NI 43-101 Preliminary Economic Assessment Era Dorada Gold Project Jutiapa, Guatemala

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DATE AND SIGNATURE

This report, titled "NI 43-101 Preliminary Economic Assessment Era Dorada Gold Project, Jutiapa, Guatemala", having an effective date of December 31, 2024, was prepared by GE21 Consultoria Mineral Ltda. on behalf of Aura Minerals Inc, and signed.

Dated at Belo Horizonte, Brazil, on June 27, 2025.

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IMPORTANT NOTICE

GE21 Consultoria Mineral Ltda. prepared this Technical Report, in accordance with NI 43-101 guidelines, for Aura Minerals Inc. The quality of information, conclusions and estimates contained herein is based on: (i) information available at the time of preparation; (ii) data supplied by outside sources, and (iii) the assumptions, conditions, and qualifications set forth in this report.

This report is intended for use by Aura, subject to terms and conditions of its individual contracts with the report authors and relevant securities legislation. The contract permits Aura to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to the Canadian Securities Administrators' National Instrument 43-101, Standards of Disclosure for Mineral Projects, Companion Policy 43-101CP and form 43-101F1 (collectively, NI 43-101). Except for the purposes legislated under provincial securities law, any other uses of this report by any third party are at that party's sole risk. The responsibility for this disclosure remains with Aura. GE21 is under no obligation to update this technical report, except as may be agreed to between Aura and GE21 by contract from time to time. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

Before this report, this deposit was named "Cerro Blanco." Aura Minerals Inc. has renamed it to "Era Dorada". Therefore, all previous studies that were used as the basis for information in this report refer to the former name Cerro Blanco.

UNITS, SYMBOLS, AND ABBREVIATIONS

Abbreviations	
3D	Three Dimensional
AA	Atomic Absorption
ABA	Acid Base Accounting
AGP	Acid Generating Potential
ANP	Acid Neutralizing Potential
Avg	Average
BAi	Bond Abrasion Index
BWi	Bond Work Index
CA-1	Central America Highway 1
CAPEX	Capital Expenditure
CCD	Counter Current Decantation
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	Carbon in Pulp
CN_T	Total Cyanide
CN_{WAD}	Cyanide Weak Acid Dissociable
COCODEs	Consejos Comunitarios de Desarrollo
CoG	Cut-off Grade
CONAP	Consejo Nacional de Áreas Protegidas
CRF	Cemented Rock Fill
CRM	Certified Reference Material
CTA	Consultoria y Tecnologia Ambiental
CV	Coefficient of Variation
DD	Diamond Drilling
DSTF	Dry Stack Tailings Facility
EDS	Dispersive Energy Spectroscopy
EIA	Environmental Impact Assessment
EMP	Environmental Management Plan
EPA	Environmental Protection Agency
EPS	Enhanced Production Scheduler
ESR	Excavation Support Ratio
FS	Feasibility Study
G&A	General and Administrative
GE21	GE21 Consultoria Mineral Ltda.
GMS	G Mining Services Inc.
GPS	Global Positioning System
IFC	International Finance Corporation
ILR	Intensive Leach Reactor
INAB	Instituto Nacional de Bosques
IRR	Internal Rate of Return
ISRM	International Society for Rock Mechanics and Rock Engineering
LH	Long Hole
LHD	Load-Haul-Dump
LoM	Life of Mine
LP	Preliminary License
MARN	Ministerio de Ambiente y Recursos Naturales
MCF	Mechanized Cut-And-Fill
MEM	Ministerio de Energía y Minería
MRE	Mineral Reserve Estimate
MSO	Mineable Shape Optimizer
NA	Not Applicable
NAD 27	North American Datum 1927
NAG	Non-Acid-Generating
NI 43-101	National Instrument 43-101
NNP	Net Neutralization Potential
NPR	Neutralization Potential Ratio

Abbreviations	
NPV	Net Present Value
NSR	Net Smelter Revenue
OPEX	Operational Expenditure
P_{80}	Passing 80%
PAG	Potentially Acid Generating
PEA	Preliminary Economic Assessment
PFS	Pre-Feasibility Study
PLT	Point Load Test
PPE	Personal Protective Equipment
QA/QC	Quality Assurance and Quality Control
QP	Qualified Person
RC	Reverse-Circulation
RMR	Rock Mass Rating
ROM	Run of Mine
RPEEE	Reasonable Prospects for Eventual Economic Extraction
RQD	Rock Quality Designation
SBS	Social Baseline Study
SEM	Scanning Electron Microscopy
SFE	Shake Flask Extraction
SG	Specific Gravity
SMC	Social Monitoring Committee
SMP	Social Management Plan
SPT	Standard Penetration Tests
SSSAG	Single Stage Semi-Autogenous Grinding
UCS	Uniaxial Compressive Strength
US	United States
UTM	Universal Transverse Mercator
VAT	Value Added Tax
WBS	Work Breakdown Structure
WMC	Water Management Consultants
WRF	Waste Rock Facility
WTP	Water Treatment Plant

Units and Symbols	
%	percentage
“	inch
<	less than
>	greater than
°	degree
°C	degree Celsius
µm	micron
bgs	below ground
CDN$/C$/CA$	Canadian dollar
cm	centimetre
E	East
Eq.	equivalent
E-W	East - West
g	gram
g/cm^3	grams per cubic centimetre
g/m	gallon per minute
g/mm^3	grams per cubic millimetre
g/t	grams per tonne
Ga	gigaannum
h	hour
ha	hectare

Units and Symbols	
K	thousand
k$	thousands of dollars
kg	kilogram
km	kilometre
koz	one thousand troy ounces
kozt	thousand tonnes
kPa	kilopascal
kt	thousands of tonnes
kV	kilovolt
kWh	kilowatt-hour
l	litre
m	metre
M	million
m^3/h	cubic metre per hour
Ma	megaannum
masl	metres above sea level
mg	milligram
mm	millimetre
Mm^3	million cubic metres
MPa	megapascal
Mt	million tonnes
Mtpa	million tonnes per annum
MUS$	million United States dollars
MVA	megavolt-ampere
MW	megawatt
N	North
n=	sample size
NE	Northeast
nm	nanometre
NW	Northwest
Oz	ounce
ozt	ounce troy
pH	hydrogen potential
ppm	part per million
Q	Quetzal (Guatemalan currency)
t/h	tonnes per hour
t/m^3	tonnes per cubic metre
tpd	tonnes per day
USD/US$	United States dollar
V	volts
w/v	weight by volume
w/w	weight by weight
μS/cm	microsiemens per centimetre
σH	horizontal tension (sigma H)
σV	vertical voltage (sigma V)

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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 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, TSX-listed company trading under the symbol "ORA," with its head office located at 78 SW 7th St., Miami, FL 33130, USA.

Aura commissioned GE21 Consultoria Mineral Ltda. (GE21) to prepare a Technical Report for the Project. The Mita Geothermal Project is not considered in this report.

This report, "NI 43-101 Preliminary Economic Assessment Era Dorada Gold Project, Jutiapa, Guatemala", has been prepared in accordance with the disclosure requirements set out in National Instrument 43-101 and follows the Mineral Resource and Mineral Reserve classification standards and definitions established by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), as adopted by the CIM Council.

The purpose of this study is to document a Mineral Resource Estimate, mine design, and preliminary economics of the Project.

The Qualified Persons (QPs) responsible for this PEA are Mr. Porfirio Cabaleiro Rodriguez, Mr. Homero Delboni Jr., and Mr. Garth Kirkham. Neither GE21 nor the authors of this PEA have had any material interest invested in Aura or any of its related entities. Their relationship with Aura is strictly professional, consistent with that held between a client and an independent consultant. This PEA was prepared in exchange for payment based on fees that were stipulated in a commercial agreement. Payment of these fees is not dependent upon the results of this PEA.

The effective date as it relates to the PEA is December 31, 2024, with the issue date of this PEA being June 27, 2025.

1.2 Reliance and Other Experts

The authors of this Technical Report are Qualified Persons (QPs) under NI 43-101 with expertise in mineral exploration, geology, mine planning, Mineral Resource and Mineral Reserve estimation, and environmental assessment. The Project complies with Guatemalan environmental regulations and holds the necessary permits for underground mine development and process plant construction, provided ongoing compliance with permit conditions. However, design changes since 2007 will require permit amendments and updated environmental baseline studies for certain infrastructure, including power lines.

Aura Minerals prepared and GE21 reviewed key technical and economic inputs, including:

- Gold and silver price assumptions based on long-term consensus forecasts from over 20 investment banks (see Section 19).
- Mineral processing methodologies tested by Kappes, Cassiday & Associates (1999–2012), with supplementary testing by SGS Lakefield, Carson GeoMin, Pocock Industrial, and others, and final test work completed in 2018 at BaseMet in Kamloops, BC.
- Environmental baseline and permitting information prepared for submission to MARN, with inputs from relevant Guatemalan authorities and the Asunción Mita municipality (see Section 20).
- The information presented regarding the tenure, status, and work permitted by permit type is based on information published by the Ministerio de Energía y Minas (MEM).

GE21 considers the economic assumptions, including metal prices and exchange rates, to be consistent with market norms and appropriate for the current Mineral Resource Estimate, Preliminary Economic Assessment (PEA), and supporting economic analysis. The authors of this independent Technical Report have not identified any significant risks in the underlying assumptions.

1.3 Property Description and Location

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, 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 NAD 27, Zone 16N. These coordinates correspond to the central portion of the mineral concession and are used for spatial reference in this report.

1.4 Accessibility, Climate, Local Resources, Infrastructure, Physiography and Socio-Economic Context

The Project is accessible 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 Project is situated in proximity to several communities, the largest of which is Asunción Mita, with a population of approximately 18,500 people. The recently constructed La Barranca power substation is located a few kilometres 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 labour will be available for employment at Era Dorada. As such, the Project intends to hire the majority of operations staff locally. It has allowed the Owner's budget to cover the cost of training programs.

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 will provide their own 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.

1.5 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. Geology & Mineralization

The Project is a classic hot springs-related, low-sulphidation 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 Au g/t and 8.1 Ag g/t (CB20-420) and 87.2 m grading 5.9 Au g/t and 32.5 Ag g/t (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 sulphides, 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

Exploration activities carried out on the Project were conducted by companies that held ownership prior to Aura’s acquisition. As such, this work is considered historical in nature for the purposes of this Technical Report. There are no current exploration activities to report on the property.

1.7 Drilling

The Era Dorada property was identified by Mar-West through 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, and on October 29, 2024, Aura acquired Bluestone Resources.

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 the acquisition from Goldcorp. Table 1-1 summarizes the historical drilling on the property.

Table 1-1: Drilling summary

Year	Company	Holes Drilled	Metres
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.

1.8 Sample Preparation & Data Verification

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 Mineral 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 modelling 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.

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

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 Mineral Resource estimation.

1.9 Mineral Processing and Metallurgical Testing

Metallurgical test work was conducted on samples from the Era Dorada deposit (formerly named "Cerro Blanco") between April 1999 and January 2012 by Kappes, Cassiday & Associates (KCA) and in 2018 by Base Metallurgical Laboratories Ltd. (BaseMet) in Kamloops, BC.

The test work programs included comminution testing, determination of head assays, grinding size assessments, gravity concentration, leach testing, tailings testing, and cyanide destruction.

Data obtained from both test work campaigns were used to estimate gold and silver recoveries, as well as to define the processing flowsheet configuration and process design criteria.

For the global composite sample, the average recoveries obtained were 96% Au and 85% Ag.

1.10 Mineral Resource Estimate

Era Dorada is a classic hot springs-related, low-sulphidation 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 Au g/t and 52 Ag g/t). Bonanza gold grades are associated with ginguru banding and carbonate replacement textures. Sulphide 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 metres 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 Au g/t. 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 NI 43-101.

1.10.1 Methodology

The Mineral Resource Estimate reported herein was prepared by Mr. Garth Kirkham, P. Geo. The Mineral Resources have been estimated in conformity with the generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices." There are 130,307 gold assays, totalling 153,078 m, which average 0.68 g/t, and 130,238 silver assays, totalling 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-metre 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 NI 43-101 requirements.

Table 1-2: Resource estimate using 2.25 Au g/t Cut-off

Resource Category	Tonnes (kt)	Au Grade (g/t)	Ag Grade (g/t)	Contained Gold (koz)	Contained Silver (koz)
Measured					
Indicated	6,349	9.31	31.54	1,901	6,439
Measured & Indicated	6,349	9.31	31.54	1,901	6,439
Inferred	605	6.02	19.68	117	383

Notes:
The Mineral Resource statement is subject to the following:
1. Mineral Resources are reported in accordance with NI 43-101.
2. Mineral Resource Estimates have been prepared by Garth Kirkham, P.Geo., a Qualified Person as defined by NI 43-101.
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 Au g/t. Cut-off grades are based on 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 NI 43-101. 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 excluding non-mineable material prior to reporting.
7. Metallurgical recoveries are reported as the average over the LoM 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. The effective date of the Mineral Resource Estimate is December 31, 2024.
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 the North Ramp entrance. Table 1-3 shows the volume and tonnage based on an unconsolidated specific gravity of 2.0 g/cm^3, along with gold and silver grades and metal content. These Resources are classified as Measured.

Table 1-3: 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.

1.11 Mineral Reserve Estimate

Mineral Resources are not Mineral Reserves and have no demonstrated economic viability. This PEA does not support an estimate of Mineral Reserves since a pre-feasibility or Feasibility Study is required for reporting Mineral Reserve Estimates. This report is based on potentially mineable material (mineable tonnes, not estimated Reserves).

Mineable tonnages were derived from the Resource model described in the previous section. Measured, Indicated, and Inferred Resources were used to establish mineable tonnes.

Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that all or any part of the Mineral Resources or mineable tonnes would be converted into Mineral Reserves.

1.12 Mining Methods

High-grade mineralization at the Era Dorada deposit is hosted within laterally stacked, sub-parallel narrow veins that generally strike northeast, with an average azimuth ranging from 25° to 50°. Vein dip varies, including both tabular and near-vertical geometries.

The average dip of high-grade structures is approximately 50° to 55°. The average vein thickness ranges from 2 to 10 m, with an average spacing of 8 m between parallel structures. The potentially mineable Resource ranges from 50 m at the lowest levels to 300 m near the surface. The mineralized system comprises more than 50 modelled veins with variable geometry along both strike and dip.

Era Dorada is proposed to be mined as an underground operation using a combination of long hole stoping (LH) and mechanized cut-and-fill (MCF) mining methods, with cemented paste and rock backfill. A target production rate of 1,500 tpd is envisioned over a mine life of 17 years, which will extract 8.9 Mt of ore. LH stoping will account for about 77% of total production, and the remaining 23% will come from MCF and development. The Era Dorada deposit will be accessed from the surface via a series of ramps, and all ore and waste rock will be trucked out of the mine. In addition to the four existing ventilation raises, two new raises will be required to circulate the required amount of air through the underground workings.

Dewatering, ventilation, and cooling are crucial aspects of mine design at Era Dorada. A series of existing and new surface dewatering wells will lower the water levels in the immediate mine area. Any remaining water underground will be captured and pumped to the surface through collection at underground sumps. For ventilation, the quantity of air required has been designed to dilute diesel particulate matter, reduce the air temperature from exposed rock, and maintain worker comfort. Mine air refrigeration will be used to maintain air temperatures in working areas below 28 °C wet bulb.

Indicated and Inferred Mineral Resources were included in the mine design and schedule optimization process. The Indicated material accounts for 78% of LoM, while Inferred material accounts for 22%.

The mine production schedule is shown in Table 16-13.

1.13 Recovery Methods

The processing plant will process 1,500 tpd, consisting of the following unit operations:

- Crushing circuit.
- Grinding circuit to a nominal P_{80} of 0.053 mm.
- Gravity concentration and intensive leaching (ILR).
- Pre-leach thickening to 50% solids (w/w).
- 2-hour pre-oxidation, 36-hour leaching, and 6-hour Carbon-in-Pulp (CIP).
- Carbon acid wash, elution, and regeneration.
- Electrowinning and refining.
- Cyanide destruction.
- Tailings thickening, filtration, and disposal in the DSTF or underground as paste backfill.

The leach circuit will have a residence time of 36 hours. The sodium cyanide (NaCN) consumption is predicted to be in the range of 0.3 to 0.5 kg/t to maintain a cyanide concentration of 500 ppm. Cyanide will be destroyed using the SO_2/Air process (Detox circuit). Tailings resulting from the Detox circuit will be transferred to a thickener, whose underflow will be pumped to the filtration circuit, where a horizontal vacuum filter will reduce the cake moisture to 18.6% (dry basis).

Most of the water consumed in the processing plant is designed to be derived from recirculation within the industrial installation.

The main reagents to be used in the Era Dorada industrial plant are sodium cyanide, hydrated lime, lead nitrate sodium hydroxide, sodium metabisulphite, hydrochloric acid, copper sulphate pentahydrate, and flocculant.

1.14 Project Infrastructure

The Project plans the installation of the following elements to support the mine and process facilities:

- 5.5 km new site access road, including an 80 m long bridge;
- 8.2 km new 69 kV power line;
- on-site substation (69 kV to 13.8 kV);
- water management facilities, including a flood protection levee, diversion channel, ditches, and collection ponds;
- process plant site pad and associated buildings;
- primary crusher pad;
- emergency power genset;
- communications system upgrade;
- rehabilitation of five existing dewatering wells;
- construction of eight new dewatering wells;
- construction of nine new reinjection wells;
- reagent warehouse and storage facilities;
- truck shop (existing facility to be used in pre-production, new shop to be constructed in Operating Year 1);
- fresh / fire water tank;
- process water tank;
- upgrade fuel storage facility;
- new helipad;
- upgrade septic system for the upgrade of sewage management;
- solid waste disposal facility;
- dry stack tailings facility (DSTF);
- temporary waste rock storage facility;
- 1.0 km North and South portal connector haul road;
- on-site access roads for plant and facilities;
- additional security facilities, including a site access control station.

The proposed site layout has been designed to support mining and plant operations while minimizing environmental and community impacts, reducing construction costs, ensuring secure access, and optimizing operational efficiency.

Existing Facilities: Administrative, technical, geology, environmental, and assay lab facilities are established and will remain operational. Logistics, security, equipment servicing, and sample processing infrastructure are also in place.

Access and Security: Current access via a gravel road with a 27-t bridge is insufficient. A new 5.5 km access road connecting directly to the Pan-American Highway with an 80 m bridge over El Achotal River will be built. The site entrance will feature a secure gate and access control. Fencing and security personnel will oversee site safety, with heightened measures in place during construction.

Power Supply: Power will be supplied via an 8.2 km, 69 kV transmission line from Energuate Barranca Honda Substation, stepping down to 13.8 kV and lower voltages as needed. Emergency power will utilize relocated diesel generators to support critical loads (~7 MVA). Temporary generators will serve construction needs.

Process Plant: Approximately 150 m x 70 m, including grinding, leaching, Merrill-Crowe, filtration, detoxification, reagent prep, dry stack tailings filtration, and electrical rooms. Enclosed milling and refinery facilities will be built to code; MCCs and control rooms will be preassembled where possible.

Water Management: Existing and new dewatering wells (totalling 24 wells) will manage groundwater inflows, with peak surface dewatering at ~795 m^3/h, supplemented by underground sumps. A new cooling pond and expanded water treatment plant (capacity 341 m^3/h) will treat mine water for process use and personnel facilities—potable water supplied by local bottled water vendors. Water reuse is emphasized through the use of reinjection wells and a zero-discharge strategy.

Maintenance Facilities: Mine truck shop near administration and North Portal with three service bays, welding/general shop, oil change bay, outdoor wash bay, parts warehouse, and offices—steel structure with 10-t crane.

Water Storage: New dual-purpose fresh/fire water tank (640,000 l) with minimum 470,000 l fire reserve; adjacent 170,000 l process water tank.

Fuel Storage: Existing diesel tanks (2 × 37,500 l) expanded by one additional tank and containment area. Capacity supports 14 days of mobile equipment or 2 days of critical loads.

Access Roads: North and South portal roads widened to 22 m, with the north road extended to the dry stack tailings facility. Temporary construction roads developed as needed.

Communications: Existing tower supplemented with fibre optic cable alongside 69 kV power line. Dedicated underground mine communication system and handheld radios for mobile equipment and security.

Emergency Services: First aid clinic under construction with training room and medical storage. Ambulance and fire truck stationed near the process plant. Site-wide safety detectors and fire extinguishers installed.

Explosives Storage: Existing explosives magazine continues in use with a capacity for 75,000 kg explosives and 10,000 detonators; monthly deliveries planned; compliant safety berms.

Sewage Treatment: Septic systems with bio-reactors for sanitary waste; existing and new units for expanded facilities; treated wastewater separated before discharge.

Surface Water Management: Separation of contact and non-contact water; contact water reused or treated; non-contact runoff directed to sediment control ponds and discharge points. Stormwater managed by lined channels, ponds, culverts, and erosion controls designed for 100-year storm events.

Dry Stack Tailings Facility (DSTF): Designed for 3 Mt (1.9 million m^3) of filtered tailings with centerline-raised embankment. Seasonal deposition for stability and runoff control. Tailings transported by haul trucks. Initial infrastructure includes impoundments, underdrains, geotextile liners, reclaim and stormwater ponds, and water recirculation systems.

Geotechnical Investigations: Extensive test pits, boreholes, SPT, and permeability testing confirmed subsurface conditions (colluvial/alluvial soils over residual sedimentary and volcanic materials) and supported foundation design and seismic performance.

Seepage System: Foundation underdrains and vertical decant towers collect seepage and runoff, directing water to reclaim and stormwater ponds with pumps returning water to the process plant, supporting zero-discharge management.

Waste Rock Facility (WRF): Located near the south portal, designed for 120,000 m^3 temporary storage, primarily for underground backfill. Preliminary geochemical tests indicate low acid generation risk; further testing and detailed geotechnical studies planned.

1.15 Market Studies and Contracts

Mineral Resources were estimated using a gold price of US$ 2,000 per troy ounce. Project economics were evaluated at a gold price of US$ 2,389 per troy ounce and a silver price of US$ 28.44 per troy ounce. All price assumptions are based on the long-term consensus forecast from over 20 investment banks.

1.16 Environmental Studies, Permitting, and Social or Community Impacts

1.16.1 Introduction

Aura's review shows that the Project has all necessary permits to proceed with the development of the underground mine and construction of the process facilities, subject to the future operation to adhere to the conditions of the existing permits.

1.16.2 Environmental Management and Permitting

Environmental studies have been conducted at Era Dorada since the Project's inception. The Environmental Impact Assessment (EIA) was submitted and approved for an underground mine in 2007 by Guatemala's Ministry of Environment and Natural Resources (MARN). However, the Project design changed since 2007 and requires permit amendments. Additionally, new baseline studies are necessary for infrastructure components such as power lines. The approved EIA includes an Environmental Management Plan (EMP), a Social Management Plan (SMP), and a Conceptual Mine Closure Plan, which have been reviewed and updated following international best practices.

1.16.3 Water Management

The Project's water management infrastructure consists of a Water Treatment Plant (WTP), pipelines, settling ponds, wells, and cooling channels. Monitoring of surface and groundwater is conducted regularly, with compliance reports submitted to MARN. Naturally occurring metals such as aluminum and arsenic are found in local waters. The WTP, designed for arsenic removal, uses ferric salt co-precipitation and ensures compliance with Environmental Protection Agency (EPA) and Guatemalan standards.

- **Surface Water Management:** Runoff is classified as “contact” or “non-contact” water. Contact water undergoes treatment before being reused or discharged, while non-contact water is diverted and monitored.
- **Groundwater Management:** Dewatering of the mine is achieved through surface wells and underground sumps. Treated water is either reused or discharged into Quebrada Tempisque. Reinjection wells are used to manage groundwater.

1.16.4 Waste Rock and Tailings Management

- **Waste Rock:** Temporary storage of waste rock occurs for a maximum of one year before being returned to underground as backfill. Based on the limited exposure time and historical geochemistry testing, it was assumed that any potential acid generation would not have sufficient time to occur. Comprehensive Environmental monitoring will indicate and anticipate any potential acid generation to prevent impacts.
- **Tailings:** Tailings are dewatered through filtration before placement in the Dry Stack Tailings Facility (DSTF). The facility, designed to prevent environmental contamination, collects runoff water for treatment. Based on tests of geochemical characterization and consistent with the approved EIA, tailings are considered to be non-acid generating (NAG).

1.16.5 Flora and Fauna

Baseline studies have been conducted in the region since 2007, documenting its biodiversity. Ongoing monitoring indicates minimal impact from the Project. The local ecosystem comprises subtropical and tropical dry forests, which support a diverse array of plant species. Wildlife monitoring shows stable populations of birds, reptiles, and aquatic fauna. Preventive conservation measures, including habitat relocation for threatened species, have been implemented.

1.16.6 Cultural and Archeological Resources

A dedicated on-site team monitors the potential impact on cultural and archaeological artifacts. Pre-construction inspections and external expert consultations ensure compliance with relevant regulations. To date, no significant historical artifacts have been identified within the Project's direct area of influence.

1.16.7 Environmental Monitoring

The Project maintains 26 monitoring stations for water quality, six for air quality, and additional noise monitoring locations. Monthly reports are submitted to regulatory authorities. While air quality and noise monitoring were not included in the original 2007 baseline study, comparisons to EPA standards ensure compliance.

1.16.8 Environmental Management Plan

The Environmental Management Plan (EMP) has been updated to incorporate lessons learned from a decade of on-site environmental data collection. The plan aligns with regulatory requirements and international best practices. It integrates corporate health, safety, and environmental programs, including emergency response strategies.

1.16.9 Social Management

Aura prioritizes strong community relationships. The Project retains a comprehensive database of community engagement activities and sustainability initiatives. The updated Social Management Plan (SMP) incorporates International Finance Corporation (IFC) performance standards and includes mechanisms for communication, grievance handling, and community engagement. A Social Monitoring Committee (SMC) is being established to ensure transparency.

1.16.10 Mine Closure

The approved EIA includes a conceptual mine closure plan, which was further refined.

Mine closure requirements include:

- Progressive underground backfilling of waste rock and tailings.
- Decommissioning of infrastructure while maintaining environmental safeguards.
- Long-term monitoring of water quality and ecosystem restoration.
- Revegetation using native plant species.

The Dry Stacking Tailings Facility (DSTF) will be constructed continuously over the life of the mine using the downstream construction method, so concurrent reclamation will not be possible. At the end of operations, exposed portions of the decant piping will be dismantled, and the decant pipes will be plugged below the final surface.

The surface of the DSTF will be contoured so that it will shed precipitation rather than impound it. Topsoil that is stockpiled from the DSTF footprint during construction will be spread over the surface of the DSTF. Native grass seed mixture will be planted to reduce erosion.

Total closure costs are estimated at MUS$ 17.2 and do not include contingencies.

1.17 Capital and Operating Costs

LoM Project capital costs total MUS$ 417.0, consisting of the following distinct phases:

- Pre-production capital costs – includes all costs to develop the property to a 1,500 tpd production. Initial capital costs total MUS$ 263.6 and are expended over a 23-month pre-production period on engineering, construction, and commissioning activities.
- Sustaining capital costs – includes all costs related to the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations. Sustaining capital costs total MUS$ 136.2 and are expended in operating Years 1 through 24.
- Closure costs – includes all costs related to the closure, reclamation, and ongoing monitoring of the mine, post operations. Closure costs total MUS$ 17.2 and are primarily incurred in Year 14, with costs extending into Year 17 for ongoing monitoring activities.

