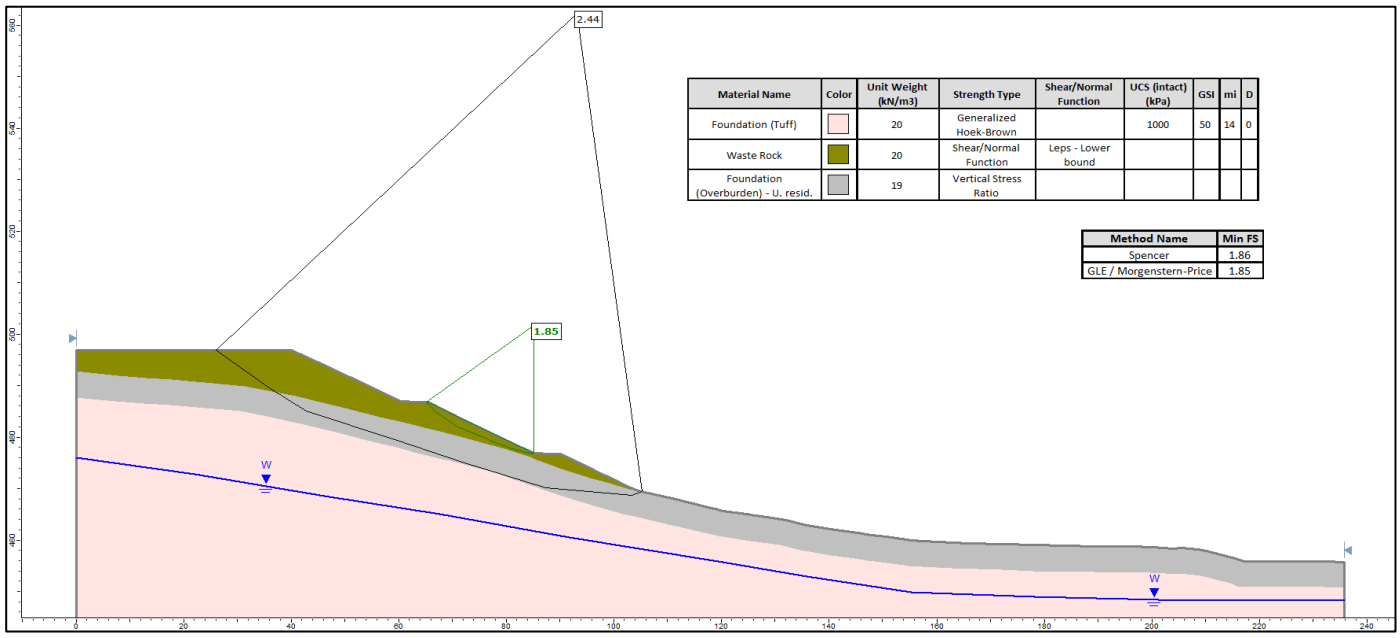
Source: Ausenco, 2025.

Figure 15-18: Stability Analysis for WRD 2, Section J-J', Earthquake 2,475 Years Condition



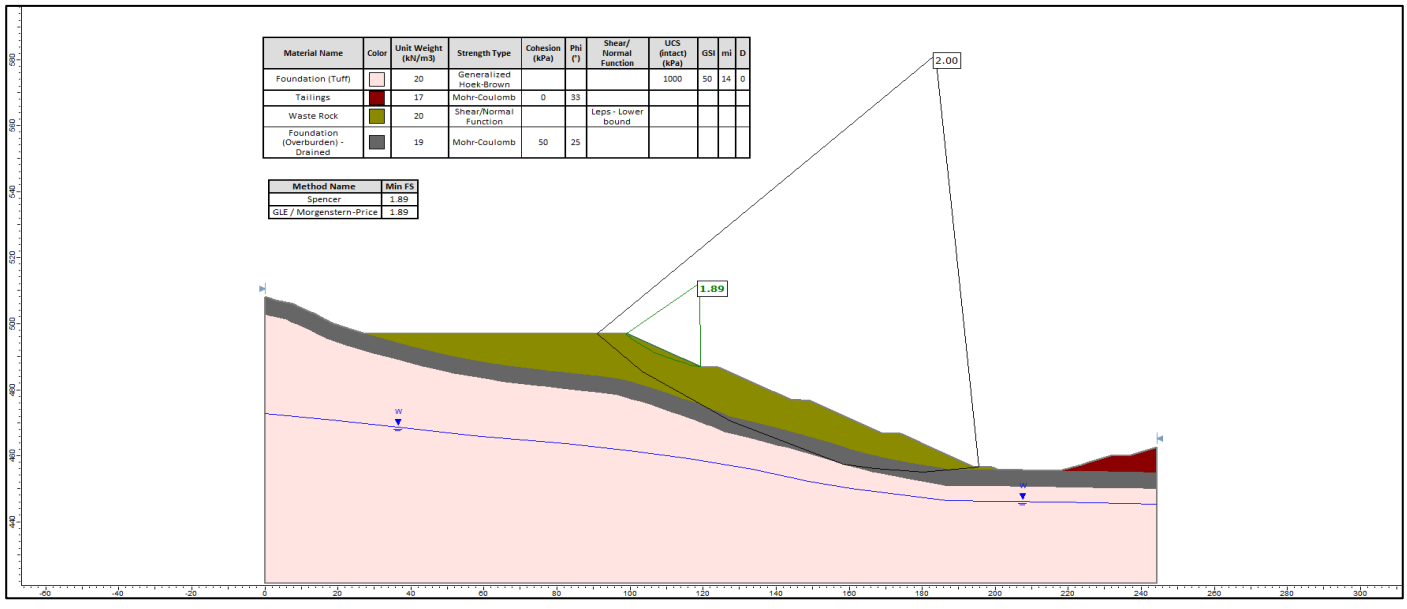
Source: Ausenco, 2025.

Figure 15-19: Stability Analysis for WRD 2, Section J-J', Post-Earthquake Condition



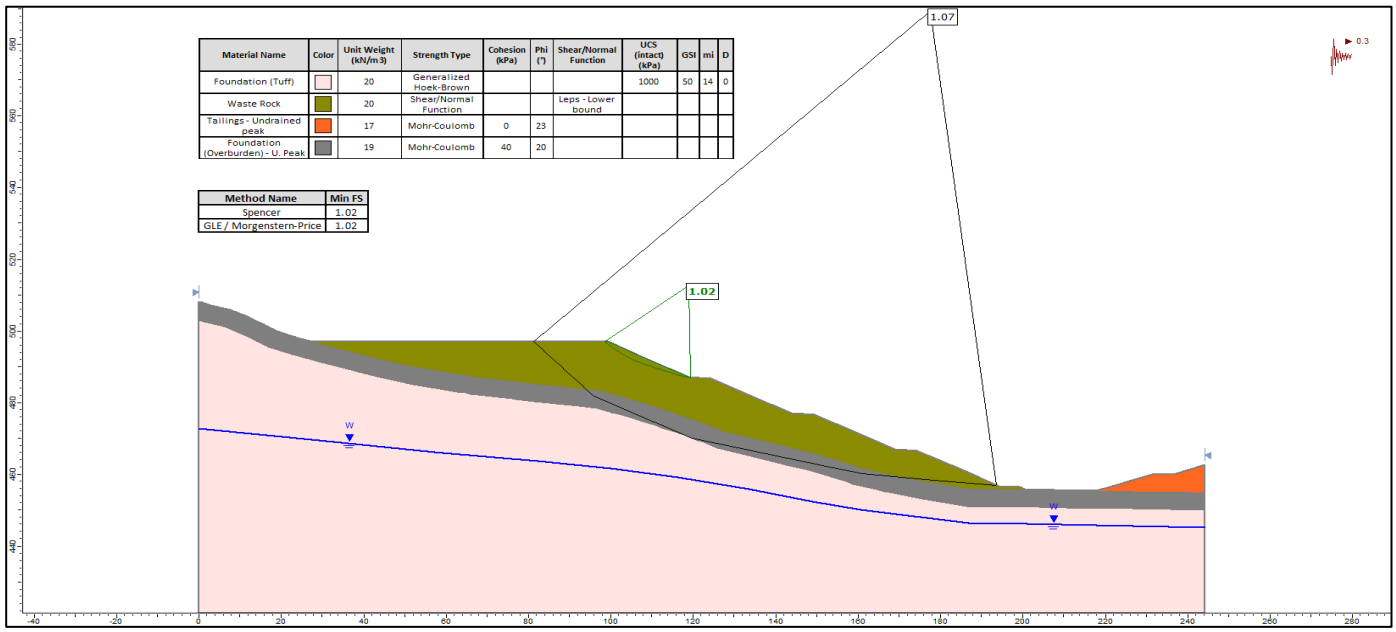
Source: Ausenco, 2025.

Figure 15-20: Stability Analysis for WRD 2 (Phase 2), Section F-F', Long Term Condition



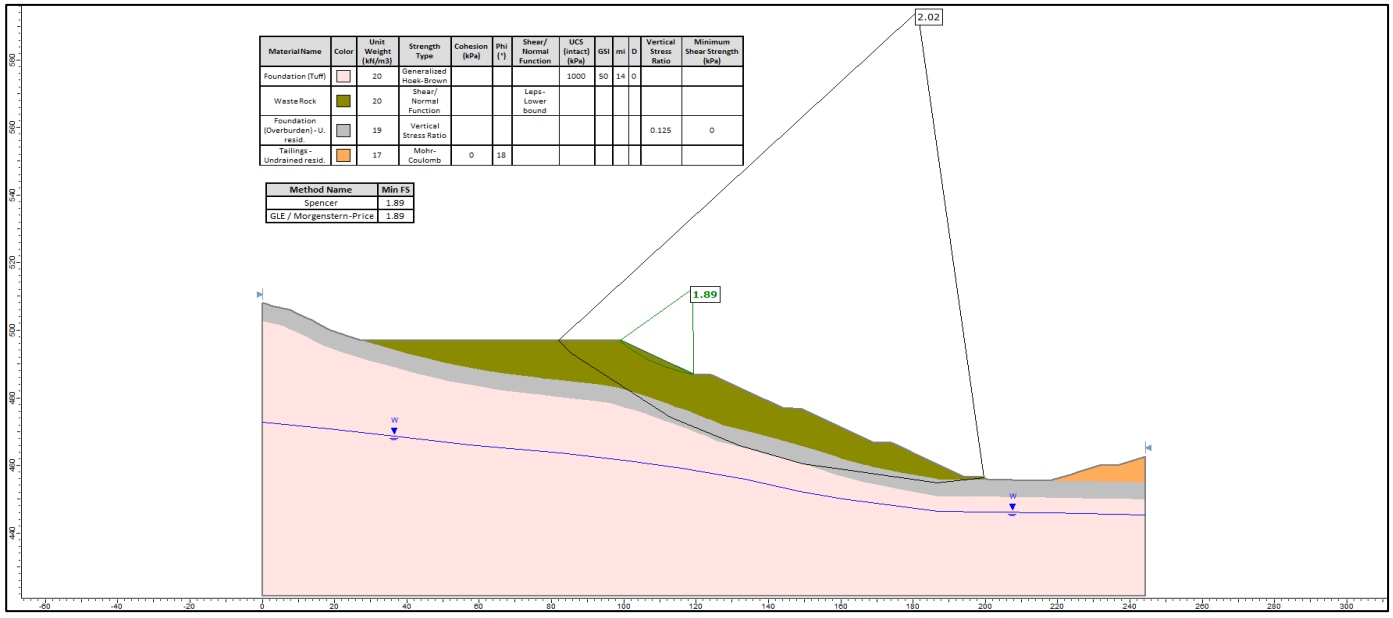
Source: Ausenco, 2025.

Figure 15-21: Stability Analysis for WRD 2 (Phase 2), Section F-F', Earthquake 2,475 Years Condition



Source: Ausenco, 2025.

Figure 15-22: Stability Analysis for WRD 2 (Phase 2), Section F-F', Post-Earthquake Condition



Source: Ausenco, 2025.

1.2.1.1 Water Management

For the Waste Rock Dumps, the surface drainage system was designed considering their lower relative criticality compared to the tailings piles, while still applying design criteria consistent with the regional rainfall regime. The following return periods (RP) were adopted:

- High-flow channels (down chutes and peripheral channels): RP of 500 years.
- Low-flow channels (berm drains and crest channels): RP of 100 year.

These parameters ensure the hydraulic stability of the system, preventing erosive processes and ensuring adequate runoff conveyance throughout the mine's operational life.

As with the tailings piles, the drainage system was designed to segregate contact water from natural runoff. For the Waste Rock Dumps, four (4) peripheral channels were designed to capture and direct runoff to the contact water collection basins.

The crown diversion channel also applies to the waste rock dumps, intercepting natural runoff from adjacent terrain and ensuring that non-contact water is discharged directly at the appropriate release point without the need for treatment.

The channels follow the same construction standards: geomembrane or concrete lining depending on slope, energy dissipation structures for erosion control, and adequate crossing structures for both light vehicles and off-road haul trucks.

Accumulated water will subsequently be sent to the Effluent Treatment Plant (ETP) due to the potential contamination associated with contact with the waste rock.

15.4.3.6 Closure Plan

The closure plan for the WRDs starts concurrently with material disposal, once proposed geometries for the facilities are conservative. This design criteria reduces risks and allows for the minimization of erosion and for the reduction of costs associated with re-grading activities. As soon as the operational berm is completed (i.e., it reaches design geometry), a 15 cm thick layer of clay material must be placed over it, and then vegetation must be implemented for erosion minimization.

15.4.4 Assessment of Displacements and Runouts Distances

15.4.4.1 Seismic Slope Deformation Analysis

Potential seismic deformation of the facility slopes was assessed using the methodology proposed by Bray & Macedo (2019), which was developed to estimate shear-induced seismic slope displacements in earth structures and natural slopes, and has been validated for application to dams and waste piles.

The earthquake magnitudes adopted for the deformation analysis are derived from the Seismic Hazard Assessment (SHA) and account for the associated reduction in probabilistic design accelerations. The corresponding spectral accelerations were obtained from the uniform hazard response spectrum defined in the SHA, evaluated at the degraded period representative of the facility slopes.

The yield coefficient, defined as the horizontal pseudo-static earthquake coefficient that results in the factor of safety of 1.0, was determined using the previously presented limit equilibrium models. The initial fundamental period of the slope was estimated as a function of the failure mass height and the shear wave velocity along the failure surface. Shear wave velocities were derived from three MASW geophysical surveys, indicating an average velocity of approximately 175 m/s for the overburden. For future waste rock and tailings deposits, average shear wave velocities of 360 m/s and 350 m/s, respectively, were assumed.

Table 15-19: Deformation Analysis Input Parameters

Section	ky	Ts (s)	TR = 2,475 years		TR = 10,000 years	
			Mw	Sa (1.3Ts) (g)	Mw	Sa (1.3Ts) (g)
A-A’	0.12	0.02	7.5	0.92	7.9	1.38
I-I’	0.14	0.04	7.5	1.20	7.9	1.70
D-D’	0.49	0.16	7.5	1.57	7.9	2.18
F-F’	0.35	0.17	7.5	1.50	7.9	2.13
J-J’	0.41	0.03	7.5	1.05	7.9	1.49

The potential displacements along the DSTFs and WRDs were calculated as shown in Table 15-20. These displacements are considered tolerable, as any movement would primarily occur along the failure plane without causing significant overall slope instability. However, localized impacts such as damage to surface drainage structures and minor alterations to slope geometry may occur and should be addressed through post-event corrective measures.

Table 15-20: Calculated Seismic Slope Displacements

Section	TR = 2,475 years			TR = 10,000 years		
	Prob. exced = 84%	Prob. exced = 50%	Prob. exced = 16%	Prob. exced = 84%	Prob. exced = 50%	Prob. exced = 16%
DSTF 1 – Section A-A’	21.9	45.7	95.0	61.9	128.6	267.4
DSTF 2 - I-I’	24.5	50.9	105.9	61.9	128.7	267.6
WRD 1 - D-D’	2,0	5.8	12.9	8.3	17.8	37.3
WRD 2 (Phase 1) - J-J’	0.5	3.4	10.3	6.1	16.7	36.6
WRD 2 (Phase 2) - F-F’	5.3	11.4	23.8	15.5	32.3	67.2

15.4.4.2 Estimation of Runout Distances

To evaluate the risks associated with hypothetical failures of the DSTFs and WRDs, runout analyses were developed. The analyses employed the methodologies proposed by Corominas (1996) and Hunter & Fell (2003). These approaches were proposed to estimate the runout distance of failed slopes of earth structures by means of empirical correlations.

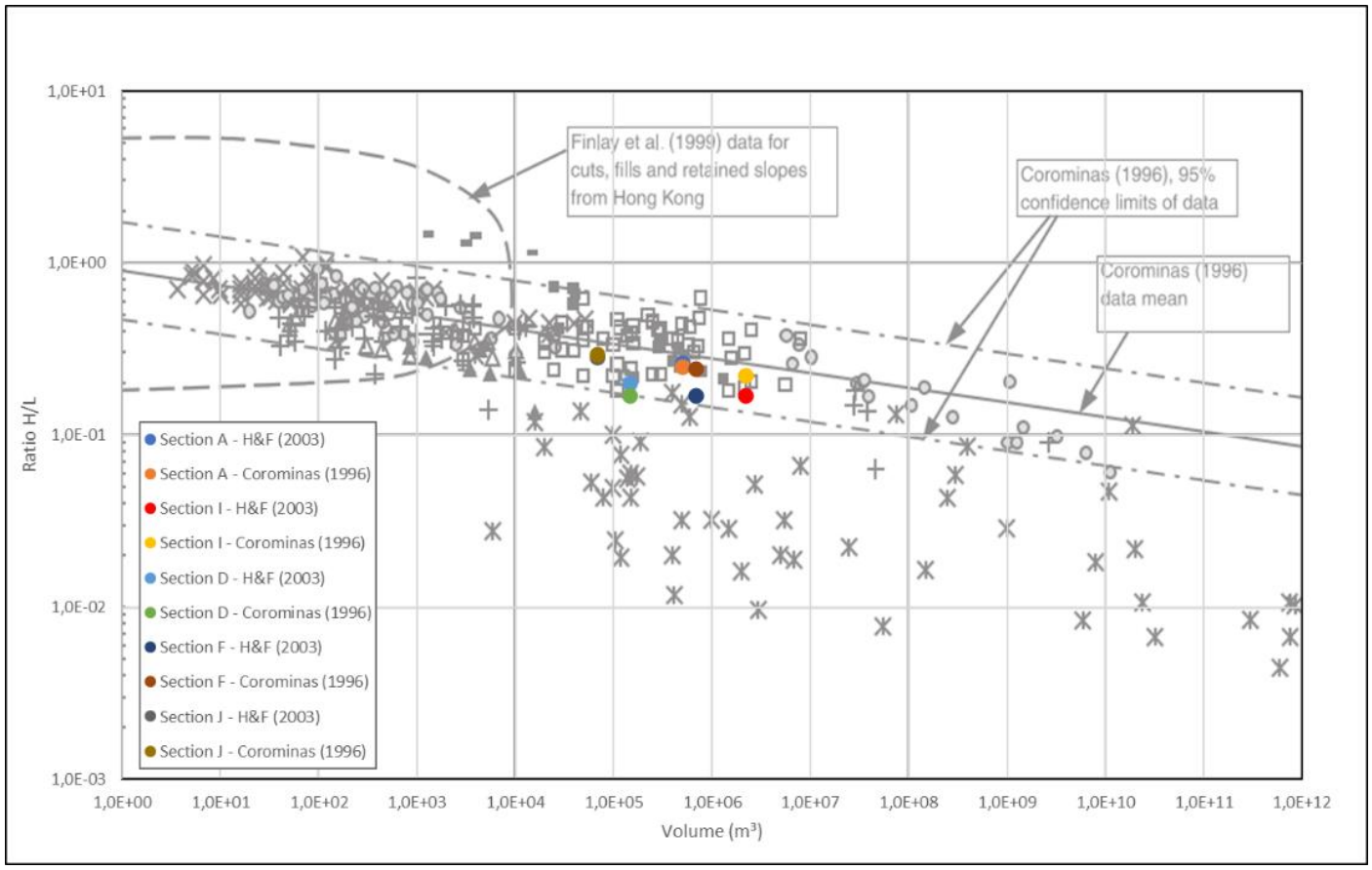
Both methods characterize the longitudinal geometry of the slide mass using the travel distance (L) and landslide height (H), measured from the crest of the source area to the distal toe of the runout. A key distinction between the two approaches is that Corominas (1996) incorporates the slide volume into its equations, whereas Hunter & Fell (2003) relies on the downslope angle downstream the source area.

In this assessment, both methodologies were applied using the equations for translational and unconfined failures, as summarized in Table 15-21. The resulting H/L ratios were analyzed in relation to the estimated slide volume and compared against the database compiled by Hunter & Fell (2003), as can be seen in Figure 15-21. This database presents H/L ratios as a function of the displaced volume (m^3) for a substantial set of historical slope-failure events. All runout estimates derived for the DSTFs and WRDs fall within the 95% confidence interval established by Corominas for this dataset, which supports the reliability of the runout assessments presented herein.

Table 15-21: Calculated Runout Estimates

	DSTF 1 A-A'	DSTF 2 I-I'	WRD 1 D-D'	WRD 2 (Phase 1) J-J'	WRD 2 (Phase 2) F-F'
Landslide height, H (m)	32.3	30.45	19.94	37	41.21
Volume (m^3)	498,489.0	2,159,136.0	145,773.0	67,107.0	683,432.0
Travel distance, L (m)	122.6	179.1	99.1	130.0	242.4
Ratio (H/L)	0.26	0.17	0.20	0.28	0.17
Travel distance, L, beyond the slope toe (m)	16.8	73.6	56.8	64.9	140.8
Travel distance, L (m)	128.2	135.9	116.2	125.1	167.8
Ratio H/L	0.25	0.22	0.17	0.30	0.25
Travel distance, L, beyond the slope toe (m)	22.5	30.4	74.0	60.0	66.2

Figure 15-23: H/L Versus Volume for Slides from Database



Source: Hunter & Fell, 2003.

Results obtained in the runout analyses indicate that sumps located adjacent to the piles could be impacted by slope failure events. The municipal road located and the beneficiation plant, located downstream the DSTF 1 and WRD 1, could also be impacted.

15.4.5 Mass and Volume Balance for Waste Rock and Tailings Facilities

15.4.5.1 Tailings Volume Balance

The tailings generated by mining activities between Year 1 and Year 16 will be deposited in multiple structures across the project site: DSTF 1 (licensed area inside mine property) and DSTF 2 (non-licensed area inside mine property). For the adopted geometry (maximum slope height: 8 m; slope inclination: 3H:1V; berm width: 5 m), the capacities of the developed arrangements for the tailings geotechnical structures are:

- DSTF 1: 498,489 m³ (regardless of the waste rock shell volume), between Year 1 and Year 4.

- DSTF 2 (to be licensed): 2,159,136 m³, between Year 4 and Year 17.

If DSTF 2 is not constructed, the peak volume gap for the tailings is approximately 2,875,000 m³, in the Year 17. The gap starts in Year 4, but the peak occurs in Year 17 because, although paste backfill will begin in Year 1, the volume required for this service is always smaller than the volume of tailings produced on an annual basis, so the gap continuously grows.

In summary, the current storage capacity for the tailings generated is not sufficient for the LOM, even if the DSTF 2 is licensed and operated. The licensed area only has capacity for the storage of the tailings generated between Year 1 and Year 4. Between Year 4 and Year 17, permitting for the DSTF 2 must be obtained. From Year 17 onwards, a new area must be licensed for the disposal of the remaining volume (approximately 450,000 m³).

15.4.5.2 Waste Rock Volume Balance

The waste rock generated by mining activities between Year -1 and Year 16 will be stored in multiple structures across the project site: WRD 1, WRD 2 (Phase 1), DSTF (as a waste rock shell), and WRD 2 (Phase 2), proposed on a non-licensed area inside the mine property. For the adopted geometry (maximum slope height: 10 m; slope inclination: 2H:1V; berm width: 5 m), the capacities of the developed arrangements for the waste rock geotechnical structures are:

- WRD 1: 145,773 m³, between Year -1 and Year 1.
- WRD 2 (Phase 1): 67,107 m³, between Year -1 and Year 1.
- DSTF 1 (waste rock shell): 90,888 m³, in Year 1.
- WRD 2 (Phase 2) (to be licensed): 616,325 m³, between Year 1 and Year 15.

If the WRD 2 (Phase 2) is not constructed, the peak volume gap for the waste rock is approximately 567,000 m³, in the Year 6. The gap starts in Year 1, but the peak occurs in Year 6 because, although rockfill will begin in Year 1, the volume required for this service is relatively small until Year 6. Thus, the capacity of the proposed WRD 2 (Phase 2) represents more than the peak gap of waste rock generated.

In summary, the storage capacity for the waste rock generated is sufficient for the LOM, if the Waste Rock Dump 2 (Phase 2) is included. However, it is emphasized that the licensed areas only have capacity for the storage of the waste rock generated in Year -1 and partially in Year 1. For Year 1 onwards, permitting for a new area must be obtained.

15.4.5.3 Summary for Waste Rock and Tailings

This item presents a summary of the current and future capacity for disposal of tailings and waste rock for the Era Dorada project (see Table 15-22).

Considering that:

- Year -1 is when waste rock production is started.
- Year 1.

- Year 2 is the predicted startup for the ore processing plant.

The following events will take place, according to the predicted schedule for the LOM:

- WRD 1 will reach its maximum capacity for waste rock disposal in Year -1.
- WRD 1 will reach its maximum capacity for low-grade ore disposal in Year 1.
- WRD 2 will reach its maximum capacity for waste rock disposal in Year 1.
- DSTF 1 will reach its maximum capacity for tailings disposal in Year 4.
- A new Waste Rock Dump (or WRD 2) will be required in Year 1 with at least 570,000 m^3 of final capacity, approximately, and this capacity is probably achievable inside the current limits of the mine property.
- A new DSTF (DSTF 2) will be required in Year 4 with at least 2,900,000 m^3 of final capacity, considering the adopted geometry for DSTF 2.
- Considering 2,159,136 m^3 of capacity for DSTF 2, it is still necessary to dispose of a remaining volume of approximately 500,000 m^3.

Table 15-22: Mass and Volume Balance for Waste Rock and Tailings at Era Dorada Project

		-1	1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
	Unit																			
Waste Rock Generation	m³	110,815.3	137,867.3	52,729.6	159,325.2	125,465.4	109,291.7	91,603.9	53,902.5	51,993.7	52,906.7	22,804.0	13,309.7	26,179.8	19,643.7	18,143.0	23,823.7	3,183.8	238.1	0.0
Waste Rock Generation (Total)	m³	110,815.3	248,682.7	301,412.3	460,737.5	586,202.9	695,494.6	787,098.5	841,000.9	892,994.6	945,901.3	968,705.3	982,015.1	1,008,194.9	1,027,838.6	1,045,981.5	1,069,805.2	1,072,989.1	1,073,227.1	1,073,227.1
Initial Low-Grade Stockpile (Total)	m³	18,578.8	43,693.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8
Waste Rock Deposit 1 - North (capacity 145,773 m³)																				
Waste Rock	m³	84,189.3																		
Waste Rock (Total)	m³	84,189.3	84,189.3	84,189.3	84,189.3	84,189.3	84,189.3	84,189.3	84,189.3	84,189.3	84,189.3	84,189.3	84,189.3	84,189.3	84,189.3	84,189.3	84,189.3	84,189.3	84,189.3	84,189.3
Low Grade Ore	m³	18,578.8	25,115.0	17,890.0																
Low Grade Ore (Total)	m³	18,578.8	43,693.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8	61,583.8
WRD 1 Total volume (WR + low grade ore)	m³	102,768.0	127,883.0	145,773.0	145,773.0	145,773.0	145,773.0	145,773.0	145,773.0	145,773.0	145,773.0	145,773.0	145,773.0	145,773.0	145,773.0	145,773.0	145,773.0	145,773.0	145,773.0	145,773.0
Waste Rock Gap (if only WRD 1 is constructed) (Total)	m³	26,626.1	164,493.4	217,223.0	376,548.2	502,013.6	611,305.3	702,909.2	756,811.7	808,805.4	861,712.1	884,516.1	897,825.8	924,005.7	943,649.3	961,792.3	985,616.0	988,799.8	989,037.9	989,037.9
Low Grade Gap	m³	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Waste Rock Dump 2 - Phase 1 (EIA) - South (capacity 67107 m³)																				
Waste Rock	m³	26,626.1	40,480.9																	
Waste Rock (Total)	m³	26,626.1	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0
WRD 2 Total volume	m³	26,626.1	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0	67,107.0
Waste Rock Gap (if only WRD 1 and WRD 2 Phase 1 are constructed) (Total)	m³		97,386.4	150,116.0	309,441.2	434,906.6	544,198.3	635,802.2	689,704.7	741,698.4	794,605.1	817,409.1	830,718.8	856,898.7	876,542.3	894,685.3	918,509.0	921,692.8	921,930.9	921,930.9
Waste Rock Shell for the DSTF 1	m³		90,888.0																	
Waste Rock Shell for DSTF 1 (Total)	m³		90,888.0	90,888.0	90,888.0	90,888.0	90,888.0	90,888.0	90,888.0	90,888.0	90,888.0	90,888.0	90,888.0	90,888.0	90,888.0	90,888.0	90,888.0	90,888.0	90,888.0	90,888.0
Waste Rock Gap (considering WRD 1, WRD 2 Phase 1 and WR shell, without Rockfill) (Total)	m³		6,498.4	59,228.0	218,553.2	344,018.6	453,310.3	544,914.2	598,816.7	650,810.4	703,717.1	726,521.1	739,830.8	766,010.7	785,654.3	803,797.3	827,621.0	830,804.8	831,042.9	831,042.9
Rockfill	t						20,470.2	2,294.6	28,076.1	97,060.7	87,730.1	121,832.7	130,728.9	128,413.6	130,678.0	114,295.5	123,851.7	185,298.8	184,701.1	0.0
Rockfill	m³						12,793.8	1,434.1	17,547.6	60,662.9	54,831.3	76,145.4	81,705.6	80,258.5	81,673.7	71,434.7	77,407.3	115,811.8	115,438.2	
Rockfill (Total)	m³						12,793.8	14,228.0	31,775.5	92,438.5	147,269.8	223,415.2	305,120.8	385,379.3	467,053.1	538,487.7	615,895.1	731,706.8	847,145.0	847,145.0
Waste Rock Gap (considering WRD 1, WRD 2 Phase 1, WR shell and Rockfill) (Total)	m³						440,516.5	530,686.2	567,041.2	558,371.9	556,447.3	503,105.9	434,710.0	380,631.3	318,601.2	265,309.5	211,725.9	99,098.0	-16,102.2	-16,102.2

		-1	1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
	Unit																			
Waste Rock Peak Gap (peak volume needed for new pile - WRD 2 Phase 2)	m³	567,041.2																		
Tailings Generation	t			42,365.9	365,431.1	365,492.5	584,131.4	584,258.2	584,472.4	584,297.8	584,465.4	584,333.0	584,459.1	584,247.0	584,371.9	584,213.4	584,413.6	583,725.6	583,870.4	384,310.7
Tailings Generation	m³			26,984.6	232,758.7	232,797.8	372,058.2	372,139.0	372,275.4	372,164.2	372,271.0	372,186.6	372,266.9	372,131.9	372,211.4	372,110.5	372,238.0	371,799.8	371,892.0	244,783.9
Tailings Generation (Total)	m³			26,984.6	259,743.3	492,541.1	864,599.4	1,236,738.3	1,609,013.7	1,981,178.0	2,353,449.0	2,725,635.6	3,097,902.5	3,470,034.4	3,842,245.8	4,214,356.2	4,586,594.2	4,958,393.9	5,330,285.9	5,575,069.8
Pastefill	t			4,467.4	155,316.3	130,768.9	274,856.6	274,916.2	275,017.0	274,934.9	275,013.7	274,951.4	275,010.7	274,910.9	274,969.7	274,895.1	274,989.3	274,665.6	274,733.7	0.0
Pastefill	m³			2,845.5	98,927.6	83,292.3	175,067.9	175,105.9	175,170.0	175,117.7	175,168.0	175,128.3	175,166.1	175,102.5	175,139.9	175,092.4	175,152.4	174,946.2	174,989.6	
Pastefill (Total)	m³			2,845.5	101,773.1	185,065.4	360,133.2	535,239.1	710,409.2	885,526.9	1,060,694.9	1,235,823.1	1,410,989.2	1,586,091.7	1,761,231.6	1,936,324.1	2,111,476.5	2,286,422.7	2,461,412.4	2,461,412.4
Tailings Gap (considering Pastefill) (Total)	m³			24,139.2	157,970.3	307,475.7	504,466.1	701,499.2	898,604.6	1,095,651.1	1,292,754.1	1,489,812.4	1,686,913.3	1,883,942.7	2,081,014.1	2,278,032.2	2,475,117.7	2,671,971.2	2,868,873.5	3,113,657.4
DSTF (capacity 498,489 m³)																				
Filtered Tailings Disposal in Pile	t			37,898.5	210,114.8	234,723.6	309,274.9	309,342.0	309,455.4	309,363.0	309,451.7	309,381.6	309,448.4	309,336.1	309,402.2	309,318.3	309,424.3	309,060.0	309,136.7	384,310.7
Filtered Tailings Disposal in Pile (Total)	t			37,898.5	248,013.3	482,736.9	792,011.8	1,101,353.8	1,410,809.2	1,720,172.2	2,029,623.9	2,339,005.5	2,648,453.9	2,957,790.0	3,267,192.2	3,576,510.5	3,885,934.8	4,194,994.8	4,504,131.5	4,888,442.2
Filtered Tailings Disposal in Pile	m³			24,139.2	133,831.1	149,505.5	196,990.4	197,033.1	197,105.4	197,046.5	197,103.0	197,058.3	197,100.9	197,029.4	197,071.5	197,018.0	197,085.5	196,853.5	196,902.3	244,783.9
Filtered Tailings Disposal in Pile (Total)	m³			24,139.2	157,970.3	307,475.7	504,466.1	701,499.2	898,604.6	1,095,651.1	1,292,754.1	1,489,812.4	1,686,913.3	1,883,942.7	2,081,014.1	2,278,032.2	2,475,117.7	2,671,971.2	2,868,873.5	3,113,657.4
Tailings Gap + Pastefill + DSTF 1 (Total)	m³			0.0	0.0	0.0	5,977.1	203,010.2	400,115.6	597,162.1	794,265.1	991,323.4	1,188,424.3	1,385,453.7	1,582,525.1	1,779,543.2	1,976,628.7	2,173,482.2	2,370,384.5	2,615,168.4
Tailings Gap Peak (volume needed for new pile - DSTF 2)	m³	2,615,168.4																		

LEGEND:	
	Adopted (approximately 100,000 m³ as agreed with client)
	Year 1
	Year 3 to Year 17
	Volumes of materil to be disposed at the end of Year 2
	Period when facilities reach ter full capacity

Materials Dry Densities	
Waste Rock	1.60 t/m³
Filtered Tailings	1.57 t/m³

General Information			
WRD 1 Capacity (EIA 2007)	145,773 m³	WRD 1 area: Waste Rock + Low Grade Ore	
WRD 2 Capacity (EIA 2007)	61,107 m³		
WRD 1 + WRD 2 Capacity	212,880 m³		
DSTF 1 Capacity	498,489 m³	782,627.73 t	
Topsoil Deposit Capacity	7,786 m³		
DSTF 2	2,159,136 m³		
Tailings excess volume (to be disposed of in a third DSTF – "DSTF 3")	456,032 m³		Even with the new pile (DSTF 2) there is around 500,000 m³ of tailings left to be disposed of.
WRD 2 Phase 2	616,325 m³		The new waste rock pile is sufficient to accommodate the excess volume.

15.5 Surface Water Management

The surface water management infrastructure was designed with the objective of minimizing and, whenever possible, separating precipitation and runoff that come into contact with potential sources of contamination from the natural surface waters within the project's watershed.

Areas considered to have contamination potential include those where filtered tailings are stored, specifically the Process Plant, the Dry Stack Tailings Facility (DSTF), and the Waste Rock Piles. Runoff from these areas is classified as "contact water." Conversely, runoff from natural drainage channels (thalwegs) and slopes is classified as "non-contact water."

In this project, the primary potential sources of contact water are the Tailings Piles, the DSTF, and the Process Plant area. In the Process Plant, filtered tailings will be produced, handled, and loaded into trucks for subsequent disposal in the designated piles.

Contact water will be directed to sediment control structures and then conveyed to a Chemical Effluent Treatment Plant, where it will undergo appropriate treatment before being discharged into the environment. Non-contact water, on the other hand, will be channeled through drainage ditches to specific monitoring points before being discharged into the nearest natural watercourse.

In the stockpile area, a total of ten peripheral stormwater drainage channel sections has been designed to convey runoff toward five contact water accumulation basins.

The stormwater drainage channels will be lined with geomembrane where constructed with riprap, while high-slope sections will be lined with concrete to minimize erosion and ensure hydraulic stability during the design storm event.

For access crossings, reinforced concrete slabs are planned to accommodate light vehicle traffic, while buried culverts, installed at approximately one meter below grade, will be used for off-highway truck crossings to avoid operational interference and reduce structural wear.

Additionally, a perimeter (diversion) channel has been designed along the upper portion of the stockpiles to separate contact and non-contact waters. This channel is intended to collect surface runoff from the natural terrain, preventing the mixing of clean and contact waters. The non-contact water collected by this system will be discharged directly into the El Tempisque River, without the need for prior treatment, as it does not pose a contamination risk.

At the discharge points of the peripheral channels, energy dissipation structures lined with riprap or concrete are proposed to control flow energy and prevent local scoring at the outlets and at the base of the accumulation basins.

Subsequently, the accumulated water shall undergo treatment due to the potential generation of acid drainage and the possible presence of arsenic concentrations exceeding the limits established by current regulations. Accordingly, the implementation of a Chemical Treatment Plant is planned to ensure that the water quality meets the required environmental standards prior to its discharge into the Óstua River.

The five contact water accumulation basins are:

- Sump WRD North
- Sump DSTF 1
- Sump DSTF 2
- South WRD South
- Sump DSTF 3

The general arrangement of the waste dumps and containment structures can be observed in Figure 15-2.

15.5.1 Hazard Considerations

Due to the proximity of the project area to two watercourses — the El Tempisque River and the Tancushapa River —, a hydrologic and hydraulic study was completed in 2018 by Stantec. The study included floodplain modeling to evaluate potential inundation areas associated with both rivers. The analysis considered 100-year return period flood events and the Probable Maximum Flood (PMF), calculated based on a Probable Maximum Precipitation (PMP) depth of 450 millimeters.

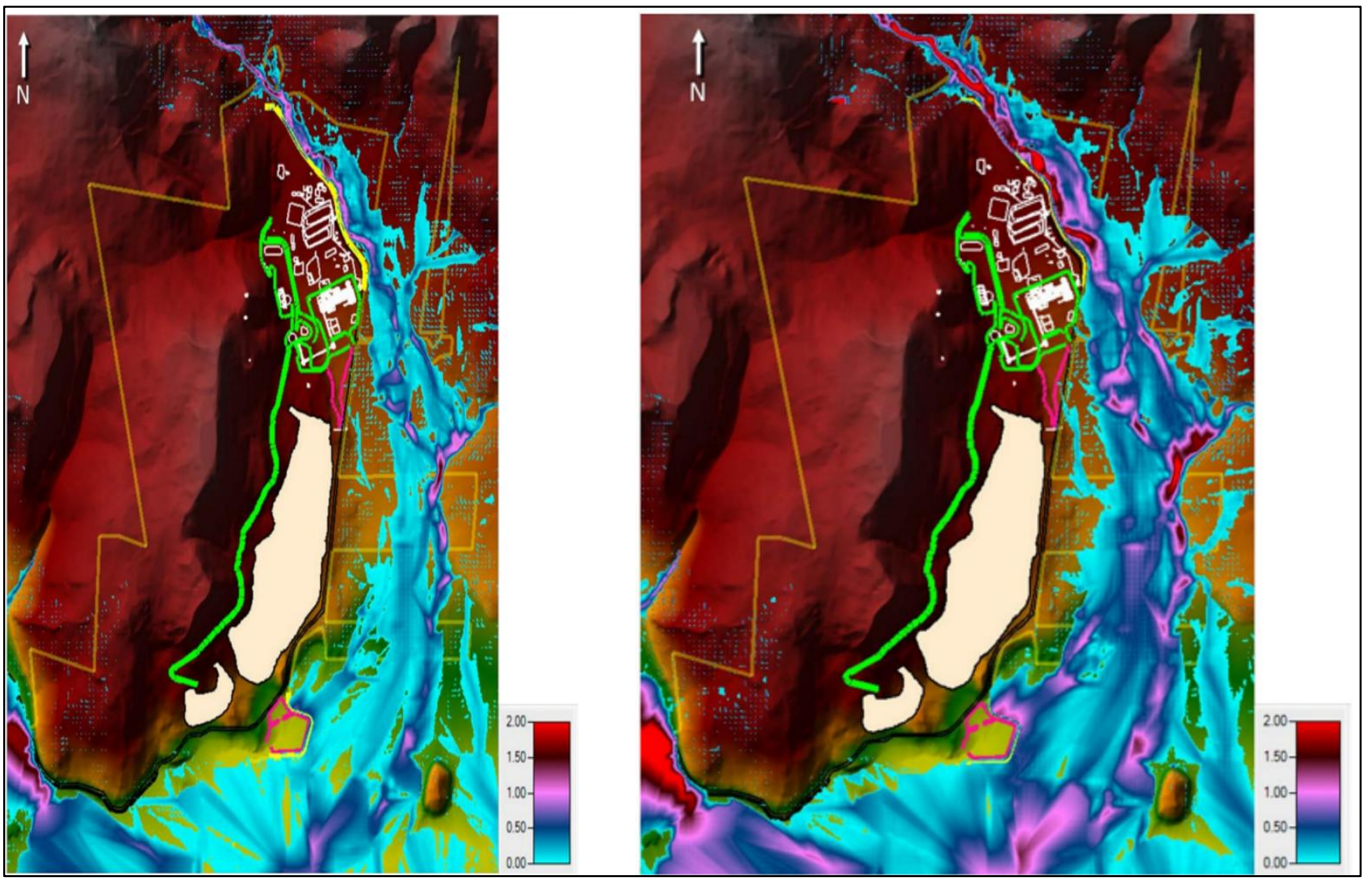
The results indicated that maximum water levels within the project area could reach approximately one meter during the 100-year event and up to two meters during the PMF event. As a mitigation measure to reduce the potential for stormwater inundation of mine infrastructure, the construction of containment dikes was proposed to manage surface runoff and protect existing facilities.

The study specified a design height of two meters above existing grade for the dikes, based on the available topographic data, but did not include detailed design parameters necessary for engineering implementation. The topographic base used in the analysis was derived from publicly available data with 15-meter contour intervals, which lack the resolution required for detailed flood assessment at the site scale.

As no updated topographic data was available at the time of this report, the results of the original Stantec study may contain uncertainties. It is recommended that a new hydrologic and hydraulic analysis be undertaken using high-resolution topographic data, to confirm the flood susceptibility of the site, identify potentially affected areas, and develop design criteria for the proposed containment structures.

The floodplain and the location of the dikes are illustrated in Figure 15-24.

Figure 15-24: Floodplain and Location of Proposed Dikes for the 100-year Return Period and PMP Events



Source: Stantec, 2018.

15.6 Water Balance and Management

The water balance was developed to quantify and control water inflows, outflows, and storage volumes within the project, supporting efficient water resources management and ensuring compliance with applicable environmental regulations.

The primary water inflows correspond to the pumping discharge from ten wells planned for the dewatering of the underground mine, as well as surface drainage, both segregated according to the existing underground access points — North Portal and South Portal. This water contains various potentially contaminating elements and therefore requires treatment prior to any type of use. Accordingly, all water collected from the mine is directed to the Wastewater Treatment Plant (WTP), where it undergoes appropriate treatment for subsequent reuse in internal processes, particularly for the Process Plant water supply.

For uses requiring potable water quality, such as administrative buildings, dining facilities, and changing rooms, the installation of a new Potable Water Treatment Plant (PWTP) was planned. Additionally, the water balance considers the supply of potable water to the local community, based on the estimated demand for approximately 10,000 inhabitants.

It is important to note that the Cerro Blanco Mine currently exhibits a water surplus, meaning that water availability exceeds operational demand. Consequently, the water balance also aimed to verify whether surplus volumes comply with the water use and discharge permits established in the Environmental Impact Study (EIA, 2007). For this project, two water rights (outorgas) were considered, associated with nearby receiving water bodies: the El Tempisque River (1,500 gpm) and the Ostua River (3,750 gpm).

The water balance modeling was performed using GoldSim 14.0 software, developed by GoldSim Technology Group. GoldSim employs Monte Carlo simulations to represent the dynamics of complex systems and is widely used in engineering and water resources management studies and it is presented in the document 108726-EG-00001-23231-010_RA. In this project, GoldSim was used to simulate reservoir volume variations, considering inflows (pumping, precipitation) and outflows (consumption, evaporation, discharge).

The main water inflows are the pumping discharges from underground wells and mine portals, initially directed to the raw water reservoirs (Pond A and Pond D). Following treatment at the WTP, water is stored in the treated water reservoirs (Pond B and Pond C), from which it is distributed via pipelines to meet the mine's operational demands, including utilities, process water, and equipment requirements.

All reservoirs also account for precipitation contributions and evaporation losses, based on data from a meteorological station located at the Cerro Blanco Mine, which provides consistent historical records from 2007 to 2025.

The treated water demands within the Cerro Blanco Mine are described below.

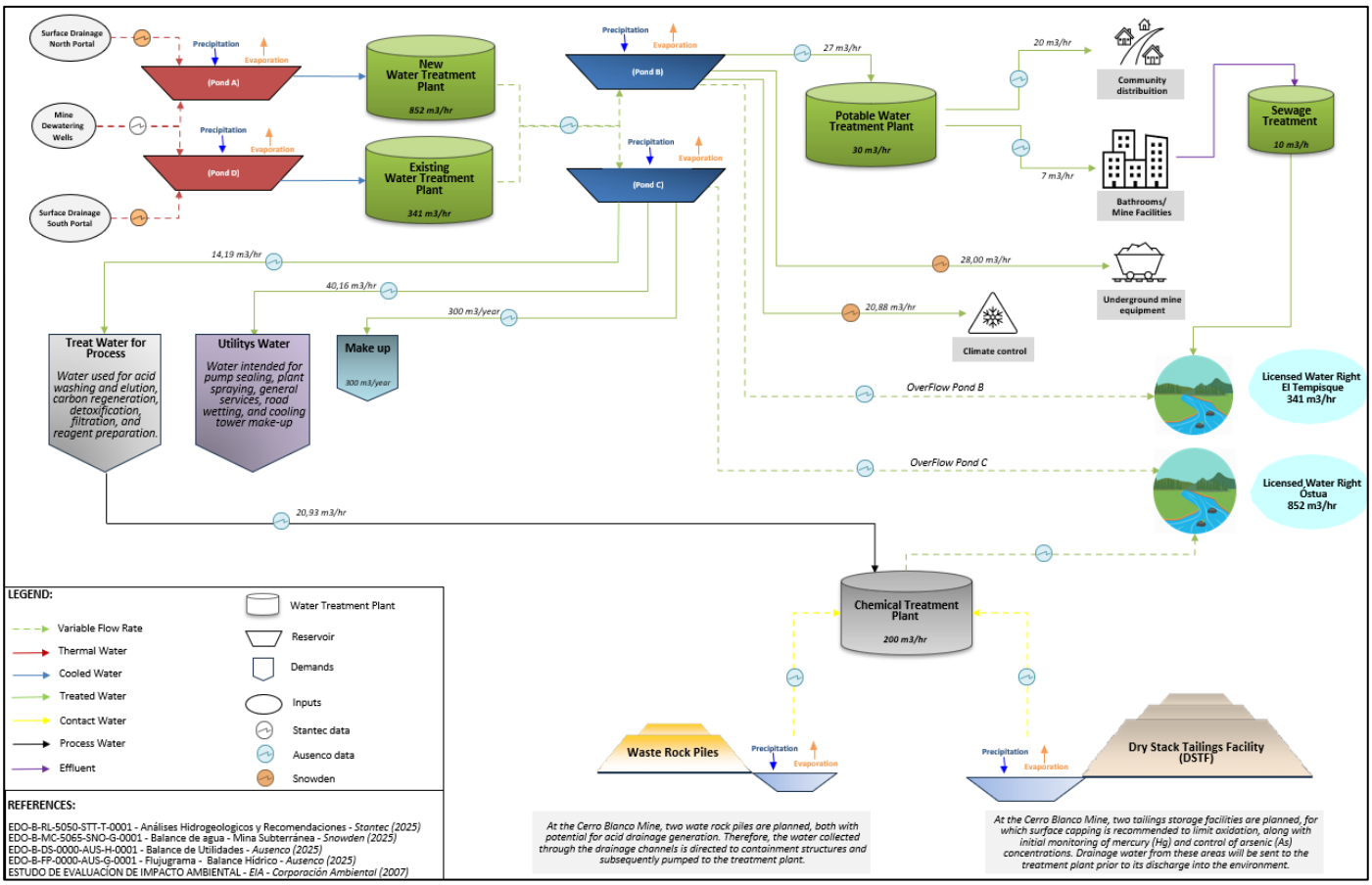
- **Utility Water Demands (40.16 m^3/h):** The volume of treated water allocated to utilities supports operational and plant infrastructure activities, including pump sealing, dust suppression spraying, general maintenance services, wetting of internal roads, and make-up water to compensate for cooling tower losses. This demand is continuous and relatively stable and is essential to ensure efficient equipment operation and adequate working conditions.
- **Process Water Demands (14.19 m^3/h):** This demand refers to the use of treated water in critical stages of the mining process, such as acid washing and elution, carbon regeneration, detoxification, filtration, and reagent preparation. The water used for these purposes must meet specific physicochemical requirements to ensure that adsorption, precipitation, and mass transfer processes perform as designed, without compromising operational efficiency or product quality.
- **Make-up Water (300 m^3/year):** The make-up volume represents compensation for unavoidable water losses in the industrial system, including evaporation, entrainment, retention in solids, and dispersed process losses. This component is essential for closing the water balance, maintaining operational reservoir levels, and preventing water deficits within the plant.
- **Community Supply and Administrative Services (24.77 m^3/h):** Water conveyed to the Potable Water Treatment Plant is intended for human consumption and internal administrative uses, such as restrooms, cafeterias, and

accommodation facilities. This use must comply with potable water and sanitary safety standards, differing from industrial demands in terms of treatment requirements and quality specifications.

- **Use in Underground Mine Equipment (27.5 m³/h):** Water consumption for underground mine operations includes supply for equipment operation and washing, as well as support for auxiliary drilling and excavation activities. This volume is critical for operational safety, environmental control, and the maintenance of productivity in the underground mine.
- **Underground Mine Cooling (20.88 m³/h):** Cooling water is used for the thermal conditioning of air circulating through underground workings, mitigating heat generated by the surrounding rock mass, mining equipment, and operational activities. This system is essential to ensure thermal comfort, occupational safety, and suitable working conditions, while also contributing to efficient equipment performance and the continuity of mining operations.

The conceptual model of the water balance, including primary inflows, internal uses, and final water disposition, is presented in Figure 15-25.

Figure 15-25: Flowchart of the Water Management System used in the Era Dorada Mine Hydrodynamic Water Balance



Source: Ausenco, 2025.

The water balance study aimed to:

- Validate whether the pumping rates reported in the Cerro Blanco Mine Dewatering and Water Discharge Report (Stantec, 2025c) comply with the water use permits established in the EIA (2007), and assess the need for new permits.Validate whether the pumping rates reported in the Cerro Blanco Mine Dewatering and Water Discharge Report (Stantec, 2025c) comply with the water use permits established in the EIA (2007), and assess the need for new permits,
- Validate and estimate the required capacity of raw water and contact water ponds under normal operating conditions.
- Assess whether the surplus water volumes generated by the mine remain within the limits of the water rights for the El Tempisque (1,500 gpm) and Ostua (3,750 gpm) rivers, and determine the need for new permits.
- Evaluate the treatment demand for contact water generated from the DSTF and Waste Rock Storage Facilities.

The storage capacity of the raw water reservoirs has been confirmed as adequate for current operational conditions. However, starting in 2027, an increase in pumping capacity will be required due to the higher volumes of water from underground wells.

Additionally, the existing water rights (permits) for pumping and discharge to the El Tempisque River (3,750 gpm) and the Ostua River (1,500 gpm) will be exceeded from 2029 onwards, making it necessary to obtain new licenses for both pumping and effluent discharge.

These actions are essential to ensure the continuous operation of the water supply system, comply with legal water rights limits, and maintain the environmental compliance of the project.

15.7 Water Treatment Infrastructure

15.7.1 Mine Water Treatment Plant

The Water Treatment Plant (WTP) currently in operation at Cerro Blanco has a treatment capacity of up to 1,500 gpm. However, a permit has been granted for the construction of a new WTP with an expanded capacity of 3,750 gpm, which will be dedicated exclusively to the treatment of water originating from the Underground Mine. The discharge of treated water is limited to a maximum rate of 1,500 gpm to the El Tempisque River and 3,750 gpm to the Óstua River.

The primary objective of the WTP is to treat water from the underground mine in accordance with the standards established by Guatemala's Acuerdo Gubernativo No. 236-2006, ensuring that the treated water meets the conditions required for safe environmental discharge. As described in the Environmental Impact Assessment (EIA, 2007), the temperature of the influent water to the WTP must be below 40°C. To comply with this requirement, cooling towers were designed upstream of the treatment system to reduce the water temperature prior to processing.

After cooling and treatment, the water will be allocated to several uses, including the industrial process, road wetting for dust control, underground mine cooling, and supplying water demands for equipment, among others. Treated

water will be pumped from the treatment facilities to the storage reservoirs (Ponds B and C) and subsequently distributed to tanks or end uses according to the requirements of each demand category.

In addition, the construction of an Industrial Chemical Treatment Plant is planned to treat process water recovered from the beneficiation plant. This water originates from the pre-leach and tailings thickeners, the overflow of the fines tank, the heat exchanger, and utility-related water streams. The effluent will initially be stored in a recovered water tank, and any excess water will not recirculate to the process; it will be directed to the IETP for treatment.

15.7.2 Potable Water Treatment

A portion of the treated water will be directed to a new Potable Water Treatment Plant (PWTP), designed to meet drinking water quality standards. The potable water produced will be used to supply the project's facilities, including administrative buildings, dining areas, locker rooms, and other support structures. Additionally, potable water will be distributed to communities located in the surrounding areas of the project site via water trucks, which will be properly licensed and operated in compliance with applicable sanitary regulations.

15.7.3 Sewage Treatment

Black water and wastewater will be managed through standard septic tank collection systems utilizing natural decomposition bioreactors prior to final discharge. Sanitary effluent generated from plant buildings and main infrastructure facilities will be directed to a buried septic system, which will be installed beneath the process area.

A bioreactor tank will be installed and connected to the facility buildings through an underground sewer network. An additional unit will be installed to support the new infrastructure included in the project.

Sewage will be properly treated, with solids separated and the liquid portion discharged in a controlled manner. Water required for septic system operation and wash-down activities will be supplied from the raw water system.

15.8 Power and Electrical

15.8.1 Power Supply

Electrical power for the plant will be sourced from a substation, located in Asunción Mita. A 69 kV single-circuit overhead transmission line, approximately 8.6 kilometers, will be constructed to connect the substation to the project site switchyard. Commissioning and operational startup of both the transmission line and the switchyard are currently targeted for Year 3.

The total estimated operating power demand for the site, considering full dewatering and injection pump operation, is provided in Table 15-23.

Table 15-23: Electrical Load Summary

Item	Unit	Value
Total Operating Load	17	MW
Total Connected Load	27	MW

15.8.2 Surface Electrical Power Distribution

The proposed electrical distribution systems are described below.

15.8.2.1 Primary Power Distribution

During the first three (3) years of plant operation, primary power distribution will be provided by a diesel power plant operating at 4.16 kV, under a lease (comodato) model. After this period, power supply will be gradually transferred to the main substation to be installed as part of the project.

The main 69 kV switchyard, to be installed at the plant site, will comprise a circuit breaker, motor-operated disconnect switches, overhead transmission structures, and step-down power transformers.

The 69 kV incoming power will pass through a main circuit breaker with a set of motor-operated disconnect switches and will then be split into two circuits. Each circuit will include a motor-operated disconnect switch and a step-down power transformer, which will feed the main 4.16 kV medium-voltage switchgear located in a pre-fabricated modular building (i.e., the main substation).

The power transformer capacity can be increased up to 30% with additional (forced) cooling. In the event that one step down transformer is out of operation, the second one would have sufficient capacity to supply full-demand power to the plant.

The main switchgear consists of two sections connected by a normally open tie-breaker. The power correction equipment, in form of two harmonic filters, will be installed as part of the main substation and fed from the 4.16 kV main switchgear. 4.16 kV is defined as the primary on-site power distribution voltage. Distribution is from the main substation to the key area substations (mine, crushing plant, process plant,paste plant, cooling plants, dewatering wells, injection wells, etc.) through a system of 4.16 kV single circuit overhead distribution lines. Each feeder line originates from the 4.16 kV switchgear in the main substation.

The existing on-site overhead power distribution lines currently operate at 4.16 kV and will remain as the standard site distribution voltage. Field verification confirmed that the line system and all associated components were originally designed and installed for 13.8 kV operation and are in good condition. Therefore, portions of the existing network will be reused where technically feasible.

The 4.16 kV system will be expanded and reinforced to supply the new process plant and auxiliary facilities. The upgraded overhead distribution network will originate from the main substation and feed the primary surface loads,

including the mill, CIP, paste fill, dewatering, and mine portal areas. New feeders and pole-mounted transformers will be installed as required to accommodate additional loads and ensure proper voltage regulation throughout the site.

15.8.2.2 Secondary Power Distribution

The selected secondary distribution voltage levels for the plant are 4.16 kV, 3-phase, 60 Hz (medium voltage) for large drives and 480 V, 3-phase, 60 Hz (low voltage) for smaller drives. The secondary distribution system originates at Area substations (i.e. Metallurgy Substation, Crushing substation, etc.) and will distribute power to each individual end user.

The Area substation step-down transformers have been sized based on the Area electrical load calculation with a 20% growth factor applied and ratings rounded up to the next higher standard transformer size.

Two masonry (conventional) substations are planned for the surface facilities: one dedicated to the crushing area and the other responsible for supplying power to the remaining process areas. Additional compact skid-mounted substations will be installed to supply power to remote areas of the plant.

In addition, substations are planned at each underground mine portal. These portal substations will be interconnected through an internal tie system, providing redundancy and ensuring that each unit has sufficient capacity to independently supply the entire underground operation, if necessary.

15.8.3 Emergency Power

The emergency power supply philosophy has dedicated generators connected to the process area substation. This emergency power plant will operate at 480 V and supply backup power to critical process loads in the event of a grid outage, ensuring safe shutdown of major equipment and maintaining operation of essential equipment and systems such as control, lighting, and safety equipment.

In addition, individual standby generator units will be installed at the substations supplying the underground mine portals. These units will provide backup power to essential underground systems such as ventilation fans and communication equipment, maintaining safe underground conditions during a loss of utility power.

The combined emergency power system is designed to ensure continuity of critical operations and to allow controlled plant shutdown in case of a total power failure. The total installed standby generation capacity will be defined during the development of the project based on the confirmed emergency load demand of each area.

15.8.4 Construction Power

During the construction phase, the contracted assembly companies must provide generators to initiate activities. Subsequently, during the final stage of construction and the initial three years of site operation, a 15 MW diesel power plant will be installed under a lease (comodato) arrangement. This temporary generation facility will be capable of supplying the entire site’s electrical demand, including both surface and underground operations.