Table 1-4: Estimated Project capital costs

WBS DESCRIPTION	Pre-Production Cost (MUS$)	Sustaining Cost/Closure (MUS$)	Project Total Cost (MUS$)
Infrastructure	8.2	8.4	16.6
Power and Electrical	16.7	-	16.7
Water Management	16.1	24.7	40.8
Surface Operations	14.7	1.7	16.4
Mining	63.4	80.2	143.6
Process Plant	49.7	16.7	66.4
Construction Indirect	38.0	4.6	42.6
General Services – Owner's Costs	21.3	-	21.3
Logistics/ Taxes/ Insurance	9.0	-	9.0
Pre-production, Start-up & Commissioning	5.0	-	5.0
Contingency	21.9	-	21.9
Closure Costs	-	17.2	17.2
Total	**263.6**	**153.5**	**417.0**

Source: Aura, 2025.

The operating cost estimate in this study includes the costs to mine and process the mineralized material to produce doré, as well as site services to maintain the site and general and administrative expenses (G&A). These items total the Project's operating costs and are summarized in Table 1-5. The target accuracy of the operating cost is -30% to +50 %. The operating cost estimate is broken into four major sections:

- Underground mining
- Processing
- Site services
- General and Administrative (G&A)

The total operating unit cost is estimated to be US$ 170/t processed. Average annual, total LoM and unit operating cost estimates are summarized in Table 1-5. The unit rates in this table include tonnes mined during the pre-production period.

Table 1-5: Estimated operating costs of the Project

Operating Costs	Avg Annual (M$)	$/t processed	LoM (M$)
Mining	38.70	100	890.01
Processing	12.38	32	284.80
Site Services	6.97	18	160.20
G&A	7.74	20	178.00
Total	**65.78**	**170**	**1,513.01**

Source: GE21, 2025.

1.18 Economic Analysis

The economic analysis for the Era Dorada Project is based on Mineral Resource Estimates, including the annual mine production schedule. As required under NI 43-101, the results of this analysis should not be interpreted as demonstrating the economic viability of the project.

The outcome of the economic analysis is subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those. The information on which this analysis is based is listed below:

- Mineral Resource Estimates
- Assumed fixed exchange rate
- Assumed known royalties
- Proposed mine production plan
- Projected mining and processing recovery rates
- Fixed installed processing plant capacity
- Assumptions on closure costs
- Assumptions on environmental, licensing, and social risks
- Changes in production costs relative to the assumptions

This analysis does not rely on:

- Unrecognized environmental risks
- Unanticipated recovery expenses
- Different geotechnical and/or hydrogeological considerations during mining
- Unexpected variations in the quantity of mineralized material, grade, metallurgical recovery efficiency, and plant recovery efficiency
- Accidents, labour disputes, and other mining industry risks
- Changes in tax rates
- Assumptions of commercial discounts that are not foreseen in the financial analysis

1.19 Interpretation and Conclusions

The Project outlines a conceptual mine plan involving the extraction of 8.9 Mt of ROM over a 17-year LoM, with a production rate of 1,500 tpd. The selected underground mining method is suitable for ensuring a stable and consistent mill feed throughout the LoM.

The Project features a comprehensive and integrated infrastructure plan that includes new access roads, power supply systems, water management facilities, a process plant, and storage facilities for tailings and waste rock. Existing support infrastructure will be leveraged, while new installations will address essential gaps in utility access, safety, and environmental control.

The total LoM capital cost is estimated at MUS$ 416.9, comprising:

- pre-production capital of MUS$ 263.6 (23-month period);
- sustaining capital of MUS$136.2 (over 17 years); and
- closure capital of MUS$17.2 (over the last 3 years of the LoM).

The cost estimate is a Class 5 estimate (±30%/±50 %) with a 12% contingency, excluding working capital, VAT, escalation, and financing. It is based on budgetary quotes, benchmarks from Latin American projects, and internal cost databases.

Operating costs were derived using first principles and local benchmarks. Processing, site services, and general and administrative (G&A) costs were carefully broken down, including labour, power, consumables, and maintenance.

The economic analysis demonstrates that the Project has robust economic potential under the developed scenario. At an average gold price of US$ 2,389/ozt, the after-tax NPV (5%) is estimated at MUS$ 485.49, an IRR of 23.8% and a payback of 3.75 years. These results indicate a financially attractive opportunity, supporting continued advancement of the Project.

1.19.1 Risks

The most significant potential risks associated with the Project include the hot water management that will be encountered during the mine dewatering effort and socio-political resistance to the development of the planned mine in Guatemala. The latter is a common risk to most mining projects. It can be mitigated, at least to some degree, with adequate planning and proactive management. The risk associated with water management is not entirely unknown, given the presence of existing dewatering wells and the continued dewatering, treatment, and discharge of underground water.

It is important to note that the current mine plan is based on a Resource model composed exclusively of Indicated and Inferred Resources, and Inferred Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. As such, there is a significant degree of uncertainty associated with the tonnages and grades used in the sequencing.

The cost of grid power is based on a market survey rather than an actual power supply agreement. A higher power cost would result in increased operating costs.

Although the local community is favourable to the development of Era Dorada as an underground mine, there is a potential risk of socio-political opposition to mine development, which could adversely impact the Project development schedule.

The ability to achieve the estimated CAPEX and OPEX costs is an important element of the Project's success. If OPEX increases, then the NSR cut-off would also increase, and all else being equal, the size of the mineable Resource would decrease, yielding fewer mineable tonnes.

1.20 Recommendations

- Additional drilling will increase Resources and improve the understanding and modelling 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 brownfield exploration program along the trend of the main deposit is recommended to explore additional gold and silver Resources that could potentially extend the project's life.
- Optimization of mine plan.
- Implementation of power generation for the cooling of the mine water.
- Mining Study detailing mining dilution for both mining methods.
- Detailed groundwater and dewatering control along LoM.
- Develop a detailed mining operating plan that respects all mining activities, accounting for project restrictions, equipment productivity, and limitations.
- Complete detailed engineering for the site infrastructure, ensuring optimization of costs, constructability, and operational integration.
- Submit final permitting documentation and ensure all facilities are compliant with local, national, and international environmental standards and regulations.
- Continue detailed geochemical testing for waste rock and tailings to confirm long-term environmental stability and support final facility design.
- Develop a phased construction plan with critical path scheduling and initiate procurement of long-lead equipment and materials.
- Maintain proactive communication with local communities and stakeholders to support social license and minimize construction-related disruptions.
- Implement a robust risk mitigation plan for infrastructure development, including contingency planning for stormwater events, equipment delays, and logistics challenges.
- Refine cost estimates to Class 5 level or higher, incorporating detailed engineering, contractor bids, and updated procurement quotes to improve accuracy and reduce contingency requirements.
- Evaluate project economics under different gold price scenarios, inflation rates, and cost escalations to test project resilience and identify key cost drivers.
- Optimize the project schedule to prioritize higher-grade zones during the initial years of operation, thereby enhancing early revenue generation.

- Evaluate alternative production scenarios involving variable feed rates throughout the LoM to improve project flexibility and economic performance.
- Conduct a Pre-Feasibility Study (PFS) or Feasibility Study (FS) for Mineral Reserve certification, considering potential variations in mining methods and/or stope geometry to identify opportunities for improved Resource recovery and economic efficiency.
- Use the current capital structure and cost estimates to support investment discussions, including potential financing, offtake agreements, or joint venture opportunities.
- Establish early procurement strategies, capital budgeting systems, and contract structures that enable cost discipline and reduce construction risk.
- Incorporate local tax regimes, VAT recoverability, depreciation schedules, and financing structures to derive a complete economic picture for stakeholders.
- Ensure that projected expenditures for G&A, environmental compliance, and social responsibility are transparently communicated and aligned with local expectations.
- Undertake a comprehensive technical and economic evaluation of the dewatering system to identify opportunities for cost reduction and efficiency improvements.
- Evaluate alternative technologies, energy-saving strategies, and hydrological modelling to minimize the operational impact of dewatering on OPEX.
- Implement a continuous monitoring strategy for gold price fluctuations, with regular updates to the economic model to assess impacts on Net Present Value (NPV), Internal Rate of Return (IRR), and payback period.
- Perform updated sensitivity analyses at key decision points to evaluate the Project’s resilience under various pricing scenarios.

2 INTRODUCTION

In January 2025, Aura Minerals Inc. (“Aura” or “the Company”) completed the acquisition of the Era Dorada Gold Project—formerly named “Cerro Blanco 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, TSX-listed company trading under the symbol “ORA,” with its head office located at 78 SW 7th St., Miami, FL 33130, USA.

Aura commissioned GE21 Consultoria Mineral Ltda. (GE21) to prepare a Technical Report for the Project. The Mita Geothermal Project is not considered in this report.

This Technical Report titled “NI 43-101 Preliminary Economic Assessment Era Dorada Gold Project, Jutiapa, Guatemala” has been prepared in accordance with the disclosure requirements of National Instrument 43-101 – Standards of Disclosure for Mineral Projects and the Mineral Resource and Mineral Reserve definitions and guidelines established by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), as adopted by the CIM Council.

The purpose of this study is to document a Mineral Resource Estimate, mine design, and preliminary economics of the Project. Bluestone Resources Inc. (Bluestone) had previously completed FS studies on the Project in 2019 and updated the results in 2022. These previous studies did not comply with NI 43-101 guidelines; however, in preparation for the PEA, any relevant information was reviewed and reused where deemed appropriate by the QPs.

This report summarizes the work carried out by several consultants, and the scope of work for each company is listed below. Combined, these make up the total Project scope.

GE21 scope of work included:

- Compiling the technical report, including information provided by other consulting companies.
- Establishing an economic framework for the PEA.
- Mine engineering, design, and scheduling.
- Designing the required site infrastructure.
- Estimating mining, process plant, and G&A OPEX and CAPEX for the Project.
- Preparing a financial model and conducting an economic evaluation, including sensitivity.
- Interpreting the results and making conclusions that lead to recommendations to improve Project value and reduce risks.

Kirkham Geosystems Ltd. (Kirkham) scope of work included:

- Mineral Resource Estimate.

HDJ’s scope of work included:

- Compiling the technical report, including information provided by other consulting companies.

- Developing a conceptual flowsheet, specifications, and selection of process equipment.
- Designing required plant facilities and other ancillary facilities.
- Establishing gold and silver recovery values for doré production on-site.
- Evaluating the status of current permits.

2.1 Qualified Persons

The Qualified Persons (QPs) responsible for this independent technical report are Mr. Porfirio Cabaleiro Rodriguez, Homero Delboni Jr., and Garth Kirkham (Table 2-1). Neither GE21 nor the authors of this independent technical report have had any material interest invested in Aura or any of its related entities.

Mr. Porfirio Cabaleiro Rodriguez, Director of GE21 Consultoria Mineral, is a mining engineer, a Fellow of the AIG (FAIG #3708), and has more than 40 years of experience in Mineral Resource and Mineral Reserve estimation. Mr. Rodriguez has sufficient experience relevant to the styles of mineralization and types of deposits under consideration to be considered a QP, as defined by NI 43-101. He is responsible for supervising all sections in this independent technical report, is individually responsible for Sections 2, 3, 4, 5, 6, 15, 16, 19, 21, 22, 23 and is co-responsible with other QPs for Sections 1, 25 and 26.

Dr. Homero Delboni Jr. is a Mining Engineer and Minerals Processing, Ph.D in Minerals Processing and Chartered Professional (Metallurgy) of the Australasian Institute of Mining and Metallurgy (AusIMM #112813) and has more than 40 years of experience in mineral processing. Mr. Delboni has sufficient and relevant experience in mineral processing industrial circuits to be considered a QP as defined by NI 43-101. He is individually responsible for Sections 13, 17, and 20 and is co-responsible with other QPs for Sections 1, 25, and 26.

Mr. Garth Kirkham, P.Geo. and Principal of Kirkham Geosystems Ltd., is a geophysicist and geologist (EGBC #30043) and has more than 30 years of experience in supplying 3D geoscience modelling, geological and geophysical consulting services to the mining, environmental and oil & gas industries. He has sufficient and relevant experience in mineral deposit geology and Resource estimation to be considered a QP as defined by NI 43-101. He is individually responsible for Sections 7, 8, 9, 10, 11, 12, 14, 24, and is co-responsible, with other QPs, for Sections 1, 25 and 26.

Table 2-1: List of QPs and related responsibilities

QP	Section Responsibility	Site Visit	Responsibility
Porfirio Cabaleiro Rodriguez, FAIG	2, 3, 4, 5, 6, 15, 16, 19, 21, 22, 23, and partially 1, 25 and 26		Author and Peer Reviewer
Dr. Homero Delboni Jr., AusIMM	13, 17 and 20 and partially 1, 25 and 26		Author and Peer Reviewer
Garth Kirkham, P. Geo.	7, 8, 9, 10, 11, 12, 14, 24, and partially 1, 25 and 26	May 8, 2017; Sep 21-22, 2017; Apr 24-28, 2018; Feb 16-22, 2020; Jan 10-15, 2021	Author and Peer Reviewer

Source: GE21, 2025.

2.2 Site Visits and Scope of Personal Inspection

Garth Kirkham, P. Geo., first visited the property on May 8, 2017, to satisfy the site visit requirements related to the 2017 Technical Report. The site visit included an inspection of the property, offices, underground vein exposures, core storage facilities, the water treatment plant, stockpiles, and a tour of major centers and surrounding villages that are 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 the interpretation and wireframe modelling, in addition to vetting and reviewing QA/QC procedures. On September 21-22, 2017, Mr. Kirkham inspected the progress of the recommended historic drill core rehabilitation program and initiated the structural studies. From April 24 to 28, 2018, the site visit focused on advancing the planning and development of sampling and drilling, as well as supporting lithological and structural modelling. From February 16 to 22, 2020, Mr. Kirkham assisted with the planning and development of advanced drilling and sampling. He provided guidance on lithology and high-grade vein modelling for Resource estimation. From January 10 to 15, 2021, Mr. Kirkham validated drill and sample data, refined high-grade models, reviewed low-grade models, and provided guidance for finalizing an open pit bulk tonnage Resource scenario.

The other QPs relied upon the observations of Mr. Kirkham, who visited the site.

2.3 Effective Date and Sources of Information

This report is based on information collected by the QPs during site visits and on additional information provided by Aura throughout the course of GE21’s analysis. Other information was obtained from the public domain. GE21 has no reason to doubt the reliability of the information provided by Aura.

Aura and its consultants provided GE21 with the information that was used to prepare this PEA, specifically during the execution of the work that is described herein. This work reflects the technical and economic conditions at the time that it was executed. The authors, whenever possible, executed independent verification of the data they received, in addition to conducting field visits to corroborate the data. This information was supplied in the form of an exploratory drilling database, certifications, maps, technical reports, and a topographical survey. The data is a combination of historical and newly generated information.

The results, images, and illustrations presented in this technical report have been generated from information provided and compiled by Aura through data organized in spreadsheets, internal and third-party technical reports, and supplemental information obtained from the Aura technical team. Exceptions will be subtitled for the source reference.

The effective date for this PEA is December 31, 2024, related to the gold price considered for the Project. The authors believe that no relevant data concerning the Mineral Resources Estimate were produced after this date.

3 RELIANCE ON OTHER EXPERTS

The authors of this report are QPs, as defined under NI 43-101, with relevant experience in mineral exploration, economic geology, data validation, mine planning, Mineral Resource and Mineral Reserve estimation, and environmental impacts.

The Project complies with Guatemalan environmental laws and holds the required permits for mine development and construction, provided it continues to meet permit conditions. However, design changes since 2007 require permit amendments and, new environmental studies and permits for infrastructure like power lines.

Aura prepared and GE21 reviewed:

1. The gold and silver price assumptions are informed by a long-term consensus forecast compiled from over 20 investment banks, effective as of the date specified in Section 19.
2. Mineral processing technologies were studied by Kappes, Cassiday & Associates (KCA) between 1999 and 2012, with additional testing by several other laboratories, including SGS Lakefield, Carson GeoMin, Pocock Industrial, Phillips Enterprises, and CyPlus. The most recent test work was completed in 2018 at Base Metallurgical Laboratories (BaseMet) in Kamloops, BC.
3. The information presented regarding new baseline studies, EIA, and permit applications to be submitted to Ministerio de Ambiente y Recursos Naturales (MARN) for approval, with input from the following Guatemalan authorities: Ministerio de Energía y Mineria (MEM), Consejo Nacional de Areas Protegidas (CONAP), Instituto Nacional de Bosques (INAB), Ministerio e Salud y Asistencia Social (Ministry of Health & Social Assistance), and the local municipality of Asunción Mita, are summarized on Section 20.

GE21 determined that the economic factors used in the determination of specific technical parameters of this Technical Report, including forecasts of gold and silver prices and the US$:R$ assumptions used, were in line with industry norms and broader Market consensus and are acceptable for use in the current Mineral Resource Estimate, current PEA, and the economic analysis presented herein. The authors of this independent Technical Report have not identified any significant risks in the underlying assumptions.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Project is located in southeast Guatemala, in the Department of Jutiapa, approximately 160 km by road from the capital, Guatemala City (Figure 4-1), 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 km^2 and lies entirely in the municipality of Asunción Mita.

Figure 4-1: Project location map

Source: Bluestone, 2021.

The location of the Mineral Resources relative to the property boundary is shown in Figure 4-2.

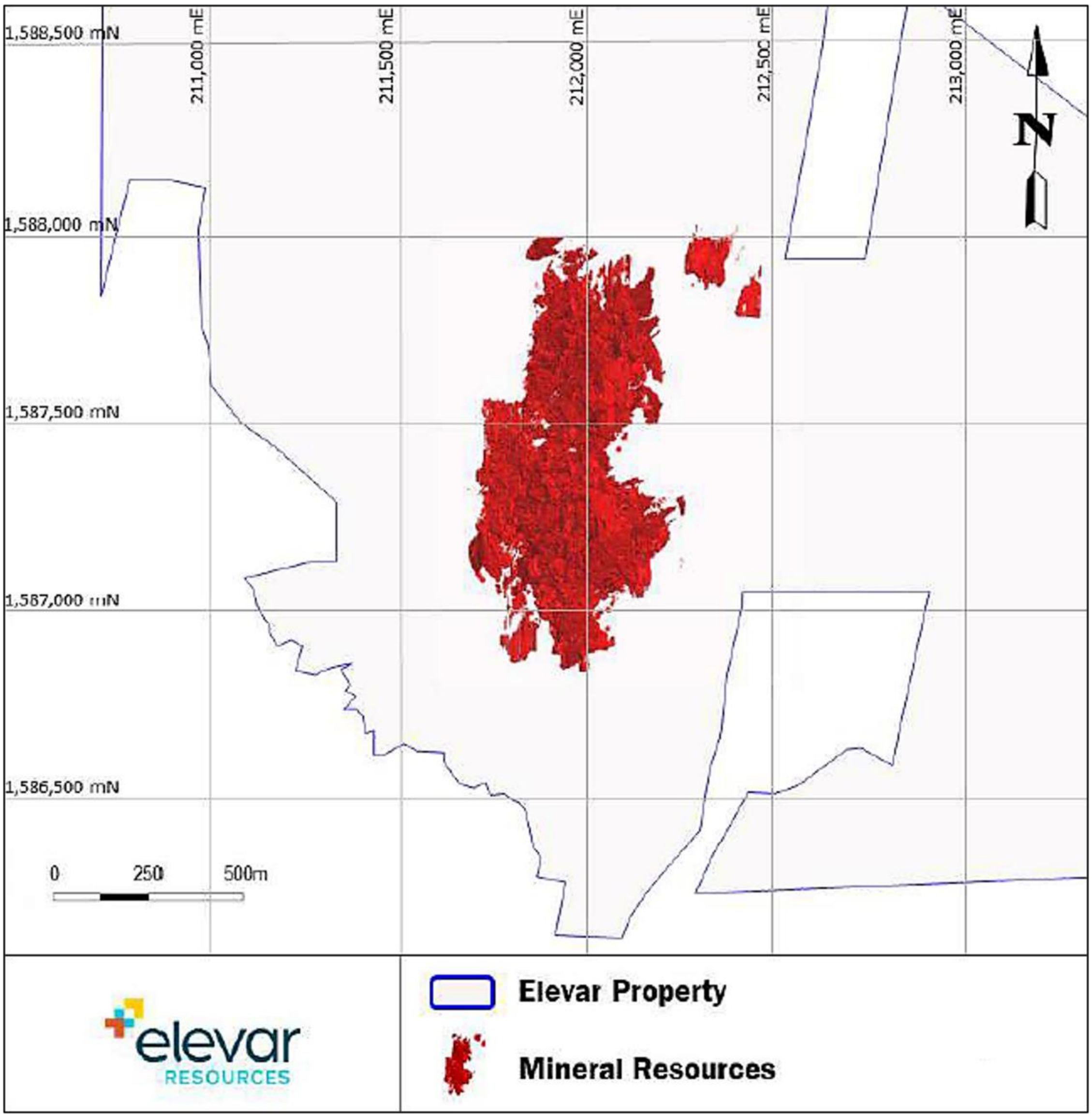

Figure 4-2: Location of Mineral Resources relative to property boundary

Source: Bluestone, 2022.

4.2 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 4-3. The perimeter of the area is described as having the UTM X and Y coordinates shown in Figure 4-3 and Table 4-1.

Table 4-1: Coordinates of exploitation license "Era Dorada"

Latitude	Longitude
210500	1589500
213000	1589500
213000	1589000
214000	1589000
214000	1585000
210500	1585000

Source: Bluestone, 2019.

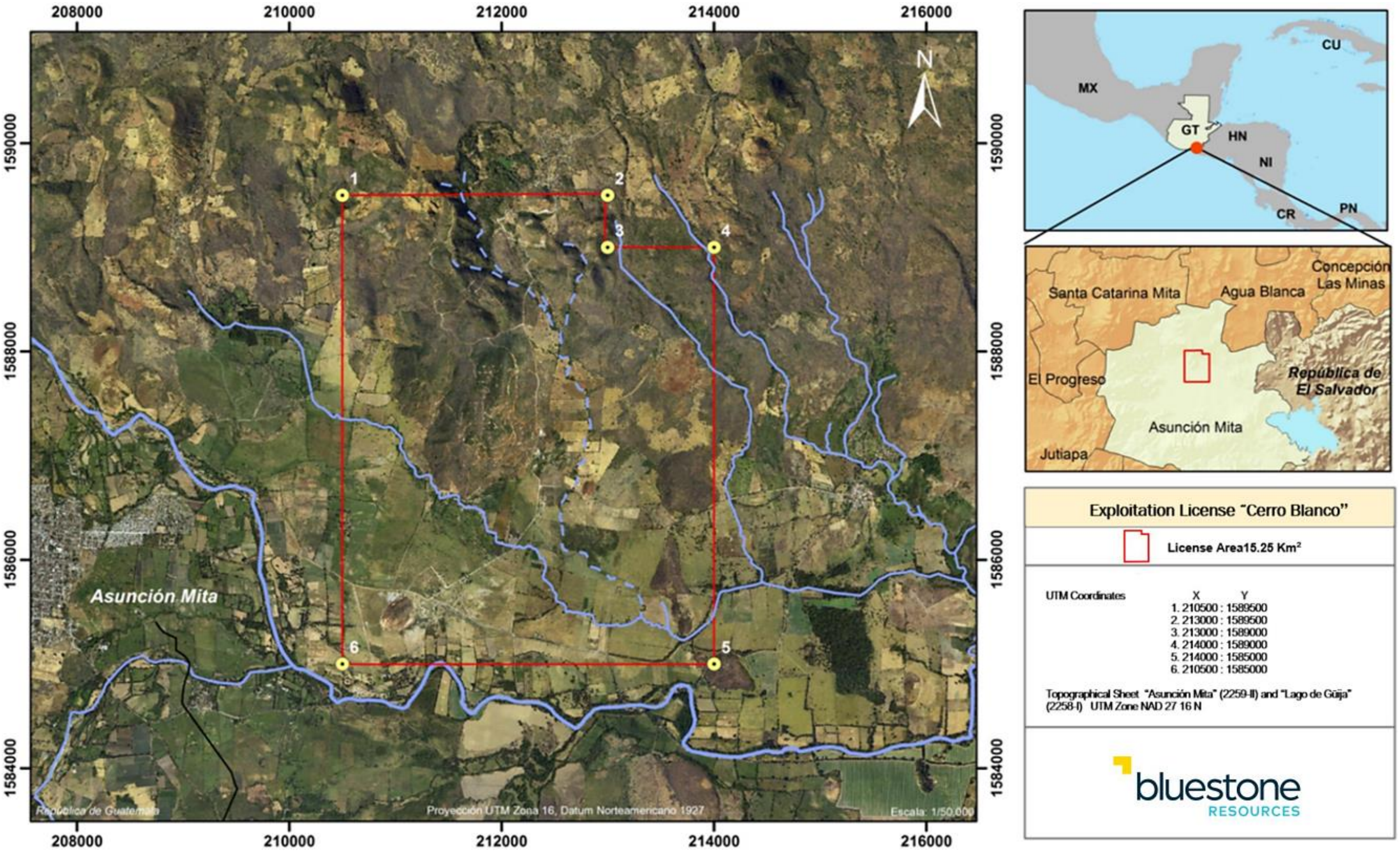

Figure 4-3: Era Dorada exploitation license coordinates

Source: Bluestone, 2019.

4.3 Royalties

Royalties in mining are financial compensation paid to the State for the right to exploit Mineral Resources, contributing to the redistribution of the benefits of mining activities. In Guatemala, the royalties due for mineral extraction amount to 5% of gross revenue. Additionally, a 1.05% rate on the Net Smelter Return is considered for royalty payments to Goldcorp Royalty.

4.4 Environmental

The Project is following Guatemala's 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 (EIA) and permits are necessary for infrastructure components such as power lines.

The current permits and permit amendments are presented in Section 20.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 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.

Guatemala has 400 km of coastline and claims its territorial waters extend 22 km outward, plus an exclusive economic zone of 370 km offshore. Hurricanes and tropical storms sometimes affect coastal regions.

The five main ports in Guatemala and their main activities are listed below:

- Atlantic Ports:
  - Puerto Santo Tomás de Castilla (containers);
  - Puerto Barrios (containers).
- 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.

The nearest airport to the region of the Project is Aurora International Airport, in Guatemala City, 114 km away from the Project, in a 2-hour car ride. It offers a large range of flights and international connections.

5.2 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.

5.3 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 5-1.

Figure 5-1: 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 exceeds 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.

5.4 Local Resources & 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.

There is no record of any previous exploitation 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 labour will be available for employment at Era Dorada. As such, the Project intends to hire the majority of operations staff locally and has allowed for the cost of training programs within the Owner's budget.

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 5-2). 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 10 kilometres to the west of the Project. The substation has a capacity to supply up to 20 MW of power.