The leased power plant will ensure reliable energy availability during construction, commissioning, and early operation, prior to the full integration of the permanent utility interconnection. Its configuration will allow progressive load transfer to the permanent electrical infrastructure as the project advances.

15.9 Fuel

15.9.1 Fuel Storage and Distribution Facilities (Existing)

An existing diesel fuel storage facility is installed on-site for the existing electric power generators. It consists of two approximately 37,500 L (10,000 gallons) tanks within a containment area constructed of concrete. This existing facility currently meets the demand and distribution of diesel for Underground Mine vehicles and equipment from a mobile vehicle fuelling station and meets the fuel consumption demand for existing generator sets.

15.9.2 Equipment that generates fuel consumption

The Era Dorada project points to the expansion of fuel consumption (Diesel) by increasing the demand and consumption of fuel by electric power generators, increasing the fleet of mobile equipment for transporting and handling ore from an underground mine and by operating mobile equipment for handling and removing material for the formation of waste piles and tailings.

15.9.3 New fuel storage and distribution facilities

The Era Dorada project will contract the supply, logistics and fuel distribution and storage infrastructure to meet the increase in fuel consumption to move the ore on the underground mining fronts and ore management in the facilities of the ore concentration plant and storage piles and permanent piles.

15.9.4 Fuel Consumption Demands for Light Vehicles

All vehicles powered by gasoline and other fuels will be fuelled off-site in Asunción, Mita – Guatemala.

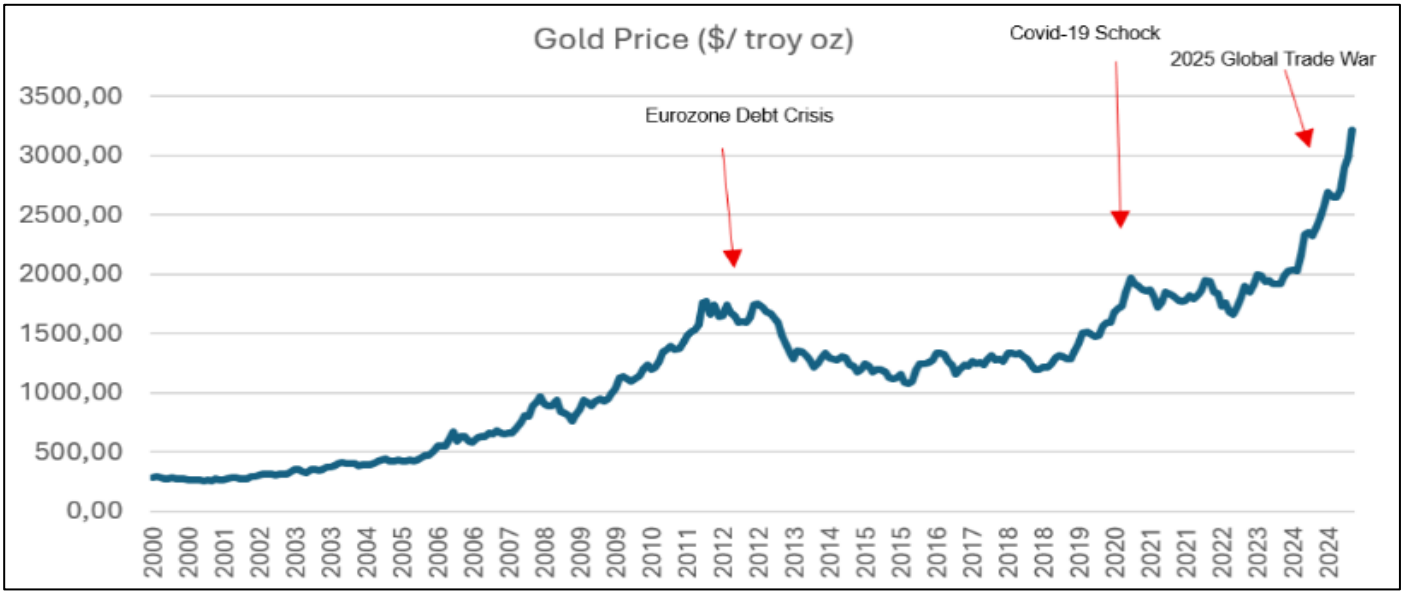
16 MARKET STUDIES

16.1 Market Studies

16.1.1 Gold Market

The global gold market operates as a well-established and highly liquid system, characterized by a diversified foundation of supply and demand. From a macroeconomic perspective, gold consistently exhibits countercyclical behavior, having historically served as a store of value under conditions of elevated financial stress, inflation volatility, and geopolitical instability. Its low to negative correlation with traditional asset classes, such as sovereign bonds and equities, significantly enhances its utility as a portfolio diversifier (Figure 16-1).

Figure 16-1: Gold Price Behavior Since 2000



Source: World Bank Group, 2025 in Aura Minerals, 2024.

16.1.2 Silver Market

Relative to global markets such gold, the global silver market is less significant in value. According to data published by the Silver Institute, it reached 680.5 million ounces (Moz) in 2024 and is projected to exceed 700 Moz in 2025 (Figure 16-2).

Figure 16-2: Silver Price Behavior Since 2000



Source: World Bank Group, 2025 in Aura Minerals, 2024.

16.2 Commodity Price Projections

16.2.1 Gold Price

Mineral Resources have been modelled at a gold price of US$ 2,000/oz. Project economics have also been assessed at a base case gold price of US$ 3,177/oz. based on the long-term consensus forecast from over 20 investment banks. Project economics at a range of gold prices are evaluated as part of project sensitivity analysis in Section 22.

16.2.2 Silver Price

A silver price of US$28/oz. was used in the mineral resource estimate, and project economics were assessed using a silver price of US$ 37.2/oz, based on the long-term consensus forecast from over 20 investment banks.

16.3 Contracts

No contracts have been entered into at the report effective date for the Era Dorada operation.

16.4 Comments on Market Studies and Contracts

The qualified person has reviewed the relevant reports and analyses and is of the opinion that the marketing and commodity price information is suitable to be used in cashflow analysis to support this Study and its Technical Report. There are currently no firm contracts in place for the execution of the project (i.e. equipment, labour, power supply), however this is appropriate for the project in this phase.

17 ENVIRONMENTAL STUDIES, PERMITTING, PLANS, NEGOTIATIONS OR AGREEMENTS WITH LOCAL INDIVIDUALS OR GROUPS

The Era Dorada Project has an Environmental Impact Assessment (EIA) approved since 2007 by the Guatemalan Ministerio de Ambiente y Recursos Naturales (MARN), referring to the operation of an underground mine, which was the basis for the licensing of the project. In addition, it has a set of licenses and management plans that have been updated over the years, based on continuous environmental monitoring and the relationship with local communities. The project has the necessary permits to proceed with the development of the underground mine and the construction of the processing facilities, and the future operation is subject to compliance with the requirements established in the current permits. To date, the project has demonstrated standards of environmental performance and social responsibility, maintaining a comprehensive and up-to-date register of permits.

17.1 Environmental Considerations

The Era Dorada Project is licensed to move forward with the development of the underground mine, the implementation of processing structures and support areas. The continuity of activities is conditional on compliance with the requirements established in the current licenses, reflecting the project's commitment to legal and regulatory compliance. Since the approval of the EIA in 2007 by the MARN, the project has maintained an up-to-date permit registry, evidencing consistent compliance with legal commitments over time. The EIA includes an Environmental Management Plan that addresses plans such as the Closure and Recovery Plan and the Social Management Plan, both of which have been revised and adjusted for internal use, in line with the reality of the project in the 2019 Feasibility Study.

The project operates with a focus on environmental prevention and control, with an emphasis on water management, the adoption of best practices for waste and waste disposal in order to minimize the impacts of these materials, and compliance with international health, safety and environmental standards. The monitoring conducted by AURA Minerals confirms that the project meets the standards of environmental performance and social responsibility.

During the Feasibility Study development was identified the need to add two new structures WRDs and a DSTF during the mining life. Those structures are not considered in the original EIA, and a request for amendment will be necessary timely.

An ancillary license for the new power line and an effluent discharge line will be necessary.

17.1.1 Baseline and Supporting Studies

Baseline studies included hydrogeological characterization, surface and groundwater quality, air quality, as well as noise levels and biodiversity. These baseline studies and monitoring studies are presented in the following reports.

Table 17-1: Baseline Studies

Study	Year	Author
Report for the month of June	2022	Elevar Resources
Environmental Impact Assessment Study	2007	Corporacion Ambiental
Meeting Minutes - Community Development Board (MDC) - Cerro Blanco Community	2025	Aura Mining
Area of influence of the Cerro Blanco mining project	2020	GeoAmbiente
Social Management Strategy Cerro Blanco Project	2023	INSUCO
Technical Study of Wastewater - WWTP-	2025	Aura Mining
Water Analysis Results	2023	Ecosistemas – Proyectos Ambientales
Terrestrial Biology Monitoring Rainy Season	2023	Elevar Resources
Aquatic Biology Monitoring Dry Season	2023	Everlife
Air Quality Report	2023	Everlife

Source: AURA, 2025.

Aura Minerals has identified the requirement for developing new environmental and social baseline studies related to infrastructures not included in the 2007 EIA, especially the new power transmission line and the effluent discharge pipeline, which will be constructed after the appropriate permits are received.

17.1.2 Environmental Monitoring

The project maintains a schedule for environmental monitoring, in accordance with the commitments established in the EIA approved in 2007 and its updates. The results of these campaigns are consolidated in monthly and annual reports, which are regularly submitted to the relevant authorities — including the Ministerio de Ambiente y Recursos Naturales (MARN), the Ministerio de Energía y Minas (MEN), the Ministerio de Salud Pública y Asistencia Social, Ministerio de Salud de Jutiapa and Ministerio de Ambiente de Jutiapa.

17.1.2.1 Water Management

The existing water management methodology of the Era Dorada Project integrates underground and surface components, with infrastructure aimed at the control, treatment and proper disposal of water. The project currently has two Wastewater Treatment Plants (WWTP): one for the treatment of groundwater and the other for the sewage generated from the administrative infrastructure. The groundwater is pumped to the WWTP at a high temperature (due to geothermal conditions) and is cooled until it reaches values below 40 C. Detailed descriptions of the WWTPs are presented in Section 15 of this report. The main objective of the treatment is to remove arsenic and other metals present in the water. The project has a license for the discharge of treated water that meets established criteria, in the nearby river course: El Tempisque River (maximum 1.500 gpm per day) and Ostúa River (maximum 3.750 gpm per day). During this process, a volume of sludge is generated, which is directed periodically to licensed disposal ponds located within the project area. The generated sludge contains elevated levels of heavy metals such as arsenic, requiring

specific and safe management to avoid any risk of adverse impacts to soil, surface or groundwater, as well as potential adverse impacts to human health and local biodiversity. The sludge disposal ponds have been designed with a geomembrane liner to minimize the potential for contaminant release and are subject to frequent monitoring by the AURA team. However, as established in the EIA, at the early stage of mining operations, sludge generated by the Treatment Plant will be used as backfill for the underground mine. It is noted that a new water treatment facility is planned, dedicated exclusively to treating, as required, contact water from mine rock waste and tailings sources (refer to Section 15 – Surface Water Management).

Currently, the project does not have a Potable Water Treatment (PWTP), and the supply of potable water is provided by a specialized contractor retained by the company. Currently, there is also no discharge of mine effluents to the receiving environment (rivers), since process plant works in a closed circuit. Wastewater currently generated at site includes effluent from treated sewage. The installation of piezometers related to the monitoring of the underground mine are to be installed once mining has commenced and a monitoring program will be developed. Since 2011, a water quality monitoring (groundwater and surface water) and sampling program has been in place, with monthly compliance reports submitted and approved by MARN. This program will be maintained during the operational phase of the project with additional monitoring locations such as piezometers, ensuring compliance with legal effluent discharge criteria and requirements.

17.1.2.2 Air Quality and Noise and Vibration Management

Although the plans for monitoring air quality, noise and vibration are not included in the Environmental Impact Assessment (EIA), these activities were incorporated later and have been conducted for the Era Dorada Project area of influence. The results of this monitoring are periodically presented to MARN.

Air quality monitoring involves the measurement of environmental variables related to particulate matter and atmospheric gases, carried out in monitoring stations strategically distributed in the project's area of influence. Air quality is determined by the ratio between the volume of filtered air and the weight of the material collected, and the results are compared with international guidelines that establish limits for these particles. In addition, the project performs noise level monitoring, which consists of measuring sound pressure levels at receiving points over a continuous period of 24 hours. These activities are conducted by specialized environmental consultants, who use certified equipment and standardized methodologies to ensure the reliability and compliance of the data obtained.

17.1.2.3 Monitoring

The Project is currently operating and reporting on a comprehensive environmental monitoring schedule. As part of the commitments under the approved EIA, the Project conducts periodic and recurring monitoring to assess water quality, air quality, and noise levels in the project's area of influence. Reports of monitoring results are prepared and presented to the Authorities (Ministerio de Ambiente y Recursos Naturales (MARN), the Ministerio de Energía y Minas (MEN), the Ministerio de Salud Pública y Asistencia Social, Ministerio de Salud de Jutiapa and Ministerio de Ambiente de Jutiapa). For the underground mine project, environmental monitoring is carried out at the following stations and with the respective periodicities (Table 17-2).

Table 17-2: Monitoring Program

ID	Profile	Periodicity
SW (totaling 12 stations)	Surface Water / Sediment	Quarterly
HS5	Groundwater	Biannual
MW1.1	Groundwater	Biannual
MW2.2	Groundwater	Biannual
GW2	Groundwater	Biannual
GW6	Groundwater	Biannual
GW7	Groundwater	Biannual
GW8	Groundwater	Biannual
DPT-1	Discharged water quality AG 236-2006	Monthly
LPT-1	Sludge AG	Quarterly
CBA (totaling 06 stations)	Air Quality	Monthly
CBR (totaling 06 stations)	Environmental Noise	Monthly

Source: Elevate Resources Monitoring Report, 2023.

17.1.2.4 Solid Waste Management

Although the Environmental Impact Assessment allows for the disposal of inert and non-hazardous waste in an on-site landfill within the Aura property during the construction and operation phases, this practice was not adopted. Instead, the project relies on licensed and specialized waste disposal companies to carry out the collection, transport and proper disposal of all waste generated, including both common and hazardous waste.

17.1.2.5 Waste Rock and Tailings Management

As presented in Section 15, the project anticipates the generation of tailings and waste rock throughout the Life of Mine (LOM), requiring solutions for storage and environmental control. These materials will be placed in specific structures designed with geometries that ensure geotechnical stability and compliance with environmental standards.

The current capacity of the licensed Dry Stacked Tailings Facility (DSTF) is 498,489 m³, sufficient only for 3 years. From Year 4 onwards, it will be necessary to license a new DSTF to meet demand. Even with this expansion, an additional area will need to be licensed to accommodate extra tailings volumes.

The waste rock will be stored in structures such as North Waste Rock Dump (WRD), South WRD, DSTF (as a support shell), and the new WRD.. With its inclusion, the capacity will be sufficient for the entire LOM, provided licensing occurs as planned.

The drainage infrastructure was designed to separate contact water (originating from areas with tailings and waste rock) from non-contact water. Contact water will be directed to sediment control structures and treated at a Chemical

Effluent Treatment Plant, ensuring compliance with environmental standards before discharge. Non-contact water, on the other hand, will be channeled through drainage ditches to specific monitoring points before being discharged into the nearest natural watercourse.

In summary, tailings will be managed through two deposition streams: one stream will be filtered and returned underground as paste backfill, while the other will be filtered and placed in surface facilities, including engineered Dry Stack Tailings Facilities (DSTFs). Most waste rock will be stored underground as either cemented rock fill (CRF) or loose rock fill (LRF). The remaining volume will be disposed of in engineered Waste Rock Dumps (WRDs) located on the surface.

17.1.2.6 Flora & Fauna Management

Since 2007, the Era Dorada Project has been conducting environmental baseline studies to record the biodiversity of the region, characterized by subtropical and tropical dry forests that are home to diverse species of plants, birds, reptiles, and aquatic fauna. Monitoring indicates minimal impact of the project's activities on biological populations. Biological monitoring covers the components of terrestrial biology and aquatic biology, carried out twice a year: once during the dry season (first trimester) and another in the rainy season (third trimester). The studies are being carried out by a company with specialized expertise, authorized by the competent authority, and include qualified professionals for each component. The main objective is to evaluate the fauna and floristic diversity in the Project Area and in its areas of influence (direct and indirect). Specifically the objectives are to identify changes in the dynamics of wild populations caused by natural or anthropic factors, to verify patterns of richness and abundance of species, to evaluate the effects of management actions, and to propose improvements as required in conservation plans. The information obtained informs mitigation and conservation measures, such as preventive actions, rescue and relocation of endangered species, and revegetation with native species.

All reports are sent to the MARN according to the established deadlines, and include evaluation criteria, comparative indexes, laboratory analyses and distribution maps of the sampling points/areas.

17.1.2.7 Cultural and Archaeological Resources

The Project has a team responsible for monitoring and evaluating possible interference in cultural and archaeological assets. Before the start of the work, preventive inspections and consultations with external experts will be carried out, ensuring compliance with the applicable legal standards and requirements. To date, no relevant historical artifacts have been identified within the area of direct influence of the project. A valid archaeological permit has been issued in 2008 by the IDEH/Ministry of Culture, covering the proposed areas of ground disturbance.

17.2 Permitting Considerations

The Era Dorada Project will be implemented within the scope approved in the 2007 Environmental Impact Assessment (EIA), which includes the operation of an underground mine and the associated environmental licenses.

However, with the evolution of the project and the identification of new technical and operational needs, it will be necessary in the future to reconcile the progress of the project and obtain specific complementary licenses for proposed modifications to ensure compliance with the legislation.

17.2.1 Environmental Impact Assessment and Permits

Since the inception of the Era Dorada Project, several environmental studies and continuous monitoring activities have been carried out in the area of influence and, due to expiration dates, specific updates to existing permits have been necessary to ensure regulatory compliance.

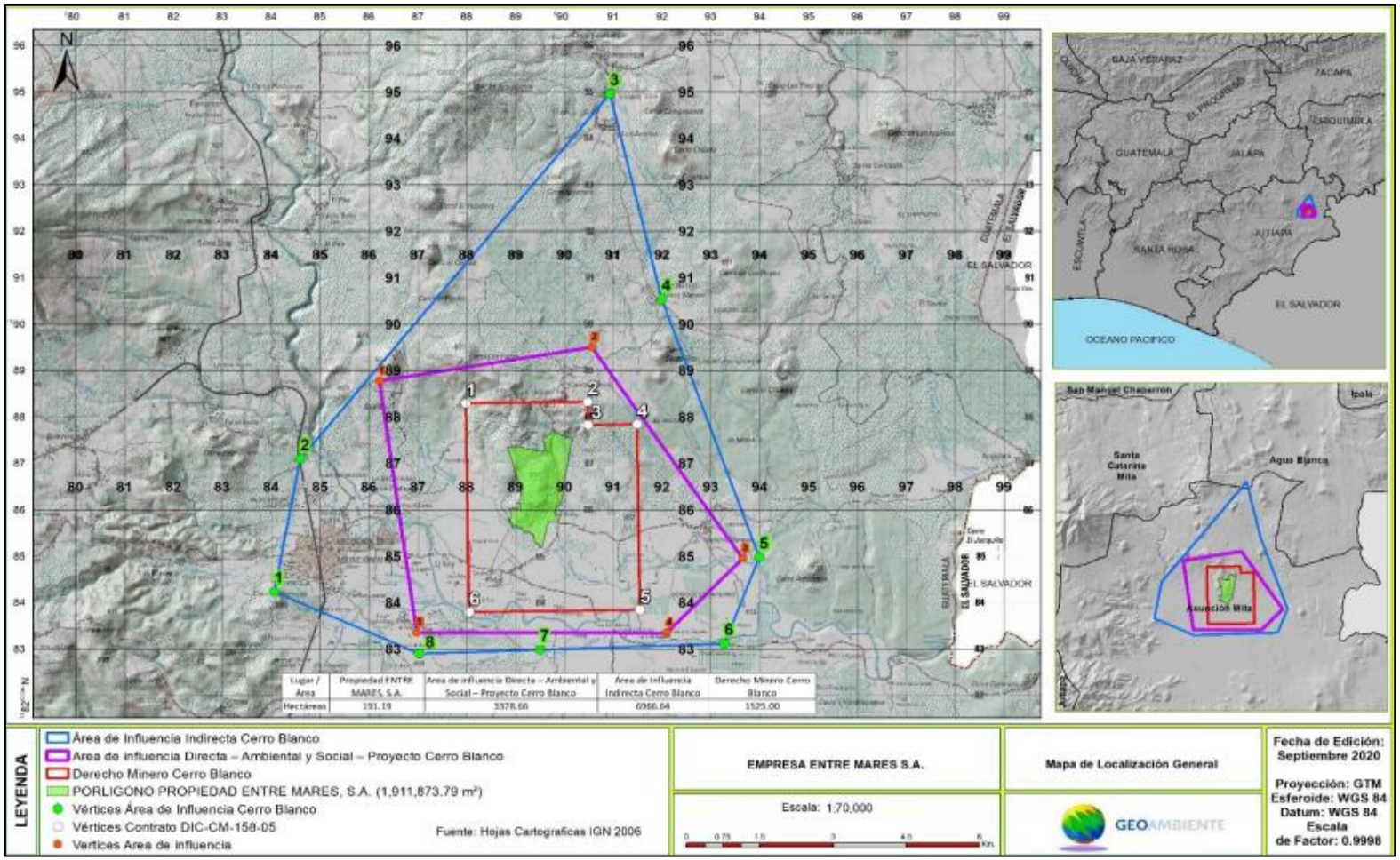
In addition, new environmental and social baseline studies aimed at environmental licensing for new project components (such as the power line) will be required since they were not contemplated in the previous studies.The originally approved EIA included a Social Management Plan and a Conceptual Mine Closure Plan that were reviewed internally, and revised, incorporating international best practices and modifications to the design as required.

17.2.1.1 EIA Areas of Influence

The Areas of Influence defined in the 2007 Environmental Impact Assessment (EIA) were updated in 2020 by a consultant (GeoAmbiental 2020) and formally communicated to the applicable environmental agency.

The new delimitation considers the direct and indirect impacts of the Era Dorada Project, reflecting changes in scope and territorial dynamics. The areas have been defined and are represented in the Figure 17-1, as follows.

Figure 17-1: Areas of Influence



Source: GeoAmbiente, 2020.

The Area of Direct Influence of the Era Dorada Project corresponds to the polygon of direct mining influence and its surroundings, covering the areas where the direct and significant environmental and social impacts of mining activity occur (classified as Level one of the Hierarchy of Social Interaction). In this area, the direct recipients of the impacts were identified and the geographic scope of the project's actions was delimited. The Area of Direct Social Influence comprises the communities, human settlements and populations located in the immediate vicinity of the project, where there is potential for socioeconomic and cultural impacts to occur. This delimitation also includes communities formally recognized as part of the area of direct influence through signed agreements. The Area of Indirect Influence, from a social point of view, covers communities classified at Level two of the Hierarchy of Social Interaction. This area does not include environmental aspects, since, according to the current configuration of the project, the occurrence of indirect environmental impacts in this region is not expected.

The communities included in each area are:

- Area of Direct Influence (IDA): (i) Cerro Blanco; (ii) El Cerrón; (iii) El Tule; (iv) Trapiche Vargas; and (v) La Lima.
- Area of Indirect Influence (AII): (i) Cabecera Municipal de Asunción Mita; (ii) San Rafael Cerro Blanco; and (iii) Las Ánimas.

17.2.2 Environmental Permits

The permitting process for the project is described below. The company submitted a formal application to the Ministry of Energy and Mines (MEM). During the review, the MEM requested technical opinions from institutions such as the National Geographic Institute, the National Forestry Institute (INAB), and the Ministry of Environment and Natural Resources (MARN), depending on the project's location. After receiving initial approval from the MEM, the company submitted the Environmental Impact Assessment (EIA) to MARN. This document detailed the potential impacts of mining activities on the environment, public health, and nearby communities, as well as the proposed prevention, mitigation, and compensation measures. It also included the citizen participation mechanisms that have been implemented. MARN carried out a technical and legal review of the EIA, requested additional information, and conducted field inspections. Once the evaluation was completed and all requirements were met, MARN issued an environmental approval resolution, authorizing the project's execution under specific conditions. This resolution was essential for the MEM to grant the mining exploitation license, marking the final step of the process.

The approved EIA and permits allow the project to proceed with the development of the underground mine and the construction of the processing facilities, provided that future operations comply with the requirements of the existing permits. As the project progresses and new technical and operational demands are identified, it will be necessary to align these developments with legal requirements by obtaining specific complementary permits for the proposed modifications. The project's current permits and licenses are summarized in the Table 17-3.

Table 17-3: Current Permits

License/Document	Number Resolution	Responsible Agency	Object	Duration	Status
EIA 2007	-	MARN	Underground Mine & Plant		
For the entire life of the project	Current; Requires amendments and update every five years				
Mining Concession	Resolution No.1942 MEM	MEM	Underground exploration	By 2032	Current
Property Registry	-	Property Registration	Mining title	2007 – 2032	Current
Environmental Underground Licence Category A	2613-2007/ECM/LP	MARN	Environmental monitoring	2028	Effective, renewal every five years
Forestry License #1 (East Zone)	No. 40-2205-155-1.6-2007	INAB	Vegetation Suppression	2030	Current, renewable every five years
Forestry License #2 (West Zone)	No. 40-2205-035-1.1.5.2020	INAB	Vegetation Suppression	2031	Current, renewable every five years
Current Diesel Tank Operating Licence, Own Consumption – 20.000 galon	Lic No. 0627	MEM / Ministry of Energy	Fuel storage	Until January 2029 (renewable)	Current
Amendment Handling and disposal of sludge	Resolution 03749-2019 - DIGARN/MOCMD/RJOP	MARN	Waste management and disposal	By 2027	Current; renewal every five years
WTP Handling and disposal of sludge	Resolution 00244-2016-DIGARN/FACD/gamc MARN	MARN	Waste management and disposal	2028	Current
Resolution: no pre-Hispanic or paleontological remains in the Project area	Opinion No. 002/mc.2008 Department of Pre-Hispanic and Colonial Monuments	IDAEH / Ministry of Culture	Cultural heritage	Issued in 2008	No relevant findings
Discharge Abatement Era Dorada Project and Environmental Management Plan - Category B2	511-2011/DIGARN/ECM/caml MARN	MARN	Tempisque – 1,500 gallons/min	Until March 2028	Current; renewal every five years

License/Document	Number Resolution	Responsible Agency	Object	Duration	Status
Medical Clinic	Sanitary License 14047 Ministry of Health and Social Assistance June 14th, 2016	Ministry of Health and Social Assistance	Outpatient clinic	Until 2026	Current
Current Era Dorada Building Permit Municipality As. Mita	-	Municipality As. Mita	Buildings used for current operation	Indefinite	Current

Source: AURA, 2025.

17.2.3 Additional Permits and Authorizations

The complementary environmental licenses provided for the continuity of the project are detailed in Table 17-4.

Table 17-4: Main Permit Amendments & New Permit Required

License	Action Required
New Power Line	New EIA and Permit – MARN Clearing Forest Permit – INAB Construction Permit – Asunción Mita Municipality
Effluent Line Discharge	Clearing Forest Permit -INAB Construction License – Asunción Mita -Municipality
Diesel Storage Tank Expansion	New Permit – MEM
Export Permit	New Permit - MEM
Project Construction Permit	New Permit - Asunción Mita -Municipality
Licenses Related to Explosives	Responsibility of a contracted company that must hold all required licenses, including: (i) License for the Use of Explosives; (ii) License for Export and Sale; (iii) License for Importation, Domestic Acquisition, and Storage; (iv) License for Manufacturing/Processing of Explosives (v) License for Transportation of Explosives (vi) Explosives Specialist License (vii) Permit for Construction Explosives Storage Facility.
Licenses Related to the new DSTF and WRD	New Permit – Asuncion Mita - Municipality

Source: AURA, 2025.

17.3 Social Considerations

The social strategy of the Era Dorada Project aims to strengthen relations with local communities, based on the transparency of management and the traceability of interactions. To this end, the project uses a database that records all relationship activities, a Social Management Plan (SMP) and a Social Baseline study, which includes communication mechanisms, response to complaints and socioeconomic monitoring. Initially presented in the 2007 Environmental Impact Assessment (EIA), the SMP has been aligned with the practices adopted by the Era Dorada Project over the years. Its purpose is to support the integral development of communities in the area of influence, promoting economic, social, cultural and political advances, in addition to mitigating negative impacts.

The SMP adopts a participatory methodology, involving planning, execution, monitoring and evaluation together with the main local stakeholders: communities, municipal government, community organizations, the company and its employees. The goal is to ensure that decision affecting communities are made in an inclusive and transparent manner.

The communities that are included in the SMP and have a direct impact are: Cerro Blanco, El Tule and Cerrón. The communities with secondary attention in the plan are: Trapiche Vargas, San Rafael Cerro Blanco and Las Ánimas. The Plan includes site visits and opportunities for dialogue with community organizations, especially with the Consejos Comunitarios de Desarrollo Urbano y Rural (COCODEs), Conselho Municipal de Desenvolvimento (COMUDE), Corporação Municipal de Asunción Mita and organized associations and groups.

Social monitoring is an essential part of the SMP and aims to monitor changes in social and cultural processes in the area of external influence. It makes it possible to identify significant transformations and propose joint solutions that benefit both communities and the company. This process also evaluates whether the conditions identified in the socioeconomic and cultural baseline have been altered by the project activities. The purpose is to verify whether the company's actions contributed positively to social, economic and cultural aspects or if there were negative impacts, based on the information collected before the start of the project.

In 2018, the update of the social baseline (Social Capital Group (2018)) was completed, incorporating the most recent official data from Guatemala, a rural census, and information obtained from meetings with local representatives. This update allowed for the review of relevant social aspects and improved engagement with stakeholders, based on international best practices and IFC standards. In the same year, the Social Management System (SMS) was developed and initiated, which includes a database to record all community engagements, activities and key information related to the relationship with communities.