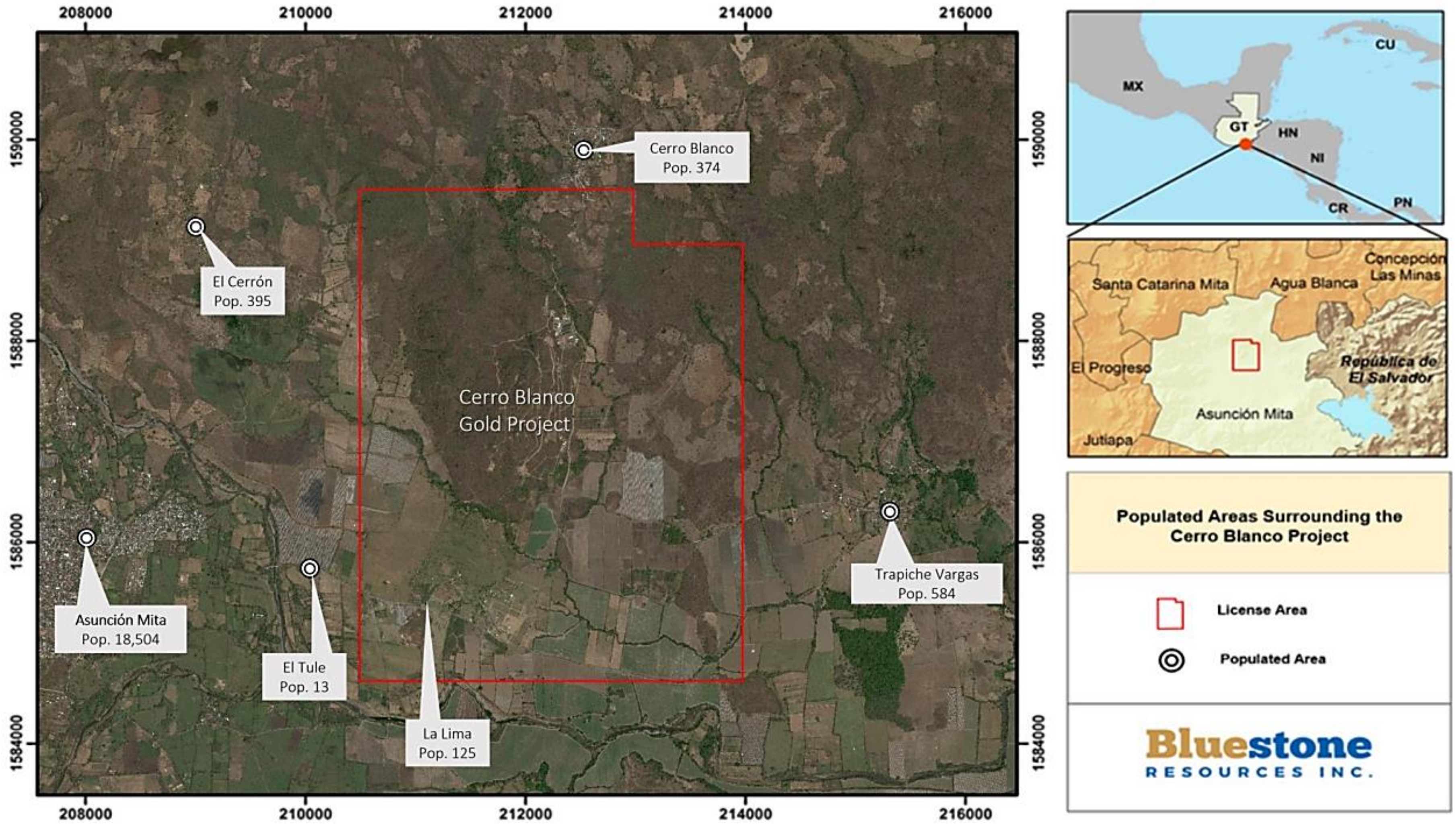

Figure 5-2: Population centers near the Project area

Source: Bluestone, 2022

6 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 its 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 Au g/t. 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 Au g/t, with the occasional 10 to 20 Au g/t 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;
- construction of the north and south ramp access and ventilation raises;
- underground 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.

6.1 Data Validation History

Historical core logging, sampling, and QA/QC procedures were reviewed by Golder in 2014. Ten 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, 10 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 6-1 summarizes the samples selected for core logging review and verification sampling.

Table 6-1: 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 6-2 is a list of the drill hole collar surveys completed by Golder.

Table 6-2: 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

Source: Goldcorp, 2014.

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 6-3, 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 6-3: 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 6-1 illustrates an example of the XY scatter plots used to compare assay results.

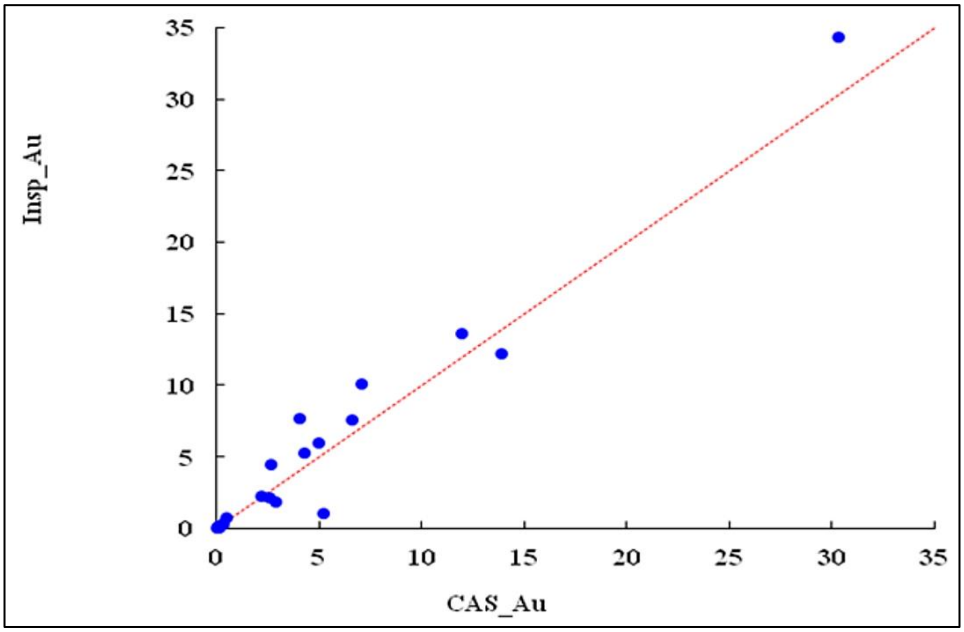

Figure 6-1: Example of XY scatter plot for hole CB34

Source: Goldcorp, 2014.

6.2 Historic Resources

Indicated and Inferred Resources were initially reported in 2008 for the Cerro Blanco Project at an 8.0 g/t gold equivalent cut-off grade as follows in Table 6-4.

Table 6-4: Indicated and Inferred Mineral Resource Estimate (2008)

Resource Type[1]	Tonnes (kt)	Gold Grade (g/t)	Contained Gold (Koz)	Silver Grade (g/t)	Contained Silver (koz)	Contained Equivalent Ounces of Gold[2]
Indicated	2,500	15.65	1,266	72	5,826	1,324
Inferred	1,400	15.3	665	59.6	2,589	691

Notes:
1. The Mineral Resources have been calculated in accordance with definitions adopted by the Canadian Institute of Mining, Metallurgy, and Petroleum on August 20, 2000. Employees of Glamis Gold Ltd., under the supervision of James S. Voorhees, Executive Vice President of Operations and Chief Operating Officer, have prepared these calculations.
2. The conversion of silver ounces to gold-equivalent ounces is at a ratio of 100 silver ounces to one gold-equivalent ounce.

Source: Voorhees, 2008.

Subsequently, Mineral Resources were reported as in-situ Resources at cut-off grades of 1.0 g/t Au and 4.0 g/t Au in 2014. Blocks were classified based on drill spacing (sample distances) and the number of drill holes. In-situ Mineral Resources are summarized in Table 6-5. No wireframing was performed around these blocks.

Table 6-5: In-*situ* Mineral Resources (2014)

In-situ Mineral Resources (veins)					
CoG (g/t)	**Mt**	**Au g/t**	**Ag g/t**	**Au Moz**	**Ag Moz**
Indicated In-situ* Mineral Resources					
1	19	3.38	13.9	2.06	8.5
4	3.85	9.71	35.2	1.2	4.4
Inferred In-situ* Mineral Resources					
1	2.3	3.1	8.5	0.22	0.6
4	0.33	10.8	19.8	0.11	0.2

Note: *Reported in-situ Mineral Resources do not consider mineral availability by underground or open pit mining methods.
Source: Goldcorp, 2014.

In 2016, Goldcorp listed Mineral Resources for Cerro Blanco within its Annual Report. The last public statements by Goldcorp outlined historical Mineral Resources of 2.05 Mt grading 12.69 g/t for 840,000 oz of gold in the Indicated category, as well as 0.75 Mt grading 9.34 g/t for 230,000 oz of gold in the Inferred category.

The Indicated and Inferred Resources are historical estimates and use the categories set out in NI 43-101. These Resources are effective as of June 30, 2016, and are disclosed in Goldcorp's press release dated October 26, 2016. Resources were estimated using US$ 1,400/oz Au and US$ 20/oz Ag. Given the source of the estimates, Bluestone considers them reliable and relevant for the further development of the Project; however, a qualified person has not done sufficient work to classify the historical estimates as current Mineral Resources or Mineral Reserves, and the Company is not treating the historical estimates as current Mineral Resources or Mineral Reserves.

In 2021, the Mineral Resource Estimate was based on a scenario that considered open-pit mining methods and is reported at a base case above a 0.4 Au g/t cut-off, as tabulated in Table 6-6.

Mineralized material from mining activities conducted up to 2021—including ramp development and access—was stockpiled on-site and segregated for future processing. Correlograms for gold and silver were created and employed to estimate the stockpile Resources using ordinary kriging. The estimate was validated using the nearest neighbour and inverse distance methods.

In addition to the open-pit Resources, high-grade vein material located below the pit shell remains a potential target for limited underground mining meth.

Table 6-6: Mineral Resource statement (2021)

Resource Category	Tonnes (kt)	Au Grade (g/t)	Ag Grade (g/t)	Contained Gold (koz)	Contained Silver (koz)
Measured	40,947	1.8	7.9	2,382	10,387
Indicated	22,595	1.0	4.2	706	3,058
Measured & Indicated	63,542	1.5	6.6	3,089	13,445
Inferred	1,672	0.6	2.1	31	112
Below Pit (Indicated)*	189	5.7	13.4	35	82
Stockpile (Measured)	30	5.4	22.6	5	22

Notes:
1. All Mineral Resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum (CIM) definitions, as required under National Instrument 43-101 (NI 43-101), with an effective date of December 31, 2020.
2. Mineral Resources reported demonstrate a reasonable prospect of eventual economic extraction, as required under NI 43-101; Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
3. *Underground Mineral Resources are reported at a cut-off grade of 3.5 Au g/t. Cut-off grades are based on a price of US$ 1,600/oz gold, US$ 20/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.
7. Mineral Resources are inclusive of Mineral Reserves.

Source: Kirkham, 2021.

6.3 Historic Reserves

The mining stope and sub-level designs with external, backfill, and planned dilution, along with the mining recovery factor applied, determined the 2019 Mineral Reserve estimate shown in Table 6-7.

Table 6-7: Mineral Reserve Estimate (2019)

Class	Diluted Tonnes (kt)	Au Grade (g/t)	Ag Grade (g/t)	Au Ounces (koz)	Ag Ounces (koz)
Proven	313	8.3	31.4	83	315
Probable	3,131	8.5	32.3	857	3,256
Total	3,444	8.5	32.2	940	3,571

Notes:
1. The Qualified Person for the Mineral Reserve Estimate is Michael Makarenko, P. Eng., of JDS Energy & Mining Inc.
2. Effective date: January 29, 2019. All Mineral Reserves have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum (CIM) definitions, as required under NI 43-101.
3. Mineral Reserves were estimated using a $ 1,250 /oz gold price and a gold cut-off grade of 3.5 g/t. Other costs and factors used for gold cut-off grade determination were mining, process, and other costs of $ 109.04/t, transport and treatment charges of $ 5.00/oz Au, a royalty of $ 24.84 /oz Au, and a gold metallurgical recovery of 95%.
4. Silver was not used in the estimation of cut-off grades, but is recovered and contributes to the Project's cash flow.
5. Tonnages are rounded to the nearest 1,000 t; metal grades are rounded to one decimal place. Tonnage and grade measurements are in metric units; contained gold and silver are reported as thousands of troy ounces.
6. Rounding, as required by reporting guidelines, may result in summation differences.

Source: Bluestone, 2019.

In 2021, Mineral Reserves for the Cerro Blanco Gold Project were estimated at 53.9 Mt at an average grade of 1.64 g/t of gold for 2,846,000 oz and 7.27 g/t of silver for 12,602,000 oz, as summarized in Table 6-8. The Mineral Reserve Estimate (MRE) was prepared by G Mining Services Inc. (GMS)

Table 6-8: Cerro Blanco Gold Project open pit Mineral Reserve Estimate (2022)

Reserve Category	Tonnage (kt)	Gold (g/t)	Gold (koz)	Silver (g/t)	Silver (koz)
Proven	37,618	1.89	2,286	8.33	10,084
Probable	16,279	1.07	560	4.81	2,518
Proven & Probable	**53,896**	**1.64**	**2,846**	**7.27**	**12,602**

Notes:
1. CIM definitions were followed for Mineral Reserves.
2. The effective date of the estimate is November 1, 2021.
3. Mineral Reserves are estimated at a cut-off grade of 0.50 g/t Au Eq.
4. Mineral Reserves are estimated using the following long-term metal prices (Au = US$ 1,550/oz and Ag = US$ 20/oz).
5. The bulk density of ore is variable but averages 2.70 t/m^3.
6. The average strip ratio is 2.7:1.
7. The average mining dilution factor is 6.7%.
8. Other costs and factors used for gold cut-off grade determination were processed, G&A, and other costs of $ 21.17/t, a royalty of $ 31.60 /oz Au, and gold and silver metallurgical recoveries of 91% and 85%, respectively.
9. Tonnages are rounded to the nearest 1,000 t; metal grades are rounded to two decimal places. Tonnage and grade measurements are in metric units; contained gold and silver are reported as thousands of troy ounces.
10. The Mineral Reserves may be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, and socio-economic factors.
11. Mineral Resources are inclusive of Mineral Reserves.

Source: GMS, 2021.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 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 Paleozoic 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 7-1), 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 7-1: Location of Era Dorada and other deposits in the Central American Volcanic

Source: Bluestone, 2020.

7.2 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 7-2). 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).

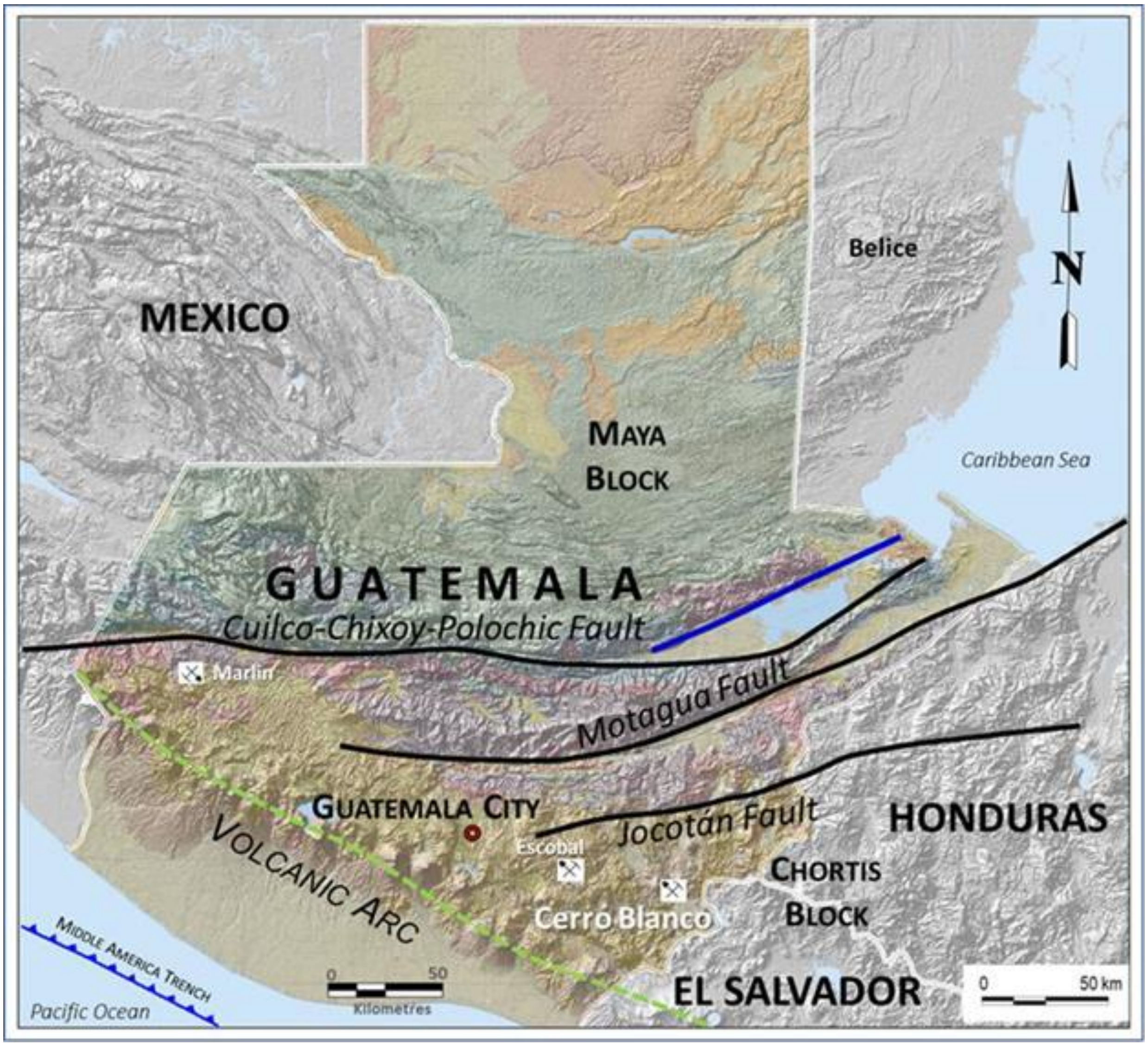

Figure 7-2: Regional structural map of Guatemala

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 Paleozoic 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 7-1), 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 4 and 8 million years. Argon-argon dating (40Ar-39Ar) of vein adularia from Era Dorada returned a date of 4.93 ± 0.47 Ma.

7.3 Local Geology

The Project deposit is a classic hot springs-related, low-sulphidation 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 7-3 shows a simplified geological map for Cerro Blanco.

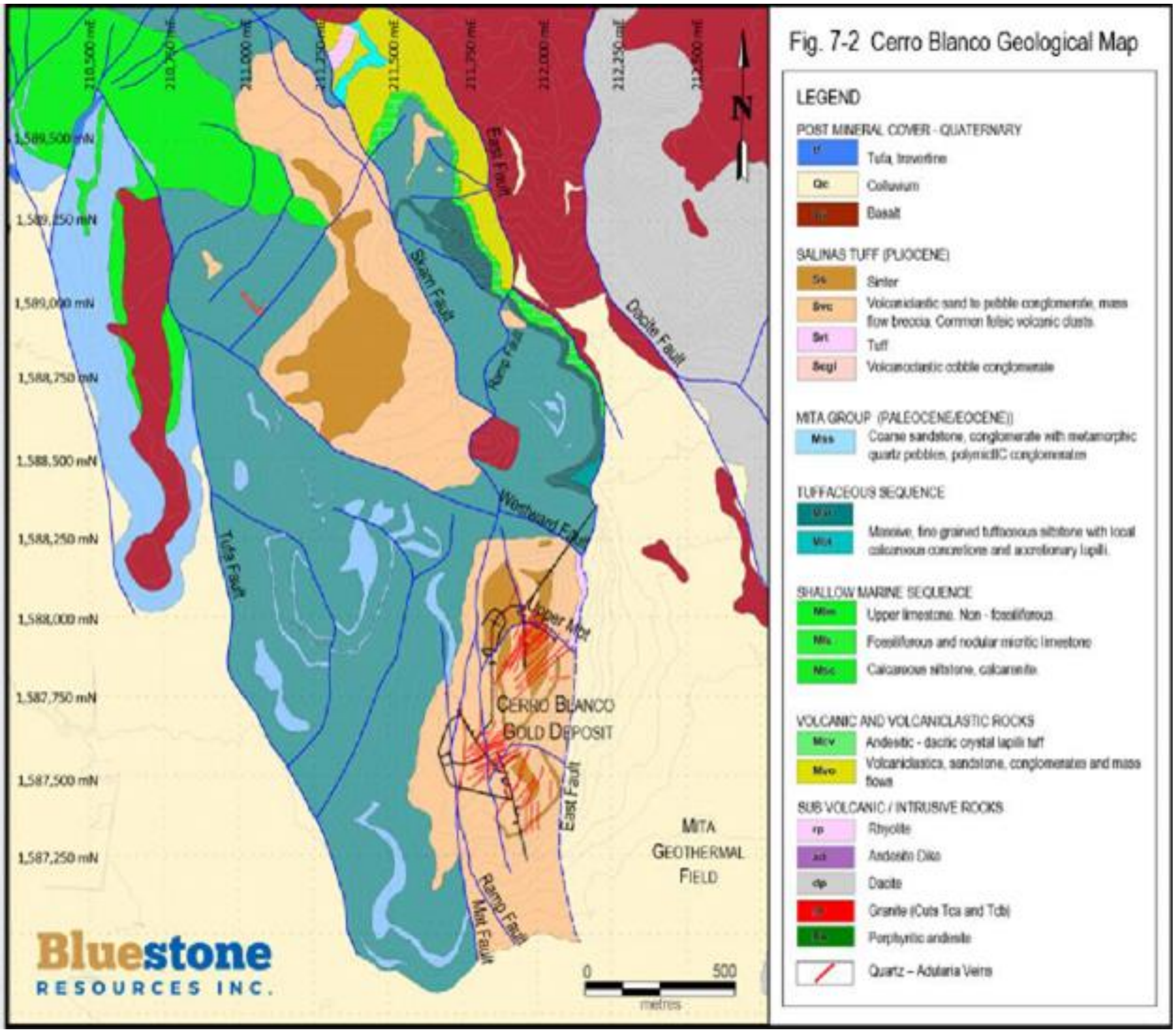

Figure 7-3: Geological map of Cerro Blanco

Source: Pratt and Gordon, 2019.

7.3.1 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 accumulation (Figure 7-4).

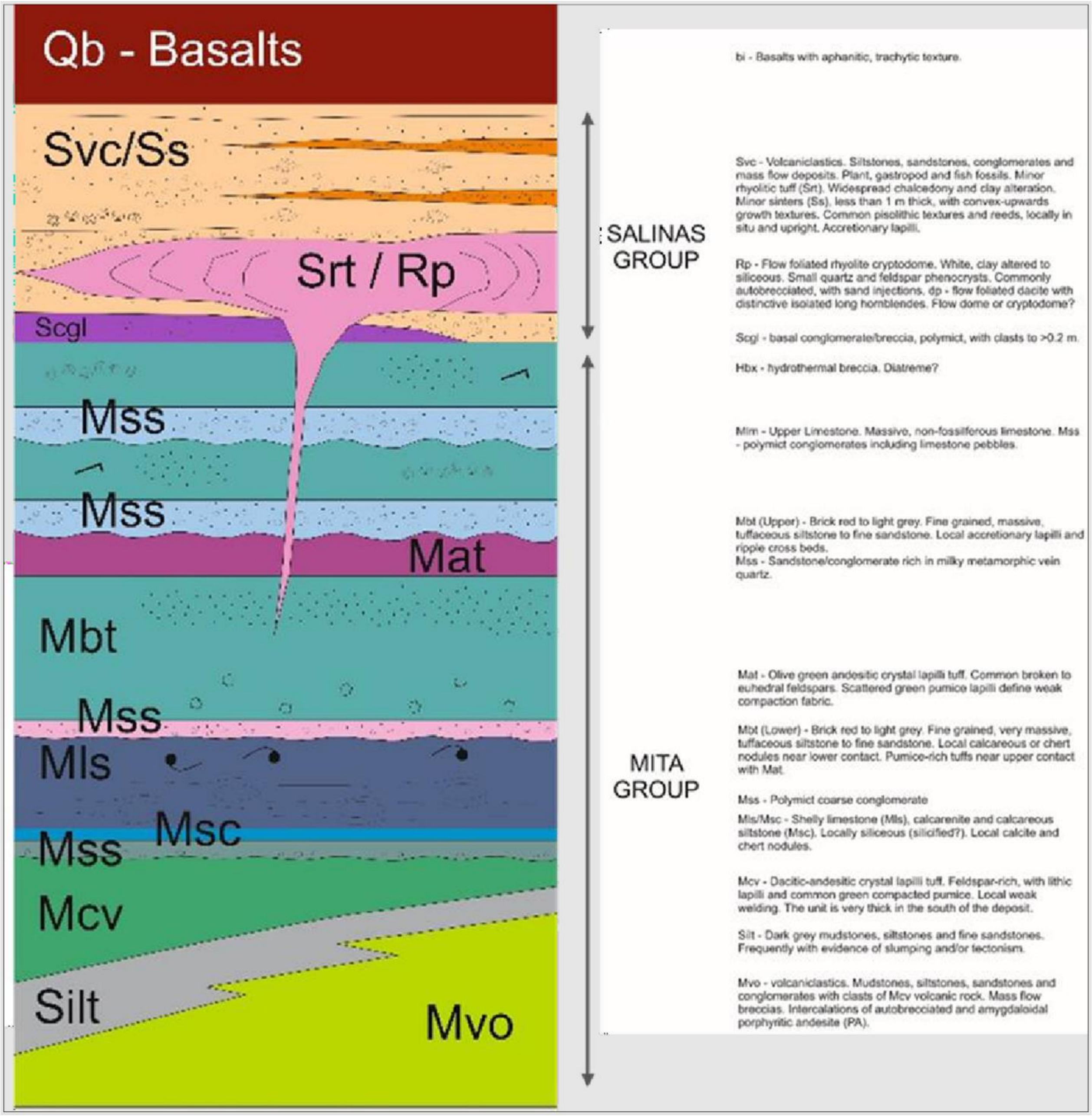

Figure 7-4: Lithostratigraphy & lithology at Era Dorada

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 7-5), 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 7-5: 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, see Figure 7-6), 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 7-3), 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).