17.4 Closure and Reclamation Planning

17.4.1 Closure and Reclamation Plans

As mentioned earlier, the approved Environmental Impact Assessment (EIA) includes a Conceptual Mine Closure Plan, which outlines the guidelines for the decommissioning of the project facilities after operations have ceased. According to regulations in Guatemala, the Closure Plan must be submitted to MARN for official approval three years before the end of the mine's activities. Therefore, at this time, the company is not yet legally required to submit a detailed closure plan to regulators and there is no requirement for post reclamation bonds.

During the development of the Feasibility Study (2019), the Conceptual Closure Plan was reviewed for Aura's internal management and costing purposes. As part of this study, an updated cost estimate for the closure was prepared, which is incorporated into the overall project cost estimate. The plan prioritizes environmental protection and the well-being of impacted communities, both in the short and long term. This includes proactively planning for the closure of operations and ensuring that the necessary financial resources will be available during the operational phase, ensuring the future implementation of the plan responsibly and effectively.

The structure of the plan is based on the technical information contained in the Feasibility Study, covering the following areas of the project: safety; general infrastructure, processing plant, water treatment plant, piping systems, dams and tanks, distribution yard and electrical substation, administrative offices and auxiliary buildings, dry tailings disposal facility (DSTF), disposal rock embankment and wells.

17.4.2 Closure Cost Estimates

Mine closure costs have been estimated based on typical closure, environmental recovery and monitoring activities of an underground mine, including the removal of surface infrastructure and underground equipment, the closure and coverage of the Waste and Tailings Deposit (DSTF), the closure of mine accesses, the removal of the transmission line and electrical substation, in addition to the re-vegetation of the impacted areas and post closure environmental monitoring. The Table 17-5 Provides a summary of the cost categories involved.

Table 17-5: Cost Estimates

Item	Estimated Cost (MUSD)
Safety	0.45
Underground mine	1
Infrastructure	0.35
Process Plant	5.74
Water Treatment Plant	1.1
Piping, ponds and tanks	1.98
Switchyard and Power Distribution	0.4
Administration Office and Ancillary Buildings	0.25
Drystack Tailings Facility (DSTF)	2.2
Waste Rock Dumps	0.43
Wells	1.2
Monitoring	2.1
Total Closure	**17.2**

Source: Aura, 2025.

17.5 Comments on Environmental Studies, Permitting and Plans, negotiations, or agreements with local individuals or groups

The project currently has all the required environmental licenses, including the Environmental Impact Assessment (EIA) approved for the underground mining operation. However, some of the proposed modifications will require updates or new regulatory authorizations. Delays in approving these changes or obtaining new permits can impact the development timeline, resulting in potential schedule delays.

The implementation of the power transmission line and effluent discharge pipeline requires special attention to the properties neighboring the project, which may be affected by the installation of the structures. It will be necessary to establish partnerships with local landowners to make these interventions viable.

As for the management of the tailings and waste rock storge facilities, they are currently assumed to be non-acid generating (NAG), based on the preliminary geochemical tests carried out to date. However, additional tests will be conducted before the detailed engineering phase to confirm this assumption. If the materials are classified as Potentially Acid Generating (PAG), the project design should be reviewed and updated to incorporate appropriate control and mitigation measures.

While the local community, in general, supports the development of the Era Dorada Project as an underground mine, there is a potential risk of socio-political opposition that could negatively impact the implementation schedule. This risk will be continuously monitored, and stakeholder engagement strategies are being considered to mitigate potential impacts.

18 CAPITAL AND OPERATING COSTS

18.1 Introduction

The capital and operating cost estimates presented in this report provide substantiated costs that support the feasibility study of Aura's Era Dorada project. The estimates considered the beneficiation plant with an average gold production capacity of 103,96 oz/a.

The capital and operating cost estimate aligns with Class 3 guidelines for an FS-level estimate with an accuracy range of +/-15% as specified by the Association for the Advancement of Cost Engineering International (AACE International). The estimate, developed in Q4 2025, is based on the proposed design for the Project, on Ausenco's budgetary quotations, in-house project and study database and Aura's inputs.

All capital and operating cost estimates are presented in US dollars (USD) and Guatemalan Quetzal (GTQ). The exchange rate applied is:

Guatemalan Quetzal (GTQ) to US dollars (USD): GTQ7.60 = USD1.00.

18.2 Mine Costs

The mining costs were estimated by Snowden-Optiro with the support of Ausenco for the quotations.

Mining costs were assessed in a monthly basis for the initial three years during the ramp-up and pay-back period and are reported in yearly intervals herein in this section.

As the mine has to be developed before the commercial production from the operation, which defines the Project capex in terms of time, the classification of expenses has to follow a timeline whereby, for the purpose of the financial valuation, all mining costs are classified as capex prior to the commercial production of the plant (even including any ore development and stoping costs, which generate a credit in the valuation process), and, after that, the costs of primary and secondary development (driven in waste), related to the establishment of the mine infrastructure (excavations) as identified in Table 18-1, are classified under the capex expenditures up to the commercial production of the plant and, from that point forward, they are classified as sustaining capital expenses; the costs of ore development and stoping (which are excavated in ore) are classified as capex expenditures up to the commercial production and, later, are classified as operational expenses (opex).

The same reasoning applies for the expenses related to the mobile fleet acquisition as identified in Table 18-3 and infrastructure costs as summarized in Table 18-4.

18.2.1 Excavation Costs Estimates

The mining costs were derived from first principles. As for the excavation costs, representing the higher proportion, data from the drill and blast patterns and those from the excavation sections standards were matched to equipment productivities for the different categories of excavations in the mine design.

The approach requires the definition of individual consumption of supplies and equipment hours necessary to excavate each meter of the linear development excavations and each tonne of the Cut-and-fill Pivots as well as the Long hole and Cut-and-fill stopes, which are (inversely) proportional to the equipment productivities for the following categories of cost:

- Supplies:
  - Drilling materials, such as drill shanks, rods and bits.
  - Explosives and initiation accessories.
  - Rock reinforcement and support elements such as rock bolts (hydrabolts), mesh, cement, cement admixtures, aggregate, and fiber for shotcrete and cement and admixtures for the pastefill.
- Equipment costs – equipment hours were estimated to complete units (meters or tonnes) for each type of excavation to define:
  - Mobile equipment maintenance costs.
  - Costs of ground engaging tools (excluding those of drilling materials already accounted for as above).
  - Diesel and DEF (Diesel Exhaust Fluid) consumption and costs.
  - Power costs for the electrohydraulic and electric systems of the mobile equipment.

An allowance of 10% of the known costs elements was applied to cover minor expenses not captured in the costs estimates.

The estimates for the excavation unit costs are shown in Table 18-1.**Erro! Fonte de referência não encontrada.**

Table 18-1: Mining Excavation Unit Costs per Category of Excavation

Reference	Type	Class	Section	Unit costs (USD/m,t)
Cost per meter (USD/m)	Ramp	Primary Development	5x5 m^2	$7.633
	Ventilation drifts		4x4 m^2	$6.115
	Exploration drifts		4x4 m^2	$6.262
	Other drifts 4x4 m^2		4x4 m^2	$6.142
	Raise Boring		3100 mm dia	$3.728
	Raises		2x2 m1	$3.865
	Access / Long hole	Secondary Devpmt	4x4 m1	$5.914
	Other drifts 5x5 m^2	Primary/Secondary Devpmt	5x5 m1	$6.643
Cost per tonne (USD/t)	C&F Pivots	Secondary Devpmt		$51
Cost per meter (USD/m)	Ore development / Long hole	Ore Development	4x4 m^2	$5.819
	Ore development / CF (Sill drives)		3.5x3.5 m^2	$969
Cost per tonne (USD/t)	Stoping LH 64 mm dia	Stoping	64 mm dia	$46
	Stoping LH 76 mm dia		76 mm dia	$45
	Stoping C&F		3.5x3.5 m^2	$51

Erro! Fonte de referência não encontrada. data were combined with the LoM plan quantities to define the estimates for the mining excavation costs per category as shown in **Erro! Fonte de referência não encontrada.**.

Table 18-2: Mining Excavation Costs per Category of Excavation

Type	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17
Early works	$3,822	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Ramp	$7,628	$14,279	$9,102	$5,219	$3,838	$1,493	$1,215	$1,574	$1,628	$0	$0	$0	$0	$0	$0	$0	$0	$0
Ventilation drifts	$5,142	$4,327	$7,172	$2,789	$1,045	$2,074	$446	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Exploration drifts	$1,360	$7,315	$3,679	$1,557	$535	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Other drifts 4x4	$1,475	$1,677	$1,056	$1,544	$301	$491	$89	$196	$6	$0	$150	$0	$300	$0	$0	$0	$0	$0
Raise Boring	$360	$0	$497	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Raises	$563	$1,220	$1,449	$736	$337	$299	$773	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Acess/ LH	$51	$384	$1,315	$803	$940	$2,324	$1,141	$551	$1,682	$699	$669	$600	$525	$949	$834	$0	$0	$0
Other drifts 5x5	$8,945	$18,368	$14,247	$12,580	$7,226	$8,260	$1,687	$2,075	$2,349	$1,777	$98	$2,311	$1,024	$1,781	$2,660	$17	$0	$0
C&F Pivots	$0	$23	$0	$91	$1,099	$966	$1,044	$1,136	$965	$317	$320	$350	$260	$120	$6,692	$0	$0	$0
Ore development/ LH	$5,481	$8,486	$16,140	$26,964	$29,710	$30,279	$28,430	$14,715	$17,573	$6,891	$6,492	$6,777	$7,002	$8,291	$8,270	$1,203	$549	$0
Ore development/ CF (Sill drives)	$0	$0	$0	$41	$136	$94	$285	$219	$106	$0	$169	$50	$194	$0	$0	$0	$0	$0
Stoping LH 64 mm	$0	$273	$7,145	$6,656	$12,444	$11,990	$12,961	$15,933	$16,149	$19,093	$18,833	$18,929	$19,072	$18,823	$19,024	$20,374	$20,433	$16,904
Stoping LH 76 mm	$0	$76	$1,978	$1,846	$3,441	$3,317	$3,589	$4,413	$4,472	$5,291	$5,221	$5,245	$5,286	$5,217	$5,271	$5,652	$5,669	$4,712
Stoping C&F	$0	$0	$0	$54	$1,081	$875	$1,552	$2,166	$636	$264	$921	$497	$290	$53	$0	$0	$0	$0

The capex component of the excavation costs, from Year -1 to commercial production in Year 1 is US$81.5 million. Once the project achieves commercial production, the costs for excavations for mine infrastructure in waste are capitalized (corresponding to sustaining capital) and the costs for excavation in ore (for ore development and stoping) are classified as operational (Opex).

18.2.2 Mobile Equipment Purchases

Aura will acquire its fleet of mobile equipment, to operate stoping on an onwner basis, while the development contractor will acquire its equipment following Aura's directions. The costs for the acquisition and replacement of Aura's fleet are detailed in Table 18-3.

Table 18-3: Mine Mobile Fleet Acquisition Costs

Equipment	Unit cost (USD)	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17
1 boom jumbo	$1,081					$1,081							$1,081						
Production drill	$1,570		$1,570								$3,140						$1,570		
LHD 10t	$1,272		$1,272			$1,272			$1,272		$1,272	$1,272			$1,272		$1,272	$1,272	
LHD 7t	$935					$935							$935						
Exploration drill / 200m holes	$628	$628						$628						$628					
Exploration drill / 100m holes	$628		$628						$628						$628				
Explosives truck	$600		$600							$600							$600		
Shotcrete transporter / mixer	$640		$640							$640							$640		
Scissor lift	$765		$765							$765									
Personnel carrier	$628		$628							$628									
Fuel/lube truck	$585		$585							$585							$585		
Backhoe	$116		$116							$116							$116		
Telehadler	$219		$219							$219									
Refuge Chambers 16 people	$135	$269																	
Refuge Chambers 20 people	$147	$442	$147			$147													
Heavy equipment + Refuge Chambers		**$1,339**	**$7,171**	**$0**	**$0**	**$3,435**	**$0**	**$628**	**$1,900**	**$3,554**	**$4,412**	**$1,272**	**$2,016**	**$628**	**$1,900**	**$0**	**$4,784**	**$1,272**	**$0**

The capex component of the fleet acquisition and replacement, from Year -1 to commercial production in Year 1 is US$8,510 million. After the project achieves commercial production, the fleet acquisition and replacement costs are classified as sustaining capital.

18.2.3 Infrastructure

The costs for mine infrastructure were also estimated for the Ventilation System, the Pumping System, Electric Substations, Electric Materials and other minor elements such as ventilation ducts, pastefill piping and the dispatch system. **Erro! Fonte de referência não encontrada.** shows the costs for the major elements of mine infrastructure.

Table 18-4: Mining Costs – Infrastructure – Major Elements

Reference	Unit	Unit costs (USD)	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17
Ventilation																				
Main Centrifugal Fans SH1-EXT-1/2	qty	$2,577	$2,577																	
Main Centrifugal Fans SH4-EXT-1	qty	$239			$239															
Main Centrifugal Fans NH6-EXT-1/2	qty	$291	$291																	
Civil Works Ventilation System Ventilation System	percent	2%	$197		$476															
Electromechanical assembly Ventilation System	percent	16%	$878		$39															
Maintenance Ventilations System	percent	5%	$16	$274	$34	$393	$393	$393	$393	$393	$393	$393	$393	$393	$393	$393	$393	$393	$393	$393
Auxiliary Fans VAUX-1	qty	$81	$244				$163					$163					$163			
Auxiliary Fans VAUX-2	qty	$59	$652	$59	$356	$59	$712	$59	$356	$59	$237	$474	$59	$356	$59	$237	$474	$59	$356	$59
Auxiliary Fans VAUX-3	qty	$122	$367				$122					$122					$122			
Vent ducts and doors			$22	$138	$128	$38	$119	$3	$3	$1										
Pumping																				
Centrifugal Pump - STH42-PP-17/18/19/2	qty	$2,394	$96																	
Centrifugal Pump - NTH37-PP-5/6/7/8	qty	$7,292		$281																
Centrifugal Pump - STH32-PP-13/14/15/16	qty	$76,427			$36															
Centrifugal Pump - NTH27-PP-1/2/3/4	qty	$76,427						$36												
Centrifugal Pump - STH21-PP-9/1/11/12	qty	$76,427						$36												
Civil Works Pumping System	percent	2%	$19	$56	$61			$122												
Electromechanical assembly Pumping System	percent	16%	$15	$45	$49			$98												
Maintenance Pumping System	percent	5%	$1	$2	$24	$34	$34	$65	$34	$34	$34	$34	$34	$34	$34	$34	$34	$34	$34	$34
Submersible Pump -Working areas-PP21@28	qty	$785	$39		$8	$8	$8													
Submersible Pump - NTH22-PP-29@3	qty	$9,455	$19																	
Submersible Pump - STH32-PP-3@31	qty	$1,723			$21															
Submersible Pump - STH22-PP-33@36	qty	$24,735	$49							$49										
Submersible Pump - STH16-PP-37@4	qty	$3,159				$63		$63												
Piping			$556	$592	$192	$33	$114	$215								$44				
Electric Subestations			$266	$125	$1,694	$851	$636	$280		$590						$875				
Electric Materials			$45	$33	$27	$32	$14	$18	$16	$4			$15			$15	$30			
Paste fill system (pilot holes and piping)			$84	$387	$978	$177	$214	$45	$384	$265	$3									
Dispatch System			$88	$12																

The capex component of the mine infrastructure, from Year -1 to commercial production in Year 1 is US$16,168 million. Once the project achieves commercial production, the mine infrastructure costs are classified as sustaining capital.

The mine Capital Costs are summarized in Table 18-5.

Table 18-5: Mining Costs – Capex Summary

Reference	Capital Costs (MUSD)
Mine – Underground	
Excavation	$81,454
Mine Mobile Equipment	$8,510
Mine Infrastructure	$16,168
Mine – Surface Infrastructure	
Mine Dewatering Pumps	$13,872
Mine Paste Fill	$11,123
Mine Refrigeration Plant	$24,183
Mine Pipeline	$138
Mine Electrical Equipment	$779
Mine Electro Mechanical Assembly	$1,458
Mine Civil Works and Architecture	$118
Total Mine Capital Costs	**$158,224**

The mine Sustaining Capital Costs and Opex are summarized in Table 18-6.

Table 18-6: Mining Costs – Sustaining Capital Costs and Opex

Reference		Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17
Excavation	Sustaining Capital	$5.9	$38.5	$25.3	$15.3	$15.9	$6.4	$5.5	$6.6	$2.8	$1.2	$3.3	$2.1	$2.9	$10.2	$0.0	$0.0	$0.0
	Opex	$3.9	$25.3	$35.6	$46.8	$46.6	$46.8	$37.4	$38.9	$31.5	$31.6	$31.5	$31.8	$32.4	$32.6	$27.2	$26.7	$21.6
Mine Mobile Equipment		$0.0	$0.0	$0.0	$3.4	$0.0	$0.6	$1.9	$3.6	$4.4	$1.3	$2.0	$0.6	$1.9	$0.0	$4.8	$1.3	$0.0
Mine Infrastructure		$0.2	$7.4	$1.7	$2.5	$2.3	$1.2	$1.4	$0.7	$1.2	$0.5	$0.8	$0.5	$2.0	$1.2	$0.5	$0.8	$0.5
Mine Surface Infrastructure																		

18.3 Capital Costs

18.3.1 Overview

The capital cost estimate aligns with Class 3 guidelines for an FS-level estimate with an accuracy range of +/- 15% as specified by the Association for the Advancement of Cost Engineering International (AACE International). The capital cost estimate, developed in Q4 2025, is based on the proposed design for the Project, on Ausenco's budgetary quotations, in-house project and study database and Aura's inputs.

The total capital cost summary is presented in Table 18-7. The total capital cost for the Era Dorada Project is US$202.36 million, of which US$197.58 million is for the Plant and US$4.79 million for the Tailings, Waste Rock and Stockpiles (with contingency).

The project's Main electrical substation, Power transmission line and Connection substation will be implemented as Sustaining Capital in Year 3 (2030). From Years -2 to 3 (2026 to 2030) the site will be powered by generators, and from Years 4 to 16 (2031 to 2043) the site will be powered by the local energy company.

Table 18-7: Capital Cost Estimate

Description	Total Cost (MGTQ)	Total Cost (MUSD)
Process Plant	681.34	89.65
On-Site Infrastructure (Earthworks, Ancillary Facilities, Laboratory, Water Treatment Stations)	156.31	20.57
Tailings, Rock Waste and Stockpiles	34.55	4.55
Off-Site infrastructure (External access roads construction)	8.48	1.12
Direct Costs	**880.68**	**115.88**
Owner's Costs	265.44	34.93
Project Indirects	210.67	27.72
Contingency	181.18	23.84
Indirect Costs	**657.29**	**86.49**
Mining costs	1204.95	158.55
Mining costs contingency	160.29	21.09
Total Capital Costs	**2,904.06**	**382.11**

Notes: Values may not sum correctly due to rounding.

18.3.2 Basis of Estimate

The capital cost estimate, developed by Ausenco in Q4 2025 USD, is based on budgetary quotations from equipment suppliers, construction contractors, an in-house database of projects and studies, and experience from similar

operations. Due to the methodology employed and feasibility level of engineering definition, the estimate has an accuracy range of +/-15%, in line with the Class 3 guidelines of the Association for the Advancement of Cost Engineering International (AACE International).

The capital cost estimate was developed in Guatemalan Quetzal (GTQ), and the exchange rate used in the estimate are presented in Table 18-8.

Table 18-8: Exchange Rate

Currency	Currency Abbreviation	Exchange Rate
United States Dollar	USD	7.60

The data used for the estimates has been obtained from numerous sources, including the following:

- quantities from preliminary Bill of Quantities prepared by the technical team;
- quantity Growth allowances were applied;
- quotations for packages;
- databases;
- Aura's inputs;
- contingency was estimated as 13.3% (P_{70}), based on a risk analysis of the project, utilizing a Monte Carlo Simulation.

The following costs and scope items are excluded from the capital cost estimate:

- scope changes, project schedule changes, and other associated costs;
- any facilities or structures not included in the project scope;
- tax benefit analysis; and
- demolition or decontamination costs for existing site.

18.3.3 Process Capital Costs

Process equipment requirements were defined based on process flowsheets and process design criteria. All major equipment was sized according to the process design criteria and mass balance to create a mechanical equipment list. Mechanical scopes of work were developed and sent to equipment suppliers for budgetary pricing. For mechanical equipment costs, 96% of the value was sourced from budgetary quotes. For platework supply, 94% of the value was sourced from budgetary quotations. The remaining costs were determined by Ausenco's recent database, adjusted to the Project's base date where applicable. Mechanical equipment costs include international freight costs and spare parts for start-up and commissioning. In-land national freight costs and spare parts for one year of operation are included in indirect costs.

Major electrical equipment was sized according to the equipment list. Scopes of work were developed to receive budgetary pricing from equipment suppliers. For electrical equipment, 48% of the value was sourced from budgetary quotations. The remaining costs were determined using Ausenco's recent database, adjusted to the Project's base date where applicable.

All structural steel required for the process plant was sized to generate material take-offs. For structural steel, 100% of the value was determined through budgetary quotations.

Process plant piping was sized to generate a material take-off, with 53% of its value determined through budgetary quotations. The remaining costs were sourced by Ausenco's recent database, adjusted to the Project's base date where applicable. For valves, 100% of the value was based on Ausenco's database.

For major piping and valves (excluding pipeline, described above), electric material and accessories 100% of the value was index-based.

For automation equipment, instruments and telecommunication systems and implementation, 38% of the value was determined through budgetary quotations. The remaining costs were index-based.

Electromechanical assembly costs cover the services required for the Project's implementation, including the assembly of electrical materials, steel structures, platework, piping, automation, and instrumentation. This cost includes all the direct and indirect expenses of the contractor performing the services. For electromechanical assembly, 77% of the value was determined through budgetary quotations. The remaining costs were sourced by Ausenco's recent database, adjusted to the Project's base date where applicable.

Budgetary quotations were obtained for 83% of the value of civil works. The remaining costs were sourced by Ausenco's recent database, adjusted to the Project's base date where applicable.

The process plant costs are presented in Table 18-9.

Table 18-9: Process Plant Capital Costs

WBS	WBS Description	Initial Capital Cost (MGTQ)	Initial Capital Cost (MUSD)
0005	Indirects	16.99	2.24
0025	Temporary Installations	27.45	3.61
1000	General Plant Services	125.84	16.56
1005	Plant Infrastructure	1.63	0.21
1008	Site Clearing	0.85	0.11
1009	Drilling and Topographic Survey	14.70	1.93
1010	Primary Crushing (except ROM Pad)	14.91	1.96
1025	Crushed Ore Stockpile	4.46	0.59
1030	SAG Milling	56.66	7.45

WBS	WBS Description	Initial Capital Cost (MGTQ)	Initial Capital Cost (MUSD)
1032	Ball Milling	28.76	3.78
1033	Pebble Circuit	9.0	1.18
1035	Pre-Leach Thickener	8.60	1.13
1045	Pre-Oxidation	25.52	3.36
1047	Adsorption / Carbon in Pulp (CIP)	18.33	2.41
1050	Carbon Elution	38.69	5.09
1055	Electrowinning	5.62	0.74
1060	Carbon Regeneration	0.28	0.04
1065	Cyanide Detoxification	7.91	1.04
1070	Cyanide Preparation and Storage	4.61	0.61
1071	Reagent Preparation and Storage	14.12	1.86
1073	Tailings Filtration	67.27	8.85
1077	Tailings Thickener	3.70	0.49
1080	Gravity Concentration	5.02	0.66
2000	Utilities	1.77	0.23
2010	Reclaim Water System	0.04	0.0
2015	Water Intake System (Pipeline)	28.02	3.69
2016	Raw Water Excavated Reservoir (Sumps and Ponds)	3.71	0.49
2017	Reuse Water Outfall	1.61	0.21
2020	Process Water System	2.83	0.37
2021	Raw Water System	1.34	0.18
2022	Gland Seal Water System	0.49	0.06
2025	Potable Water Distribution System	0.24	0.03
2030	Water Treatment System - WTP	30.96	4.07
2035	Sewage Treatment System - STP	0.0	0.0
2037	Industrial Effluent Treatment System - IETS	0.82	0.11
2040	Compressed Air System	1.84	0.24
2041	Compressed Air System - Instruments	0.33	0.04
2050	Fire Protection and Fighting System - FPFS	1.22	0.16
2055	Cooling System	3.74	0.49
2060	Oxygen Plant	24.87	3.27
3015	Metallurgy Secondary Substation	31.74	4.18
3020	Crushing Secondary Substation	3.12	0.41
3055	Recovered Water Intake Secondary Substation	4.52	0.60

WBS	WBS Description	Initial Capital Cost (MGTQ)	Initial Capital Cost (MUSD)
3060	Filtration Secondary Substation	0.04	0.01
3075	Internal Power Distribution Network	14.49	1.91
4000	Site Infrastructure	0.84	0.11
4005	Construction Yard	7.88	1.04
4015	Disposable Material Center	0.06	0.01
4020	Warehouse	0.63	0.08
4025	Physical Laboratory	0.13	0.02
4030	Main Gate	1.46	0.19
4035	Truck Scale	0.59	0.08
4045	Medical Center	0.94	0.12
4050	Plant Locker Room	0.18	0.02
4060	Dining Facility	2.10	0.28
4065	Central Office	2.49	0.33
4075	Plant Maintenance Workshop	1.86	0.24
5035	Workshop	4.37	0.58
Total Process Plant		**682.17**	**89.76**

Note: Values may not sum correctly due to rounding.

18.3.4 On-site Infrastructure Capital Costs

Material take-offs were developed for earthworks and geotechnical services to obtain pricing for the capital cost estimate.

For Laboratory costs, 100% of the value was provided by Aura.

For Water Treatment costs, 81% of the value was determined through budgetary quotations. The remaining costs were sourced by Ausenco's recent database, adjusted to the Project's base date where applicable.

For Earthworks, 66% of the total value was determined through Ausenco's budgetary quotations. The remaining costs were sourced by Ausenco's recent database, adjusted to the Project's base date where applicable.

The project's Main electrical substation will be implemented as Sustaining Capital in Year 3 (2030). From Years -2 to 3 (2026 to 2030) the site will be powered by generators, and from Years 4 to 16 (2031 to 2043) the site will be powered by the local energy company.

The on-site infrastructure costs are presented in Table 18-10.

Table 18-10: On-Site Infrastructure

WBS	WBS Description	Initial Capital Cost (MGTQ)	Initial Capital Cost (MUSD)
0025	Temporary Installations	0.98	0.13
0030	Access Roads	11.0	1.45
1006	Infrastructure - Earthworks	21.20	2.79
1007	Infrastructure - Drainage	8.30	1.09
2030	Water Treatment System - WTP	73.27	9.64
2035	Sewage Treatment System - STP	3.61	0.48
2037	Industrial Effluent Treatment System - IETS	17.25	2.27
4025	Physical Laboratory	20.69	2.72
Total On-Site Infrastructure		**156.31**	**20.57**

Note: Values may not sum correctly due to rounding.

18.3.5 Off-site Infrastructure Capital Costs

The off-site infrastructure costs consist of External access roads construction. The quantities were provided by Aura, and 100% of values were sourced by Ausenco's recent database, adjusted to the Project's base date where applicable.

The project's Power transmission line and Connection substation will be implemented as Sustaining Capital in Year 3 (2030).

The off-site infrastructure costs are presented in Table 18-11.

Table 18-11: Off-Site Infrastructure Capital Cost

WBS	WBS Description	Initial Capital Cost (MGTQ)	Initial Capital Cost (MUSD)
0030	Access Roads	6.99	0.92
1007	Infrastructure - Drainage	1.40	0.18
1008	Vegetation Suppression	0.09	0.01
Total Off-Site Infrastructure		**8.48**	**1.12**

Note: Values may not sum correctly due to rounding.

18.3.6 Stockpile Capital Costs

Stockpile costs includes tailings storage, waste rock storage and low grade ore stockpile. Budgetary quotations make up 14% of the capital costs. The remaining costs were determined by Ausenco's recent database, adjusted to the Project's base date where applicable.

The piles costs are presented in Table 18-12.

Table 18-12: Stockpiles Capital Costs

WBS	WBS Description	Initial Capital Cost (MGTQ)	Initial Capital Cost (MUSD)
5006	Low-Grade Stockpile	4.49	0.59
5010	Waste Dump	21.10	2.78
6020	Tailings Storage and Fines Dike	8.55	1.12
Total Stockpiles Infrastructure		**34.13**	**4.49**

Note: Values may not sum correctly due to rounding.

18.3.7 Indirect Capital Costs

The indirect costs include project-related indirect costs, owner's costs, and provision costs, as presented in the items below. Indirect costs are summarized in Table 18-13.

Table 18-13: Indirect Costs Estimate

Description	Initial Capital Cost (MGTQ)	Initial Capital Cost (MUSD)
Owner Costs	265.44	34.93
Contingency	181.28	23.85
EPCM (Engineering, Procurement, and Construction Management)	145.95	19.20
Spare Parts for First year Operation	21.54	2.83
In-land National Freight	12.78	1.68
Insurance	9.03	1.19
First Fill	8.51	1.12
Vendor Assembly Supervision	6.43	0.85
Vendor Commissioning and Start-up	6.43	0.85
Total	**657.39**	**86.50**

Note: Values may not sum correctly due to rounding.

18.3.7.1 Owner Capital Costs

Owner's costs were provided by Aura. These costs include labor, vehicles, fuel, indirect field constructions (including generators for electricity), furniture for permanent buildings, environmental management and community engagement, pre-operational labor, administration and human resources, IT and legal.

18.3.7.2 Contingency Capital Costs

Contingency was estimated as 13.3% (P70), based on a risk analysis of the project, utilizing a Monte Carlo Simulation. The index was applied to the overall sum of direct and indirect costs.

18.3.7.3 EPCM Capital Costs

Estimate inputted by aura. Contemplates Engineering, Procurement, Construction Management, as well as Commissioning costs.