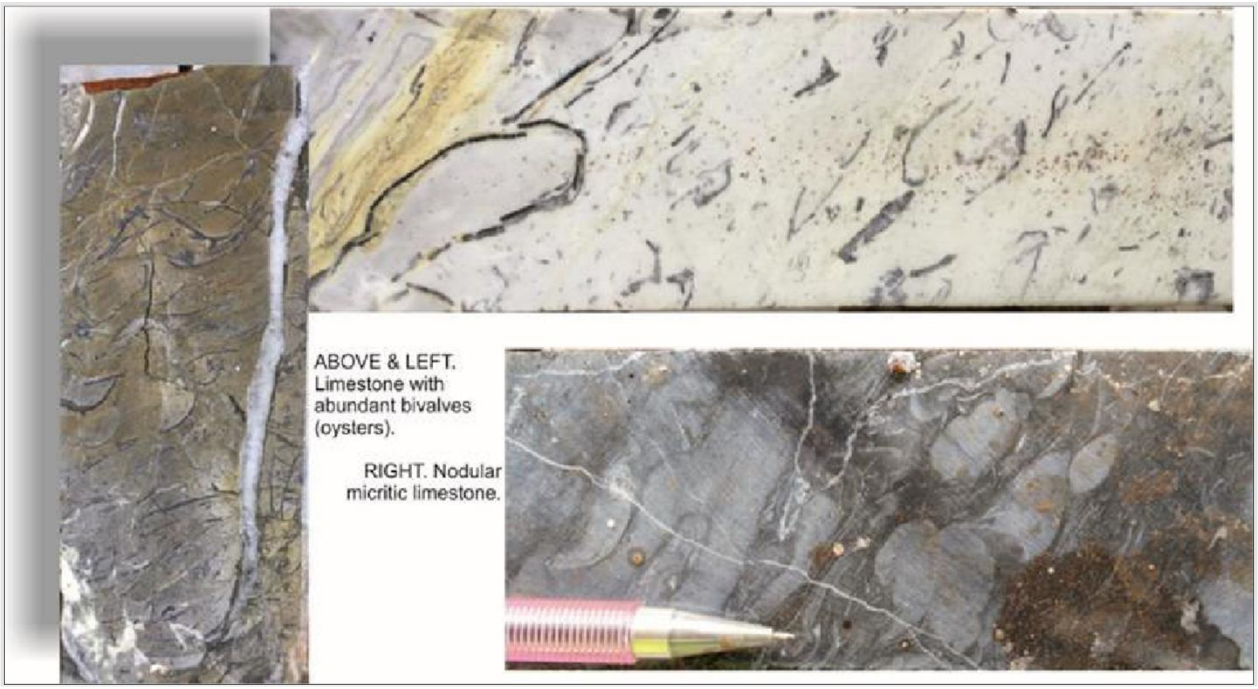

Figure 7-6: Examples of limestones (Mls)

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 7-7). Where the paleo-groundwater table was several metres 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 H_2S-bearing steam into downward-percolating groundwater. The overall result is an interlayered sequence of sinter, silicified conglomerate, and steam-heated alteration (Figure 7-7).

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 7-7: 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 7-4). 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 7-8, 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).

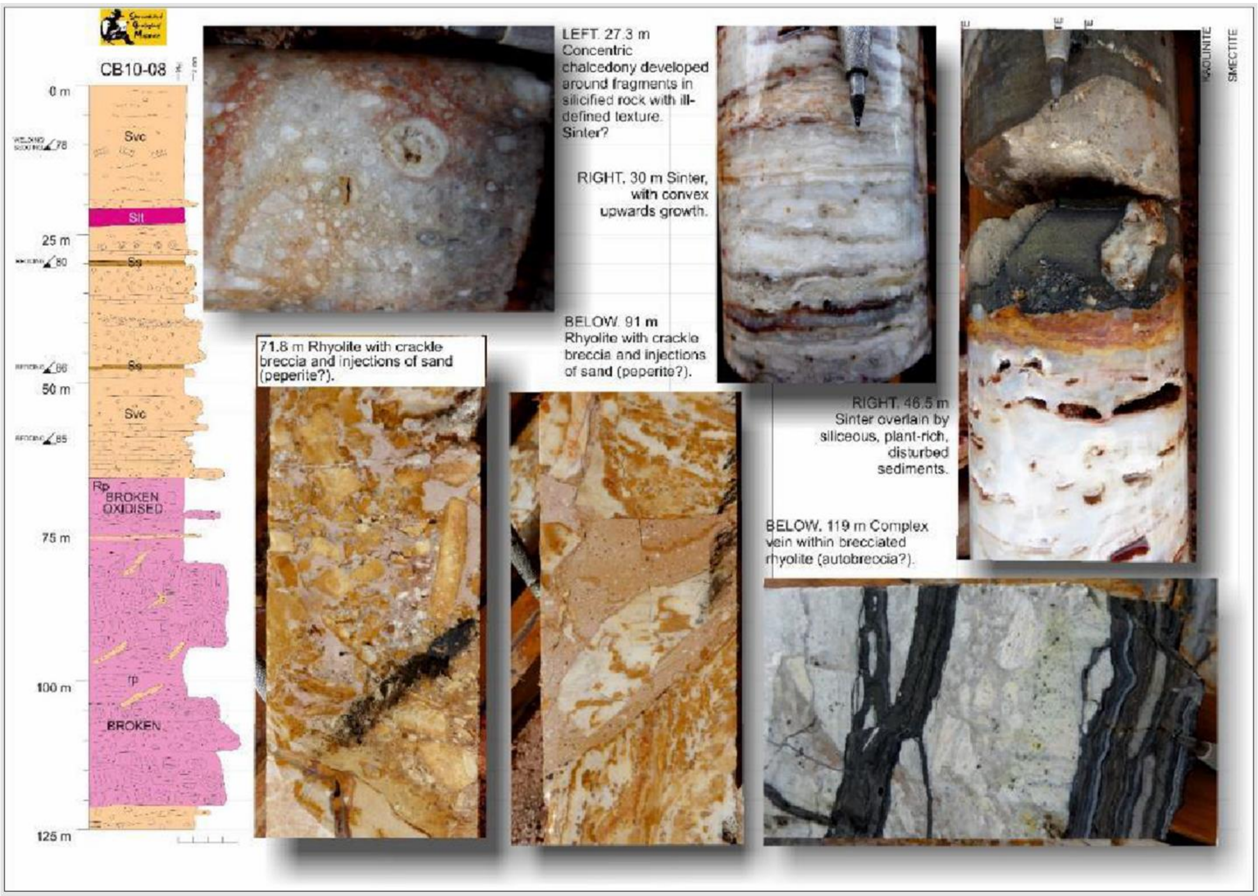

Figure 7-8: Example drill log from the Salinas Group

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 infill 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 7-9). 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.

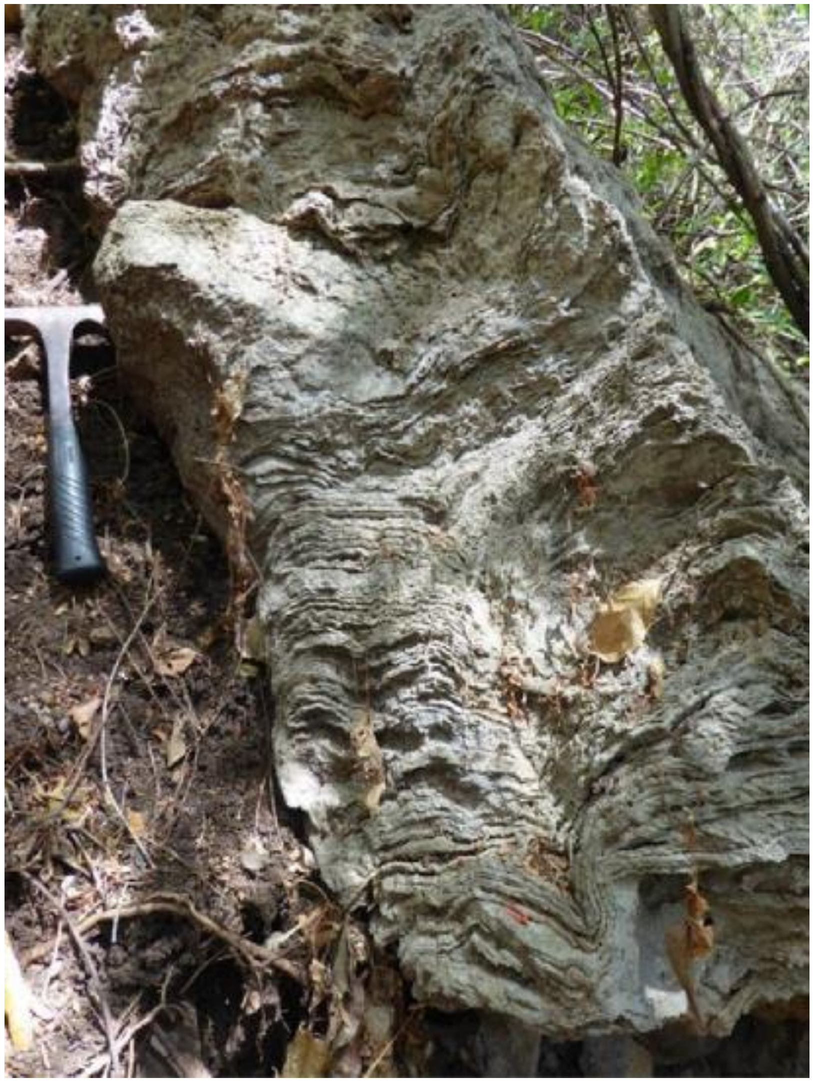

Figure 7-9: Recent travertine exposure

Source: Pratt and Gordon, 2019.

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

7.3.2 Structure

The gold mineralization at the Project is hosted within a broadly north-south-striking graben. The East Fault (Figure 7-10), also referred to as the "East Horst Fault" in previous studies, is cut by several drill holes 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.

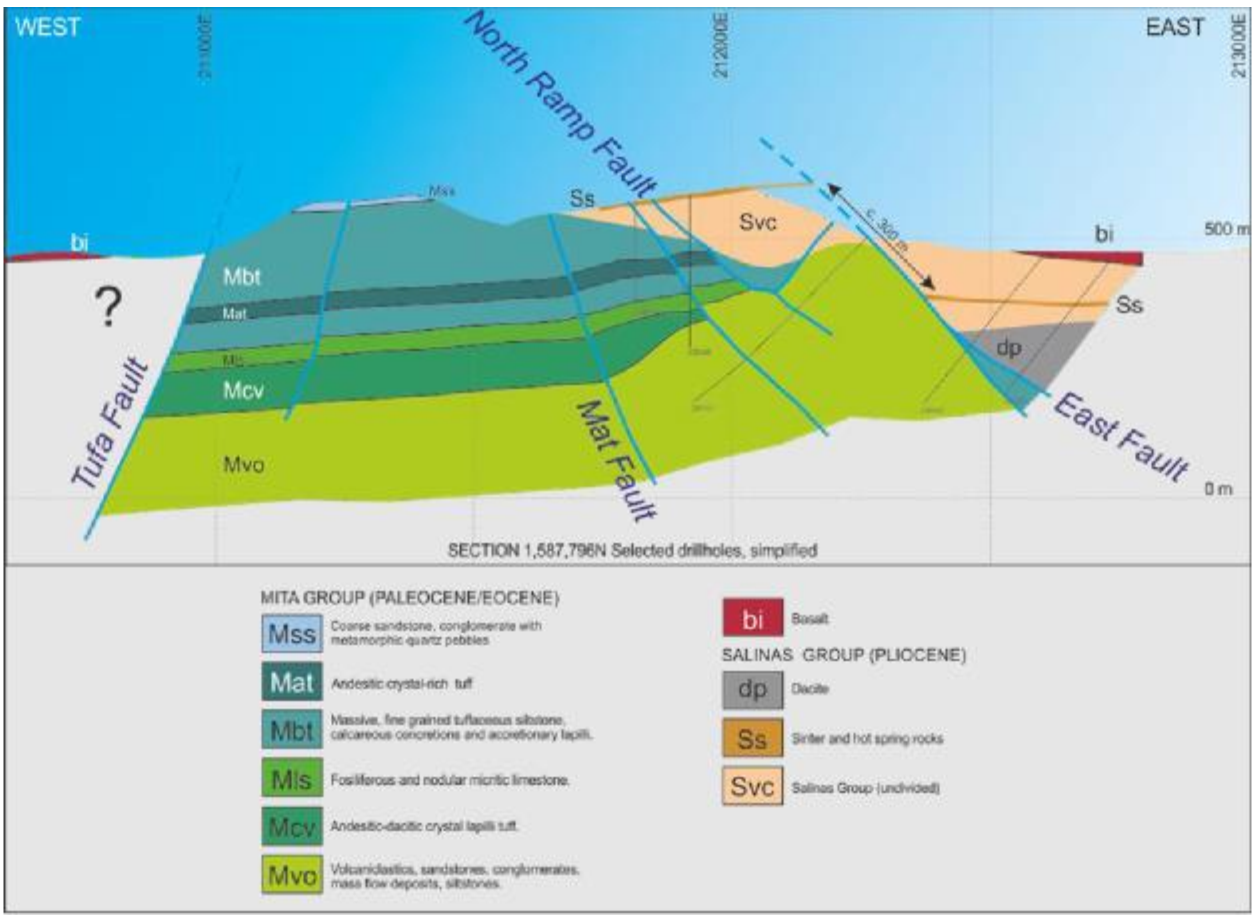

Figure 7-10: Simplified west-west cross-section across Era Dorada

Legend: Many drill holes 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-sulphidation epithermal vein deposits, which normally comprise a single, relatively continuous vein. At Era Dorada, there are sheeted vein swarms that resemble a duplex. Figure 7-11 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.

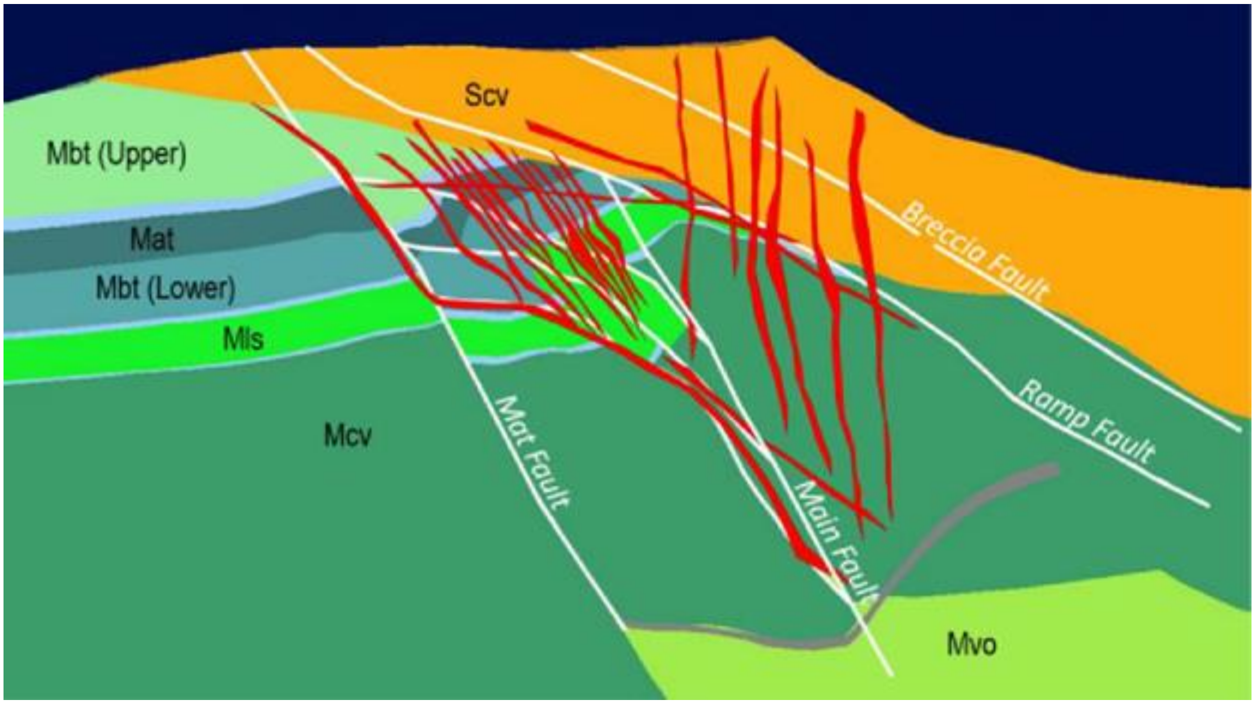

Figure 7-11: East-west cross-section of the South zone, Era Dorada looking North

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 7-10).
3. Major extensional faulting with downthrows to the east of up to several hundred metres. 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 (centimetre 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 7-12 shows structural measurements from the underground workings for faults and veins. However, most understanding of the principal faults comes from 3D modelling, 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 7-13.

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 favourable 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).

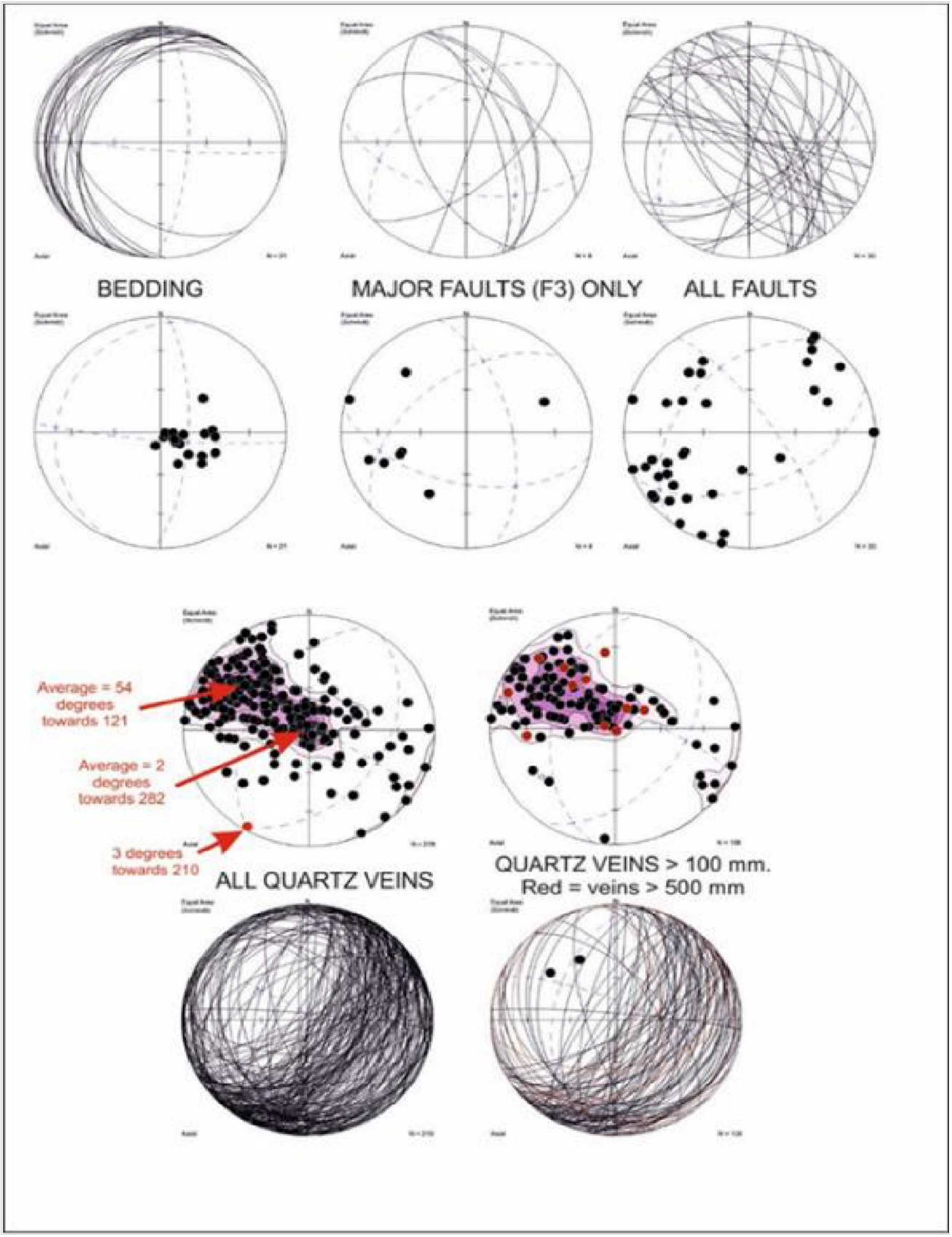

Figure 7-12: Stereograms (equal area) showing poles & great circles for faults & veins

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

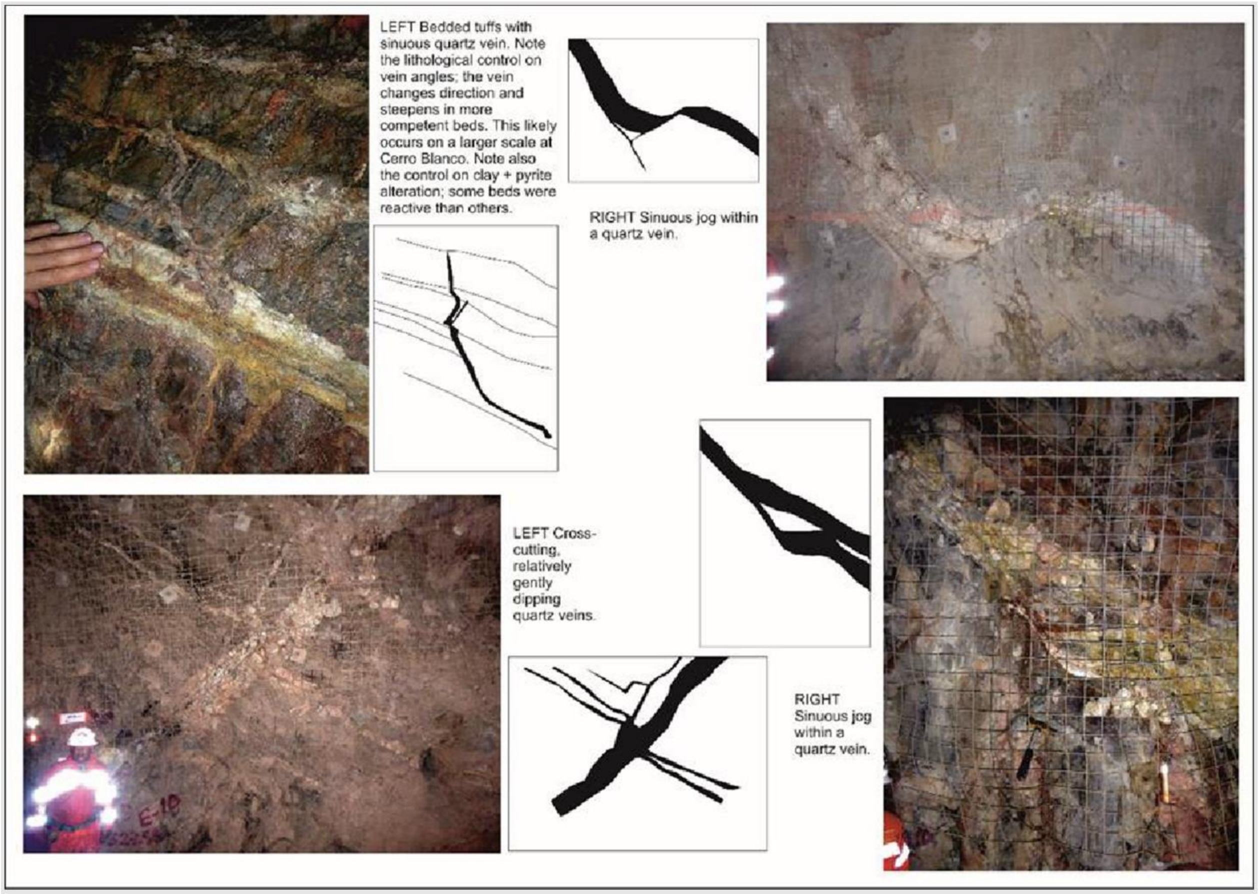

Figure 7-13: Photographs with sketches of veins exposed underground

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 7-14 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 7-15 shows a series of horizontal slices at different elevations. The gap between the South and North Resource areas mostly comprises the triangular wedge of Upper Mbt stratigraphy between the Upper Mbt and Ramp faults. This seems to have been unfavourable for veining.

Underground mapping supports the 3D modelling; 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.

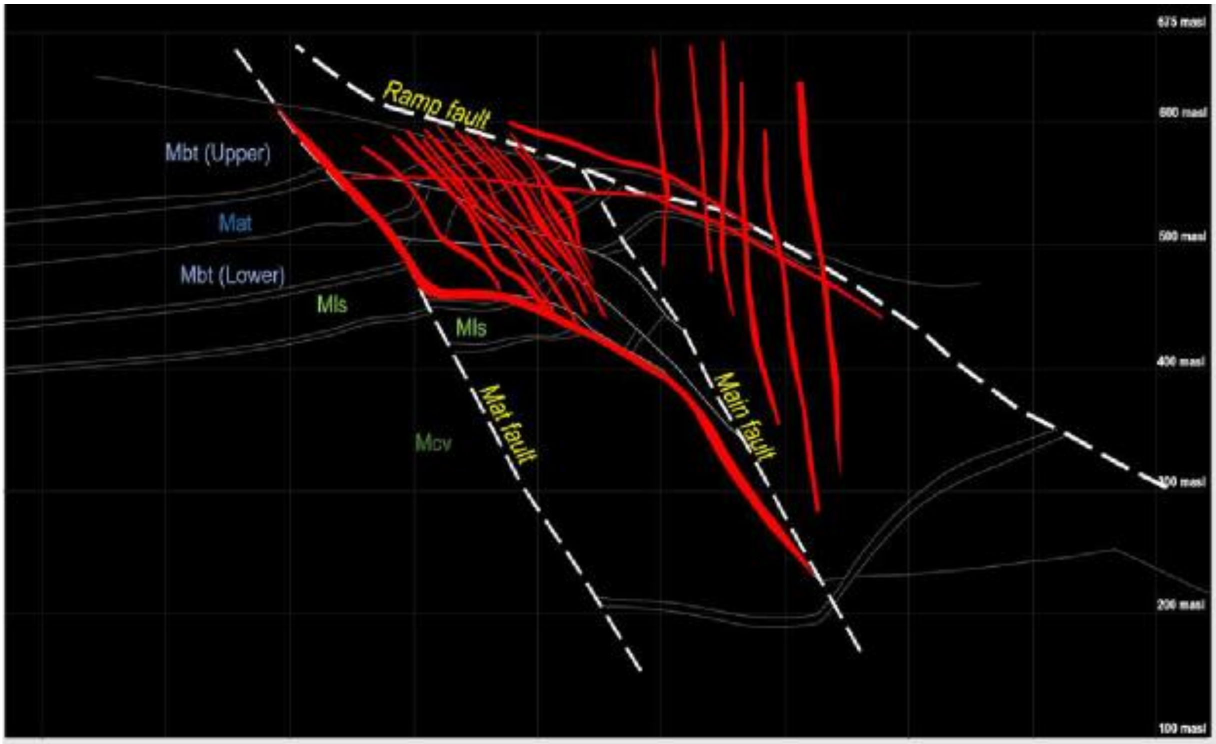

Figure 7-14: Annotated, vertical east-west cross-section across the south ramp (looking North)

Source: Bluestone, 2020.

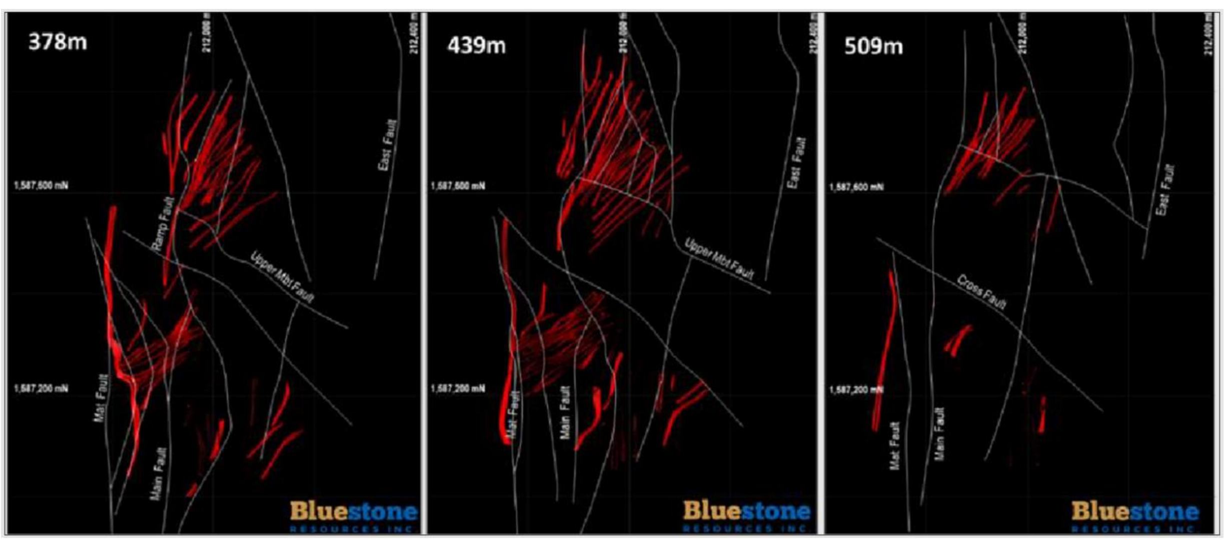

Figure 7-15: Horizontal slices at different elevations through Era Dorada

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

There are some secondary (conjugate) vein directions, but stereograms for sub-areas (Figure 7-16) 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.

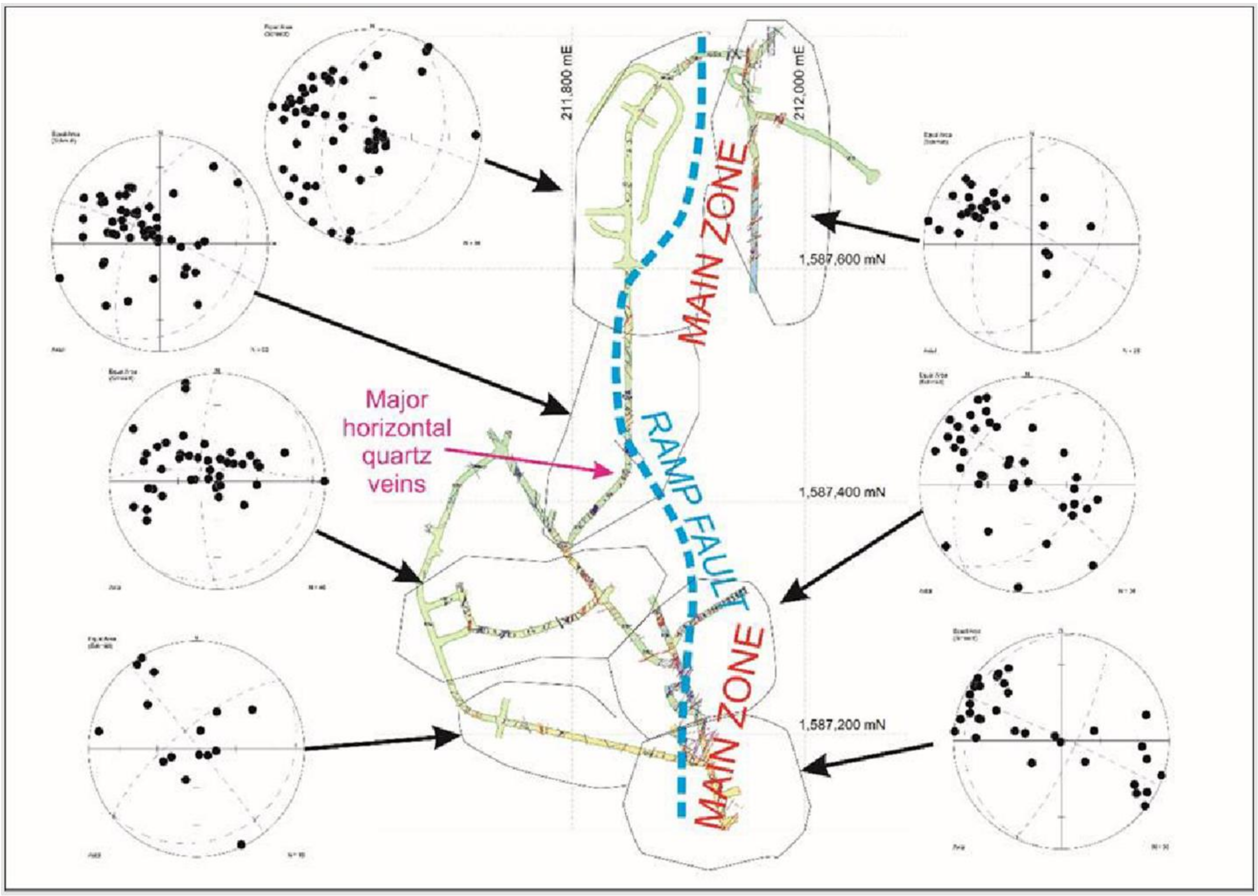

Figure 7-16: Stereograms for more detailed sub-areas in underground mapping

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 that 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 modelling, 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.

7.4 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-sulphidation 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 Au g/t and 22 Ag g/t at a depth of 634 m. The upper 6 m of this zone averages 23.9 Au g/t and 79 Ag g/t. 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.

7.4.1 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, sulphides, 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 7-17 shows a high-grade intercept in drill hole CB-20-430 with banded chalcedony-adularia-acanthite and visible gold that assayed 144 Au g/t and 282 Ag g/t.

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 sulphides, 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. Sulphides 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.

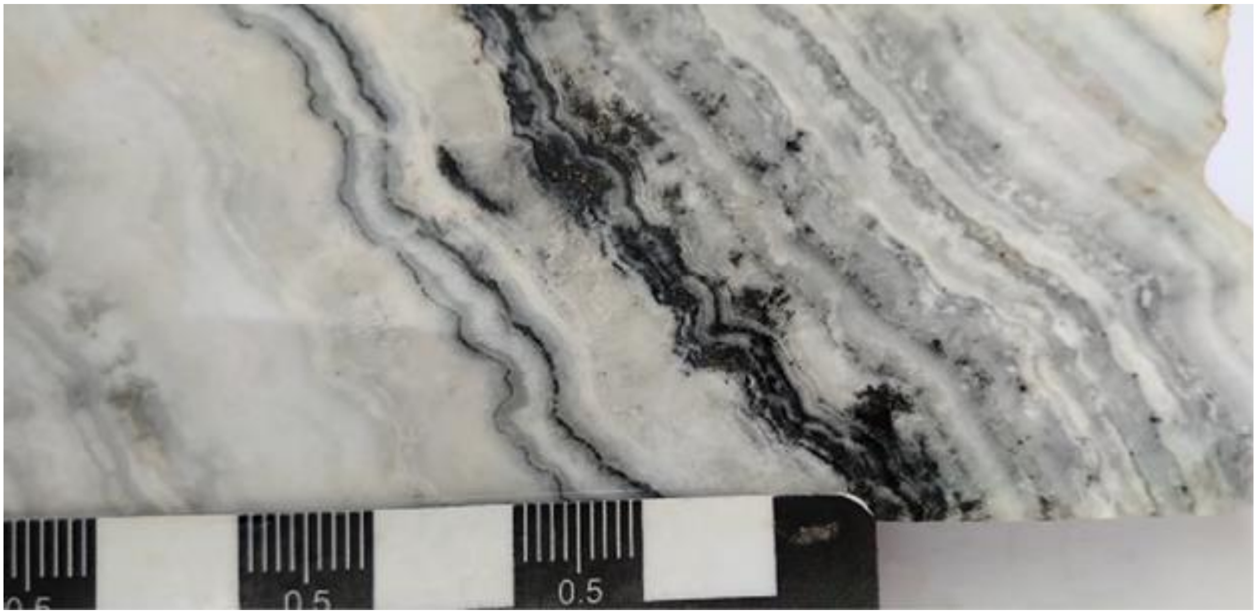

Figure 7-17: High-grade drill hole intercept hole CB20-430 – 144 g/t Au, 282 g/t Ag (227.3 to 228.9 m)

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 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 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, drill holes 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 in width, do appear in the vicinity of the north zone ramp (Figure 7-18) and in deeper levels of the south zone. Closely spaced veins or zones of convergence form wide zones of high-grade mineralization (Figure 7-18).

Figure 7-18: View of veins VN-05, 06, 07 in the north ramp underground workings

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

Figure 7-19 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 sulphides/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 are mostly pseudomorphed to cryptocrystalline silica phases.

Figure 7-19: 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 7-20). 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-sulphidation 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 7-20: Example of geopetal structure

Source: Pratt and Gordon, 2019.

7.4.2 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 7-21). 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 they contain strongly argillic-altered lithologies and fault gouge zones.

Figure 7-21: Salinas Unit – examples of disseminated mineralization rock types, Salinas Unit

Source: Bluestone, 2020.

7.4.3 Hydrothermal Alteration

Many low-sulphidation 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 sulphide content (Figure 7-22). 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 1 equilibrium assemblage of quartz (chalcedony) adularia-calcite- ankerite. Sulphides are located mostly in ginguro bands that consist of fine-grained pyrite, chalcopyrite, tetrahedrite, and acanthite. Stage 2 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.

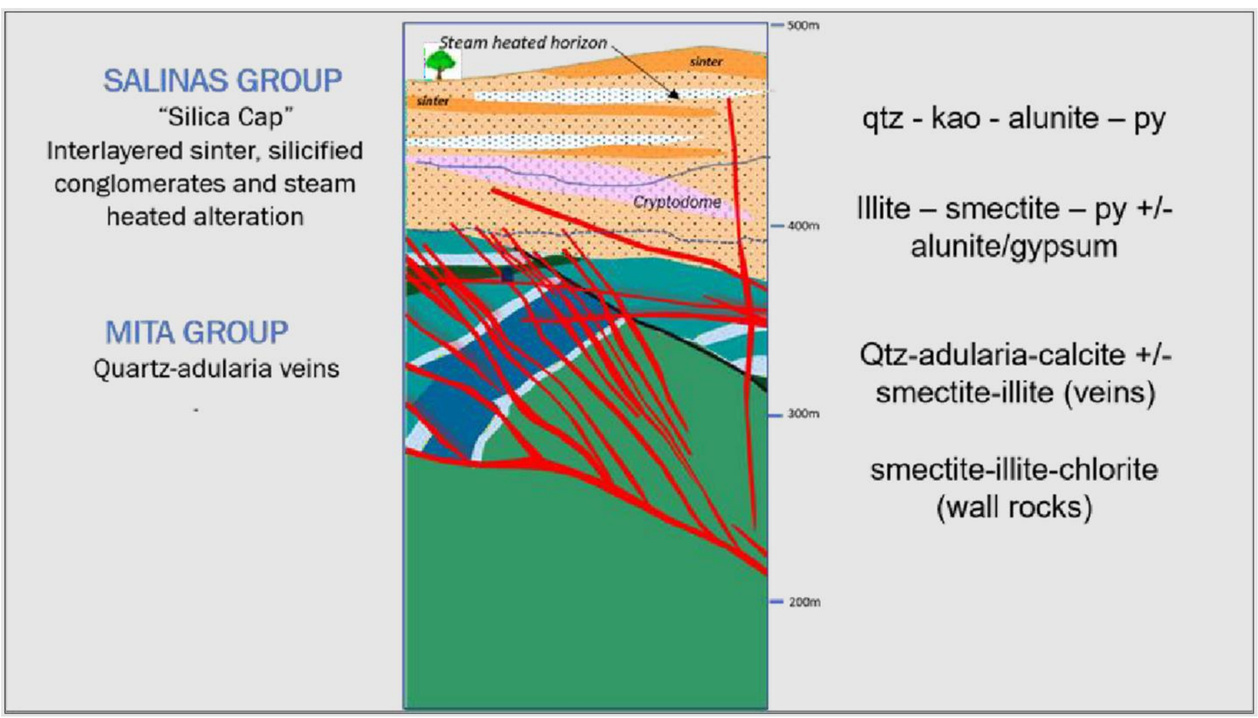

Figure 7-22: Vertical alteration profile through Era Dorada

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 metres 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 metres 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 H_2S-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 7-23.

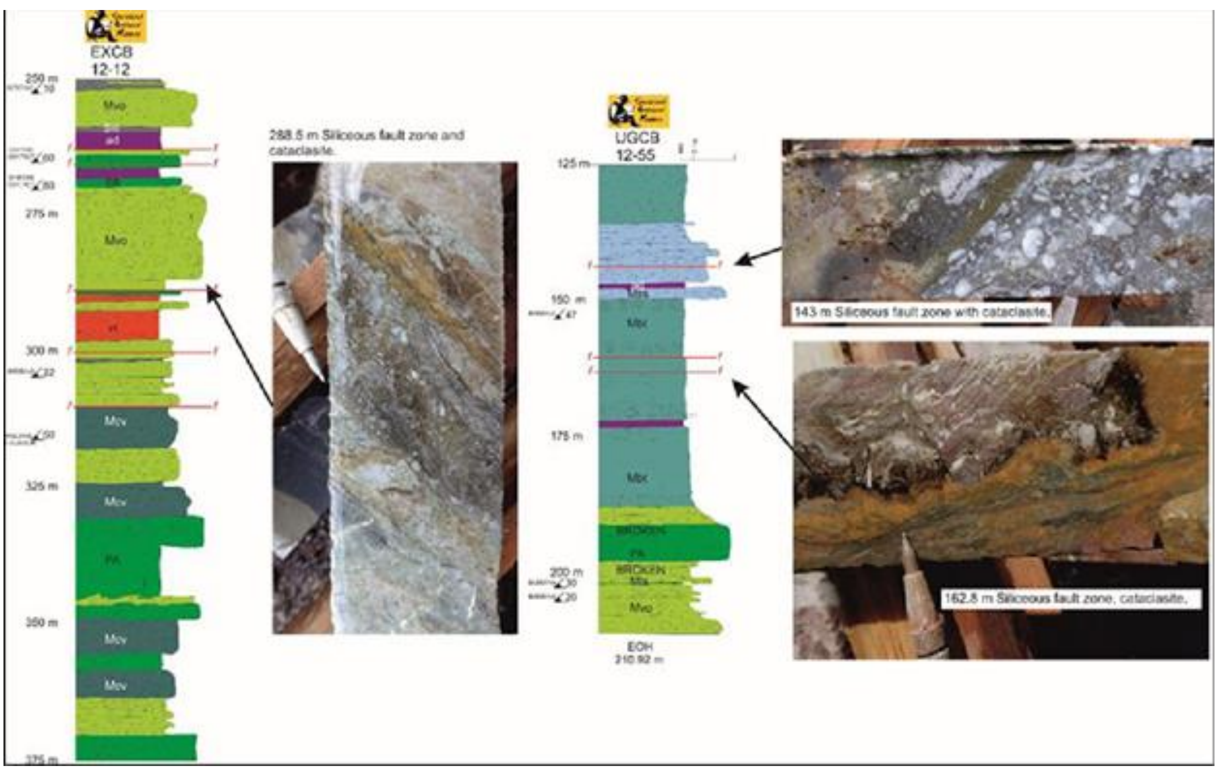

Figure 7-23: Examples of sealed, silicified fault zones

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-sulphidation 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 favourable rock types. Overall, the Era Dorada silica cap averages 400 m wide and is up to 150 m deep for at least a kilometre 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%.

8 DEPOSIT TYPES

The low sulphide content and near absence of base metals in the Era Dorada veins confirm it as a classic hot springs-related, low-sulphidation epithermal deposit. In common with most low-sulphidation 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-sulphidation gold deposits occur at the highest crustal levels, most removed from inferred intrusion source rocks. Figure 8-1 shows the generalized deposit model.

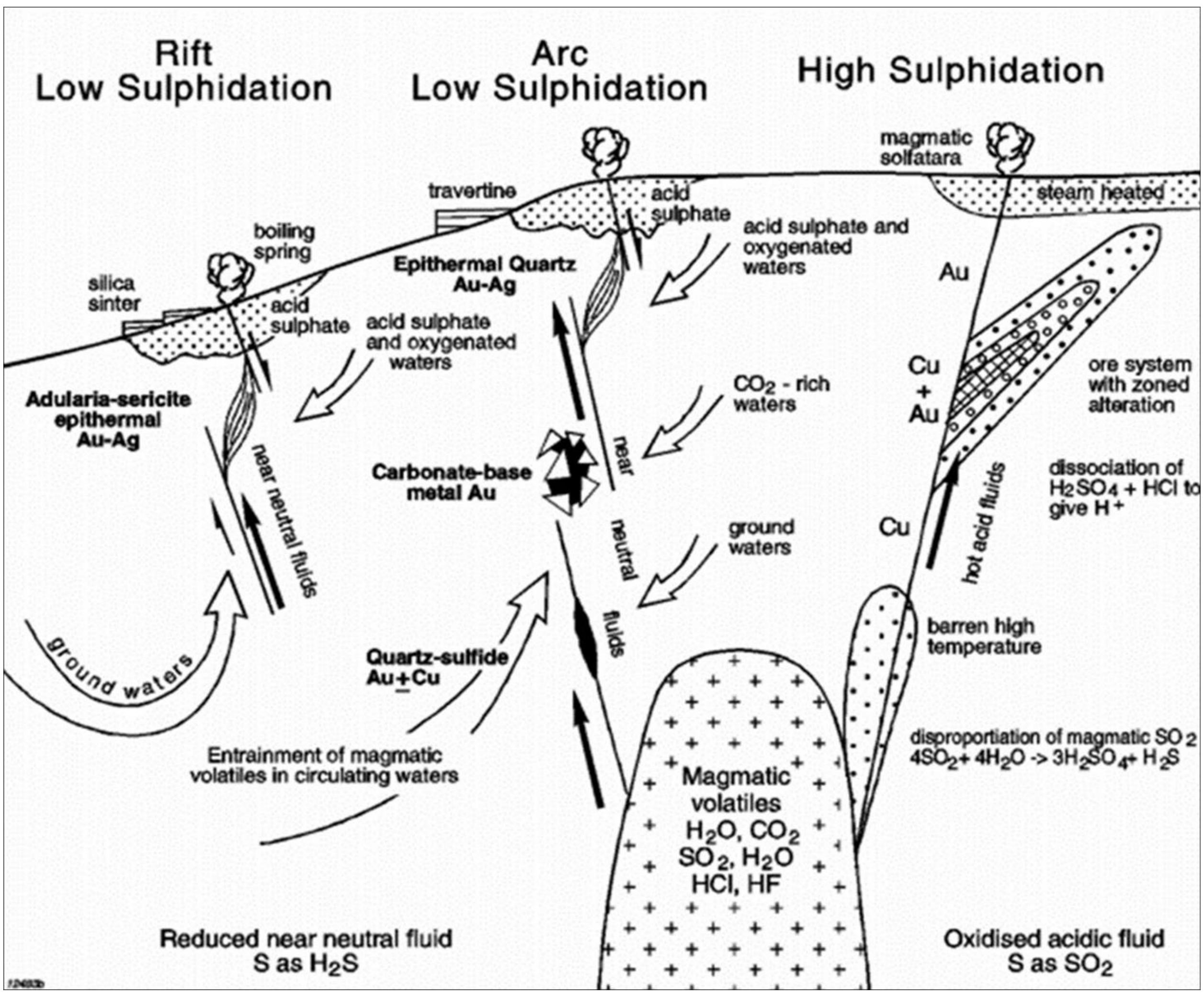

Figure 8-1: Generalized deposit model schematic

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 sulphate (low pH), groundwater is also favoured 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 sulphate alteration comprising silica (chalcedony, opal), kaolin, and local alunite, and these acid sulphate 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 sulphidation. The low-sulphidation 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 kilometres 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 sulphide-rich sediment—reinforce the interpretation.

8.1 Era Dorada Deposit Geology

The Era Dorada deposit is a classic hot springs-related, low-sulphidation 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 and 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 metres 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.

9 EXPLORATION

Exploration activities carried out on the Project were conducted by companies that held ownership prior to Aura's acquisition. As such, this work is considered historical in nature for the purposes of this Technical Report.

There are no current exploration activities to report on the property.

10 DRILLING

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 10-1 summarizes historical drilling on the property.

Table 10-1: Drilling summary

Year	Company	Holes Drilled	Metres
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 10-1 shows a plan view of drill hole locations. Figure 10-2 and Figure 10-3 show representative section views of the drilling along with gold, as say data and topography.

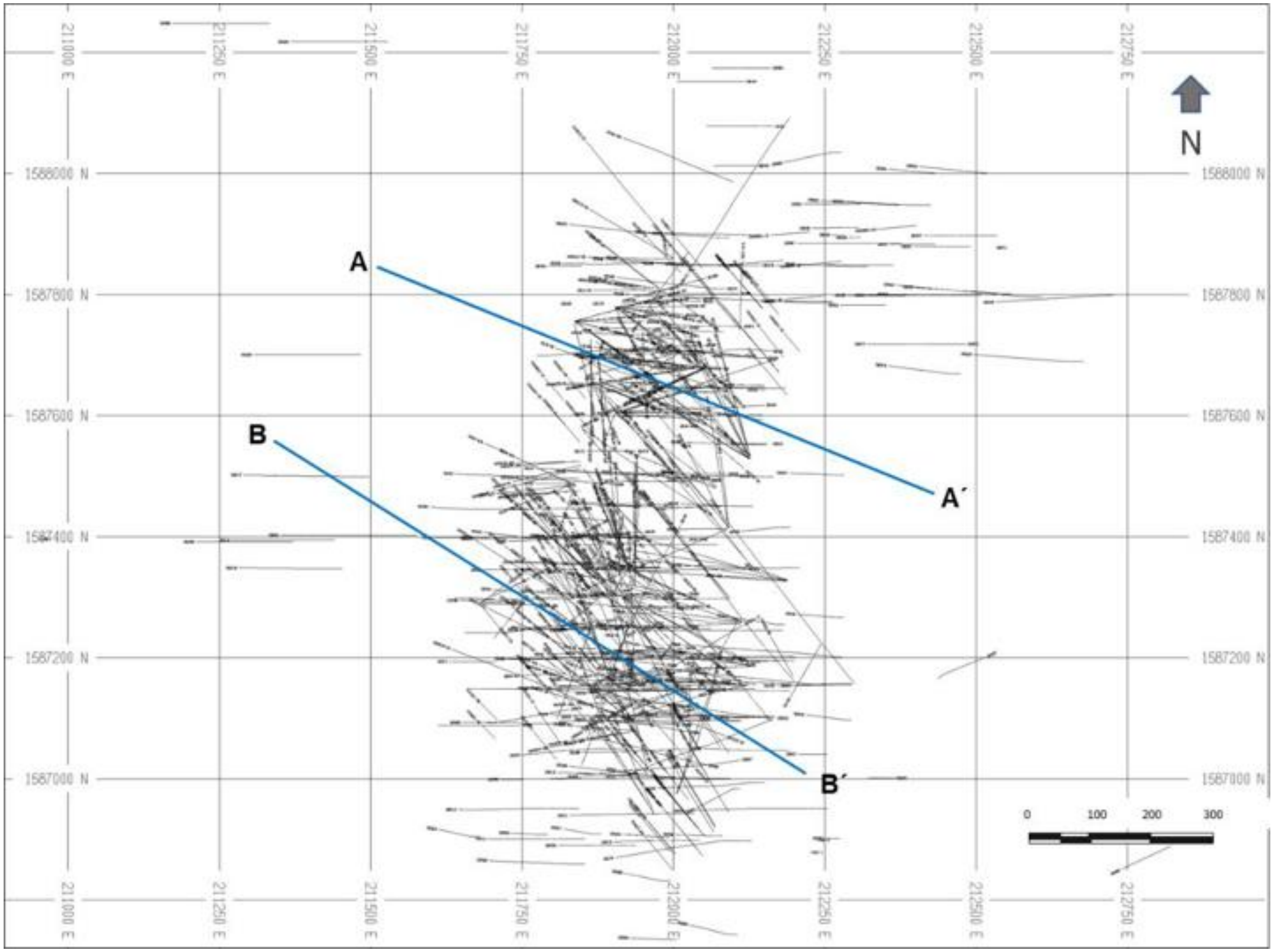

Figure 10-1: Plan view of drill hole locations

Source: Kirkham, 2021.

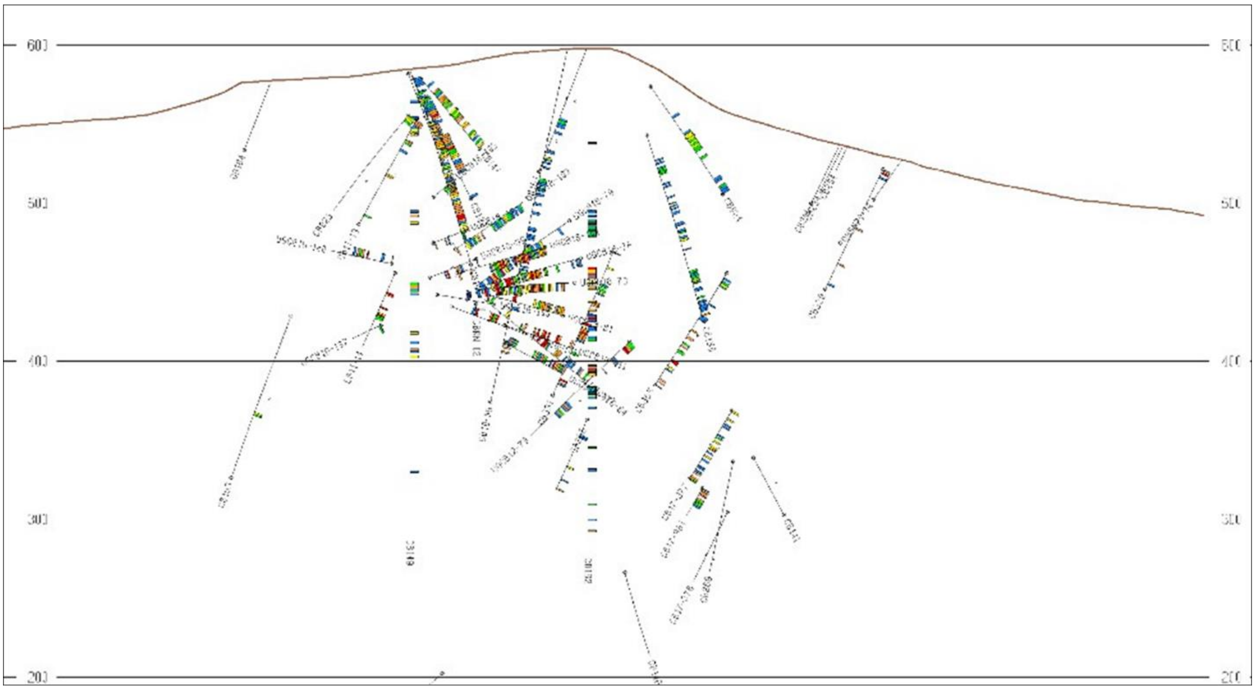

Figure 10-2: Section view A-A' (azimuth 110°)

Source: Kirkham, 2021.