18.3.7.4 Spare Parts for First Year Operation

These costs include spare parts for the first year of operation for all mechanical and electrical equipment, and are based on costs provided by the suppliers with their budgetary quotations. For cases in which the supplier did not provide a quote, it was considered an index-based application of 4% over the equipment's cost.

18.3.7.5 In-land National Freight Costs

These costs include in-land freight inside Guatemala for equipment and materials, considering the cost provided by the suppliers on budgetary quotations. For cases in which the supplier did not provide a quote, an index-based application was considered.

18.3.7.6 Insurance Costs

Insurance costs were provided by Aura, and are comprised of property insurance, civil liability insurance, directors & officers (D&O) insurance and species/precious metals insurance.

18.3.7.7 First Fill Costs

The costs for first fills include chemicals, fuels, lubricants, and consumables needed to establish the inventory levels necessary to begin operations.

For this item, a 1% rate was applied to the overall sum of direct costs.

18.3.7.8 Vendor Assembly Supervision Capital Costs

Vendor representatives are required on-site during construction to verify that major equipment installation complies with vendor requirements. Their presence will also be required during pre-commissioning.

For this item, a 2% rate was applied to the overall cost of mechanical and electrical equipment supply.

18.3.7.9 Vendor Commissioning and Start-up Costs

Costs for vendor supervision during pre-commissioning and commissioning supervision were estimated at 2% of the electromechanical supply costs.

18.3.8 Sustaining Capital

Plant sustaining costs consider the purchase and assembly of the main electrical substation, power transmission line 69 kV, Asunción Mita's electrical connection substation, underground mine water treatment and decontamination station, as well as tailings and waste rock piles and owner costs.

The project's main electrical substation, power transmission line and connection substation will be implemented as Sustaining Capital in Year 3 (2030). From Years -2 to 3 (2026 to 2030), the site will be powered by generators, and from Years 4 to 16 (2031 to 2043) the site will be powered by the local energy company.

The second underground mine water treatment and decontamination station will be implemented in 2028 to absorb the growing demand for mine water pumping.

Indirect costs were estimated at 0.5% of the Plant and Piles sustaining costs.

The contingency rate of 20% has been applied to the sum of Plant and Indirect sustaining costs.

The total sustaining cost is US$43.70 million, distributed as shown in Table 18-14.

Table 18-14: Sustaining Capital Costs

Description	Year -1 (MUSD)	Year 1 (MUSD)	Year 2 (MUSD)	Year 3 (MUSD)	Year 4 (MUSD)	Year 5 (MUSD)	Year 6 (MUSD)	Year 7 (MUSD)	Year 8 (MUSD)
Plant Costs	**-**	**9.45**	**-**	**4.91**	**-**	**-**	**-**	**-**	**-**
Main Substation	-	-	-	4.91	-	-	-	-	-
Underground Mine Water Treatment and Decontamination Station	-	9.45	-	-	-	-	-	-	-
Off-Site Infrastructure Costs	**-**	**-**	**-**	**3.30**	**-**	**-**	**-**	**-**	**-**
Power Transmission Line 69kv	-	-	-	2.0	-	-	-	-	-
Asunción Mita's Electrical Connection Substation	-	-	-	1.30	-	-	-	-	-
Mine Costs	**6.07**	**53.71**	**27.12**	**26.89**	**18.37**	**8.34**	**8.95**	**10.98**	**8.51**
Dewatering Pumps	-	7.69	-	5.45	-	-	-	-	-
Mine Development	6.07	46.02	27.12	21.44	18.37	8.34	8.95	10.98	8.51
Piles Costs	**-**	**1.95**	**9.05**	**-**	**-**	**3.66**	**-**	**-**	**3.66**

Description	Year -1 (MUSD)	Year 1 (MUSD)	Year 2 (MUSD)	Year 3 (MUSD)	Year 4 (MUSD)	Year 5 (MUSD)	Year 6 (MUSD)	Year 7 (MUSD)	Year 8 (MUSD)
Tailings Waste Piles	-	1.95	6.05	-	-	3.66	-	-	3.66
Waste Rock Piles	-	-	3.0	-	-	-	-	-	-
Indirect Costs	**0.27**	**0.06**	**0.05**	**0.02**	**-**	**0.02**	**-**	**-**	**0.02**
Owner Costs	0.27	-	-	-	-	-	-	-	-
Other indirects	-	0.06	0.05	0.02	-	0.02	-	-	0.02
Contingency	**1.27**	**13.03**	**7.24**	**7.02**	**3.67**	**2.4**	**1.79**	**2.2**	**2.44**
Plant Contingency	0.05	1.9	0.01	0.99	-	0.	-	-	0.
Off-Site Infrastructure Contingency	-	-	-	0.66	-	-	-	-	-
Mine Contingency	1.21	10.74	5.42	5.38	3.67	1.67	1.79	2.2	1.7
Piles Contingency	-	0.39	1.81	-	-	0.73	-	-	0.73
Project Total	**7.62**	**78.2**	**43.46**	**42.15**	**22.04**	**14.42**	**10.74**	**13.18**	**14.62**

Description	Year 9 (MUSD)	Year 10 (MUSD)	Year 11 (MUSD)	Year 12 (MUSD)	Year 13 (MUSD)	Year 14 (MUSD)	Year 15 (MUSD)	Year 16 (MUSD)	TOTAL (MUSD)
Plant Costs	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Main Substation	-	-	-	-	-	-	-	-	-
Underground Mine Water Treatment and Decontamination Station	-	-	-	-	-	-	-	-	-
Off-Site Infrastructure Costs	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Power Transmission Line 69kv	-	-	-	-	-	-	-	-	-
Asunción Mita's Electrical Connection Substation	-	-	-	-	-	-	-	-	-
Mine Costs	**3.13**	**6.22**	**3.34**	**6.82**	**11.52**	**5.4**	**2.17**	**0.49**	**208.03**
Dewatering Pumps	-	-	-	-	-	-	-	-	13.15
Mine Development	3.13	6.22	3.34	6.82	11.52	5.4	2.17	0.49	-
Piles Costs	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**3.66**
Tailings Waste Piles	-	-	-	-	-	-	-	-	3.66
Waste Rock Piles	-	-	-	-	-	-	-	-	-
Indirect Costs	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**0.02**
Owner Costs	-	-	-	-	-	-	-	-	-
Other indirects	-	-	-	-	-	-	-	-	0.02
Contingency	**0.63**	**1.24**	**0.67**	**1.36**	**2.3**	**1.08**	**0.43**	**0.1**	**48.89**

Description	Year 9 (MUSD)	Year 10 (MUSD)	Year 11 (MUSD)	Year 12 (MUSD)	Year 13 (MUSD)	Year 14 (MUSD)	Year 15 (MUSD)	Year 16 (MUSD)	TOTAL (MUSD)
Plant Contingency	-	-	-	-	-	-	-	-	2.96
Off-Site Infrastructure Contingency	-	-	-	-	-	-	-	-	0.66
Mine Contingency	0.63	1.24	0.67	1.36	2.3	1.08	0.43	0.1	41.61
Piles Contingency	-	-	-	-	-	-	-	-	3.66
Project Total	**3.75**	**7.46**	**4.01**	**8.19**	**13.82**	**6.48**	**2.6**	**0.58**	**293.34**

Note: Values may not sum correctly due to rounding.

18.4 Operating Costs

18.4.1 Overview

Operating costs include the ongoing costs of operations related to processing, tailings and waste rock disposal, water treatment stations, as well as general and administrative activities for the operation of the enterprise, scheduled for 16 years.

Costs are expressed in United States dollars (USD), using the exchange rate of GTQ7.60 = USD1.00. The estimate covers the following items:

- Labor
- General and Administrative (G&A)
- Laboratory
- Access Maintenance
- Mobile Equipment Fleet
- Reagents
- Consumables
- Maintenance, Fuel and Lubricants
- Power
- Water Treatment
- Tailings, and Rock Waste Piles
- Mine Costs

A summary of the operating costs is presented in Table 18-15 and Table 18-16.

Table 18-15: Operating Cost Summary (USD/t ROM basis)

Item	Year -1 (Ramp-up)	Year 1 (Ramp-up)	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t
Processing	371.2	245.7	254.6	307.9	219.4	179.3	180.4	164.0	166.4
Infrastructure	46.5	19.5	10.2	19.1	11.9	11.9	11.9	11.9	11.9
Mining	122.5	71.6	80.4	108.8	86.8	86.7	87.7	71.3	73.7
Total Operating Costs	**540.2**	**336.8**	**345.2**	**435.8**	**318.1**	**277.9**	**280.1**	**247.3**	**252.1**

Item	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t
Processing	155.5	154.6	154.5	155.6	156.2	155.7	156.7	154.7	220.8
Infrastructure	12.0	11.9	11.9	11.9	11.9	11.9	12.7	12.7	22.9
Mining	62.1	61.3	61.3	62.3	62.8	62.9	58.1	56.2	74.1
Total Operating Costs	**229.6**	**227.8**	**227.7**	**229.9**	**230.9**	**230.5**	**227.5**	**223.6**	**317.8**

Note: Values may not sum correctly due to rounding.

Table 18-16: Operating Cost Summary

Item	Year -1 (Ramp-up)	Year 1 (Ramp-up)	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD
Processing	13.5	14.7	77.8	112.5	128.1	104.7	105.4	95.9	97.3
Infrastructure	1.7	1.2	3.1	7.0	7.0	7.0	7.0	7.0	7.0
Mining	4.5	4.3	24.6	39.8	50.7	50.7	51.3	41.7	43.1
Total Operating Costs	**19.6**	**20.1**	**105.5**	**159.3**	**185.8**	**162.4**	**163.7**	**144.5**	**147.3**

Item	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	TOTAL
	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD
Processing	90.6	90.3	90.3	90.9	91.2	91.0	86.1	85.0	77.3	90.6
Infrastructure	7.0	7.0	7.0	7.0	7.0	7.0	7.0	7.0	8.0	7.0
Mining	36.2	35.8	35.8	36.4	36.7	36.8	31.9	30.9	25.9	36.2
Total Operating Costs	**133.8**	**133.2**	**133.1**	**134.3**	**134.9**	**134.7**	**125.0**	**122.8**	**111.2**	**133.8**

Note: Values may not sum correctly due to rounding.

18.4.2 Basis of Estimate

The following assumptions are common to all operating cost estimates:

- Operation estimated to start with a four-month ramp-up period.
- Cost estimates are based on Q4 2025 pricing, without allowances for inflation.
- Costs are expressed in USD, using the exchange rate of GTQ7.60 = USD1.00.
- Equipment and materials will be purchased as new.
- Reagent consumption rates were determined by metallurgical test results.

18.4.3 Mine Operating Costs

Mine operating costs per ton processed, including total costs and costs by individual mine facilities, are presented in Table 18-17.

Table 18-17: Total Mine Operating Costs (Including Mine Infrastructure)

Description	Unit	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17
Mine total	US$/t processed	105.1	79.0	108.8	86.8	86.7	87.7	71.3	73.7	62.1	61.3	61.3	62.3	62.8	62.9	54.7	52.9	68.3
Mine	US$/t processed	93.4	70.4	98.5	80.9	80.5	80.9	64.8	67.4	54.9	54.9	54.7	55.2	56.2	56.5	47.4	46.4	58.1
Paste fill	US$/t processed	1.0	0.7	0.7	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.7
Well - Stantec	US$/t processed	0.3	0.7	2.4	0.9	1.3	1.9	1.6	1.4	2.3	1.5	1.7	2.1	1.7	1.5	2.4	1.5	2.6
Refrigeration plant - BBE	US$/t processed	10.4	7.2	7.2	4.5	4.5	4.5	4.5	4.5	4.5	4.5	4.5	4.5	4.5	4.5	4.5	4.5	6.9

18.4.4 Process Operating Costs

18.4.4.1 Labor, General and Administrative Operating Costs

Staffing estimates were provided by Aura. The labour costs include the requirements for plant operations such as management, maintenance, site services, assay laboratory and operations.

The average operational labour force distributed over the LoM of the project is shown in Table 18-18.

Table 18-18: Operational Labor Roster

Item	Average Number of Employees
Plant and Maintenance	117
Health, Safety, Environment, and Communities	16
Administrative	47
Total Labour Force	**180**

General and administrative (G&A) costs are expenses not directly related to production and exclude mining, processing, external refining, and transportation costs.

The Project's G&A operational costs were allocated to the following cost centers: general expenses, health, safety and environment, human resources, administration, and outsourced services, based on the projected number of personnel, infrastructure, and services required during the operational period. G&A costs were provided by Aura. A detailed breakdown of all G&A costs is presented in Table 18-19 and Table 18-20.

Table 18-19: G&A Detailed Costs

Item	Year -1 (Ramp-up)	Year 1 (Ramp-up)	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD
General expenses	0.20	0.20	1.02	1.63	1.63	1.63	1.63	1.63	1.63
Health, Safety and Environment	0.16	0.16	0.81	1.30	1.30	1.30	1.30	1.30	1.30
Human Resources	0.20	0.20	0.99	1.58	1.58	1.58	1.58	1.58	1.58
Administration	0.28	0.28	1.39	2.22	2.22	2.22	2.22	2.22	2.22
Outsourced services	0.26	0.26	1.28	2.05	2.05	2.05	2.05	2.05	2.05
Other	0.02	0.02	0.09	0.15	0.15	0.15	0.15	0.15	0.15
Total Operating Costs	**1.12**	**1.12**	**5.58**	**8.93**	**8.93**	**8.93**	**8.93**	**8.93**	**8.93**

Item	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	TOTAL
	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD
General expenses	1.63	1.63	1.63	1.63	1.63	1.63	1.63	1.63	1.63	**25.88**
Health, Safety and Environment	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30	**20.69**
Human Resources	1.58	1.58	1.58	1.58	1.58	1.58	1.58	1.58	1.58	**25.08**
Administration	2.22	2.22	2.22	2.22	2.22	1.88	2.22	2.22	1.62	**34.26**
Outsourced services	2.05	2.05	2.05	2.05	2.05	2.05	2.05	2.05	2.05	**32.58**
Other	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	**2.35**
Total Operating Costs	**8.93**	**8.93**	**8.93**	**8.93**	**8.93**	**8.59**	**8.93**	**8.93**	**8.34**	**140.85**

Note: Values may not sum correctly due to rounding.

Table 18-20: G&A Detailed Costs (USD/t ROM Basis)

Item	Year -1 (Ramp-up)	Year 1 (Ramp-up)	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t
General expenses	5.60	3.41	3.33	4.46	2.79	2.79	2.79	2.79	2.79
Health, Safety and Environment	4.48	2.73	2.66	3.57	2.23	2.23	2.23	2.23	2.23
Human Resources	5.43	3.31	3.23	4.32	2.70	2.70	2.70	2.70	2.70
Administration	7.62	4.64	4.53	6.07	3.80	3.79	3.79	3.79	3.79
Contracted Services	7.05	4.30	4.20	5.62	3.51	3.51	3.51	3.51	3.51
Other	0.51	0.31	0.30	0.41	0.25	0.25	0.25	0.25	0.25
Total Operating Costs	**30.70**	**18.70**	**18.26**	**24.44**	**15.29**	**15.29**	**15.28**	**15.29**	**15.28**

Item	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t
General expenses	2.80	2.79	2.79	2.79	2.79	2.79	2.97	2.97	4.66
Health, Safety and Environment	2.24	2.23	2.23	2.23	2.23	2.23	2.37	2.37	3.72
Human Resources	2.71	2.70	2.70	2.70	2.70	2.70	2.88	2.88	4.51
Administration	3.80	3.79	3.79	3.79	3.79	3.21	4.04	4.03	4.64
Contracted Services	3.52	3.51	3.51	3.51	3.51	3.51	3.74	3.74	5.87
Other	0.25	0.25	0.25	0.25	0.25	0.25	0.27	0.27	0.42
Total Operating Costs	**15.32**	**15.28**	**15.29**	**15.28**	**15.29**	**14.70**	**16.26**	**16.25**	**23.83**

Note: Values may not sum correctly due to rounding.

18.4.4.2 Laboratory Operating Costs

The operating cost estimate for laboratory activity was informed by Aura, and comprises mobilization and demobilization, as well as annual fixed and variable costs for operating.

18.4.4.3 Access Maintenance – site roads

Access maintenance for site roads costs comprises the leasing of mobile equipment, fuel and operators.

The fleet sized for this activity is shown in Table 18-21.

Table 18-21: Access Maintenance Fleet

Item	Quantity
(Bulldozer	1
Motor Grader	1
4x4 Backhoe Loader	1
Fuel and Lubrication Service Truck	1
Excavator	1
Water Truck	1
Compactor Roller	1
Dump Trucks	3

18.4.4.4 Mobile Equipment Fleet Operating Costs

Mobile equipment costs comprise the leasing of mobile equipment, fuel and operators for the Filtration and Crushing areas.

The fleet sized for this activity is shown in Table 18-22.

Table 18-22: Mobile Equipment Fleet

Item	Quantity
Crushing Area	
Wheel Loader	1
Skid Steer Loader	1
Dump Trucks	1
Filtration Area	
Wheel Loader	1
Dump Trucks	3

18.4.4.5 Reagents Costs

This item includes the costs of reagents for the plant. Quantities were defined by Ausenco, based on metallurgical test results, flowsheets and mass balance calculations. The prices for reagents were based on budgetary quotations, information provided by Aura and Ausenco's recent database.

18.4.4.6 Consumables Costs

This item includes the costs of consumables for the plant. Quantities were defined by Ausenco, based on metallurgical test results, flowsheets and mass balance calculations. Quantities for SAG mill balls and Ball mill balls were inputted by Aura.

The prices for consumables were based on budgetary quotations, information provided by Aura and Ausenco's recent database.

18.4.4.7 Maintenance, Fuel and Lubricants Costs

Equipment maintenance costs include annual expenses for spare parts and maintenance of process and handling equipment, as well as third-party services.

Consumable materials and Fuel and Lubricants include all expenses for materials, as well as the consumption of diesel, gasoline, and lubricants for processing plant equipment and industrial support vehicles.

The indexes used to calculate the costs for these items were derived from similar projects in Ausenco's recent database and are shown in Table 18-23.

Table 18-23: Maintenance, Fuel and Lubricants Indexes

Description	%	Source
Maintenance Parts and Materials	3.5%	Index applied on the total costs of process, mechanical and electrical equipment (CapEx and Sustaining).
Consumable Materials	1.2%	
Fuel and Lubricants	1.0%	

There is an increase in cost as of 2031 related to the Main substation, Asunción Mita's substation and Power transmission line 69 kV to be implemented in 2030. The detailed costs are shown in Table 18-24.

Table 18-24: Detailed Maintenance, Fuel and Lubricants Operating Costs

Item	Year -1 (Ramp-up)	Year 1 (Ramp-up)	Year 1	Year 2 - 3	Year 4 - 16
	MUSD	MUSD	MUSD	MUSD/a	MUSD/a
Maintenance Parts and Materials	0.19	0.19	0.94	1.12	1.41
Consumable Materials	0.06	0.06	0.32	0.39	0.48
Fuel and Lubricants	0.05	0.05	0.27	0.32	0.32
Total Operating Costs	**0.30**	**0.30**	**1.52**	**1.83**	**2.22**

Note: Values may not sum correctly due to rounding.

18.4.4.8 Power Costs

The power costs of the process plant were estimated from the installed power in the mechanical equipment list with factors applied for availability and utilization (power study).

The project's Main electrical substation, Power transmission line and Connection substation will be implemented as Sustaining Capital in Year 3 (2030). From Years -2 to 3 (2026 to 2030) the site will be powered by generators, and from Years 4 to 16 (2031 to 2043) the site will be powered by the local energy company.

18.4.4.8.1 Generator Power Costs

The Power costs for Years 2026 to 2030 (considering as OpEx cost from November/2027), is comprised of the generator's lease, demand and fuel costs, as shown in Table 18-25 and Table 18-26.

Aura informed the unit cost for diesel as US$0.93 per liter.

Table 18-25: Power Operating Costs – Generators (MUSD)

Item	Year -1 (Ramp-up)	Year 1 (Ramp-up)	Year 1	Year 2	Year 3
	MUSD	MUSD	MUSD	MUSD	MUSD
Leasing costs	0.93	0.96	4.79	5.88	5.88
Demand costs	0.10	0.17	0.97	1.24	1.33
Fuel and Lubricants	2.22	3.73	21.75	27.86	29.74
Total Operating Costs	**3.25**	**4.86**	**27.52**	**34.98**	**36.94**

Note: Values may not sum correctly due to rounding.

Table 18-26: Power Operating Costs – Generators (USD/t ROM basis)

Item	Year -1 (Ramp-up)	Year 1 (Ramp-up)	Year 1	Year 2	Year 3
	USD/t	USD/t	USD/t	USD/t	USD/t
Leasing costs	25.59	16.05	15.67	16.08	10.06
Demand costs	2.72	2.79	3.17	3.40	2.27
Fuel and Lubricants	61.04	62.54	71.16	76.24	50.92
Total Operating Costs	**89.36**	**81.38**	**90.01**	**95.72**	**63.25**

Note: Values may not sum correctly due to rounding.

18.4.4.8.2 Power Operating Costs – Local energy company

The power cost is divided into two parts, one for consumption and one for demand. Aura informed the unit power cost for consumption and demand, as presented in Table 18-27.

Table 18-27: Power Operating Costs - Local Energy Company - Unit Costs

Description	Power Cost
Demand	USD 19.52/kW
Consumption	USD 0.095/kWh

18.4.4.9 Water Treatment Operating Costs

The water treatment operating costs consists of the operation of the following structures:

- Potable Water Treatment Plant (PWTP)
- Sewage Treatment Plant (STP)
- Groundwater Treatment and Decontamination Plant for Mine (GWTP)
- Chemical Effluent Treatment Plants (CETP)
- Water and Oil Separator.

Water treatment are a significant part of the site operating costs. GWTP's are critical for the project as all the underground water pumped from the mine (average 5,206,888 m^3/a) must be treated to remove the heavy metals before returning to the river or to be retreated by the PWTP to supply the plant and the surrounding communities with potable water.

The quantities of water to be treated were determined by Ausenco and Aura, and the unit prices for treatment were based on information from the treatment plant suppliers.

The detailed costs for the water treatment structures are shown in Table 18-28 and Table 18-29.

Table 18-28: Water Treatment Operating Costs (MUSD)

Item	Year -1 (Ramp-up)	Year 1 (Ramp-up)	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD
Potable Water Treatment Plant (PWTP)	0.0	0.0	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Sewage Treatment Plant (STP)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Groundwater Treatment and Decontamination Plant for Mine (GWTP) – Operational cost	0.34	0.43	2.16	4.14	4.25	5.25	5.33	5.36	5.36
Groundwater Treatment and Decontamination Plant for Mine (GWTP) – Fixed cost	0.01	0.01	0.04	0.05	0.05	0.05	0.05	0.05	0.05
Chemical Effluent Treatment Plants (CETP)	0.08	0.08	0.38	0.46	0.46	0.46	0.46	0.46	0.46
Water and Oil Separator	0.12	0.12	0.62	0.74	0.74	0.74	0.74	0.74	0.74
Potable Water Treatment Plant (PWTP)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total Operating Costs	**0.55**	**0.64**	**3.21**	**5.41**	**5.51**	**6.51**	**6.59**	**6.62**	**6.62**

Item	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	TOTAL
	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD
Potable Water Treatment Plant (PWTP)	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	**0.16**
Sewage Treatment Plant (STP)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**0.02**
Groundwater Treatment and Decontamination Plant for Mine (GWTP) – Operational cost	5.39	5.45	5.44	5.45	5.50	5.51	5.50	5.50	5.49	**81.83**
Groundwater Treatment and Decontamination Plant for Mine (GWTP) – Fixed cost	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	**0.78**
Chemical Effluent Treatment Plants (CETP)	0.46	0.46	0.46	0.46	0.46	0.46	0.46	0.46	0.46	**7.42**
Water and Oil Separator	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	**11.99**
Potable Water Treatment Plant (PWTP)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**0.05**
Total Operating Costs	**6.65**	**6.71**	**6.70**	**6.71**	**6.76**	**6.78**	**6.76**	**6.76**	**6.75**	**102.25**

Note: Values may not sum correctly due to rounding.

Table 18-29: Water Treatment Operating Costs (USD/t ROM basis)

Item	Year -1 (Ramp-up)	Year 1 (Ramp-up)	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t
Potable Water Treatment Plant (PWTP)	0.04	0.03	0.03	0.03	0.02	0.02	0.02	0.02	0.02
Sewage Treatment Plant (STP)	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Groundwater Treatment and Decontamination Plant for Mine (GWTP) – Operational cost	9.33	7.23	7.06	11.33	7.27	8.98	9.11	9.17	9.17
Groundwater Treatment and Decontamination Plant for Mine (GWTP) – Fixed cost	0.22	0.13	0.13	0.13	0.08	0.08	0.08	0.08	0.08
Chemical Effluent Treatment Plants (CETP)	2.10	1.28	1.25	1.26	0.79	0.79	0.79	0.79	0.79
Water and Oil Separator	3.40	2.07	2.02	2.03	1.27	1.27	1.27	1.27	1.27
Potable Water Treatment Plant (PWTP)	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Total Operating Costs	**15.12**	**10.75**	**10.50**	**14.79**	**9.43**	**11.14**	**11.28**	**11.33**	**11.33**

Item	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t
Potable Water Treatment Plant (PWTP)	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.03
Sewage Treatment Plant (STP)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Groundwater Treatment and Decontamination Plant for Mine (GWTP) – Operational cost	9.24	9.32	9.31	9.33	9.41	9.44	10.01	10.01	15.68
Groundwater Treatment and Decontamination Plant for Mine (GWTP) – Fixed cost	0.08	0.08	0.08	0.08	0.08	0.08	0.09	0.09	0.14
Chemical Effluent Treatment Plants (CETP)	0.79	0.79	0.79	0.79	0.79	0.79	0.84	0.84	1.31
Water and Oil Separator	1.27	1.27	1.27	1.27	1.27	1.27	1.35	1.35	2.12
Potable Water Treatment Plant (PWTP)	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Total Operating Costs	**11.41**	**11.48**	**11.47**	**11.49**	**11.57**	**11.60**	**12.31**	**12.31**	**19.29**

Note: Values may not sum correctly due to rounding.

18.4.4.10 Tailings and Rock Waste Piles Operating Costs

Tailings and Rock Waste Piles costs comprise the leasing of mobile equipment, fuel and operators for material placement and compaction in the tailings, waste and stockpiles.

Table 18-30: Tailings and Rock Waste Piles Operating Costs

Item	Quantity
Crushing Area	
Wheel Loader	1
Skid Steer Loader	1
Dump Trucks	1
Filtration Area	
Wheel Loader	1
Dump Trucks	3

Detailed costs are shown in Table 18-31 and Table 18-32.

Table 18-31: Tailings and Rock Waste Piles Detailed Costs

Item	Year -1 (Ramp-up)	Year 1 (Ramp-up)	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD
Tailings waste piles	0.45	0.38	0.38	2.27	2.27	2.27	2.27	2.27	2.27
Waste rock piles	0.35	0.30	0.30	1.80	1.80	1.80	1.80	1.80	1.80
Stockpiles	0.35	-	-	-	-	-	-	-	-
Total Operating Costs	**1.16**	**0.68**	**0.68**	**4.06**	**4.06**	**4.06**	**4.06**	**4.06**	**4.06**

Tailings Waste Piles Waste Rock Piles	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	TOTAL
	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD	MUSD
Stockpiles	2.27	2.27	2.27	2.27	2.27	2.27	2.27	2.27	2.83	**35.77**
Tailings waste piles	1.80	1.80	1.80	1.80	1.80	1.80	1.80	1.80	2.21	**28.34**
Waste rock piles	-	-	-	-	-	-	-	-	-	**0.35**
Total Operating Costs	**4.06**	**4.06**	**4.06**	**4.06**	**4.06**	**4.06**	**4.06**	**4.06**	**5.04**	**64.46**

Note: Values may not sum correctly due to rounding.

Table 18-32: Tailings and Rock Waste Piles Detailed Costs (USD/t ROM basis)

Item	Year -1 (Ramp-up)	Year 1 (Ramp-up)	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t
Tailings waste piles	12.48	6.33	1.24	6.20	3.88	3.88	3.88	3.88	3.88
Waste rock piles	9.73	5.02	0.98	4.92	3.08	3.08	3.08	3.08	3.08
Stockpiles	9.73	-	-	-	-	-	-	-	-
Total Operating Costs	**31.95**	**11.34**	**2.22**	**11.12**	**6.96**	**6.96**	**6.95**	**6.96**	**6.95**

Tailings Waste Piles Waste Rock Piles	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t
Stockpiles	3.89	3.88	3.88	3.88	3.88	3.88	4.13	4.12	8.09
Tailings waste piles	3.09	3.08	3.08	3.08	3.08	3.08	3.27	3.27	6.31
Waste rock piles	-	-	-	-	-	-	-	-	-
Total Operating Costs	**6.97**	**6.95**	**6.96**	**6.96**	**6.96**	**6.96**	**7.40**	**7.40**	**14.41**

Note: Values may not sum correctly due to rounding.

19 ECONOMIC ANALYSIS

19.1 Forward-Looking Information Cautionary Statements

The results of the economic analyses discussed in this section represent forward-looking information as the results depend on inputs that are subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. Forward-looking information includes:

- Mineral Resource estimates
- The proposed mine production plan
- Smelting and Refining Terms
- Assumed gold and silver prices and exchange rates
- Capital Costs
- Projected mining and process recovery rates
- Sustaining costs and proposed operating costs
- Assumptions as to closure costs
- Assumptions as to environmental, permitting, and social risks
- Leasing
- Royalties
- Taxes
- Working Capital
- Closure Costs
- Salvage Value
- Inflation (assumed to be 0% as estimates are long term).