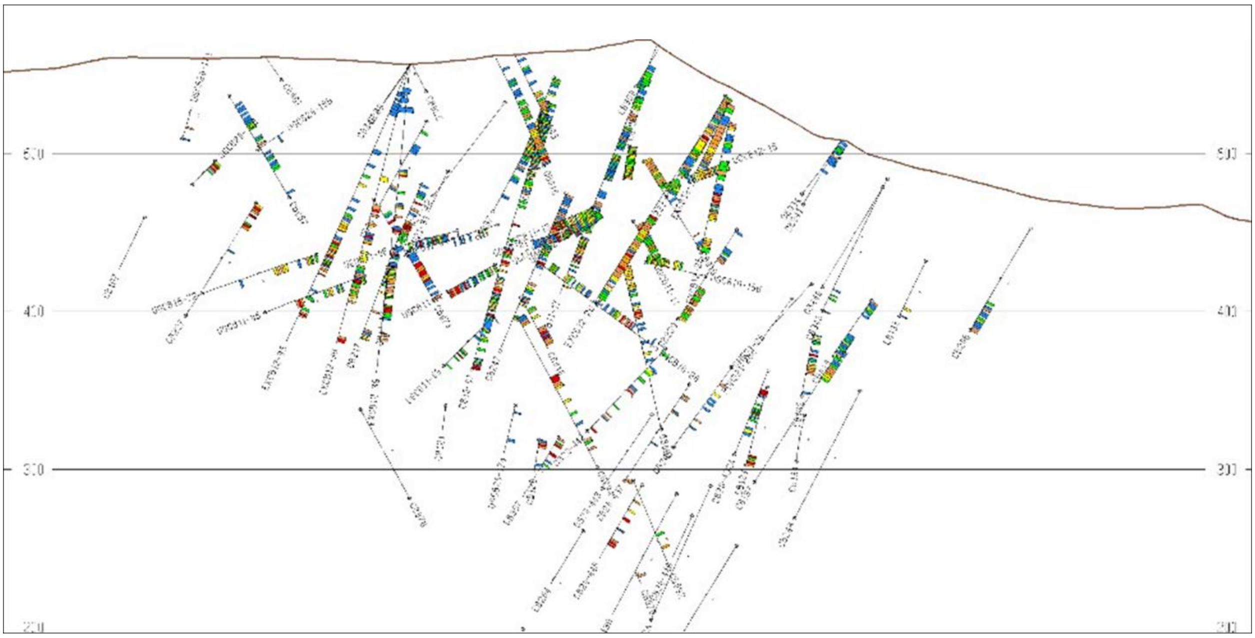

Figure 10-3: Section view B-B' (azimuth 110°)

Source: Kirkham, 2021.

10.1 Goldcorp & Glamis Drilling (Pre-2017)

Prior to Bluestone's ownership, reverse-circulation (RC) and diamond drilling (DD) were 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, the drilled widths of many veins were not representative. The most common vein intersections occur 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).

10.2 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 a 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, 10 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 10-2 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 10-3 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 10-2: Verification 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 10-3: 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 10-4, 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.

External 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. Figure 10-4 illustrates an example of the XY scatterplots used to compare assay results.

Table 10-4: Drill holes 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.

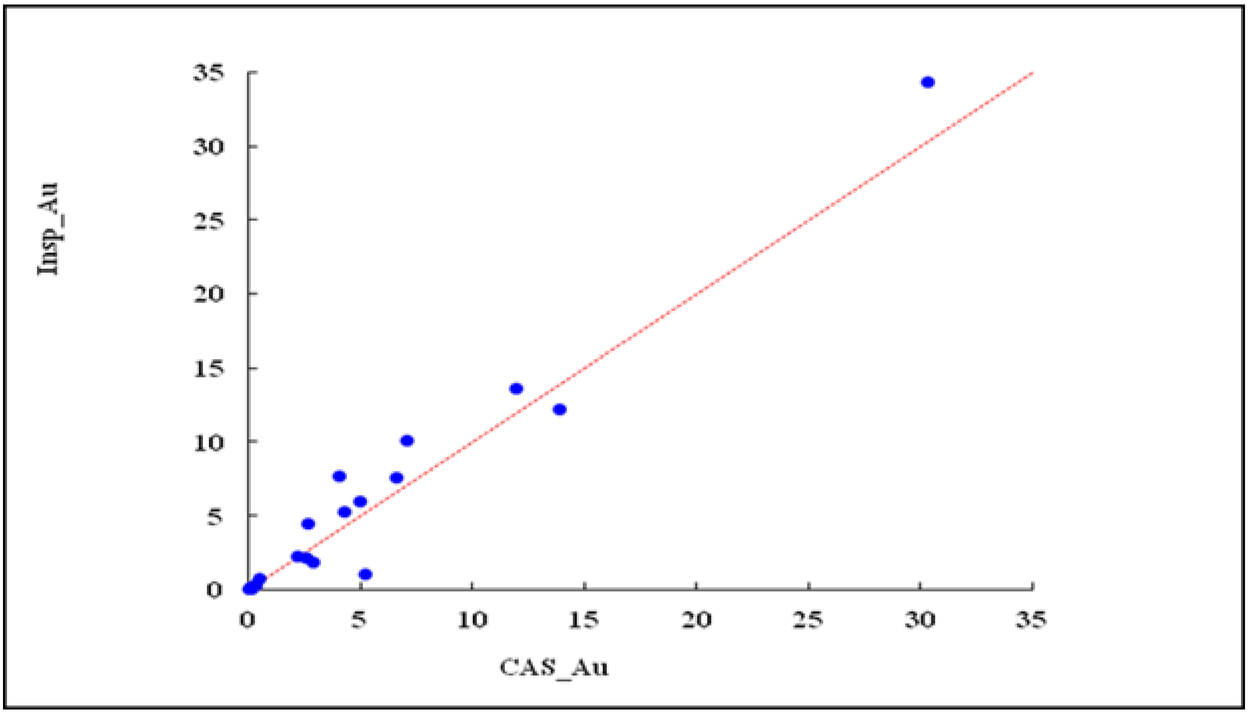

Figure 10-4: Example of XY scatterplot for hole CB34

Source: Goldcorp, 2014.

10.3 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 Mineral 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 program using Reflex ACT III downhole survey equipment.

10.4 Significant Assay Results

Table 10-5 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 10-5: 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
Hole	Company	From	To	Length (m)	Au (g/t)	Ag (g/t)
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
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

Hole	Company	From	To	Length (m)	Au (g/t)	Ag (g/t)
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
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
Hole	Company	From	To	Length (m)	Au (g/t)	Ag (g/t)
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

Hole	Company	From	To	Length (m)	Au (g/t)	Ag (g/t)
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
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
Hole	Company	From	To	Length (m)	Au (g/t)	Ag (g/t)
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
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

Hole	Company	From	To	Length (m)	Au (g/t)	Ag (g/t)
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

Source: Goldcorp, 2014; Bluestone, 2021.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sampling Method & Approach

11.1.1 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 were 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-labelled 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 the 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-labelled 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 laboratory in Tegucigalpa, Honduras, by CAS personnel, and all sample preparation and analyses were conducted at CAS Honduras.

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 1 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 its 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, sulphide 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 sulphide 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 the 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.

11.1.2 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 labelled 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 into MapInfo™ and MineSight™ software for geological modelling.

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 modelling and estimation.

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.

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

The drill core from the surface and underground was stored in labelled wooden boxes (Figure 11-1) at the drill site and transported to the surface core logging facility. Before core splitting and logging commenced, 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 11-1: 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 modelling.

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 modelling. The samples were subject to the same QA/QC protocols as the drill core and were deemed suitable for use in calculating Mineral Resources. Figure 11-2 shows a saw-cut channel sample across a mineralized vein in the South Ramp of the Era Dorada underground workings.

Figure 11-2: 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 ISO 17025-accredited laboratories. Gold and silver were analyzed by a 30 g charge with atomic absorption with a gravimetric finish for values exceeding 5 Au g/t and 100 Ag g/t.

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 MapInfo™ and MineSight™ 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.

11.2 Quality Assurance & Quality Control

11.2.1 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 Mineral 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.

11.2.2 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 11-1) were assigned for QA/QC purposes, accounting for approximately 20% of the total samples taken during the program.

Table 11-1: 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 11-2) and inserted by the logging geologists.

Table 11-2: 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 is 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 11-3 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 11-3 shows a control plot for standard CDN-GS-6E.

Table 11-3: 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.

Source: Bluestone, 2020.

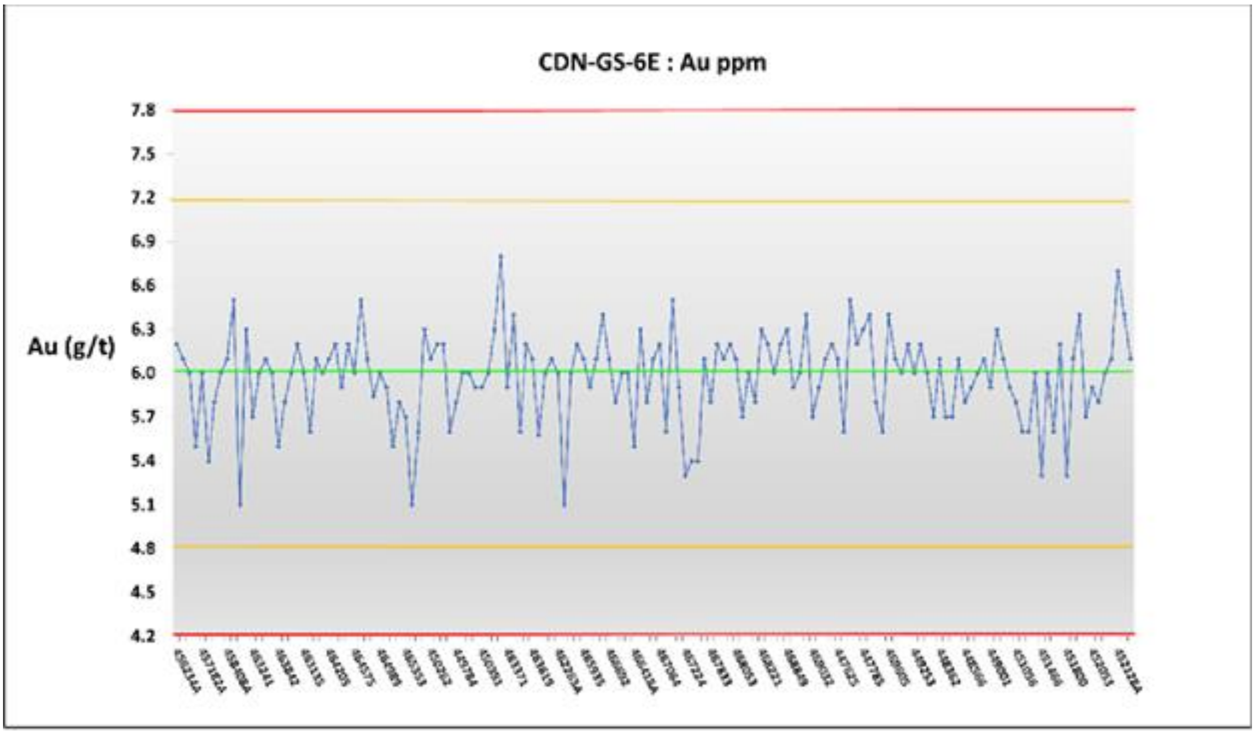

Figure 11-3: Batch plot of standard CDN-GS-6E

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 Au g/t) utilized early in the Bluestone drill program plotted consistently along the highest acceptable threshold for fire assay with instrumental finish. The 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 11-4, 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 11-5) consistently yielded gold values near or below the detection limit of the primary laboratory. No sample contamination was detected.

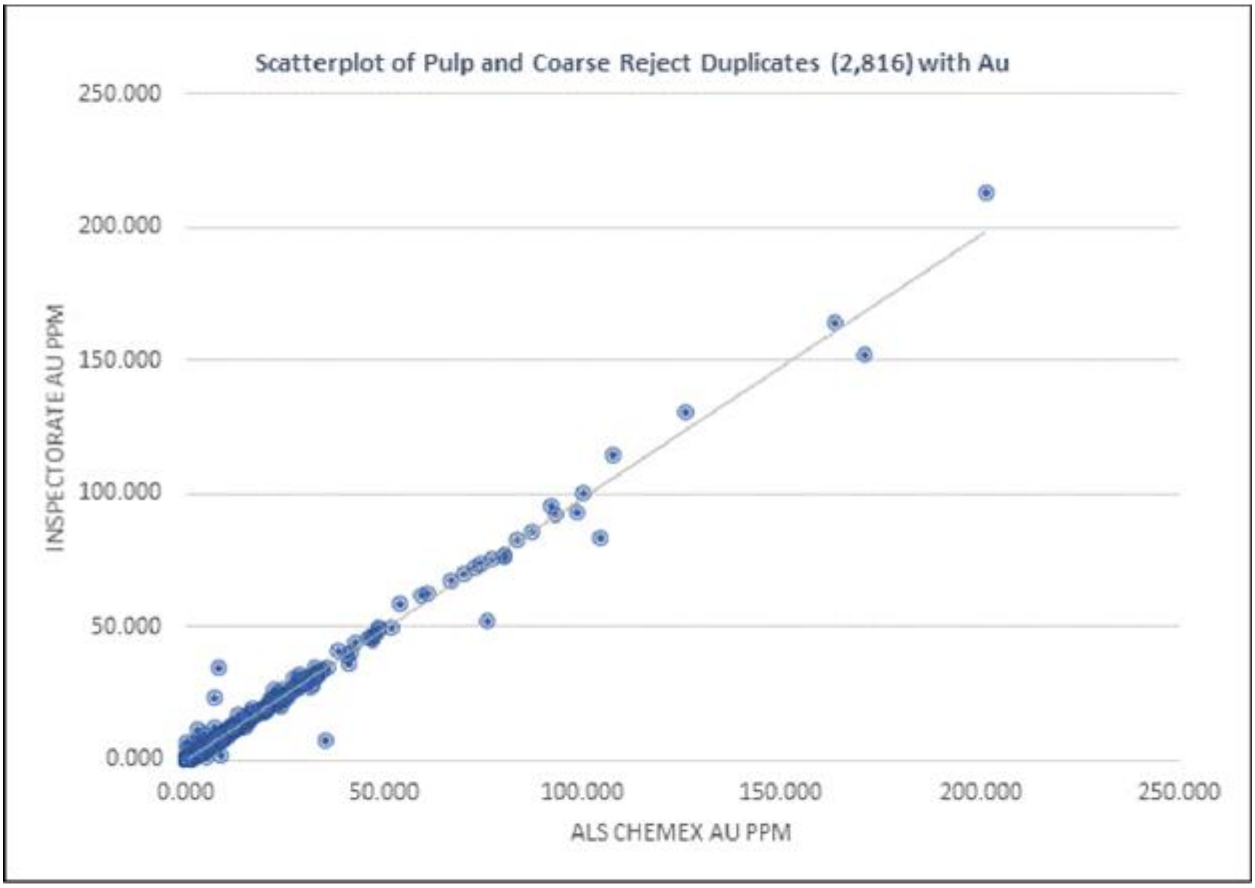

Figure 11-4: Plot of pulp & coarse reject duplicates (Bluestone 2017-2021)

Source: Bluestone, 2021.

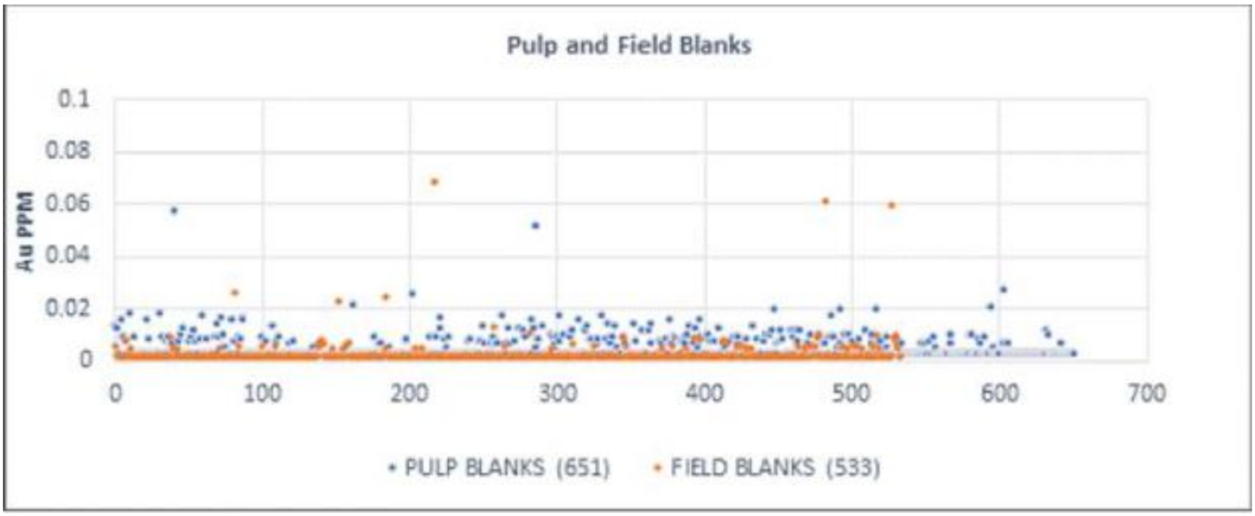

Figure 11-5: Pulp & field blanks (Bluestone 2017 to 2021)

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 Mineral 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.

12 DATA VERIFICATION

A geological 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 the Canadian Institute of Mining and Metallurgy is to fulfill the requirements specified under NI 43-101 and to familiarize oneself with the property, 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.

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.

12.1 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 Mineral 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.

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 modelling 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 modelling.

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 modelling.

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 Mineral 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.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Various metallurgical testing campaigns were conducted on Era Dorada (formerly named "Cerro Blanco") samples by Kappes, Cassiday & Associates (KCA) between 1999 and 2012, together with additional testing carried out by SGS Lakefield Research Ltd., Carson GeoMIn Inc., Pocock Industrial Inc., Phillips Enterprises Inc., and CyPlus GmbH. The most recent test program was completed in 2018 and was carried out at Base Metallurgical Laboratories Ltd. (BaseMet) in Kamloops, BC. The following reports include all metallurgical testing programs carried out so far on Era Dorada samples.

- **Kappes, Cassiday & Associates (1999):** "Cerro Blanco Project, Results of Cyanide Leach Tests" (Issued: April 8, 1999).
- **Kappes, Cassiday & Associates (2000a):** "Cerro Blanco Project, Results of Cyanide Bottle Roll Tests" (Issued: January 12, 2000).
- **Kappes, Cassiday & Associates (2000b):** "Cerro Blanco Project, Bottle Roll Tests" (Issued: August 24, 2000).
- **Kappes, Cassiday & Associates (2002):** "Cerro Blanco Project, Results of Leaching Tests and Gravity Concentration Tests" (Issued: 8 October 2002).
- **SGS Lakefield Research Ltd. (2005):** "Cerro Blanco North Zone Samples for Met Testing at SGS Lakefield" (Issued: August 2005).
- **Kappes, Cassiday & Associates (2005):** "Cerro Blanco Project" (Issued: December 15, 2005).
- **Carson GeoMIn Inc. (2005):** "Mineralogy of Ore Composites and Related Cyanide Tailings from the Cerro Blanco Gold Project" (Issued: December 29, 2005).
- **Kappes, Cassiday & Associates (2006a):** "Cerro Blanco Project" (Issued: January 18, 2006).
- **Kappes, Cassiday & Associates (2006b):** "Cerro Blanco Project" (Issued: April 21, 2006).
- **Phillips Enterprises LLC (2011):** "Comminution Tests, Cerro Blanco" (Issued: June 11, 2011).
- **Pocock Industrial Inc. (2011):** "Sample Characterization and PSA, Flocculant Screening, Gravity Sedimentation, Pulp Rheology, Vacuum Filtration and Pressure Filtration Studies Conducted for Kappes, Cassiday & Associates Cerro Blanco Project" (Issued: October 2011).
- **Kappes, Cassiday & Associates (2012):** "Cerro Blanco Project, Report of Metallurgical Test Work, January 2012" (Issued: January 25, 2012).
- **Base Metallurgical Laboratories Ltd. (2018):** "BL0246: Generation of Cyanide Detox Tailings – Cerro Blanco Project" (Issued: August 3, 2018).

The following sections show the selected reports and respective testing results for designing the recovery method and equipment for the Project.

13.2 Selected Test works

13.2.1 KCA (2012) – Leach Tests

The six pallets received by KCA in April 2011 included a total of 55 cloth bags containing half-split HQ and PQ drilling core material from five samples. These five samples were assayed, together with a master composite sample. The latter was assembled on the basis of the same five samples. The head assay results of both Au and Ag are summarized in Table 13-1.

Table 13-1: Head assays

KCA Sample No.	Description	Average Au Assay (g/t)	Average Ag Assay (g/t)
48901	MbT	9.4	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	7.7	37.86

Source: KCA, 2012.

The experimental program included bottle roll leach testing on the Master Composite sample according to different grinding sizes. The summarized testing conditions and gold extraction results are listed in Table 13-2, which indicates that for P_{80} (80% passing) equal to or finer than 0.085 mm, the overall gold extractions ranged from 92% to 94%.

Table 13-2: Gold extraction summary

KCA Sample No.	KCA Test No.	P_{80} – Milled Size (µm)	Target NaCN (g/l)	Calculated Head – Au (g/t)	Gold Extracted – Au (%)	Leach Time (h)
48907	48914 A	160	1.0	6,537	86	96
48907	48913 A	138	1.0	6,328	91	96
48907	48913 B	85	1.0	5,822	93	96
48907	48913 C	77	1.0	6,497	94	96
48907	48914 B	76	1.0	5,887	92	96
48907	48913 D	72	1.0	5,628	94	96
48907	48916 C	71	1.0	5,788	92	96
48907	48915 B	57	1.0	6,124	93	96
48907	48916 B	55	1.0	8,663	94	96
48907	48917 B	44	1.0	5,169	92	96
Average		—	1.0	6,244	92.1	96

Source: KCA, 2012.

13.2.2 Phillips Enterprises (2011) – Comminution Tests

The same five samples used in the KCA testing campaign listed in Section 13.2.1 were split and used for comminution testing at Phillips Enterprises LLC in Golden, Colorado. Comminution testing included the Bond Work index for the ball mill (BWi), Bond Work index for the rod mill (RWi), as well as Bond Abrasion index (Ai). The results obtained are summarized in Table 13-3.

Table 13-3: Comminution test results

KCA Sample No.	Sample Description	Bond Rod Mill Work Index - BWi (kWh/t)	Bond Ball Mill Work Index - RWi (kWh/t)	Bond Abrasion Index - BAi
48901	MbT	17.08	20.27	0.193
48902	Mcv	13.91	16.37	0.104
48903	Svc	18.26	22.24	0.328
48904	Msc	16.90	21.45	0.329
48905	Cbx	15.52	18.95	0.246
Average		**16.33**	**19.86**	**0.240**

Source: Phillips Enterprises, 2011.

13.2.3 Pocock Industrial (2011) – Solid / Liquid Separation Tests

Solid/liquid separation tests were conducted on “Fresh Milled” and “Leached and Detoxed” samples as resulting from the KCA test work previously described in Section 13.2.1. The program included testing for supporting solid/liquid separation equipment design and sizing. All testing was conducted by Pocock Industrial at their laboratory facilities located in Salt Lake City, Utah, in October 2011.

The summary of solid/liquid separation testing was as follows.

- The flocculant concentration varied by individual sample and thickener type or application, but was in the overall range of 35 to 55 g/t for the Fresh Milled sample, as well as 30 to 55 g/t for the Leached and Detoxed sample within the tested pH range.
- For conventional thickener sizing, Pocock recommended a minimum unit area design basis of 0.30 to 0.40 m^2/tpd for the Fresh Milled and 0.25 to 0.35 m^2/tpd for the Leached and Detoxed material.
- 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 both the Fresh Milled and Leached and Detoxed samples for high performance.
- The overall maximum underflow density range for the Leached and Detoxed material was 53 to 57%. However, the range was narrowed to 53 to 55% with rake torque considerations based on unsheared data.
- The designed pressure filter for a stipulated 1,250 tpd plant throughput resulted in a minimum requirement of 190 chambers for both Leached and Detoxed samples, a horizontal recess plate type press, equipped with 1,500 mm plates and 15 mm recess (30 mm full chamber) with no cake wash to achieve 18.3% moisture. The alternative design indicated equipment with 181 chambers to achieve 18.9% moisture with pH adjusted to 10.5.

13.2.4 BaseMet (2018) – Chemical Assays

The 2018 testing campaign carried out at BaseMet was based on the following two samples.

- Approximately 90 kg of half and quarter-cut drill core.
- About 590 kg of bulk rock, consisting of 180 individual intervals.

Each drill core interval was stage crushed to 3.36 mm. The crushed material was blended and split to obtain a representative sub-sample of the Global Composite. The head assay results are shown in Table 13-4.

Table 13-4: Head assays

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

Source: BaseMet, 2019.

13.2.5 BaseMet (2018) – Gravity Concentration

The same testing campaign described in Section 13.2.4 comprised gravity concentration testing. Accordingly, sub-samples of the Global Composite sample were ground in a laboratory rod mill to three targeted P_{80} grind sizes of 0.050 mm, 0.075 mm and 0.100 mm. Each one of these three samples was tested at a Knelson MD-3 centrifugal gravity concentrator. Knelson concentrates were panned, targeting a 0.1% to 0.5% mass recovery. Gravity concentration results shown in Table 13-5 indicate average recoveries of 19.1% Au and 7.8% Ag.

Table 13-5: Gravity concentration results

Test No.	Grind Size (µm)	Mass Recovery (%)	Au Recovery (%)	Ag Recovery (%)
4	50	0.317	22.5	6.3
10	50	0.186	21.1	6.9
11	50	0.23	15.1	6.5
17	53	0.319	20.8	9.4
18	53	0.301	17.9	8.5
19	53	0.274	16.7	6
20	53	0.326	21.7	9.1
21	75	0.185	16.3	5.4
2	75	0.239	29.8	16.5
6	75	0.27	14.7	5.4
7	75	0.314	20.7	6.4
8	75	0.398	20	6.6
3	75	0.48	17.7	10.2
12	75	0.291	15.9	4.1
5	100	0.534	15.6	10.2
Average		**0.311**	**19.1**	**7.8**

Source: BaseMet, 2019.

13.2.6 BaseMet (2018) – Leach Tests

Leaching test work was also carried out in the 2018 BaseMet testing campaign. In this case, the tests consisted of direct cyanide leaching on two samples, i.e., fresh milled product and gravity tailings. All tests were conducted in closed rolling bottles with monitoring and control of cyanide level, dissolved oxygen (DO), and pH. Sampling for kinetic assessments was conducted during test periods of 2, 6, 24, 48, and 72 hours. The leach test results are summarized in Table 13-6. The average recoveries were 93.24% Au for 24 hours of residence time and 94.96% Au for 72 hours of residence time.

Table 13-6: Bottle roll leach results

Test No.	Grind Size (µm)	Reagent Consumption		Gravity Au Recovery (%)	Cumulative Gold Extraction (%)				Final Gold Recovery (%)	Final Ag Recovery (%)
		NaCN (kg/t)	Lime (kg/t)		2 h	6 h	24 h	48 h	72 h	72 h
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
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
Average		**0.76**	**0.96**	**19.10**	**73.06**	**84.81**	**93.24**	**93.94**	**94.96**	**81.07**

Source: BaseMet, 2019.

According to the results listed in Table 13-6, finer grinding sizes resulted in higher Au and Ag recoveries.

Figure 13-1 shows a graph of gold extraction as a function of the leaching period (residence time) for different grind sizes. The plotted curves include gravity recovery. The average gold extraction for a P_{80} grind size of 0.050-0.053 mm was 95% for 24 hours, as well as 97% for 72 hours.