Additional risks related to forward-looking information include:

- Changes to costs of production from what is assumed
- Unrecognized environmental risks
- Unanticipated reclamation expenses

- Unexpected variations in the quantity of mineralized material, grade, or recovery rates
- Geotechnical or hydrogeological considerations being different during mining from what was assumed
- Failure of mining methods to operate as anticipated
- Failure of plant, equipment or processes to operate as anticipated
- Changes to assumptions as to the availability of electrical power and the power rates used in the operating cost estimates and financial analysis
- Ability to maintain the social license to operate
- Accidents, labour disputes, and other mining industry related risks
- Changes to interest rates
- Changes to tax rates.

Calendar years used in financial analysis are provided for conceptual purposes only. Permits still must be obtained in support of operations and approval for development must be granted by the Aura's Board.

19.2 Methodologies Used

An economic engineering model was developed to estimate annual pre-tax and post-tax cash flows and sensitivities of the Project based on a 5 % discount rate. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the post-tax results are only approximations.

To reflect the time value of money, annual net cash flow (NCF) projections are discounted back to the beginning of the project execution period. The discounted present values of the cash flows are then summed to obtain the net present value (NPV) of the project.

Capital and operating cost estimates are presented in Section 18 of this Report and are expressed in Q4 2025 USD. The economic analysis was run on a constant dollar basis with no inflation.

Where key elements used in the economic analysis are discussed elsewhere in the report, the reader is referred to those sections rather than repeating information. Specifically, reference is made to Section 11 (Mineral Resources), Section 12 (Mineral Reserves), Section 13 (mining methods), Section 10.4 (metallurgical Recoveries), Section 16 (Marketing Studies) and Section 18 (capital and operating costs).

19.3 Financial Model Parameters

19.3.1 Revenue

Mine revenue is derived from the sale of doré bars into the international precious metals market. Refining is assumed to be conducted on a market basis, with no binding refining contracts in place at the time of this study. Table 19-1 presents the NSR parameters used to estimate revenue in the economic analysis.

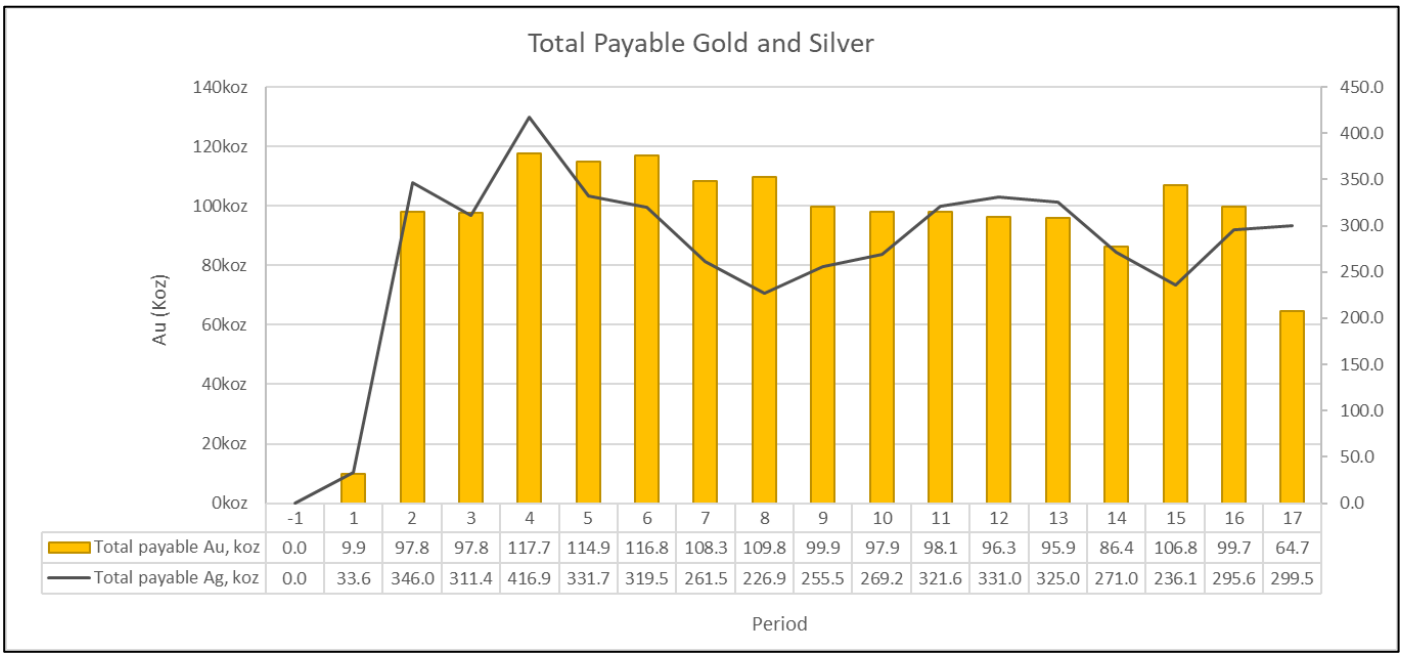
Table 19-1: NSR Parameters

Parameter	Unit	Value
Gold (Au) Recovery	%	96.0
Silver (Ag) Recovery	%	85.0
Gold (Au) Payable	%	99.90
Silver (Ag) Payable	%	99.50
Gold (Au) Refining Charge	US$/payable oz	0.55
Silver (Ag) Refining Charge	US$/payable oz	0.50

Source: Ausenco, 2025.

Figure 19-1 shows the payable quantities of gold (Au) and silver (Ag) over the life of mine (LOM). A total of 1,618.7 koz of gold and 4,852 koz of silver are projected to be sold over the LOM. Gold accounts for approximately 97% of gross project revenue, with silver contributing the remaining 3%.

Figure 19-1: LOM Payable Gold and Silver



Source: Ausenco, 2025.

19.3.2 Gold and Silver Pricing

Gold and silver prices were based on market prices obtained from CIBC Global Mining Group, Analyst Consensus Commodity Price Forecast.

The forecasts used are intended to represent expected prices for gold and silver throughout the life of the Project, considering the Consensus scenario (Dic. 2025). See Table 19-2 below.

Table 19-2: Gold and Silver Pricing

Precious Metals	2025	2026	2027	2028	LT
Gold (US$/oz)	$3,368	$3,930	$3,827	$3,689	$3,140
Silver (US$/oz)	$37.5	$45.2	$42.8	$40.0	$36.9

19.3.3 Working Capital

A high-level estimation of working capital has been incorporated into the cash flow based on accounts receivable (30 days of revenue), inventories (30 days of operating costs), and accounts payable (60 days of operating costs). Net working capital was calculated as the sum of receivables and inventories minus payables.

19.3.4 Closure Costs

Closure costs are assumed to be incurred at the end of the LOM. The closure cost estimate was provided by Aura Gold and is estimated at US$ 17.2 million.

The closure cost estimate includes activities related to site safety, underground mine closure, infrastructure decommissioning, process plant and water treatment facilities, piping, ponds and tanks, power distribution systems, administrative and ancillary buildings, dry stack tailings facility (DSTF), waste rock dumps, wells, and post-closure monitoring.

A summary of the closure cost components is presented in Table 19-3, which outlines the estimated costs by major facility and activity.

Table 19-3: Closure Costs

Closure Costs	USDM
Safety	0.45
Underground mine	1
Infrastructure	0.35
Process Plant	5.74
Water Treatment Plant	1.1
Piping, ponds and tanks	1.98
Switchyard and Power Distribution	0.4
Administration Office and Ancillary Buildings	0.25
Drystack Tailings Facility (DSTF)	2.2
Waste Rock Dumps	0.43
Wells	1.2
Monitoring	2.1
Total	**17.2**

19.3.5 Taxes

The Project has been evaluated on a post-tax basis to provide a more indicative, although still approximate, assessment of the potential Project economics. A tax model was prepared by Ausenco based on information provided by Aura Gold and a report prepared by EY for Aura, which includes information related to the tax regime applicable in Guatemala. Current tax pools were used in the analysis.

The tax model incorporates the following assumptions:

- Income tax is calculated as the lesser of 25% on net taxable income or 7% on gross income.
- Value-added tax (VAT) has been modelled.
- Withholding taxes are assumed to be 0% on royalties and interest.
- Depreciation has been modelled in accordance with Guatemalan legislation, applying different depreciation methods depending on the nature of the capital assets, including straight-line depreciation and depreciation based on production.
- Total taxes for the Project amount to US$315 million.

1.2.2 Royalties

The Era Dorada Project is subject to two contractual royalties and a statutory royalty. In addition, a voluntary royalty is paid by the Project. All applicable royalties have been incorporated into the economic analysis and cash flow model.

The royalty terms applied in the economic evaluation are summarized in Table 19-4. Total royalties payable over the LOM are estimated at US$369.5 million.

Table 19-4: Royalties Included in Economic Analysis

Parameter	Unit	Value
Contractual Royalty		
Newmont Royalty	% NSR	1.00
Goldcorp Royalty	% NSR	1.05
Statutory Royalty		
Guatemalan Mining Royalty	% Revenue	1.00
Guatemalan Voluntary Royalty	% Revenue	4.00
Municipality	*% Revenue*	*1.5*
Central Government	*% Revenue*	*1.5*
MARN	*% Revenue*	*0.1*
MEM	*% Revenue*	*0.1*
Project financed directly by the company	*% Revenue*	*0.8*

19.4 Economic Analysis

The economic analysis was performed using a 5% discount rate. Cash flows were discounted to the beginning of construction, assuming that the execution decision and major financing occurred at that time. The start of the operation begins 1.8 years (21 months) thereafter.

On a pre-tax basis, the present net value discounted at 5% (NPV 5%) is US$1,535.2 million, with an internal rate of return (IRR) of 38.5%, and a payback period of 2.7 years. On a post-tax basis, the NPV 5% is US$1,344.5 million, the IRR is 35.6%, and the payback period is 2.8 years.

A summary of the project economics is included in Table 19-5 and illustrated in Figure 19-2.

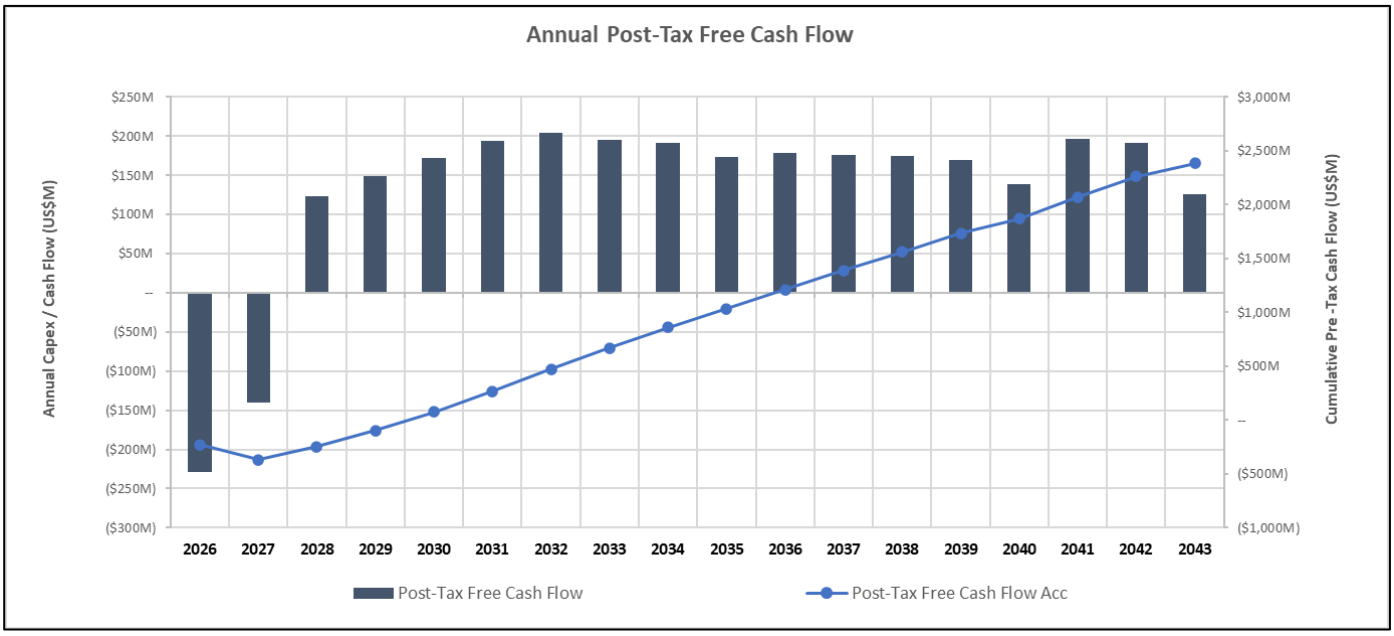
Table 19-5: Economic Analysis Summary

General	Unit	LOM Total Value/Average
Gold realized Price	US$/oz	$3,177
Silver realized Price	US$/oz	$37.2
Mine Life	years	16.8
Production – LOM		
Ore to Plant	kt	8,747
Total Recovered Gold	koz	1,620.4
Total Payable Gold	koz	1,618.7
Total Recovered Silver	koz	4,876.4
Total Payable Silver	koz	4,852.0
Operating Costs		
Mining Cost	US$/t processed	$70.54
Processing Cost	US$/t processed	$87.92
Tailing costs	US$/t processed	$7.37
G&A Cost	US$/t processed	$10.54
Total Operating Costs	US$/t processed	$176.37
Refining & Transport Cost	US$/oz	$11.06
Cash Costs *	US$/oz Au	$993.1
AISC **	US$/oz Au	$1,178.0
Capital Costs		
Initial Capital	US$M	$382.1
Sustaining Capital	US$M	$293
Closure Capital	US$M	$17.2
Financials - Pre Tax		

General	Unit	LOM Total Value/Average
NPV (5%)	US$M	$1,535.2
NPV (0%)	US$M	$2,701.5
NPV (10%)	US$M	$904.0
IRR (%)	%	38.5%
Payback (years)	years	2.7
Financials - Post Tax		
NPV (5%)	US$M	$1,344.5
NPV (0%)	US$M	$2,386.8
NPV (10%)	US$M	$781.1
IRR (%)	%	35.6%
Payback (years)	years	2.8

* Cash costs consist of mining costs, processing costs, international transport cost and royalties.
** AISC includes cash costs plus sustaining capital and closure cost.

Figure 19-2: Post-Tax-Free Cash Flow



Source: Ausenco, 2025.

Table 19-6: Cashflow Statement on an Annual Basis

General	Unit	LOM	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Production Summary																				
Total Material Mined	kt	8,717	40	71	365	365	584	584	584	584	584	583	584	584	584	584	584	549	549	350
Developed	kt	1,420	40	65	154	183	171	199	175	80	99	38	41	35	42	53	41	1	3	--
Ore mined	kt	7,297	--	6	212	183	413	385	410	504	485	545	543	549	543	531	544	548	547	350
Au Head Grade	g/t	6,0	--	7,60	8,68	8,68	6,53	6,38	6,48	6,01	6,09	5,56	5,43	5.45	5.34	5.32	4.80	5.93	5.54	5.53
Au Contained	koz	1,688	--	10.35	102.03	101.97	122.70	119.81	121.77	112.91	114.51	104.13	102.07	102.31	100.41	100.01	90.10	111.35	103.97	67.49
Au Recovery	%	96,0%	--	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	0.96
Au Recovered	koz	1,620.4	--	9.9	97.9	97.9	117.8	115.0	116.9	108.4	109.9	100.0	98.0	98.2	96.4	96.0	86.5	106.9	99.8	64.8
Au Payable	koz	1,618.7	--	9.9	97.8	97.8	117.7	114.9	116.8	108.3	109.8	99.9	97.9	98.1	96.3	95.9	86.4	106.8	99.7	64.7
Ag Head Grade	g/t	20.3	--	29.15	34.82	31.33	26.25	20.88	20.11	16.46	14.28	16.12	16.94	20.24	20.83	20.46	17.05	14.87	18.62	29.01
Ag Contained	koz	5,736.9	--	39.71	409.09	368.20	492.97	392.16	377.83	309.14	268.32	302.09	318.29	380.28	391.38	384.27	320.39	279.12	349.52	354.17
Ag Recovery	%	85.0%	--	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	0.85
Ag Recovered	koz	4,876.4	--	33.75	347.73	312.97	419.03	333.34	321.15	262.77	228.07	256.78	270.55	323.24	332.67	326.63	272.33	237.25	297.09	301.04
Ag Payable	koz	4,852.0	--	33.6	346.0	311.4	416.9	331.7	319.5	261.5	226.9	255.5	269.2	321.6	331.0	325.0	271.0	236.1	295.6	299.5
Prices																				
Gold Price	USD/oz	3,232		3,827	3,689	3,140	3,140	3,140	3,140	3,140	3,140	3,140	3,140	3,140	3,140	3,140	3,140	3,140	3,140	3,140
Silver Price	USD/oz	38		43	40	37	37	37	37	37	37	37	37	37	37	37	37	37	37	37
Revenues																				
Gold Revenue	US$/M	5,143		38.0	361.0	307.1	369.5	360.8	366.7	340.0	344.8	313.5	307.4	308.1	302.3	301.1	271.3	335.3	313.1	203.2
Silver Revenue	US$/M	180		1.4	13.8	11.5	15.4	12.2	11.8	9.6	8.4	9.4	$9.9	11.9	12.2	12.0	10.0	8.7	10.9	11.1
Operating Costs																				
Mine Costs	MUSD	(617)		(4.5)	(28.9)	(39.8)	(50.7)	(50.7)	(51.3)	(41.7)	(43.1)	(36.2)	(35.8)	(35.8)	(36.4)	(36.7)	(36.8)	(31.9)	(30.9)	(25.9)
Processing Costs	MUSD	(833)		(8.3)	(59.2)	(66.9)	(71.6)	(48.2)	(48.3)	(48.3)	(48.3)	(48.5)	(48.6)	(48.6)	(48.6)	(48.7)	(48.7)	(48.3)	(48.3)	(46.1)
G&A Costs	MUSD	(92)		(0.7)	(4.4)	(5.9)	(5.9)	(5.9)	(5.9)	(5.9)	(5.9)	(5.9)	(5.9)	(5.9)	(5.9)	(5.9)	(5.5)	(5.9)	(5.9)	(5.3)
Refining Charges, Transportation Cost & Royalties																				
Refining charges costs	MUSD	(18)		(0.1)	(1.1)	(1.1)	(1.3)	(1.3)	(1.3)	(1.2)	(1.2)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.0)	(1.2)	(1.1)	(0.7)
Royalties	MUSD	(369)		(2.7)	(26.0)	(22.1)	(26.7)	(25.9)	(26.3)	(24.3)	(24.5)	(22.4)	(22.0)	(22.2)	(21.8)	(21.7)	(19.5)	(23.9)	(22.5)	(14.9)
EBITDA																				
EBITDA	MUSD	3,393		23.1	255.2	182.8	228.7	241.1	245.5	228.3	230.2	208.8	203.9	206.4	200.7	199.1	169.8	232.9	215.4	121.4
Capital Expenditures																				
Initial Capital	MUSD	(382)	(229.3)	(152.8)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Sustaining Capital	MUSD	(293)	--	(7.6)	(78.2)	(43.5)	(42.1)	(22.0)	(14.4)	(10.7)	(13.2)	(14.6)	(3.8)	(7.5)	(4.0)	(8.2)	(13.8)	(6.5)	(2.6)	(0.6)
Closure Cost	MUSD	(17)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	(17.2)
Salvage Value	MUSD	1	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	0.8
Change in Working Capital																				
Change in Working Capital	MUSD	--	--	(3.2)	(27.6)	4.6	(5.4)	1.0	(0.4)	2.4	(0.3)	2.5	0.5	(0.2)	0.4	0.1	2.6	(5.2)	1.6	26.6
Pre-Tax Unlevered Free Cash Flow																				
Pre-Tax Unlevered Free Cash Flow	MUSD	2,701	(229.3)	(140.6)	149.4	144.0	181.1	220.0	230.6	220.0	216.7	196.7	200.6	198.7	197.2	191.0	158.6	221.2	214.4	131.1
Pre-Tax Cumulative Unlevered Free Cash Flow	MUSD	--	(229.3)	(369.9)	(220.4)	(76.5)	104.6	325	555	775	992	1,189	1,389	1,588	1,785	1,976	2,135	2,356	2,570	2,701
Cost KPI's																				
Cash Costs	USD/oz	993.1	--	1,395.3	992.8	1,180.9	1,117.0	953.2	946.2	932.3	936.3	952.7	965.5	957.5	979.1	986.4	1,087.1	857.2	895.4	1,204.0
AISC	USD/oz	1,178.0	--	2,134,8	1,762.5	1,609.3	1,460.9	1,138.8	1,065.9	1,028.8	1,053.5	1,094.9	1,002.6	1,030.8	1,019.1	1,068.5	1,241.3	916.4	920.6	1,452.5

19.5 Sensitivity Analysis

A sensitivity analysis was conducted on the pre-tax and post-tax NPV and IRR of the Project. Sensitivities to changes in the discount rate were evaluated first on both a pre-tax and post-tax basis, and the results are presented in Table 19-7. Additional sensitivities were then assessed for key economic variables, including gold price, sustaining capital costs, initial capital costs, and operating costs. The analysis indicates that the Project is most sensitive to fluctuations in gold prices, with lower sensitivity to changes in initial capital costs, operating costs, and sustaining capital costs, as shown in Table 19-8 and Figure 19-3.

Table 19-7: Sensitivities to Changes in the Discount Rate

Pre-Tax Unlevered NPV (MUSD)

Discount Rate \ Gold Price, US$/oz	(25%)	(10%)	--	10%	25%
1.0%	$1,321	$1,971	$2,404	$2,837	$3,487
3.0%	$1,019	$1,556	$1,915	$2,273	$2,810
5.0%	$786	$1,236	$1,535	$1,835	$2,284
8.0%	$530	$880	$1,114	$1,347	$1,698
10.0%	$404	$704	$904	$1,104	$1,404

Pre-Tax Unlevered IRR (%)

Discount Rate \ Gold Price, US$/oz	(25%)	(10%)	--	10%	25%
1.0%	23.8%	32.8%	38.5%	44.0%	52.0%
3.0%	23.8%	32.8%	38.5%	44.0%	52.0%
5.0%	23.8%	32.8%	38.5%	44.0%	52.0%
8.0%	23.8%	32.8%	38.5%	44.0%	52.0%
10.0%	23.8%	32.8%	38.5%	44.0%	52.0%

Pre-Tax Unlevered Payback

Discount Rate \ Gold Price, US$/oz	(25%)	(10%)	--	10%	25%
1.0%	4.3	3.2	2.7	2.3	1.9
3.0%	4.3	3.2	2.7	2.3	1.9
5.0%	4.3	3.2	2.7	2.3	1.9
8.0%	4.3	3.2	2.7	2.3	1.9
10.0%	4.3	3.2	2.7	2.3	1.9

Post-Tax Unlevered NPV (MUSD)

Discount Rate \ Gold Price, US$/oz	(25%)	(10%)	--	10%	25%
1.0%	$1,123	$1,723	$2,121	$2,519	$3,116
3.0%	$859	$1,354	$1,683	$2,013	$2,507
5.0%	$655	$1,069	$1,344	$1,620	$2,034
8.0%	$431	$753	$968	$1,183	$1,506
10.0%	$320	$597	$781	$966	$1,243

Post-Tax Unlevered IRR (%)

Discount Rate \ Gold Price, US$/oz	(25%)	(10%)	--	10%	25%
1.0%	21.5%	30.1%	35.6%	40.8%	48.4%
3.0%	21.5%	30.1%	35.6%	40.8%	48.4%
5.0%	21.5%	30.1%	35.6%	40.8%	48.4%
8.0%	21.5%	30.1%	35.6%	40.8%	48.4%
10.0%	21.5%	30.1%	35.6%	40.8%	48.4%

Post-Tax Unlevered Payback

Discount Rate \ Gold Price, US$/oz	(25%)	(10%)	--	10%	25%
1.0%	4.5	3.3	2.8	2.4	2.0
3.0%	4.5	3.3	2.8	2.4	2.0
5.0%	4.5	3.3	2.8	2.4	2.0
8.0%	4.5	3.3	2.8	2.4	2.0
10.0%	4.5	3.3	2.8	2.4	2.0

Table 19-8: Sensitivity Analysis Pre- Tax

Pre-Tax NPV Sensitivity To Opex

Opex \ Gold Price, USD/oz	(25.0%)	(10.0%)	--	10.0%	25.0%
(20.0%)	$592	$1,042	$1,341	$1,640	$2,090
(10.0%)	$689	$1,139	$1,438	$1,738	$2,187
--	$786	$1,236	$1,535	$1,835	$2,284
10.0%	$884	$1,333	$1,632	$1,932	$2,381
20.0%	$981	$1,430	$1,729	$2,029	$2,478

Pre-Tax IRR Sensitivity To Opex

Opex \ Gold Price, USD/oz	(25.0%)	(10.0%)	--	10.0%	25.0%
(20.0%)	19.5%	28.9%	34.7%	40.3%	48.4%
(10.0%)	21.7%	30.8%	36.6%	42.2%	50.2%
--	23.8%	32.8%	38.5%	44.0%	52.0%
10.0%	25.8%	34.7%	40.4%	45.8%	53.7%
20.0%	27.9%	36.6%	42.2%	47.6%	55.5%

Pre-Tax Payback Sensitivity To Opex

Opex \ Gold Price, USD/oz	(25.0%)	(10.0%)	--	10.0%	25.0%
(20.0%)	5.0	3.6	3.0	2.6	2.1
(10.0%)	4.6	3.4	2.8	2.4	2.0
--	4.3	3.2	2.7	2.3	1.9
10.0%	4.0	3.0	2.5	2.2	1.8
20.0%	3.7	2.8	2.4	2.1	1.7

Pre-Tax NPV Sensitivity To Capex

Initial Capex \ Gold Price, USD/oz	(25.0%)	(10.0%)	--	10.0%	25.0%
(20.0%)	$715	$1,164	$1,464	$1,763	$2,213
(10.0%)	$751	$1,200	$1,500	$1,799	$2,248
--	$786	$1,236	$1,535	$1,835	$2,284
10.0%	$822	$1,271	$1,571	$1,870	$2,320
20.0%	$858	$1,307	$1,607	$1,906	$2,355

Pre-Tax IRR Sensitivity To Capex

Initial Capex \ Gold Price, USD/oz	(25.0%)	(10.0%)	--	10.0%	25.0%
(20.0%)	20.1%	28.1%	33.1%	37.9%	44.9%
(10.0%)	21.8%	30.3%	35.6%	40.7%	48.2%
--	23.8%	32.8%	38.5%	44.0%	52.0%
10.0%	26.0%	35.8%	41.9%	47.9%	56.5%
20.0%	28.7%	39.3%	46.0%	52.5%	61.9%

Pre-Tax Payback Sensitivity To Capex

Initial Capex \ Gold Price, USD/oz	(25.0%)	(10.0%)	--	10.0%	25.0%
(20.0%)	4.8	3.6	3.1	2.7	2.2
(10.0%)	4.5	3.4	2.9	2.5	2.1
--	4.3	3.2	2.7	2.3	1.9
10.0%	4.0	2.9	2.5	2.1	1.7
20.0%	3.7	2.7	2.3	1.9	1.5

Pre-Tax NPV Sensitivity To Sustaining Capex

Sustaining CAPEX \ Gold Price, USD/oz	(25.0%)	(10.0%)	--	10.0%	25.0%
(20.0%)	$743	$1,192	$1,492	$1,791	$2,240
(10.0%)	$765	$1,214	$1,513	$1,813	$2,262
--	$786	$1,236	$1,535	$1,835	$2,284
10.0%	$808	$1,258	$1,557	$1,857	$2,306
20.0%	$830	$1,279	$1,579	$1,878	$2,328

Pre-Tax IRR Sensitivity To Sustaining Capex

Sustaining CAPEX \ Gold Price, USD/oz	(25.0%)	(10.0%)	--	10.0%	25.0%
(20.0%)	22.4%	31.3%	37.0%	42.4%	50.4%
(10.0%)	23.1%	32.0%	37.7%	43.2%	51.2%
--	23.8%	32.8%	38.5%	44.0%	52.0%
10.0%	24.5%	33.5%	39.3%	44.8%	52.8%
20.0%	25.2%	34.3%	40.1%	45.6%	53.6%

Pre-Tax Payback Sensitivity To Sustaining Capex

Sustaining CAPEX \ Gold Price, USD/oz	(25.0%)	(10.0%)	--	10.0%	25.0%
(20.0%)	4.5	3.4	2.8	2.4	2.0
(10.0%)	4.4	3.3	2.8	2.4	1.9
--	4.3	3.2	2.7	2.3	1.9
10.0%	4.1	3.1	2.6	2.3	1.8
20.0%	4.0	3.0	2.5	2.2	1.8

Table 19-9: Sensitivity Analysis Post-Tax

Post-Tax NPV Sensitivity To Opex

Opex \ Gold Price, USD/oz	(25.0%)	(10.0%)	--	10.0%	25.0%
(20.0%)	$471	$875	$1,152	$1,428	$1,841
(10.0%)	$563	$972	$1,248	$1,524	$1,937
--	$655	$1,069	$1,344	$1,620	$2,034
10.0%	$750	$1,165	$1,441	$1,716	$2,130
20.0%	$847	$1,261	$1,537	$1,813	$2,228

Post-Tax IRR Sensitivity To Opex

Opex \ Gold Price, USD/oz	(25.0%)	(10.0%)	--	10.0%	25.0%
(20.0%)	17.4%	26.0%	31.6%	37.0%	44.8%
(10.0%)	19.4%	28.1%	33.6%	38.9%	46.6%
--	21.5%	30.1%	35.6%	40.8%	48.4%
10.0%	23.5%	32.1%	37.5%	42.7%	50.2%
20.0%	25.6%	34.1%	39.4%	44.5%	52.0%

Post-Tax Payback Sensitivity To Opex

Opex \ Gold Price, USD/oz	(25.0%)	(10.0%)	--	10.0%	25.0%
(20.0%)	5.3	3.8	3.2	2.7	2.2
(10.0%)	4.9	3.6	3.0	2.6	2.1
--	4.5	3.3	2.8	2.4	2.0
10.0%	4.2	3.1	2.7	2.3	1.9
20.0%	3.8	2.9	2.5	2.2	1.9

Post-Tax NPV Sensitivity To Capex

Initial Capex \ Gold Price, USD/oz	(25.0%)	(10.0%)	--	10.0%	25.0%
(20.0%)	$594	$1,005	$1,281	$1,557	$1,970
(10.0%)	$625	$1,037	$1,313	$1,588	$2,002
--	$655	$1,069	$1,344	$1,620	$2,034
10.0%	$686	$1,101	$1,376	$1,652	$2,065
20.0%	$717	$1,132	$1,408	$1,683	$2,097

Post-Tax IRR Sensitivity To Capex

Initial Capex \ Gold Price, USD/oz	(25.0%)	(10.0%)	--	10.0%	25.0%
(20.0%)	18.3%	25.9%	30.7%	35.3%	42.0%
(10.0%)	19.8%	27.8%	32.9%	37.9%	44.9%
--	21.5%	30.1%	35.6%	40.8%	48.4%
10.0%	23.4%	32.7%	38.6%	44.3%	52.5%
20.0%	25.8%	35.9%	42.3%	48.5%	57.3%

Post-Tax Payback Sensitivity To Capex

Initial Capex \ Gold Price, USD/oz	(25.0%)	(10.0%)	--	10.0%	25.0%
(20.0%)	5.0	3.8	3.2	2.8	2.3
(10.0%)	4.7	3.5	3.0	2.6	2.2
--	4.5	3.3	2.8	2.4	2.0
10.0%	4.2	3.1	2.6	2.3	1.9
20.0%	4.0	2.9	2.4	2.1	1.7

Post-Tax NPV Sensitivity To Sustaining Capex

Sustaining CAPEX \ Gold Price, USD/oz	(25.0%)	(10.0%)	--	10.0%	25.0%
(20.0%)	$613	$1,025	$1,302	$1,578	$1,991
(10.0%)	$634	$1,047	$1,323	$1,599	$2,012
--	$655	$1,069	$1,344	$1,620	$2,034
10.0%	$676	$1,090	$1,366	$1,641	$2,055
20.0%	$698	$1,111	$1,387	$1,663	$2,077

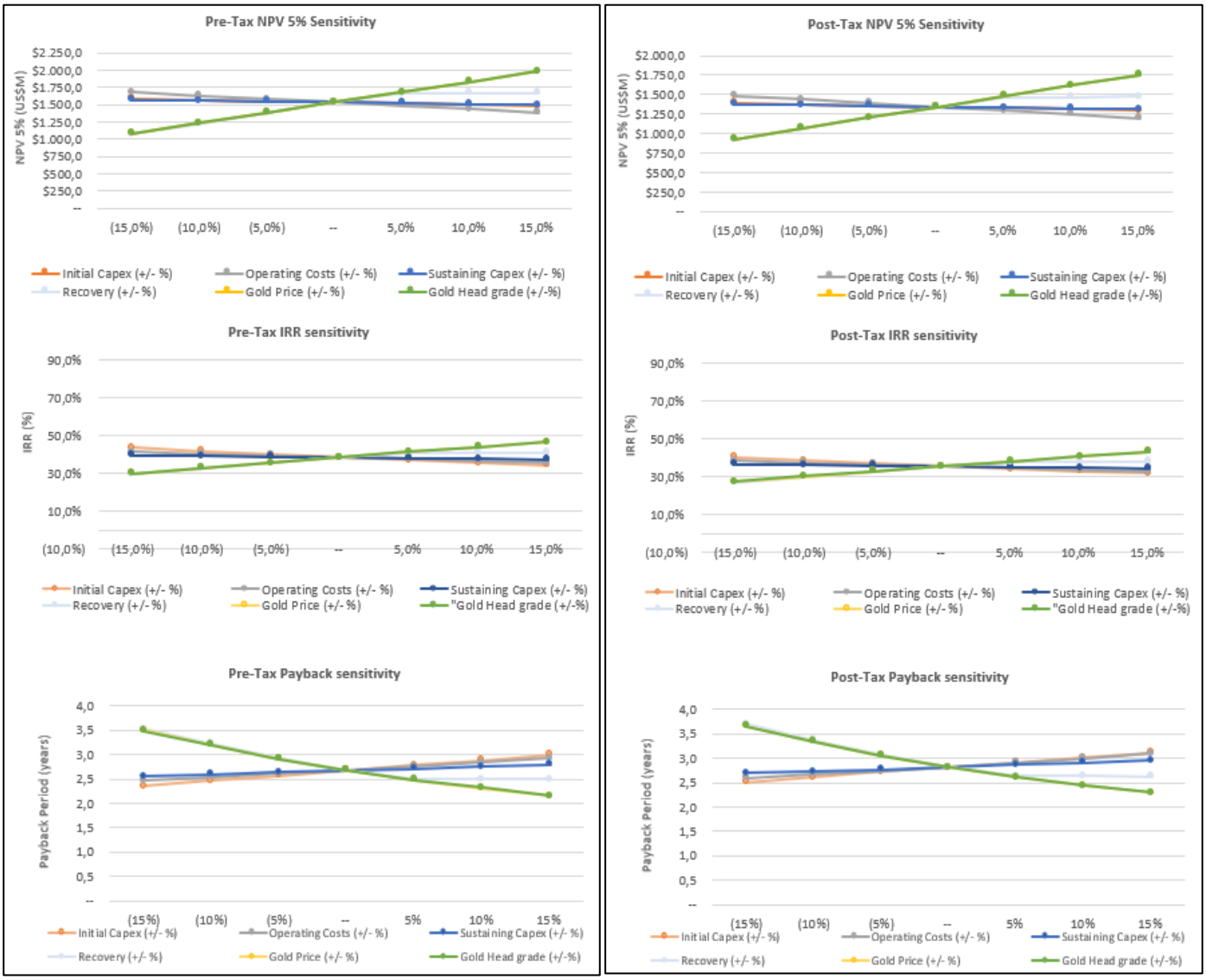
Post-Tax IRR Sensitivity To Sustaining Capex

Sustaining CAPEX \ Gold Price, USD/oz	(25.0%)	(10.0%)	--	10.0%	25.0%
(20.0%)	20.1%	28.6%	34.0%	39.3%	46.8%
(10.0%)	20.8%	29.3%	34.8%	40.0%	47.6%
--	21.5%	30.1%	35.6%	40.8%	48.4%
10.0%	22.2%	30.8%	36.3%	41.6%	49.2%
20.0%	22.9%	31.6%	37.1%	42.4%	50.0%

Post-Tax Payback Sensitivity To Sustaining Capex

Sustaining CAPEX \ Gold Price, USD/oz	(25.0%)	(10.0%)	--	10.0%	25.0%
(20.0%)	4.8	3.6	3.0	2.6	2.1
(10.0%)	4.6	3.5	2.9	2.5	2.1
--	4.5	3.3	2.8	2.4	2.0
10.0%	4.3	3.2	2.7	2.4	2.0
20.0%	4.2	3.1	2.6	2.3	1.9

Figure 19-3: Sensitivity Analysis Pre-Tax and Post-Tax



Source: Ausenco, 2025.

19.5.1 Indicative Financing Scenario Comments on Economic Analysis

In addition to the unlevered base case economic analysis, an indicative financing scenario was evaluated to illustrate the potential impact of debt financing on Project financial performance. This scenario assumes a capital structure comprising 50% debt on total initial capital with an upfront debt facility of US$191 million used to partially fund construction capital expenditures.

The loan is assumed to have a 5-year term, with 2-year grace period and 7% annual interest rate. Repayment is scheduled to begin after the grace period and will be completed over the life of the Project.

This analysis is presented for illustrative purposes only and does not form part of the Project base case economic evaluation.

The financial metrics presented in Table 19-10 presents leveraged, equity-level financing results under the indicative financing scenario and does not represent unlevered project results. Based on the assumptions and parameters presented in this report, the PFS shows positive economics.

Table 19-10: Parameters for Financing – 50% Debt

Indicative Loan assumptions (interest-only grace period)	Value
Gearing (% of initial CAPEX)	50.0%
Loan Principal, MUS$	$191.06
Grace period, years	2
Repayment periods, years	3
Years term, years	5
Annual interest rate, %	7.0%

Table 19-11: Summary Results for Financing – 50% Debt

General Financials - Pre Tax	Unit	LOM Total Value / Average Unleveraged	LOM Total Value / Average Leveraged
NPV (5%)	US$M	1,535	1,525
NPV (0%)	US$M	2,702	2,661
NPV (10%)	US$M	904	917.1
IRR (%)	%	38.5%	55.1%
Payback (years)	years	2.7	2.7
NPV (5%)	US$M	1,345	1,335
NPV (0%)	US$M	2,387	2,346
NPV (10%)	US$M	781	794
IRR (%)	%	35.6%	49.8%
Payback (years)	years	2.8	2.9

19.6 Comments on Economic Analysis

Based on the assumptions and parameters presented in this Report, the FS economic analysis yields positive results. However, these results remain subject to the assumptions and uncertainties described elsewhere in this Report.

20 ADJACENT PROPERTIES

There are no adjacent properties relevant to the scope of this report.

21 OTHER RELEVANT DATA AND INFORMATION

There is no further relevant data or information to be submitted.

22 INTERPRETATION AND CONCLUSIONS

22.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this Report.

22.2 Geology and Mineral Resources

Era Dorada is a classic hot springs-related, low-sulfidation epithermal gold-silver deposit comprising both high-grade vein and low-grade disseminated mineralization. Most of the high-grade mineralization is hosted in the Mita unit as two upward-flaring vein swarms (north and south zones) that converge downwards and merge into basal feeder veins where drilling has demonstrated widths of high-grade mineralization (e.g., 15.5 m at 21.4 g/t Au, and 52 g/t Ag). Bonanza gold grades are associated with ginguru banding and carbonate replacement textures. Sulfide contents are low, typically <3% by volume. Low-grade disseminated and veinlet mineralization in wall rocks around the high-grade veins is well documented in drilling since discovery of the deposit, with grades typically ranging from 0.3 to 3.0 g/t Au.

The Mita rocks are overlain by the Salinas unit, a sub-horizontal sequence of volcanogenic sediments and sinter horizons approximately 100 m thick that form the low-lying hill at the Project. Low-grade disseminated and veinlet mineralization within and as halos around the high-grade vein swarms is well documented in drilling since discovery of the deposit, with grades typically ranging from 0.3 to 1.5 g/t Au. The overlying Salinas cap rocks are also host to low-grade mineralization associated with silicified conglomerates and rhyolite intrusion breccias.

Mineral exploration activities performed at Era Dorada have been performed in accordance with “CIM Mineral Exploration Best Practice Guidelines” dated November 23, 2018.

The mineral resource has a footprint of 800 x 400 m between elevations of 525 and 200 masl. The mineral resource estimate is the result of 153,003 m of drilling by Bluestone and previous operators (totaling 1,256 Drill holes and channel samples). There are 130,307 gold assays which average 0.68 g/t and 130,238 silver assays or 153,003 m total which average 3.75 g/t. Bulk densities were assigned to individual rock types and assigned on a block-by-block basis using measurement data by lithology and mineralized vein.

The 3.4 km of underground infrastructure allowed for underground mapping, sampling, and over 30,000 m of underground drilling enhanced the understanding and validation of the Era Dorada geological model. The mineral resource estimate included an estimate of dilutive material, some of which has proven to be economic and have a reasonable prospect of economic extraction. Therefore, improved and refined geological models of the lithological units was required. These broad mineralized lithologies are host to the high-grade veins that have been the focus of the potential underground mining scenario. The resulting domain models and estimation strategy were designed to accurately represent the grade distribution.

The estimate was completed using MineSight™ software using a 3D block model (5 m by 5 m by 5 m). Interpolation parameters have been derived based on geostatistical analyses conducted on 1.5-meter composited drill holes. Block grades have been estimated using ordinary kriging (OK) methodology and the mineral resources have been classified based on proximity to sample data and the continuity of mineralization in accordance with SEC Regulation S-K Subpart 1300, CIM's "Definition Standards for Mineral Resources and Mineral Reserves" dated May 19, 2014, and "CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" dated November 29, 2019. The mineral resources are presented in at a 2.25 g/t Au/t cut-off grade.

Table 22-1: Resource Estimate using 2.25 g/t Au Cut-off Inclusive of Reserves

Resource Category	Tonnes (kt)	Au Grade (g/t)	Ag Grade (g/t)	AuEq Grade (g/t)	Contained Gold (koz)	Contained Silver (koz)	Contained AuEq (koz)
Measured							
Indicated	7,059	9.03	30.66	9.36	2,049	6,958	2,125
Measured & Indicated	7,059	9.03	30.66	9.36	2,049	6,958	2,125
Inferred	736	5.94	19.22	6.16	141	455	146

Table 22-2: Resource Estimate using 2.25 g/t Au Cut-off Exclusive of Reserves

Resource Category	Tonnes (kt)	Au Grade (g/t)	Ag Grade (g/t)	AuEq Grade (g/t)	Contained Gold (koz)	Contained Silver (koz)	Contained AuEq (koz)
Measured							
Indicated	2,460	6.36	22.76	6.61	503	1,801	523
Measured & Indicated	2,460	6.36	22.76	6.61	503	1,801	523
Inferred	736	5.94	19.22	6.16	141	455	146

Notes: The mineral resource statement is subject to the following:

1. Mineral Resources are reported in in accordance with S-K 1300.
2. Mineral resource estimates have been prepared by Garth Kirkham, P.Geo., a Qualified Person as defined by SK-1300.
3. The Mineral Resource estimate is reported on a 100% ownership basis.
4. Underground mineral resources are reported at a cut-off grade of 2.25 g/t Au. Cut-off grades are based on a assumed metal prices of US$ 2,500/oz gold and US$ 28/oz silver, and assumed metallurgical recovery, mining, processing, and G&A costs.
5. Mineral Resources are reported without applying mining dilution, mining losses, or process losses.
6. Resources are constrained within underground shapes based on reasonable prospects of economic extraction, in accordance with SK-1300. Reasonable prospects for economic extraction were met by applying mining shapes with a minimum mining width of 2.0 m, ensuring grade continuity above the cut-off value, and by excluding non-mineable material prior to reporting.
7. Metallurgical recoveries reported as the average over the life of mine and are assumed to be 96% Au and 85% Ag, respectively.
8. Bulk density is estimated by lithology and averages 2.47, 2.57 and 2.54 g/cm3 for the Salinas, Mita and mineralized vein domains, respectively.
9. Mineral resources are classified as Indicated, and Inferred based on geological confidence and continuity, spacing of drill holes, and data quality.
10. Effective date of the mineral resource estimate is November 30, 2025.
11. Tonnage, grade, and contained metal values have been rounded. Totals may not sum due to rounding.
12. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

Source: Kirkham, 2025.

In addition, there has been mixed grade material mined during the creation of the extensive, existing ramp network which has been stockpiled adjacent to North Ramp entrance. Table 22-3shows the volume and tonnage based on an unconsolidated specific gravity of 2.0 along with gold and silver grades and metal content. These resources are classified as measured.

Table 22-3: Stockpile Resource Estimate (Measured Resource)

Volume (BCM)	Mine (t)	Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)
14,863	29,726	5.35	22.59	5,108	21,590

Source: Kirkham, 2019.

Era Dorada represents a well-defined, hot springs–related low-sulfidation epithermal gold–silver system with demonstrated geological continuity, robust grade distribution, and a substantial high-grade vein component supported by extensive surface and underground data. The mineral resource estimate is underpinned extensive drilling, detailed underground mapping and sampling, and compliance with SEC Regulation S-K Subpart 1300. The resulting Indicated and Inferred resources, together with measured stockpile material, are constrained within shapes demonstrating reasonable prospects for economic extraction at conservative cut-off grades and metal price assumptions. Based on the quality, quantity, and validation of the geological, analytical, and modeling data, it is the conclusion of the Qualified Person that the Era Dorada Project is at a sufficiently advanced stage to support mineral resource disclosure and to form a sound basis for continued technical and economic evaluation.

22.3 Metallurgical Testwork

Metallurgical testwork was conducted on samples from the Era Dorada deposit between April 1999 and January 2012 by Kappes, Cassiday & Associates (KCA) in Reno, NV. The most recent test program, completed in 2018 in support of this FS, was carried out at Base Metallurgical Laboratories Ltd. (BaseMet) in Kamloops, B.C.

The focus of the recent test program was to optimize the flowsheet and generate tailings for geochemistry, geotechnical and paste backfill testing. A global composite from drill core was created to run the optimization test program. The testwork included grind extraction optimization, gravity, leach optimization, tailings generation and cyanide destruction. Bulk samples from the underground workings were collected and two composites were created to represent the North and South areas of the deposit. The final flowsheet and test parameters determined in the optimization phase were used to generate tailings samples from the North and South zones for physical and chemical characterization to be used in defining DSTF and backfill applications.

Based on the results from BaseMet (2018), gold and silver doré can be produced at a primary grind size of 80% passing (P80) 53 µm followed by gravity concentration, 2-hour pre-oxidation, a 36-hour cyanide leach at a sodium cyanide concentration of 500 mg/L, 6-hour carbon-in-pulp (CIP) adsorption, carbon desorption, electrowinning and refining. For the global composite, this recovery method achieved average precious metal recoveries of 96% Au and 85% Ag.

Although testing has been limited, arsenic has been identified as a deleterious element that will require treatment and removal from any water discharged from site. No other deleterious elements have been identified that may impair bullion quality, although testing has been limited.

22.4 Mineral Reserve Estimate

The Mineral Reserve estimates were completed using industry-standard methodologies and software, and the Mineral Reserve is reported in accordance with S-K 1300 requirements. The mine plan supports an economically viable underground operation, subject to the stated assumptions, modifying factors, and risk controls.

The Mineral Reserve estimate was subject to Legal and Permitting constraints and other modifying factors such as the plant and infrastructure timing and capacities, the metallurgical recoveries for gold and silver, economic factors as gold and silver prices, costs and exchange rates as well as technical modifying factors derived from geotechnical constraints, mining methods and productivity.

The Mineral Reserve was estimated from the Measured and Indicated Mineral Resources which demonstrate economic viability, incorporating dilution allowances and mining recovery factors.

Any Inferred Mineral Resources enclosed in feasible Indicated Mineral Resources envelopes either inside the stope shapes, dilution material or forced mine development was treated as waste, i.e., such material carried its excavation costs and no revenue was accounted for.

The economic assumptions which underpin the conversion of Indicated Resources to Probable Reserve were defined in the beginning of the Feasibility Study. The gold and silver prices are consistent with Aura guidance and were deemed adequate at that stage by the QP.

Assumptions for gold and silver recoveries and mining costs are based on previous studies for the deposit and are assessed as adequate, within industry standards, by the QP.

The Mineral Reserve estimate is summarized in Table 22-4.

Table 22-4: Mineral Reserves

	Tonnage (kt)	Au grade (g/t)	Au metal (koz)	Ag grade (g/t)	Ag metal (koz)	Au Equiv grade (g/t)	Au Equiv metal (koz)
Proven	30	5.35	5	22.59	22	5.60	5
Probable	8,717	6.01	1,684	20.39	5,715	6.23	1,746
Proven + Probable	8,747	6.01	1,689	20.40	5,736	6.23	1,751

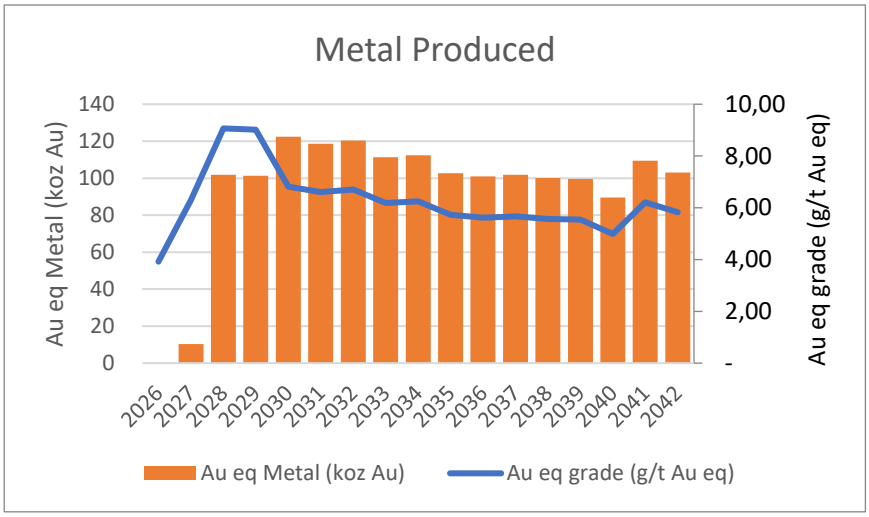
Mineral Reserve Notes:

1. The Mineral Reserve was estimated and classified in accordance with the USA S-K 1300 standards.
2. Mineral Reserve has an effective date of December 5, 2025. The Qualified Person for the estimate is Ruy Lacourt, BSc. Mining Engineering, MSc., Registered Member of the SME, an Associate of Snowden Optiro.
3. The Mineral Reserve was estimated using metal prices of US$2,000/oz Au and US$25/oz Ag, and metallurgical recoveries of 96% Au and 85% Ag. Underground mining costs were assumed as US$100/t (Long Hole mining) and US$115/t (Cut-and-Fill mining), with processing, site services and G&A costs as of US$32/t, US$18/t and US$20/t, respectively. Royalties comprise 1.05% NSR to the previous owners plus a 1.0% gross government royalty. Cutoff grades in gold equivalent are 2.82g/t Au eq for underground Long Hole mining and 3.07 g/t Au eq for Cut-and-fill.
4. The formula for gold equivalent is: Au eq = Au grade + 0.011 * Ag grade.

5. The Mineral Reserve is presented on a 100% ownership basis fully attributable to Aura Minerals.
6. Tonnages and grades have been rounded in accordance with reporting guidelines. Tonnages are rounded to the nearest 1,000 t, metal grades are rounded to two decimal places. Tonnage and grade are in metric units, containing gold and silver are reported as thousands of troy ounces. Totals may not sum due to rounding.
7. The existing surface stockpile (29,726 t, dry basis, at 5.35 g/t Au and 22.59 g/t Ag) were evaluated using the same economic parameters as the underground Mineral Reserve and is classified as Proven Mineral Reserve.

The mine plan was designed to achieve a target production rate of 1,600 t/d to deliver a metal equivalent production averaging 100 koz of gold equivalent for 15 years for a total mine life of 18 years.

Figure 22-1: Gold Production and Grades



Source: Snowden, 2025.

22.5 Mining Methods

22.5.1 Mine Geotechnical

The geotechnical assessment carried out for the Era Dorada underground project indicates that the mining method, stope geometry and development strategy proposed for the feasibility level are technically viable, provided that the recommendations outlined in this report are incorporated into the next design stages and mine operation.

A review of the geotechnical database allowed to establish a new geomechanical model based on a parameter-driven RMR reassessment – replacing the lithology only based model, producing geotechnical domains that are more representative of *in-situ* behaviour and suitable for engineering analyses.

Empirical assessments indicate that Domains 2 and 3 are appropriate for Long hole stoping under the geometries studied. Domain 1 rock masses require systematic support (cable bolts) for dips below 60°.

Dilution estimates indicate that when mining–filling sequences are applied backfill plays a dominant role in confinement, significantly reducing the potential for overbreak. Therefore, adherence to the prescribed mining and filling sequence is essential for stability.

Crown pillar stability assessment shows that a 10-m-thick crown pillar provides adequate stability for the expected rock-mass conditions. Pillars between stopes present acceptable safety margins for 6m pillars, and their long-term stability is strongly dependent on fill performance and adherence to the mine-fill sequence.

Development reinforcement and support recommendations preclude that all excavations should require rock bolts, with modifications in surface support depending on domain and span. Swellex Mn12 (galvanized) or similar bolts and 5-cm shotcrete should be used as a baseline, welded mesh can replace the shotcrete in selected areas. Intersections will require cable bolts due to greater spans and operational practicality.

In summary, the Era Dorada underground mine is geotechnically feasible under the conditions evaluated. Continued generation of geomechanical data, proper sequencing of mining and backfilling, updated numerical modelling, and refinement of support strategies are essential to ensure safe and efficient development of the operation.

22.5.2 Hydrogeology Analysis and Dewatering

Groundwater management and mine dewatering have a pivotal role to secure the safety of the operations, enabling mining below the water table.

Modelling results indicate that the current permitted discharge capacity of approximately 5,250 gpm (about 330 L/s) will become insufficient as pumping requirements increase with mine development. Total dewatering demand is projected to reach about 6,080 gpm (roughly 384 L/s) by 2029, exceeding the current discharge limit. Under the maximum development scenario, ten dewatering wells will be operating simultaneously by January 2031, with a combined pumping rate on the order of 7,600 gpm (approximately 480 L/s).

To accommodate projected excess flows and maintain operational flexibility two dedicated reinjection wells, each designed to handle about 1,000 gpm (approximately 63 L/s), starting in the fifth year of operation will be required. The reinjection wells will operate in parallel with existing surface discharge infrastructure, providing additional capacity to manage peak dewatering rates beyond 2031 and reducing dependence on a single discharge pathway. Expansion of discharge permits and planning for additional disposal options are required to ensure that increased pumping rates can be managed without constraining mine production.

Although these reductions are relevant in hydrogeologic terms, the current discharge permit capacity exceeds the magnitude of modeled baseflow depletion. In way that the treated effluent from the mine can generate a net surplus of water requiring disposal. As a result, the volumes of treated water discharged to receiving streams are expected to surpass the reduction in natural baseflow, helping maintain downstream flows within regulatory and environmental criteria.

The geothermal nature of the system introduces specific design and operational risks. The presence of groundwater at temperatures up to 190 °C creates potential for steam flashing if pressures are not adequately controlled in wells, pipelines and underground workings.

22.5.3 Mining Methods

The Mineral Reserve will be mined using a combination of long hole stoping (LH), Cut-and-fill (MCF) and minor Room-and-pillar, utilizing paste fill and cemented rockfill. Long hole stoping is the main method, accounting for approximately 98.5% of total metal production, while Cut-and-fill will contribute with 1.2% and room and pillar with only 0.1%.

The mining method selection was primarily guided by geotechnical rock quality, vein geometry, and orebody continuity. Long hole stoping was applied as the preferred mining method due to its safer working conditions, higher productivity and lower unit mining costs relative to MCF. Where geotechnical or geometric conditions are required, mechanized Cut-and-fill was otherwise applied.

The preferred mining method for Era Dorada is sublevel long hole stoping (LH), owing to its safer working conditions, higher productivity and lower operating costs. LH stoping will be applied wherever geotechnical and geometric conditions allow for efficient stope design and operation.

Longitudinal and transverse long hole stoping configurations will be used: Longitudinal layouts will be applied for 78% of the long hole mining, while transverse stoping will be employed in the wider zones of the deposit exceeding 20 m.

Cut-and-fill is planned for areas with less favorable rock quality and/or where the mineralization geometry is not suitable for long hole (LH) stoping.

Both the Long hole and Cut-and-fill areas will be extracted using overhand (bottom-up) sequences.

All the stopes will be backfilled with cemented paste fill or cemented rockfill to provide structural confinement.

The primary mine geometry will be based on panels 100 m high, each composed by four sublevels of 20 m vertically, plus a sill pillar, also of 20m vertically that will be reclaimed at the end of the mine life. For a given mine area in the South and North zones, each panel can be operated independently to allow for increased operational flexibility and secure production rates.

22.5.4 Mine Infrastructure

The ventilation, cooling, and underground pumping systems were designed at a Feasibility Study level. Capital and operating costs have been estimated with Feasibility Study accuracy and integrated into the mine plan and economic analysis. The implementation of the mine infrastructure systems for ventilation, thermal conditions, and underground water management will enable industry standards for mine safety, production rates and economic performance over the life of the mine.

The ventilation system for Era Dorada is a push-pull system, with all the main fans to be installed on surface.

The cooling solution will consist of three cooling systems located in the intake raises (SH02, SH03, and NH05) to be commissioned early in the mine life.

On the South Zone, the two existing raises (SH02 and SH03) will be used as intake raises. The SH02 raise will be equipped with intake fans in conjunction with a 6 MWR plant, while the SH03 model will have a 1 MWR cooling plant.

On the North Zone, an intake shaft (NH05, built) will be equipped with main fans in conjunction with a 3.6 MWR cooling plant and an exhaust shaft (NH06, built) will be equipped with two centrifugal fans.

Centrifugal exhaust fans will be installed due to system resistance and air characteristics, which have high humidity and a high risk of corrosion; the fans must have wear plates and paint schemes suitable for humid, abrasive, and corrosive environments.

The underground pumping system to the surface includes main pumping stations for both the South and North zones of the mine: the South Zone will have three main pumping stations on levels 210, 320 and 420. Six pumps with a designated duty point and power consumption of 285 kW will meet the required flow rate as of 50.0 liters per second. The North Zone will have two main pumping stations on levels 270 and 370. Four pumps operating at a single duty point and consuming 38 kW will handle a 7.0 liters per second flow. Pumps and tubes were designed to withstand high water temperatures.

22.6 Recovery Plan

The selected flowsheet aligns with conventional practices in the industry. Comminution, gravity concentration, precious metal extraction and recovery of precious metals, destruction of free cyanide and handling of tailings are achieved through conventional processes that are commonly used in the industry for similar projects with no significant elements of technological innovation. Previous studies, coupled with historical and new testwork results and financial evaluations, were used to develop the resulting flowsheet suitable for the blend of rock groups and the feed grades expected over the LOM.

The plant has the capacity of 1600 t/d, with overall availability and utilization of 92%. The primary crusher circuit design is set at 75%, and the grinding, gravity concentration, leach/CIP circuit, cyanide detox and tailings filtration is set at 92% availability and utilization. The project has an estimated life of 17 years.

22.7 Infrastructure

22.7.1 Geotechnical Mine Waste Facilities

The information available for the development of the Geotechnical design of the waste rock and tailings' facilities is not sufficient to assess long term stability of these structures. For example, the foundation needs a complementary investigation campaign to be adequately characterized. The same is valid for the tailings, once the beneficiation plant is not even constructed. Therefore, the mine waste facilities design was treated as sufficient for a PFS design level only, and the recommendations presented must be followed so that the project can gain geotechnical maturity.