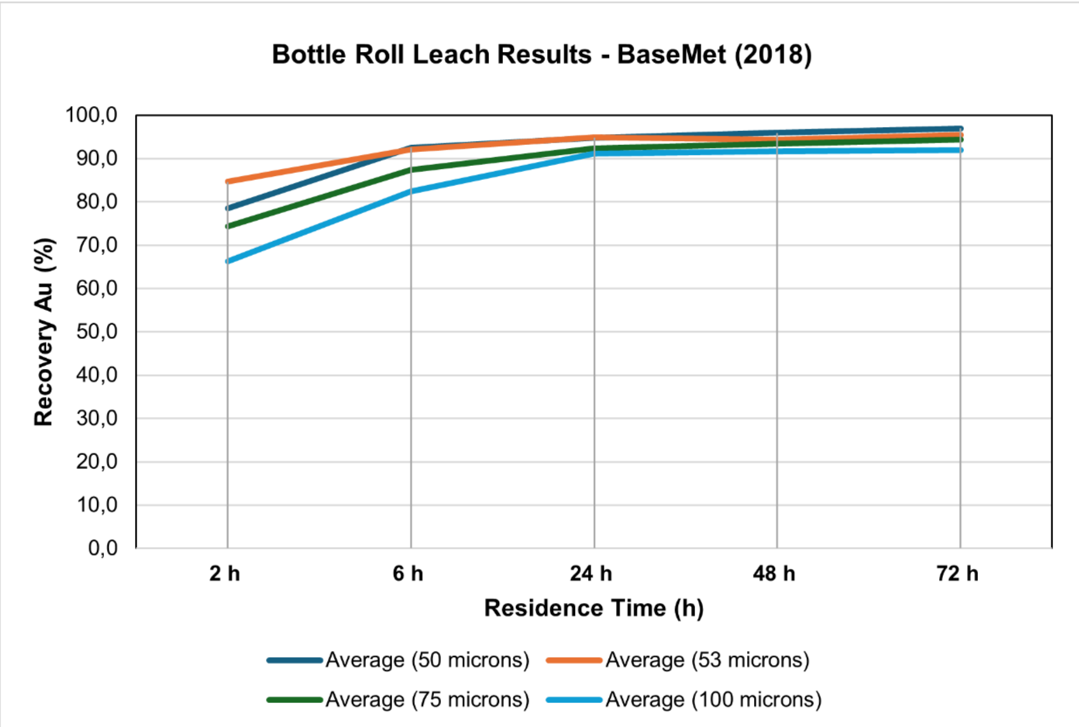

Figure 13-1: Effect of grind size on average gold extraction

Source: Author; BaseMet, 2019.

One of the Composite samples previously listed in Section 10.2.4 was further tested to determine the adsorption of Au and Ag on carbon and, therefore, the basis for a future Carbon in Pulp (CIP) circuit. The conditions for CIP testing were as follows:

- Sample grinding: P_{80} of 0.053 mm.
- Pulp Density: 33% solids (w/w).
- Pulp pH: 10.5 to 11 maintained with lime.
- Cyanide Concentration: 0.5 g/l NaCN maintained.
- Retention Time: Total 54 h (48 h leach, 6 h carbo1n absorption).
- Carbon Concentration: 25–50 g/l.
- Lead Nitrate Concentration: 0–250 g/t.

The recoveries obtained for Au and Ag are indicated in Table 13-7. Tests carried out at 50 g/l carbon concentration (T-25, T-26, and T-27) resulted in higher Au and Ag recoveries. It was also observed that the addition of lead nitrate improved Ag recovery (T-26 and T-27).

Table 13-7: Bottle roll leach results (CIP)

Test ID	Grinding P_{80}	Total Leaching Time	NaCN	Consumption $PbNO_3$	Carbon Concentration	Au Calculated	Gravity Au	Extraction (%)	
	(µm)	(h)	(g/l)	(kg/t)	(g/l)	(g/t)	(%)	Au	Ag
T-21	53	54	0.5	0	25	6.95	16.3	94.7	67.2
T-25	53	54	0.5	0	50	6.95	33.2	97.1	81.1
T-26	53	54	0.5	250	50	6.64	28.6	97.3	90.0
T-27	53	54	0.5	250	50	6.59	25	96.6	85.4

Source: BaseMet, 2019.

The Au recovery figures listed in Table 13-7 were plotted in a graph, as shown in Figure 13-2. Overall, Au recovery for a 36-hour leaching period was estimated as 96% (Tests T-25, T-26, and T-27).

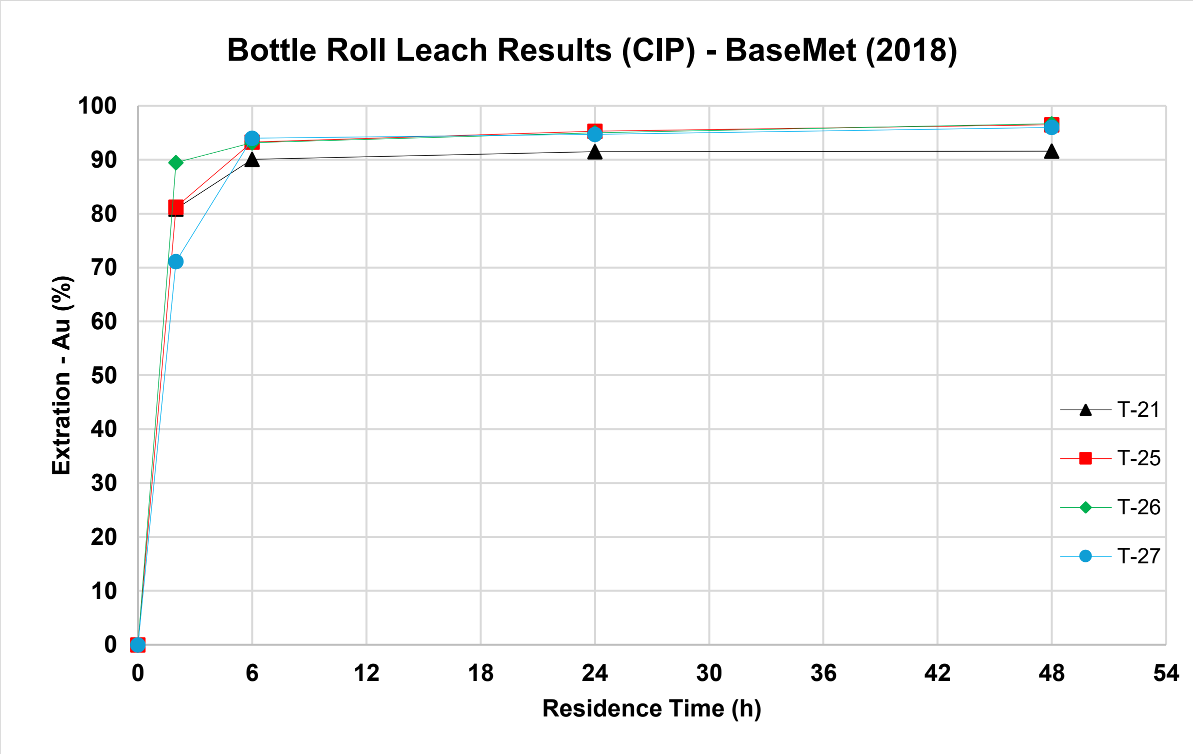

Figure 13-2: Gold recovery as a function of residence time (CIP)

Source: BaseMet, 2019.

13.2.7 BaseMet (2018) – Cyanide Destruction Tests

A series of continuous cyanide destruction tests were conducted to assess the efficiency of the selected detox process for the leaching test tailings.

Samples from leach tests described in Section 10.2.6 were filtered, and the resulting solution was analyzed. The results indicated 283 mg/l total cyanide (CN_T) and 270 mg/l weak acid dissociable cyanide (CN_{WAD}).

The SO_2/air method was assessed for destructing cyanide contained in leached tailings. Such a method, also known as the Inco or Detox method, uses sulphur dioxide (SO_2) to remove weak acidic dissociable cyanide down to concentrations smaller than 5 mg/l.

The cyanide pulp resulting from leach tests performed adequately to the SO_2/Air cyanide destruction process, resulting in final tailings with less than 1 mg/l CN_{WAD}, as well as less than 4 mg/l total cyanide (CN_T). The results shown in Table 13-8 indicate that the Detox tests resulted in CN_{WAD} concentrations smaller than 5 mg/l in the final tailings.

Table 13-8: Cyanide destruction results

Test No.	Retention Time (min)	pH	Reagent Addition (g / g CN_{WAD})		Cu (mg/l of solution)	Final Solution Composition	
			SO_2 Equivalent	Lime		CNT (mg/l)	CN_{WAD} (mg/l)
CND-C1	90	8.5	7	4.6	100	4.59	1.8
CND-C2	90	8.5	5.5	2	100	2.96	0.17
CND-C3	90	8.3	4	2.2	100	0.49	0.22
CND-C4	90	8.4	4	1.6	50	2.94	0.14
CND-C5	90	8.5	4	1.4	25	3.02	0.24
CND-C6	90	9	4	-	0	18.3	4.18
CND-C7	60	8.5	4	0.8	25	3.56	0.48

Source: BaseMet, 2019.

13.3 Summary and Conclusions

The summary and main conclusions of the selected test work carried out on Era Dorada samples were as follows.

- **Comminution**: Bond Ball Mill Work index (BWi) of 19.9 kWh/t and a Bond Abrasion index (Bai) of 0.24.
- **Grinding size:** P_{80} of 0.053 mm.
- **Gravity Concentration:** Based on the gravity test results, a gravity concentration circuit was included in the grinding circuit.
- **Leach Results:** Oxygen pre-oxidation should be incorporated into the process design, as a significant impact was observed during the first 24 hours of leaching. The recommended residence time of the pre-oxidation stage was 2 hours at a targeted cyanide concentration of 500 ppm.
- **Leach Results (CIP):** The results and test work parameters obtained from the four leach tests were used to develop the process design criteria and estimated Au and Ag recoveries for the leach/CIP circuits.
- **Cyanide Destruction:** The conditions used in tests CND-C7 were adopted as process design for the cyanide destruction circuit for reducing the CN_{WAD} concentration to less than 1 mg/l.
- **Leach Results:** The relatively small increase in Au recovery for a leaching period of 72 hours, as compared with the 36-hour leaching period, resulted in the adoption of the latter for the Era Dorada industrial plant. Detailed figures are listed in Table 13-9. The leached slurry will then flow through a carbon-in-pulp (CIP) circuit for the adsorption of the Au and Ag cyanide complexes onto the pores of activated carbon. The loaded carbon will be processed through desorption and refining circuits. The adopted Au and Ag recoveries should be 96% and 85%, respectively.

Table 13-9: Preliminary recovery estimative

Test ID	Residence Time 54 h Recovery (%)		Residence Time 36 h Estimated Recovery (%)
	Au	Ag	Au
T-25	97.1	81.1	95.8
T-26	97.3	90.0	95.8
T-27	96.6	85.4	95.4
Average	97	86	96

Source: Author; BaseMet, 2019.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

This section describes the work undertaken by Kirkham Geosystems Ltd (KGL), including key assumptionsand 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-sulphidation 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 Au g/t and 52 Ag g/t). Bonanza gold grades are associated with ginguru banding and carbonate replacement textures. Sulphide contents are low, typically < 3 volume %.

The Mita rocks are overlain by the Salinas unit, a sub-horizontal sequence of volcanogenic sediments andsinter horizons approximately 100 m thick that form the low-lying hill at the project. Low-grade disseminatedand 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 Au g/t. 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 infrastructureallowed for underground mapping, sampling, and over 30,000 m of underground drilling that enhanced thecurrent understanding and validation of the Era Dorada geological model. The Mineral Resource Estimateis based on a scenario that considers open pit mining methods and, therefore, requires improved and refinedgeological models of the lithologic units. 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 modelsand estimation strategy were designed to accurately represent the grade distribution.

Several Mineral 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)

The first three reports and Mineral Resource Estimates were for an underground mining scenario. The last two Mineral Resource Estimates were for the open pit scenario. All estimates were authored by Qualified Person, Garth Kirkham, P.Geo.

All five technical reports are filed on the System for Electronic Document Analysis and Retrieval (SEDAR+).

14.2 Data

The drill hole database was supplied in electronic format (i.e., Microsoft Excel and Access) by Bluestone. 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 14-1). Figure 14-1 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 14-1: Lithology units & 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
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.

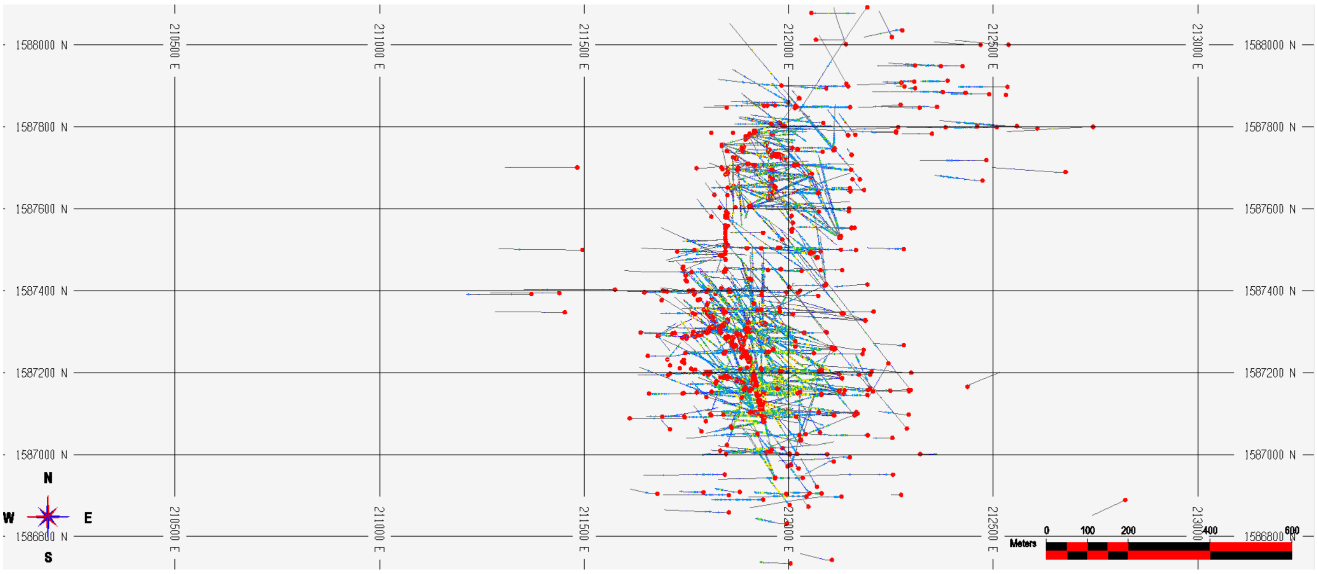

Figure 14-1: Plan view of drill holes

Source: Kirkham, 2025.

14.3 Data Analysis

Table 14-2 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 varies depending on the parameter being analyzed 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 14-2: Statistics for weighted gold & 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 14-3 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 Au g/t and 38.92 Ag g/t. The coefficient of variation is relatively high, with 3.3 for gold and 4.0 for silver. These are reviewed once compositing and cutting are 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, which 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 treated within the domains, they will be cut appropriately to ensure that they reasonably represent the estimated grades.

Table 14-3: Statistics for weighted gold & silver assays for quaternary and cross-cutting rock types

Code	Lith	Code	Metal	Valid	Length (metres)	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
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 14-4: Statistics for weighted gold & silver assays for the Salinas Group rocks

Code	Lith	Metal	Valid	Length (metres)	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 14-4 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 modelling 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 in Figure 14-2. Therefore, observations and statistical analysis support the resultant domaining for the Salinas of the Sinter, Basal Conglomerate and the remaining sedimentary units with the Volcanic Sediments (Svc) as the predominant rock type.

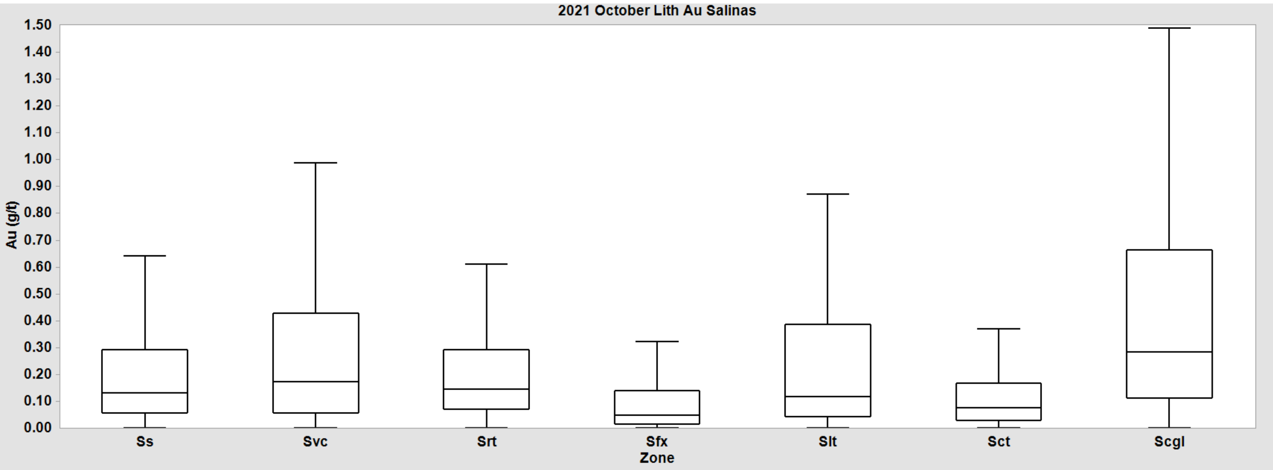

Figure 14-2: Box plot gold assays for the Salinas Group rocks

Source: Kirkham, 2025.

Table 14-5: Statistics for weighted gold & silver assays for the Mita Group rocks

Code	Lith	Metal	Valid	Length (metres)	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
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 14-5 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 modelling 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 14-3 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 14-3. Further analysis and modelling for the purpose of grouping and domaining take these observations and conclusions into account.

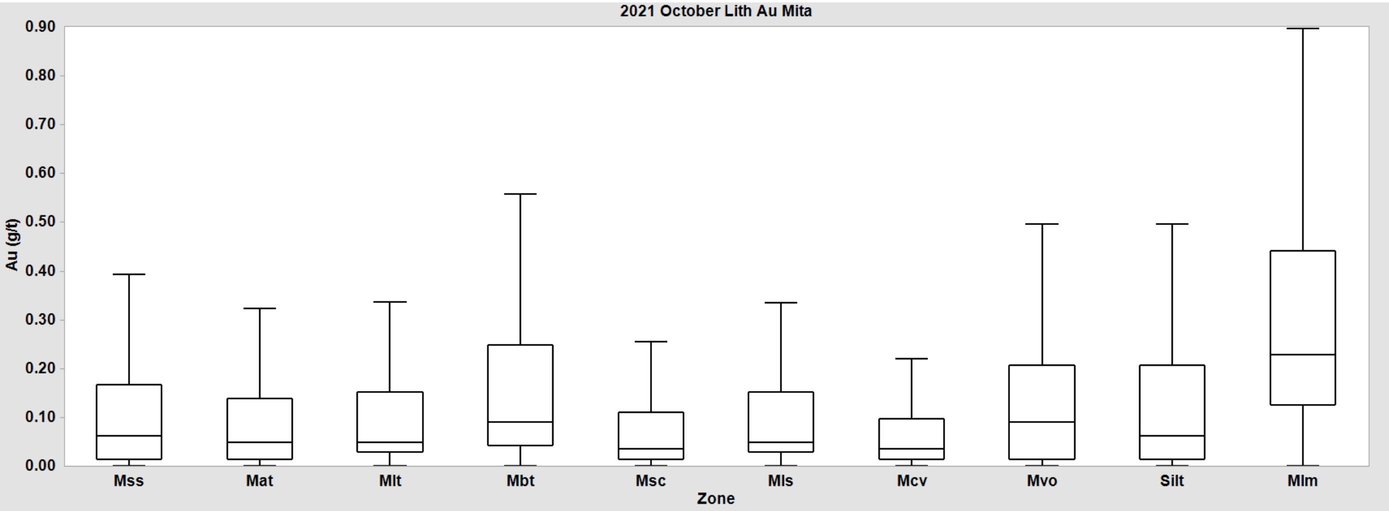

Figure 14-3: Box plot gold assays for the Mita Group rocks

Source: Kirkham, 2025.

Table 14-6: Statistics for weighted gold & silver assays

Code	Lith	Metal	Valid	Length (metres)	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 14-6 above shows intervals that intersect the Tempisque Volcanic Complex are primarily treated as waste.

14.4 Geology & Domain Model

A three-phased modelling approach was taken to creating geology and estimation domains, which includeda lithostratigraphic model, detailed vein modelling, and domain modelling 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 14-4.

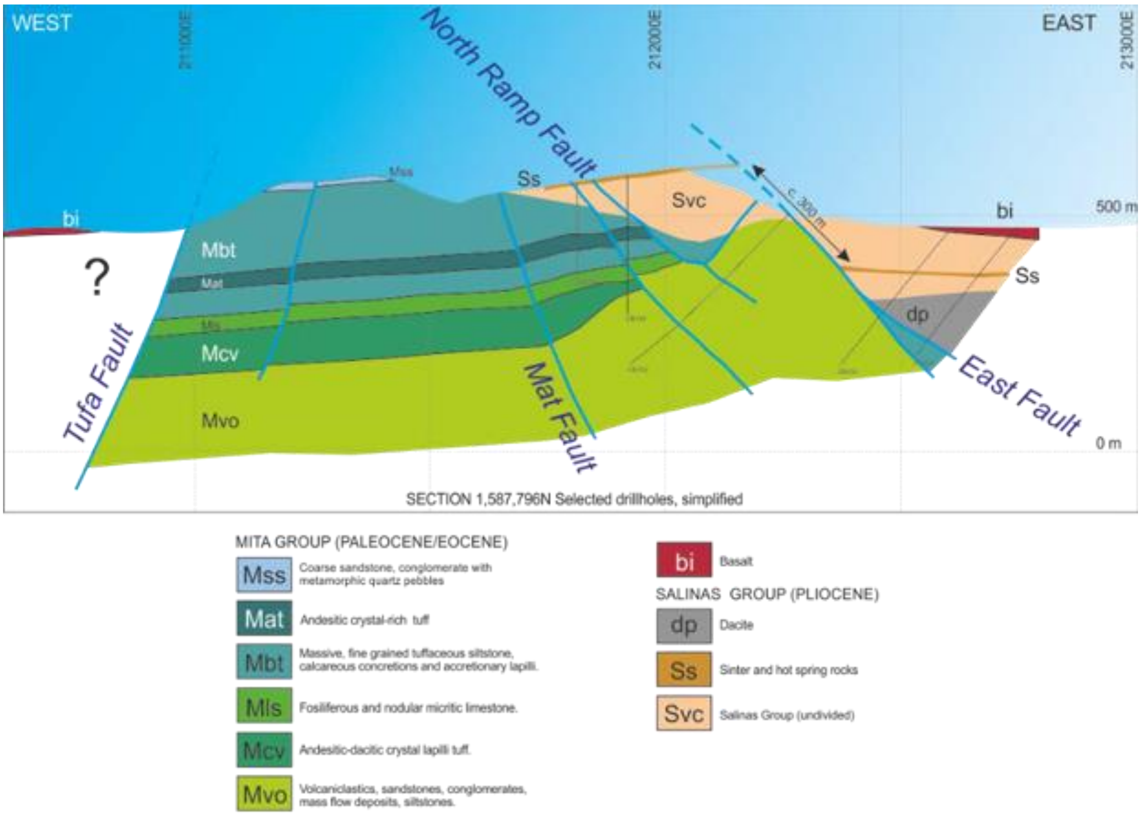

Figure 14-4: Section view schematic of lithology for the Era Dorada Deposit

Source: Kirkham, 2025.

The models were created from the first principals within LeapFrog™ and refined in MineSight™ for statistical analysis and to be used for the estimation process. Figure 14-4 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 modelling 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 the plan in Figure 14-5, with the north veins in yellow and the south veins in blue, respectively. In preparation for the creation of thevein models, a comprehensive structural model was developed that incorporated the current drilling, underground sampling, mapping, and extensive re-logging of the drill core. The models were also created from firstprinciples using the lithostratigraphic models and the structural modelling as guides by Bluestone staff within LeapFrog™ under the supervision of the independent QP. This was done utilizing 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 utilized to constrain the block model by matching assays to those within the zones.

The orientation and ranges (distances) utilized for the search ellipsoids used in the estimation process were omnidirectional 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 14-5. 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.

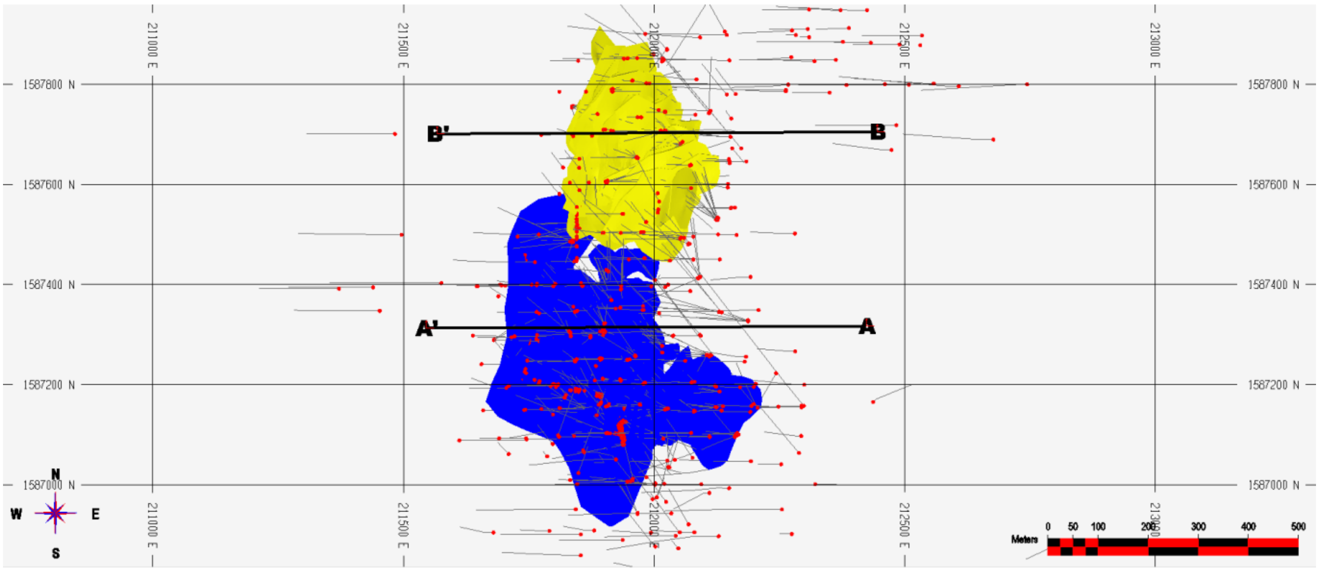

Figure 14-5: Plan view of drill holes & vein solids

Legend: 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 modelled as Salinas, Sinter, Basal Conglomerate. Within the Mita Group, the moderately mineralized volume that envelops the North and South vein clusters are predominantly the Mbt and Mcv units.