Additionally, considering the extension of the mine's operational life and consequently the increased generation of waste rock and tailings, combined with the restricted area licensed under the 2007 EIA, it becomes mandatory to pursue expansion of the licensed boundaries, evaluate the acquisition of adjacent land, and assess the initiation of

disposal activities in the currently licensed areas, while accounting for space limitations, logistical constraints, and constructability challenges during the rainy season.

22.7.2 Water Management

The integrated assessment of the surface water management system, water balance, and treatment infrastructure demonstrates that the project is supported by appropriately designed systems capable of ensuring effective water management, segregation of contact and non-contact flows, and compliance with applicable environmental and operational requirements. The hydrological and hydraulic modeling indicates the need for updated topographic surveys to reduce uncertainties related to potential flood hazards and reline the design of the proposed protective structures.

The water balance analysis confirms that, although the current storage capacities are adequate for the initial operational conditions, increases in groundwater inflows will require enhanced pumping capacity and revisions to existing water rights in the coming years. Additionally, the water treatment infrastructure—including the mine water treatment plant, process water treatment facilities, potable water supply system, and sanitary wastewater treatment—meets the operational needs of the project and adheres to the relevant environmental standards, ensuring that treated water is suitable for reuse and controlled discharge.

Collectively, these systems provide a robust technical and operational framework that supports the continuity of mining activities, ensures regulatory compliance, and strengthens the project's commitment to sustainable water resource management.

The integrated assessment of the surface water management system, water balance, and treatment infrastructure demonstrates that the project is supported by appropriately designed systems capable of ensuring effective water management, segregation of contact and non-contact flows, and compliance with applicable environmental and operational requirements. The hydrological and hydraulic modeling indicates the need for updated topographic surveys to reduce uncertainties related to potential flood hazards.

The water balance analysis confirms that, although the current storage capacities are adequate for the initial operational conditions, increases in groundwater inflows will require enhanced pumping capacity and revisions to existing water rights in the coming years. Additionally, the water treatment infrastructure—including the mine water treatment plant, process water treatment facilities, potable water supply system, and sanitary wastewater treatment—meets the operational needs of the project and adheres.

1.2.3 Power and Electrical

The proposed electrical infrastructure has been designed to ensure reliability, operational flexibility, and scalability to meet both current and future project requirements. The strategy provides a secure transition from temporary generation to permanent utility interconnection, guaranteeing continuous power availability from construction through full operation. The system architecture comprising the main substation, primary and secondary distribution networks, emergency power systems, and redundancy measures offers robust support for critical loads, optimized

performance, and safe operation. This design establishes a solid foundation for the plant's electrical supply, aligned with industry best practices and the technical standards required for the successful execution of the project.

22.7.3 Fuel

The current diesel storage infrastructure, consisting of two 37,500-liter tanks within a containment area, meets existing operational needs. However, project expansion will result in a significant increase in fuel consumption due to greater use of generators, an expanded fleet for ore transport and handling, and the operation of equipment for waste and tailings management. To ensure adequate supply, additional infrastructure for fuel procurement, logistics, distribution, and storage will be contracted, guaranteeing availability for underground mining fronts and ore handling at processing and storage facilities. Light vehicles powered by gasoline or other fuels will continue to be refueled off-site in Asunción Mita, Guatemala.

22.8 Environmental, Permitting and Social Considerations

The project currently has all the required environmental licenses, including the Environmental Impact Assessment (EIA) approved for the underground mining operation. However, some of the proposed modifications to the project will require updates or new regulatory authorizations including the construction and operation of the planned power transmission line and effluent discharge pipeline. Surface rights for portions of the adjacent properties will need to be acquired to support the construction and operation of the powerline and effluent discharge pipeline.

Regarding the management of the tailings, waste rock storage facilities, and underground workings the assumption that waste rock and tailings are non-acid generating will need further study for the purpose of developing geochemical source terms that can be used to better predict effluent quality, water treatment requirements, and potential downstream ecological and human health risks. The results of the future geochemistry study may result in modifications to tailings and waste rock infrastructure and management resulting in increased capital and operating costs.

Currently it is reported that the local community, in general, supports the development of the Era Dorada Project as an underground mine, however, there is a potential risk of socio-political opposition that could negatively impact the permit approval and construction schedule.

22.9 Capital Cost Estimate

The capital cost estimate was developed in Q4 2025 using existing project designs which were updated with recent 2025 mine plans supplied by Snowden Optiro. The costing has been built up by:

- Budgetary quotes and data from projects from internal databases for mechanical equipment
- Preliminary layouts for architectural, civil and structural disciplines, priced from vendor quotes and historical data from relevant reference projects
- Piping and electrical factored from mechanical equipment installation

- Contributions from Snowden Optiro for geotechnical and mining respectively have been incorporated into this estimate

22.10 Operating Cost Estimate

The operating cost estimate was developed in Q4 2025 using data from vendor quotations, projects, studies and previous operations from internal databases. The operating cost estimate is approximately ±15% accurate. The estimate covers the mining, processing, maintenance, power and general and administrative activities. Section 18 includes a summary of the operating expenses.

The average mining cost is $71/t. The average process plant operating cost is $87.92/t processed, and the annual G&A cost is $6.2 million.

22.11 Economic Analysis

An engineering economic model was developed to estimate annual cash flows and sensitivities for the Project. Pre-tax estimates of Project values were prepared for comparative purposes, while after-tax estimates were developed and are likely to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

Capital and operating cost estimates were developed specifically for the Project and are summarized in Sections 17 and 18 of this report, presented in constant 2025 dollars. The economic analysis was performed on a constant-dollar basis with no inflation.

The economic analysis was performed by assuming a 5% discount rate. Cash flows have been discounted to the start of construction, assuming that the project execution will be made and major project financing will be carried out at this time.

The pre-tax NPV discounted at 5% is $1,535 million; the IRR is 38.5%, and payback period is 2.7 years. On a post-tax basis, the NPV discounted at 5% is $1,335 million, the IRR is 35.6%, and the payback period is 2.8 years. Cumulative post-tax unlevered free cash flow totals $2, 701 million.

The sensitivity analysis revealed that the Project's NPV and IRR are most sensitive to fluctuations in gold prices, with lower sensitivity to changes in initial capital costs, operating costs, and sustaining capital costs.

22.12 Risks and Opportunities

22.12.1 Risks

22.12.1.1 Geology and Resource Estimation

The most significant project risks are summarized below:

- Commodity Prices (Gold, Silver) – Lower commodity prices will change the size and grade of the potential targets. Conversely, increased commodity prices will improve economics and resources.
- Although there is a relatively high degree of confidence related to geological continuity and grade variability, vein models and grade distributions may adjust with further data and structural interpretations.

22.12.1.2 Mineral Processing and Metallurgical Testing

No current metallurgical testing has been completed to support this study. Samples previously tested may not align with initial production areas. Plant throughput and recoveries may not meet design values during this key period.

Current recovery modeling is an average for both gold and silver and may not reflect changes in head grades or variations that may occur from different production areas of the mine.

22.12.1.3 Infrastructure

22.12.1.3.1 Geotechnical Mine Waste Facilities

The main risks associated with the geotechnical facilities scope for the project include:

- Possible environmental issues with the permitting process of piles WRD 2 (Phase 2) and DSTF 2.
- Possible unavailability of areas outside the current property for tailings disposal.
- There is no designated licensed area for drying tailings prior to their disposal in the DSTFs, inside the current property.
- Possible (and probable) issues related to the disposal and compaction of tailings during wet seasons. The main consequence associated with this risk is the need to start a new licensing process for DSTF 2 as soon as mining operation begins, basically. This should be considered as a high probability risk, once the adequate compaction of tailings will probably be severely impacted during rainy seasons.
- Possible worse laboratory results related to the strength or the mechanical behaviour of the foundation materials and the materials to be disposed of, especially tailings that could imply modifications to the volumes proposed for the DSTFs.

22.12.1.4 Environmental, Permitting, Social and Community Considerations

The main risks associated with the permitting schedule for the project include:

- Although the project is licensed, amendments and new permits (for pipeline, transmission line) are needed, permitting delays may impact the project timeline.
- Local community support currently exists for the project, but potential opposition could affect the overall schedule.

- Further geochemistry studies and additional data and effluent modelling results may result in the requirement for modifications to effluent treatment as well as additional engineering design for project infrastructure and processes.
- Commence negotiations on a timely and expedited basis with local landowners to secure the required surface land rights for the purpose of tailings storage facility (DSTF) expansion.

Currently it is reported that the local community, in general, supports the development of the Era Dorada Project as an underground mine, however, there is a potential risk of socio-political opposition that could negatively impact the permit approval and construction schedule.

22.12.2 Opportunities

22.12.2.1 Geology and Resource Estimation

Continued elevated commodity prices for gold and silver increase the proportion of economically viable resources and will expand the size and grade of the potential targets.

22.12.2.2 Metallurgical Testwork

Gravity concentration test results are at the lower end of the range when gravity concentration is included in the flowsheet. Additional testing can confirm if its inclusion is merited. Removal from the flowsheet will result in capital and operating costs. Leach retention times should be reviewed to determine if they can be reduced.

22.12.2.3 Infrastructure

22.12.2.3.1 Geotechnical Mine Waste Facilities

Regarding the Geotechnical scope of the mine waste facilities, the following opportunities should be evaluated:

- Due to great heterogeneity of local conditions and relatively scarce information about the subgrade material, Ausenco's design had to be based on cautiously pessimistic design properties of the subgrade material. Consequently, the design produced a relatively low sidehill dry stack facility capable of storing 50,000 m^3/ha. In principle.
- The possibility of constructing DSTF 2 as a temporary stockpile for tailings disposal during wet seasons.
- The possibility of covering parts of the DSTFs operational areas, to enable tailings disposal during wet seasons.
- The possibility of evaluating alternative options for tailings disposal, such as co-disposal or commingling, for example.

22.12.2.4 Environmental, Permitting, Social and Community Considerations

The opportunities listed below should be considered as the project continues through feasibility design:

- Immediate action and ongoing engagement with regulators and stakeholders regarding future permitting requirements.
- To mitigate the potential of socio-political risk, it will be necessary to strengthen rightsholder and stakeholder relationships by means of focussed engagement and impact benefit discussions.
- Commence negotiations on a timely and expedited basis with local landowners to secure the required surface land rights for the purpose of transmission line and effluent pipeline construction and operation.
- The study of socio-economic impacts (both positive and negative) and traditional uses of the land within and adjacent to the project study area will aid in the advancement of discussions with rightsholders and stakeholders.
- Regarding hydrological, hydrogeological, and geochemical studies, there are opportunities to work closely and collaborate with the geotechnical, water resources, and processing engineering teams and hence, reduce effort and costs.

23 RECOMMENDATIONS

23.1 Introduction

All of the required test work is completed for this FS and the additional work necessary to start EPCM stage is summarized below:

Table 23-1: Recommended Work Program - Summary

Program Component	Estimated Total Cost (MUSD)
Geology and resource estimates	7.75
Mining methods	0.26
Metallurgical Testing	0.60
Hydrogeology	0.15
Infrastructure facilities	0.30
Water management	0.10
Environmental studies	0.40
Total	**9.56**

This FS presents a project that is ready for submission for financial and other support necessary to advance through EPCM.

23.2 Geology and Resource Estimates

Additional drilling will increase resources and improve understanding and modeling of lithological units. Definition drilling ahead of blasting will improve the definition of grade boundaries between high-grade veins and low-grade disseminated mineralized material and help minimize unplanned dilution.

A review of mineral resource classification and grade distributions is prudent to ensure accuracy and certainty.

For geotechnical purposes, it is available to characterize and model the geotechnical parameters as domains and placement into the estimation block model.

A comprehensive brownfields exploration program along trend of the main deposit is recommended to explore additional gold and silver resources that could potentially extend the project's life.

23.3 Mineral Processing and Metallurgical Testing

Conduct a variability testing program based on samples reflecting the latest mine plan. The samples should focus on the first five years of production. The results will be used to derive a more comprehensive recovery model, determine if there are any spatial or lithological recovery dependencies.

The testing program should also include a Year 1-3 production composite to validate thickener sizes, filter sizes and confirm leach and cyanide detox parameters. Historical gravity concentration tests were not done with the extended gravity recoverable gold (E-GRG) protocol. The test results have high mass recoveries and low to mid range gold recoveries. Using results from an E-GRG test will allow for modeling of plant scale performance and confirm if gravity concentration is still merited in the flowsheet.

Materials handling testing is also recommended to ensure that the crushed ore stockpile geometry promotes maximum flow and minimizes dead volume. The testing will also recommend chute designs and lining materials to prevent blockage during wet season.

The estimated cost for the recommended metallurgical testing program is US$0.6 million, excluding sample acquisition costs.

Table 23-2: Phase 1 Recommended Work Program

Program Component	Estimated Total Cost (MUSD)
Drilling	5.00
Geotechnical Work	2.20
Environmental Studies	0.75
Total	**7.75**

23.4 Mineral Reserve

Vein geometry and continuity interpretation are challenging for the Era Dorada deposit. Detailed geometry and quality characterization via Infill drilling and channel sampling will enable Mineral Resources conversion from Inferred and Indicated to Proven Reserve to secure the successful operation of the mine.

Mining in the early stages will confirm assumptions regarding rock mass quality to secure the prevalence of Long hole mining over less safe, less productive and more expensive Cut-and fill mining under acceptable safety, stability and dilution parameters and must be monitored cautiously

Near-surface Mineral Resources not included in the underground mining plan can be converted to Mineral Reserves if a future open pit evaluation to demonstrate technical and economic viability to expand the Reserve base and be brought to an integrated mine plan and has to be pursued.

23.5 Mining Methods

The Feasibility Study was developed under Class 3 estimate standards for the mine components, once the final investment decision is made, detailed engineering design will be required for the mine infrastructure, namely:

- The main ventilation and cooling facilties at surface;
- The underground pumping system;
- Electrical facilities underground;
- The supervision, communication and control system for the undreground mine.

The costs for the detailed engineering studies for the mine are estimated as of US$200,000.

Likewise, operational readiness aspects must be detailed, comprising:

- Personnel recruiting, mobilization and training;
- Short-term operational mine planning;
- Detailed procurement plan;
- Detailed health and safety planning meeting Aura standards;
- Detailed risk assessment;
- Planing and controls.

The latter can be addressed by the development of a detailed Project Execution Plan for the mine components, and its costs are estimated as of US$60,000.

23.5.1 Mine Geotechnical

Geotechnical studies indicate that the mining–filling sequences play a dominant role in confinement, significantly reducing the potential for overbreak, enabling the control of stability, securing safe operations and acceptable dilution. Adherence to the prescribed mining and filling sequence is essential for stability and must be implemented from the early stages of mining.

The following are recommendations for further rockmass characterization and modelling:

- Additional laboratory testing is required to secure reliability of geotechnical parameters for MBT, MLS, and MVO, including UCS, E, v, and triaxial tests.
- Joint set characterization should be integrated into support design to refine assessments of planar and wedge failures.

- Future stability analyses must incorporate 3D numerical models for selected areas with detailed mining sequencing to capture stress redistribution and filling effects with accuracy.

23.6 Hydrogeology

Based on the current understanding of the hydrogeologic system and model results, the following actions are recommended to support mine planning and water management:

- Conduct Sensitivity and Uncertainty Analyses: Apply formal sensitivity and uncertainty methods to the groundwater numerical model to identify the key parameters and most sensitive zones controlling model behavior, and to quantify the plausible range of groundwater inflows. This will improve confidence in predictions used for mine planning and water management.
- Refine Study Area Characterization: Based on the sensitivity and uncertainty outcomes, define priority target areas for additional field investigation. Where feasible, existing wells and piezometers should be leveraged for enhanced monitoring and testing, complemented by focused in situ investigations (for example, geophysical surveys, packer testing, and long term well pumping to aquifer test). These activities should target major fault zones, high permeability or preferential flow pathways, and areas with geothermal upflow characteristics.
- Develop an updated structural geological model and revise the hydrogeologic model accordingly: Given that fault-controlled upward flows represent one of the largest sources of uncertainty, and potentially one of the greatest sensitivities, in the hydrothermal system, it is recommended to develop a dedicated structural geological model and integrate it into an updated hydrogeologic framework. This refinement will improve the representation of fault geometry, connectivity and transmissivity, particularly in zones where ascending geothermal flux is believed to occur, thereby reducing uncertainty and improving predictive reliability.zIntegrate thermal–hydraulic analysis: Incorporate heat-transport modeling to evaluate coupled thermal–hydraulic processes, including changes in water viscosity, heat exchange between groundwater and mine infrastructure, and temperature-dependent pump performance under geothermal conditions.
- Optimize dewatering system design and operation: Use additional scenario testing to assess alternative excavation sequences, dewatering well activation schedules and ramp-up strategies. Refine well spacing, consider deepening selected wells (particularly in the southern sector) and stage the activation of new wells to improve hydraulic control and minimize residual inflows as mining progresses.
- Increase and diversify disposal capacity: Expand existing discharge permits and evaluate potential additional surface discharge locations and operational reuse options to provide sufficient capacity for projected flows beyond 2031. Deep reinjection wells may be considered as a supplemental alternative; however, reinjection effectively acts as artificial recharge and introduces additional complexity. Planning must explicitly address (i) the risk of degrading groundwater quality in the receiving aquifer, (ii) the possibility that reinjected water may re-enter the dewatering capture zone and increase long-term pumping requirements, and (iii) potential pressure buildup in the system that could require reassessment of pump sizing and operational setpoints. Any reinjection scheme should therefore be supported by dedicated hydrogeologic investigation, predictive modeling and regulatory review.
- Manage environmental and social water impacts: Recognize that sustained aquifer drawdown and streamflow depletion, particularly in the Río Ostua and smaller tributaries such as the Río Tacunshapa, may affect surface-

water availability for downstream users and ecosystems. The surface-water and groundwater monitoring network should be maintained and, where necessary, expanded to track changes in water levels and flows at locations relevant to local communities, ecology and water-use points. Monitoring data should be routinely compared to pre-mining baseline conditions to identify material impacts on water supply, aquatic habitat and riparian vegetation. Where significant reductions in flow are confirmed, the operator should evaluate and implement appropriate mitigation or compensation measures, which may include augmentation or replacement of affected flows, provision of alternative water supplies to nearby communities, and adaptive adjustment of dewatering rates or infrastructure, in coordination with regulators and stakeholders.

- Validate high-temperature equipment performance: Carry out field testing of ESP systems, backpressure control equipment and well/liner configurations under representative thermal and hydraulic conditions to confirm reliable operation and to reduce the risk of steam flashing or thermal–mechanical failures. In addition, pilot wells should be constructed to intercept the principal modeled fault zones and evaluate the technical feasibility of pumping under expected geothermal conditions.
- Develop and maintain a contingency plan: Prepare a comprehensive contingency plan defining operational responses for excess inflows, temporary treatment plant outages, reinjection well underperformance and failures of high-temperature components. This plan should be supported by ongoing groundwater monitoring and periodic updates of the numerical model, ensuring that dewatering design, mining methods and water-management strategies remain aligned with observed field conditions, environmental commitments and regulatory requirements throughout the LOM.
- Assess underground-based dewatering: Evaluate the feasibility of initiating staged dewatering from advancing underground workings to provide a flexible, progressive approach to hydraulic control. The assessment should address expected inflows, high-temperature conditions, safety requirements and supporting infrastructure to determine whether this strategy can effectively complement or partially replace surface wells.
- Mine Design: The key risks for this project include the complexity and uncertainty associated with vein positions. Incorrect interpretation or deviation in the estimation of vein geometry may lead to high dilution and/or an inability to mine adjacent stopes, particularly on sublevels where multiple sub-parallel veins occur. Ongoing refinement of geological models through infill drilling and operational reconciliation is recommended to support dilution control, stope continuity, and the protection of recovered grade and fair value, particularly in areas with closely spaced veins.

Estimated costs for in situ investigations and testing (for example, geophysical surveys, packer testing, long term well pumping to aquifer test) can only be provided after the numerical groundwater flow model has been updated and the sensitivity and uncertainty analyses have been completed. The estimated cost for the model update plus sensitivity and uncertainty analyses is approximately US$150,000.

23.7 Infrastructure Facilities

The following activities are recommended to take place prior to the next stage of DSTFs and WRDs designs, with an estimated cost of US$300,000.00:

- Perform the supplementary geotechnical investigations' campaign already proposed by Ausenco in 2025, for foundation characterization, including: undisturbed sampling of soil units with Shelby tube samplers as well as block samples from test pits or Drill holes for laboratory testing. The following tests are recommended:
    - Undisturbed sampling of soil units with Shelby tube samplers as well as block samples from test pits or drill holes for laboratory testing. The following tests are recommended:
        - Triaxial static (strength testing, deformability and undrained shear behavior)
        - Triaxial cyclic shear (cyclic testing to evaluate liquefaction potential)
        - Permeability testing
        - Consolidation tests.
    - Perform geophysical surveys (for the determination of shear and compressional wave velocities).
- As soon as the beneficiation plant starts its operation, laboratory tests must be conducted on remoulded tailings from at least three samples with different void ratios, to characterize the material regarding the critical state soil mechanics. The following tests are recommended:
    - Permeability tests
    - Static and cyclic triaxial tests
    - PN triaxial tests (Neutral Pressure)
    - Consolidation tests
    - Bender Element testing at an array of densities

It is recommended that the triaxial tests be performed using confining pressures of 100, 200, 400 and 800 kPa.

Additionally, complete tailings characterization tests (including geochemical) are mandatory through LOM to assess possible variations of materials' characteristics that could imply different mechanical behaviour or possible environmental hazards (e.g., sulphide content).

- Perform laboratory tests on waste rock samples for embankment materials. The following tests are recommended:
    - Unconfined compressive strength.
    - Resistance to degradation of large-size coarse aggregate by abrasion.
    - Wetting and drying durability tests.

- In-situ density.

- Conduct a laboratory testing program for potential construction materials such as rockfill, gravel and sand. Based on this laboratory testing program, specifications will be developed for appropriate selection of these materials. After completing the field investigation and laboratory testing:
  - Perform stress–strain analyses to evaluate the expected deformations and displacements associated with the construction of the DSTFs and WRDs, supporting future definition of monitoring criteria, as well as providing input for selecting the geosynthetics required for the basal impermeabilization system of these structures.
  - Perform seepage analyses considering rainfall infiltration into the DSTFs to assess the need for additional internal drainage measures, such as intermediate blanket drains.
  - Evaluate the need for an underdrainage system for natural springs or pore pressures under the bottom impermeabilization system
  - Re-evaluate tailings disposal methodology, considering that compaction during rainy seasons may be impracticable, as well as considering construction and tailings production rates, access roads and other relevant aspects.

23.8 Water Management

The following recommendations are provided for the Water Management and Water Balance component with the purpose of strengthening the overall project design and ensuring that all required actions are implemented to maintain compliance with current water-use permits and associated discharge authorizations. These recommendations aim to enhance operational efficiency, improve the robustness of hydrological evaluations, and support long-term regulatory alignment of the project's water management strategy, with the recommended activities estimated to cost approximately US$ 100,000.00.

- **New hydrological-hydraulic study:** Conduct a high-resolution topographic survey (metric scale) to replace the low-precision public data used in 2018, ensuring accurate identification of runoff and water accumulation areas and enabling proper sizing of drainage and containment structures.
- **Floodplain reassessment:** Consider 100-year return period (TR) and PMF events using updated topographic data and new interventions from the current master plan, allowing identification of changes in flood-prone areas and ensuring project adequacy under extreme scenarios.
- **Annual GoldSim model update:** Incorporate actual operational data, recent meteorological series, and new water demand projections, providing reliable monitoring and support for water management decisions.
- **Licensing for pumping and discharge:** Immediately initiate the process to obtain new water use permits, as the current limits (1,500 gpm for the El Tempisque River and 3,750 gpm for the Ostúa River) will be exceeded from 2029 onward, ensuring legal compliance and continuous operation of the project.

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

The following recommendations are made regarding the design and implementation of environmental and socioeconomic studies as well as conducting stakeholder engagement and rightsholders negotiations. Qualified professionals should be retained to implement these recommendations based on best practice as defined by local regulatory requirements and international best practices.

Regarding environmental studies and associated modeling:

- Complete a review of surface and groundwater monitoring data (quantitative and qualitative) to assess the adequacy of the current data for the refinement of the mine water balance model and associated predictive models for effluent quality and quantity. This would include a review of potential impacts to local groundwater resources due to dewatering and advancement of underground workings and potential impacts to local ecological resources and current use of surface and groundwater by nearby community members for the purpose of agriculture and human consumption. Special emphasis should be placed on potential aquatic impacts related to subsurface geothermal groundwater impacts to surface water and their mitigation.
- Based on the above results, additional surface water and groundwater monitoring should be considered that would support the further development of the conceptual groundwater model and development of a future three-dimensional numerical groundwater model that will support future design phases and permitting including potential impacts from in-mine disposal of waste and surface disposal of tailings. The model should provide emphasis on seasonal recharge of the freshwater aquifers within and near the Project area and the potential drawdown from future underground development and dewatering activities.aA geochemical gap assessment of the ARD/ML risk for the Project should be implemented utilizing the existing geological model and available geochemical results from previous studies. A study should be designed and undertaken that utilizes the sampling of existing drill core samples that are representative of lithological, mineralogical, and structural variation of mine rock and surface soils. The range of analytical tests that should be considered include: elemental analysis; acid-base accounting; shake flask extraction (short term leach); NAG pH; minerology; and humidity cell testing. Development of source terms for the weathering of waste rock, ore, tailings, and underground contact water for use in refinement of existing water balance model, mine rock management practices and water treatment. fComplete a review and gap assessment of existing seasonal baseline vegetation/ecosystem data to ensure that that the presence/absence of listed and threatened species remains current for current and future mine footprint and study area. Based on the results of this study consider revising/enhancing existing biological compensation measures.oBaseline conditions for air quality and noise should be established for near field and further afield operations.aRegarding permitting, socio-economic, cultural baseline studies and stakeholder/rightsholders engagement:aImmediate action and ongoing engagement with regulators are recommended to support any new permit requirements (for pipeline and transmission lines).
- A geochemical gap assessment of the ARD/ML risk for the Project should be implemented utilizing the existing geological model and available geochemical results from previous studies. A study should be designed and undertaken that utilizes the sampling of existing drill core samples that are representative of lithological, mineralogical, and structural variation of mine rock and surface soils. The range of analytical tests that should be considered include: elemental analysis; acid-base accounting; shake fComplete a review and gap assessment of

existing seasonal baseline vegetation/ecosystem data to ensure that that the presence/absence of listed and threatened species remains current for current and future mine foTo mitigate the potential of socio-political risk, it will be necessary to strengthen rightsholder and stakeholder relationships by means of focussed engagement and impact benefit discussions.fCommence negotiations on a timely and expedited basis with local landowners to secure the required surface land rights for the purpose of transmission line and effluent pipeline construction and operation.fThe study of socio-economic impacts (both positive and negative) and traditional uses of the land within and adjacent to the project study area will aid in the advancement of discussions with rightsholders and stakeholders.fReview results and scope of completed baseline archaeological and cultural resource studies to ensure adequacy and expand the scope of those studies as necessary to address the entire proposed mine infrastructure disturbance area. Ensure that an archaeological chance find protocol is developed for use during project disturbance activities.

- To mitigate the potential of socio-political risk, it will be necessary to strengthen rightsholder and stakeholder relationships by means of focussed engagement and impact benefit discussions.
- Commence negotiations on a timely and expedited basis with local landowners to secure the required surface land rights for the purpose of transmission line and effluent pipeline construction and operation.
- The study of socio-economic impacts (both positive and negative) and traditional uses of the land within and adjacent to the project study area will aid in the advancement of discussions with rightsholders and stakeholders.
- Review results and scope of completed baseline archaeological and cultural resource studies to ensure adequacy and expand the scope of those studies as necessary to address the entire proposed mine infrastructure disturbance area. Ensure that an archaeological chance find protocol is developed for use throughout the implementation phase of the project.

The estimated cost for the recommended work is approximately US$400,000.

24 REFERENCES

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CIBC Global Mining Group (2025)

25 RELIANCE ON INFORMATION PROVIDED BY THE REGISTRAR

25.1 Introduction

The QPs have relied on information provided by Aura including expert reports, in preparing its findings and conclusions regarding the following modifying factors: macroeconomic information, marketing information, legal matters, environmental matters, accommodations the registrant commits or plans to provide to local individuals or groups in connection with its mine plans, and governmental factors.

The QPs consider it reasonable to rely on Aura for this information since they have obtained opinions from appropriate experts.

25.2 Property Agreements, Mineral Tenure, Surface Rights and Royalties

The Qualified Person relied upon the registrant for accurate and complete disclosure related to property agreements, mineral tenure details, environmental impacts, surface rights and royalties.

25.3 Environmental, Permitting, Closure, and Social and Community Impacts

The QPs have fully relied upon, and disclaim responsibility for, information supplied by Aura and experts retained by Aura for information related to environmental (including tailings and water management) permitting, permitting, closure planning and related cost estimation, and social and community impacts as follows:

Corporación Ambiental, Environmental Impact Assessment Study – Cerro Blanco Mining Project, 2007.

This information is used in Section 17 of the Report. The information is also used in support of the Mineral Resource estimate in Section 11, the Mineral Reserve estimate in Section 12, capital and operating costs in Section 18 and the economic analysis in Section 19.

25.4 Markets

The QPs have not independently reviewed the market studies, pricing or contract information. The QPs have fully relied upon, and disclaim responsibility for, information derived from Aura and experts retained by Aura for this information through the following documents:

- Analyst Consensus Commodity Price Forecast: December 01, 2025.

Metals price forecasting is a specialized business requiring knowledge of supply and demand, economic activity and other factors that are highly specialized and requires an extensive global database that is outside of the purview of a QP. The QPs consider it reasonable to rely upon Consensus to provide metal price forecasts and marketing information on the base metal concentrates as they sought expert input for this information.

This information is used in Section 16 of the Report. The information is also used in support of the Mineral Resource estimate in Section 11, the Mineral Reserve estimate in Section 12, and economic analysis in Section 19.