Figure 14-6 and Figure 14-7 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 to accurately account for the low- and high-grade components of each block along with the waste.

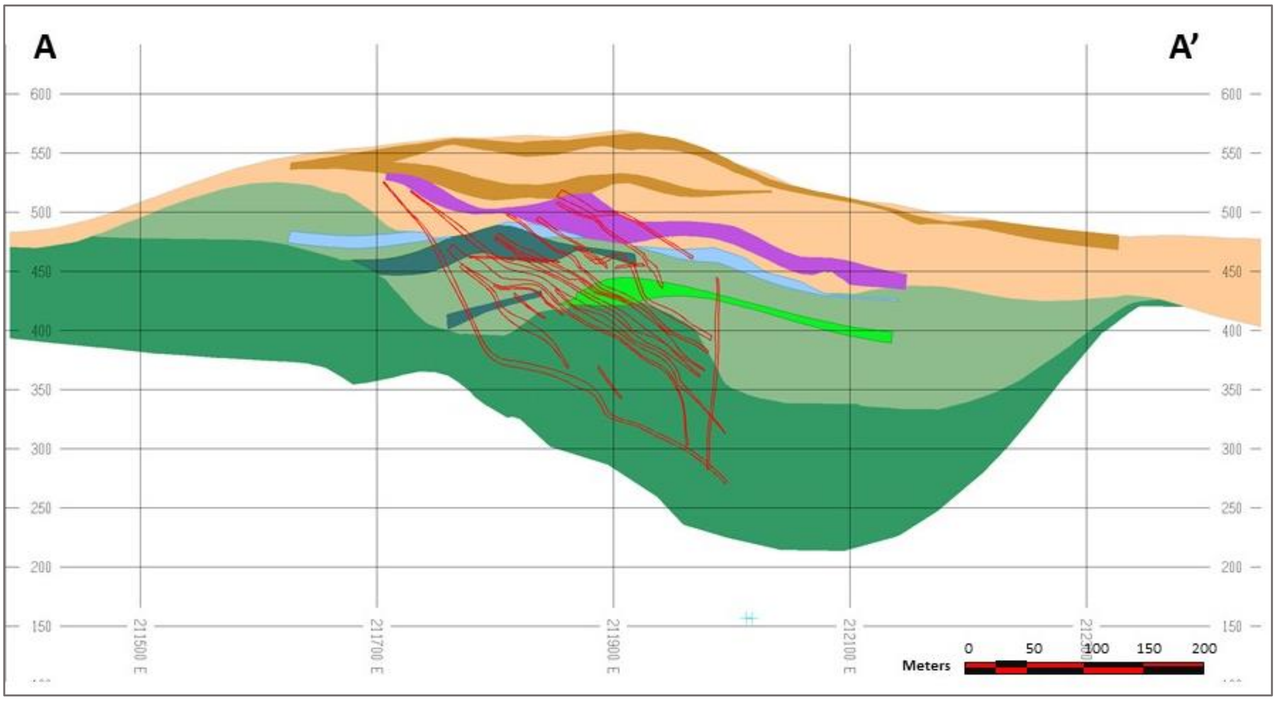

Figure 14-6: South area section A-A' view of drill holes, vein solids with Salinas and Mita Units

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.

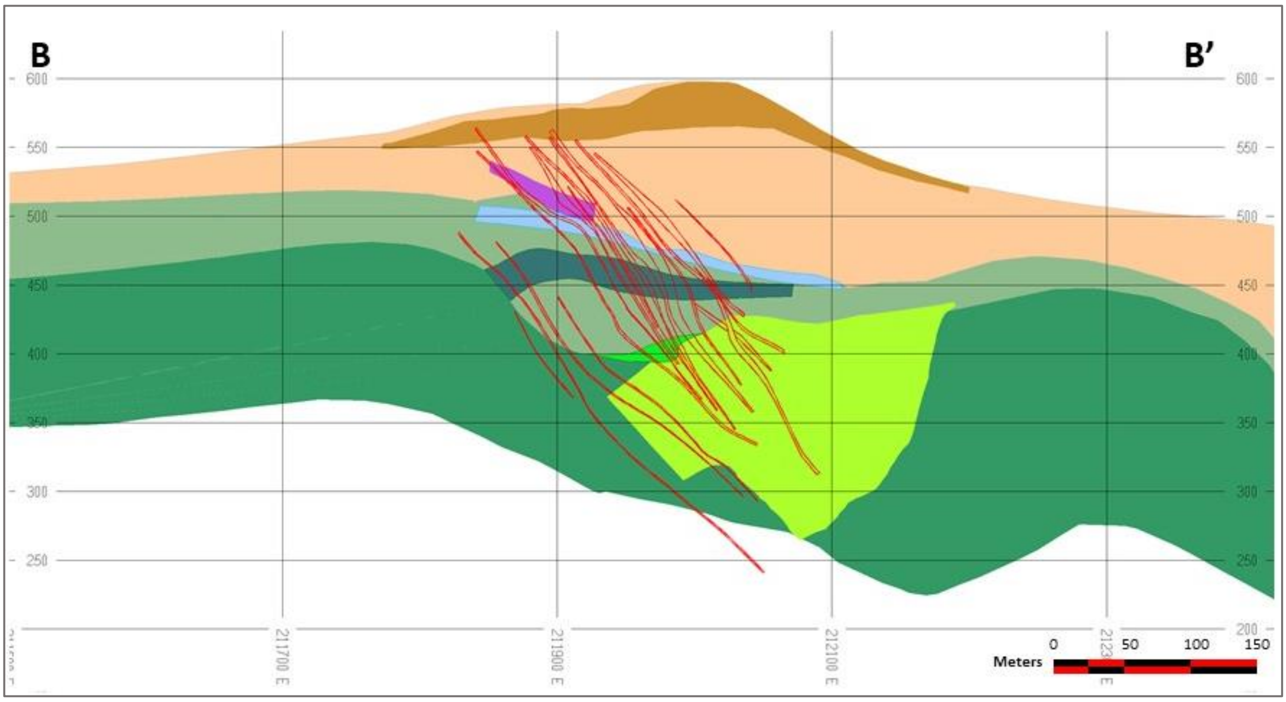

Figure 14-7: North area B-B' section view of vein solids with Salinas and Mita Units

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.

14.5 Composites

It was determined that the 1.5 m composite lengths offered the best balance between supplying common support for samples and minimizing the smoothing of grades. Figure 14-8 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 14-9 shows the histogram of 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 14-10 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 was also consistent with the distribution of sample lengths. It should be noted thatalthough 1.5 m is the composite length, any residual composites of greater than 0.75 m in length and lessthan 1.5 m remained to represent a composite, while any composite residuals less than 0.75 m were combined with the composite above.

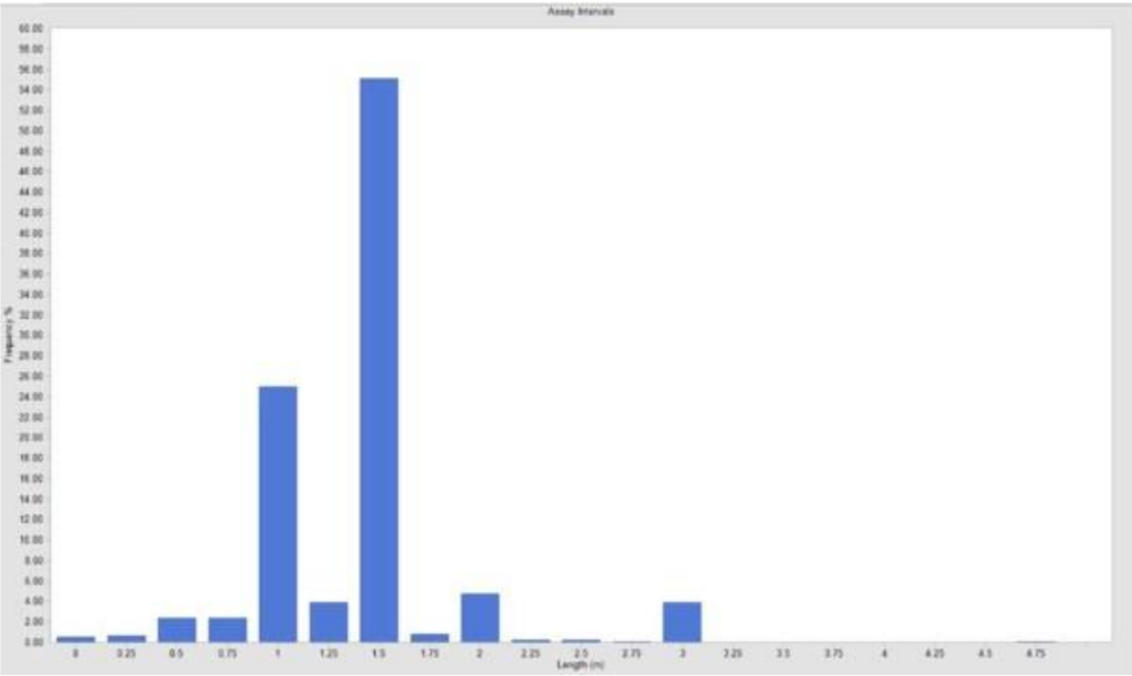

Figure 14-8: Histogram of assay interval lengths in metres

Source: Kirkham, 2025.

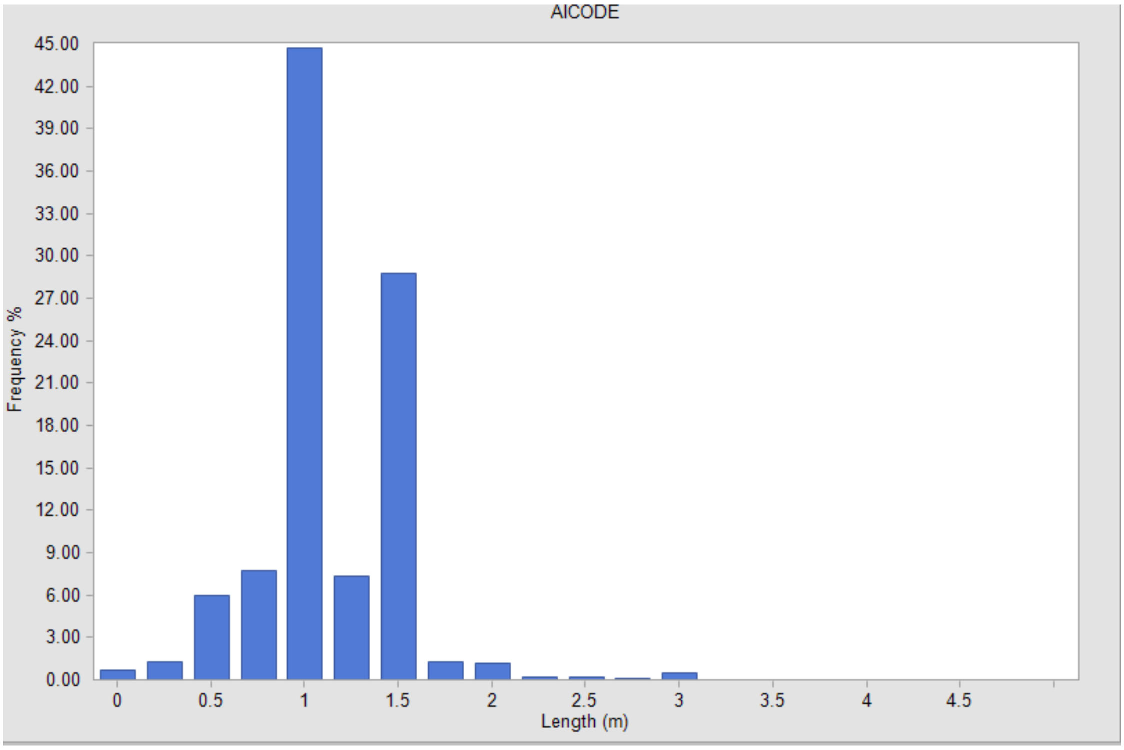

Figure 14-9: Histogram of assay interval lengths within veins in metres

Source: Kirkham, 2025.

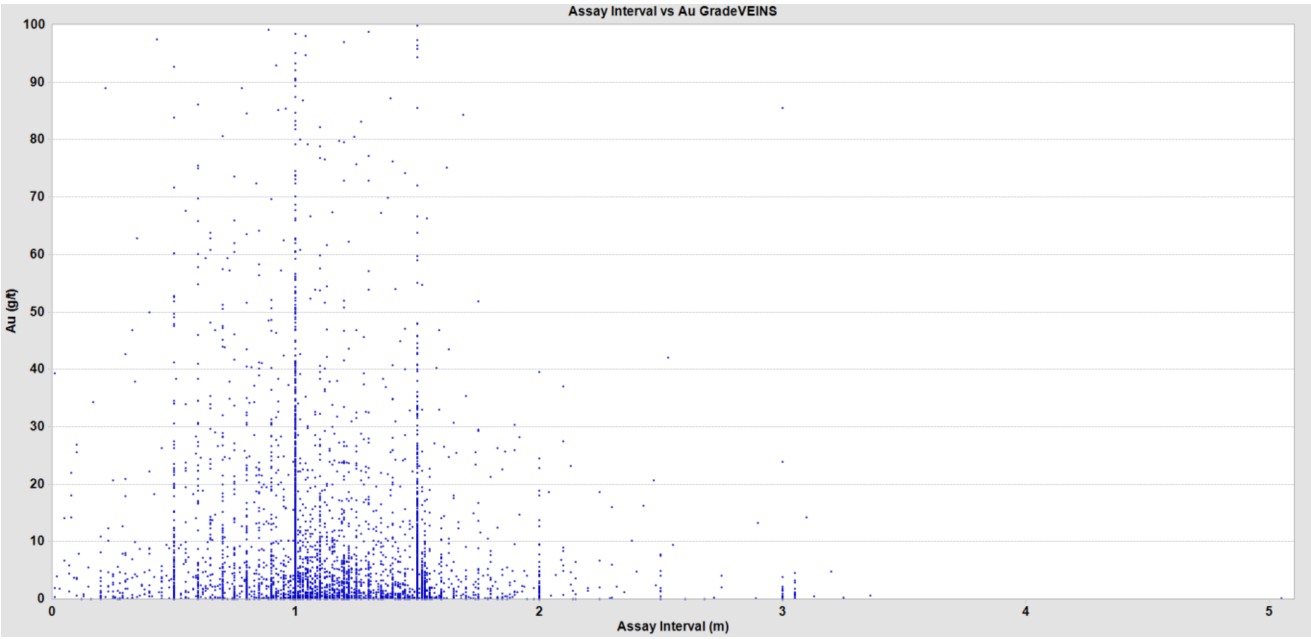

Figure 14-10: Scatterplot of assay interval lengths within veins in metres versus gold grade

Source: Kirkham, 2025.

Figure 14-11 and Figure 14-12 show histograms of the gold composite values for all composites and for thosethat are assigned to the high-grade veins, respectively.

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

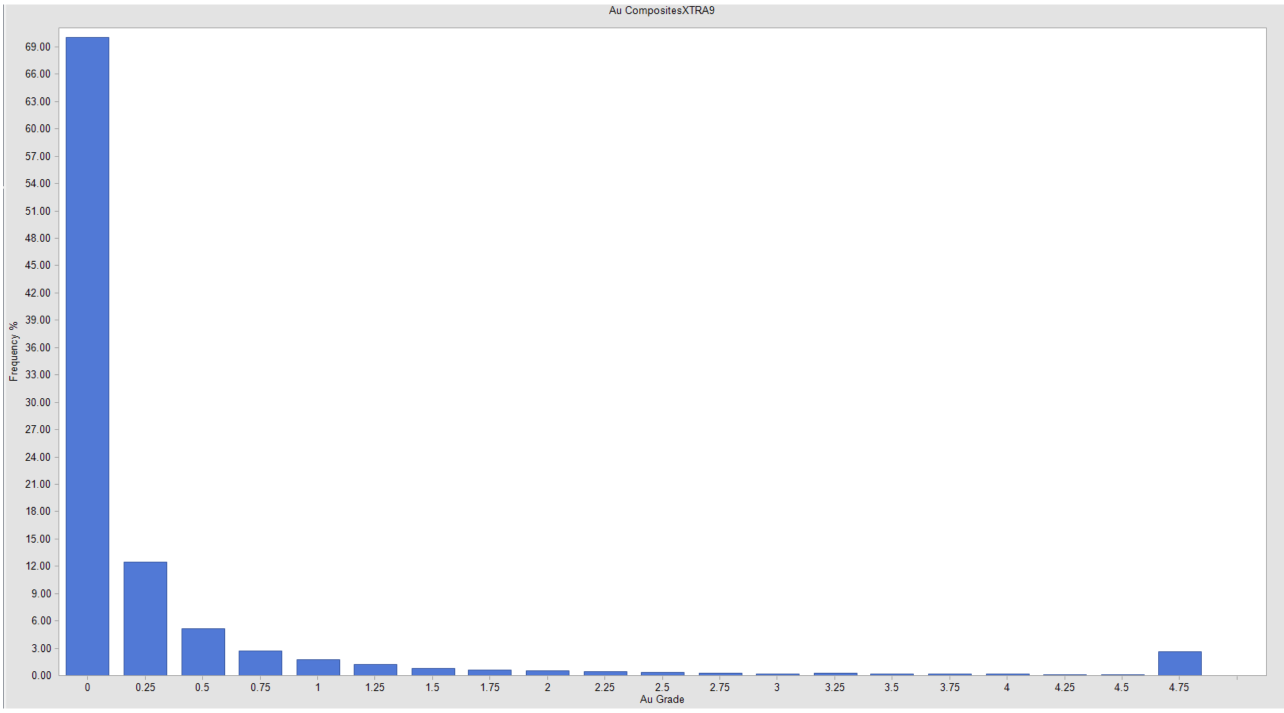

Figure 14-11: Histogram of gold composite grades (g/t)

Source: Kirkham, 2025.

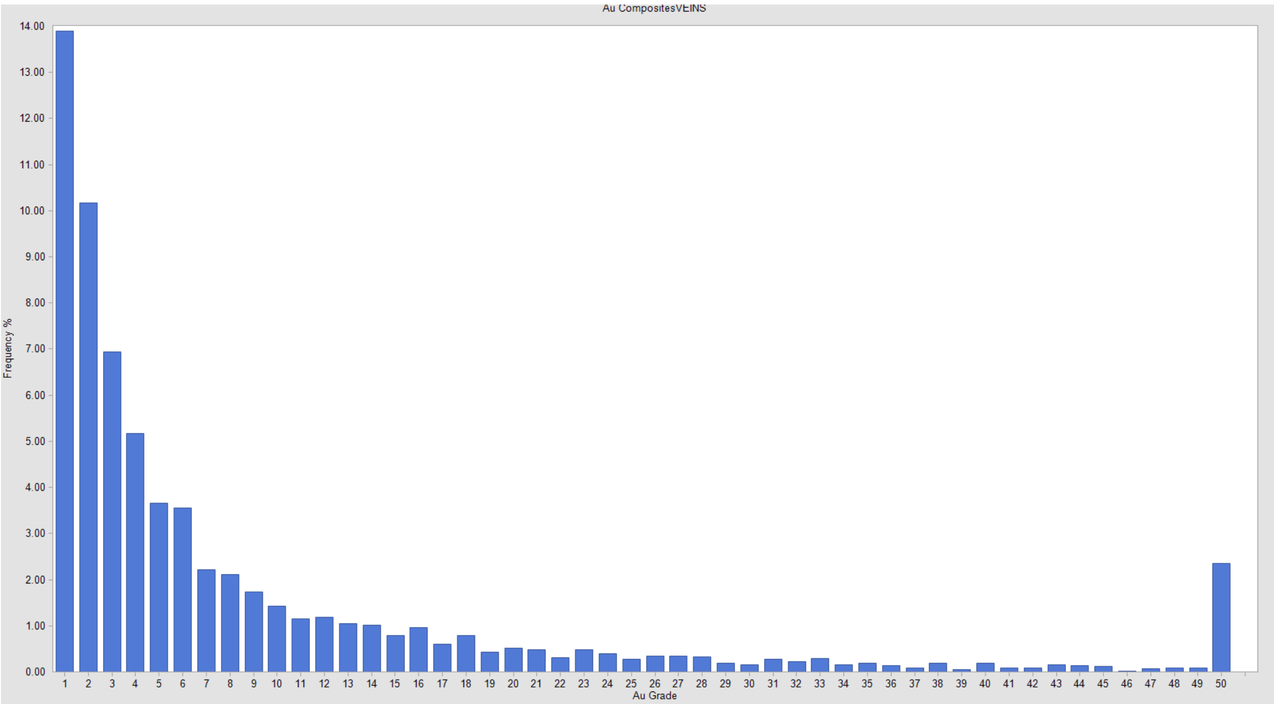

Figure 14-12: Histogram of gold composite grades (g/t) with vein zones

Source: Kirkham, 2025.

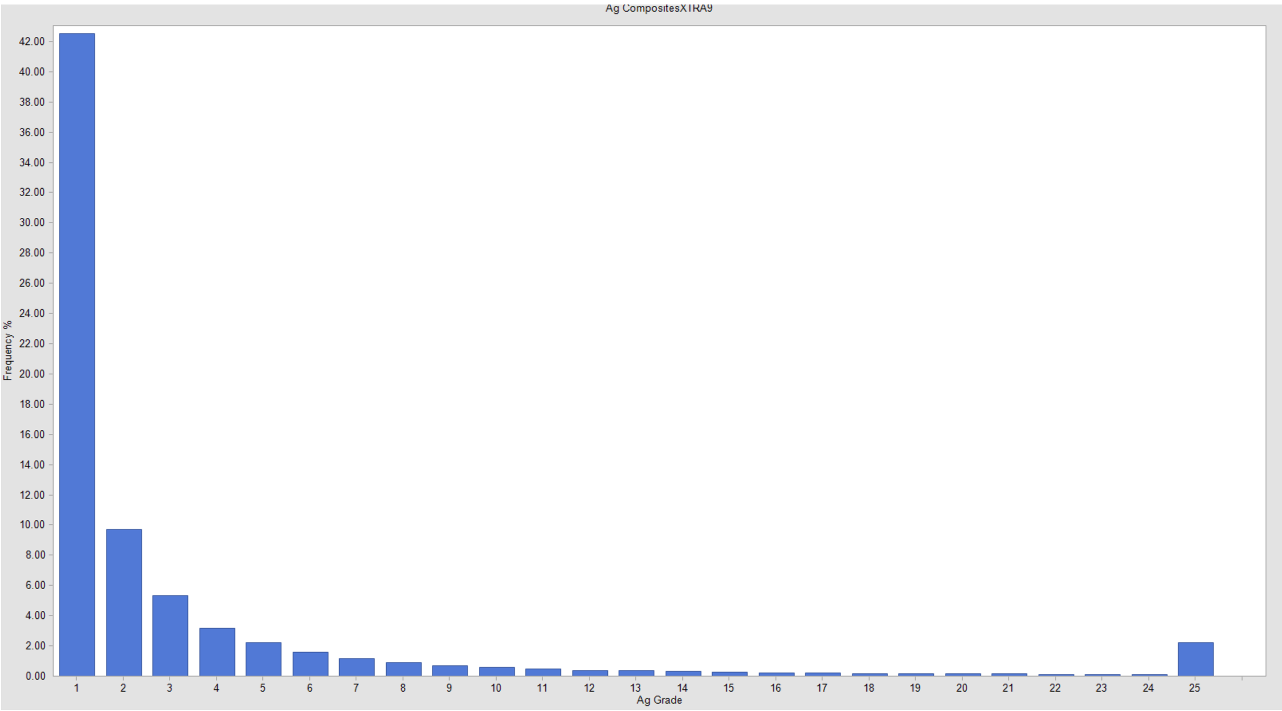

Figure 14-13: Histogram of silver composite grades (g/t)

Source: Kirkham, 2025.

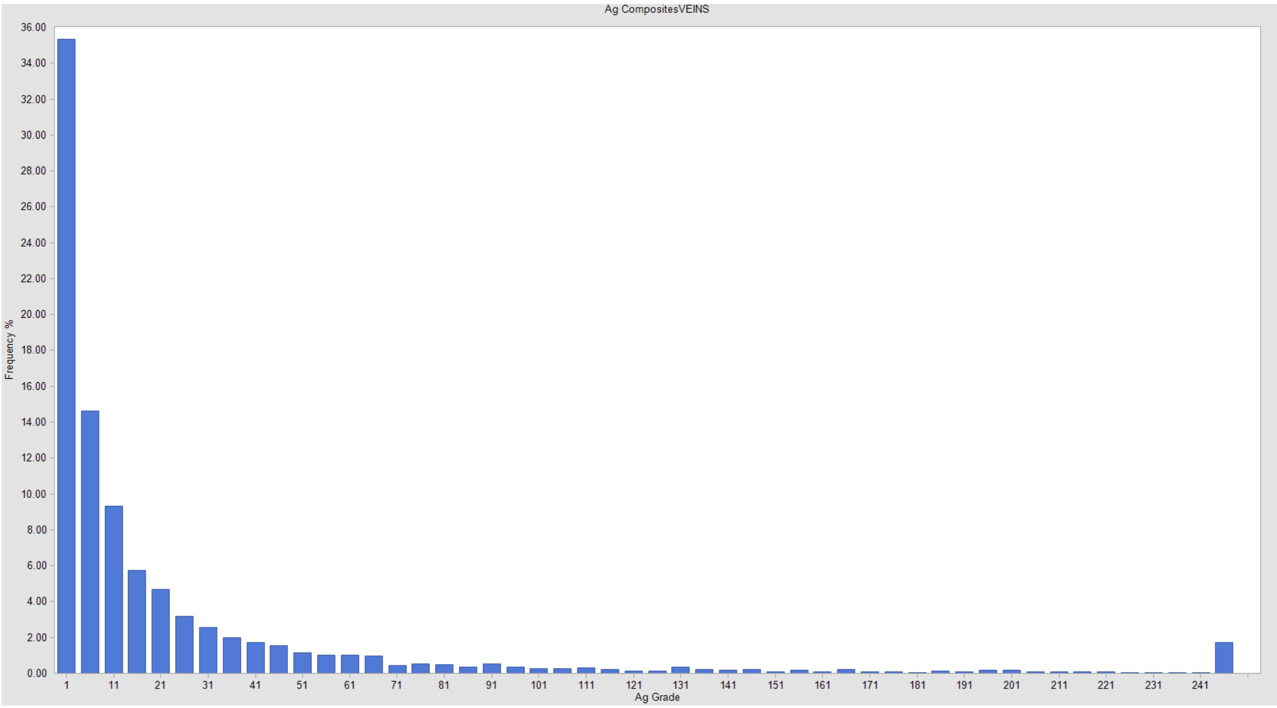

Figure 14-14: Histogram of silver composite grades (g/t) with vein zones

Source: Kirkham, 2025.

14.5.1 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 14-7, where there are two vein groups in the north and six groups in the south.

Table 14-7: Vein groupings 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 14-15 and Figure 14-16 show the box plots and basic statistics for the grouped gold and silver composites, respectively, for the high-grade vein domains. Table 14-8 and Table 14-9 show the basic statistics for the 1.5 m gold and silver composite grades within the mineralized 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 are 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 CVs 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 is 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) arehigher in the south zone. This may be due to significant outlier grades in the south, which has a maximum composite value of 792.3 Au g/t, 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, respectively. 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 Ag g/t in the South within VS-106 versus 1,257 g/t in the north within VN-5.

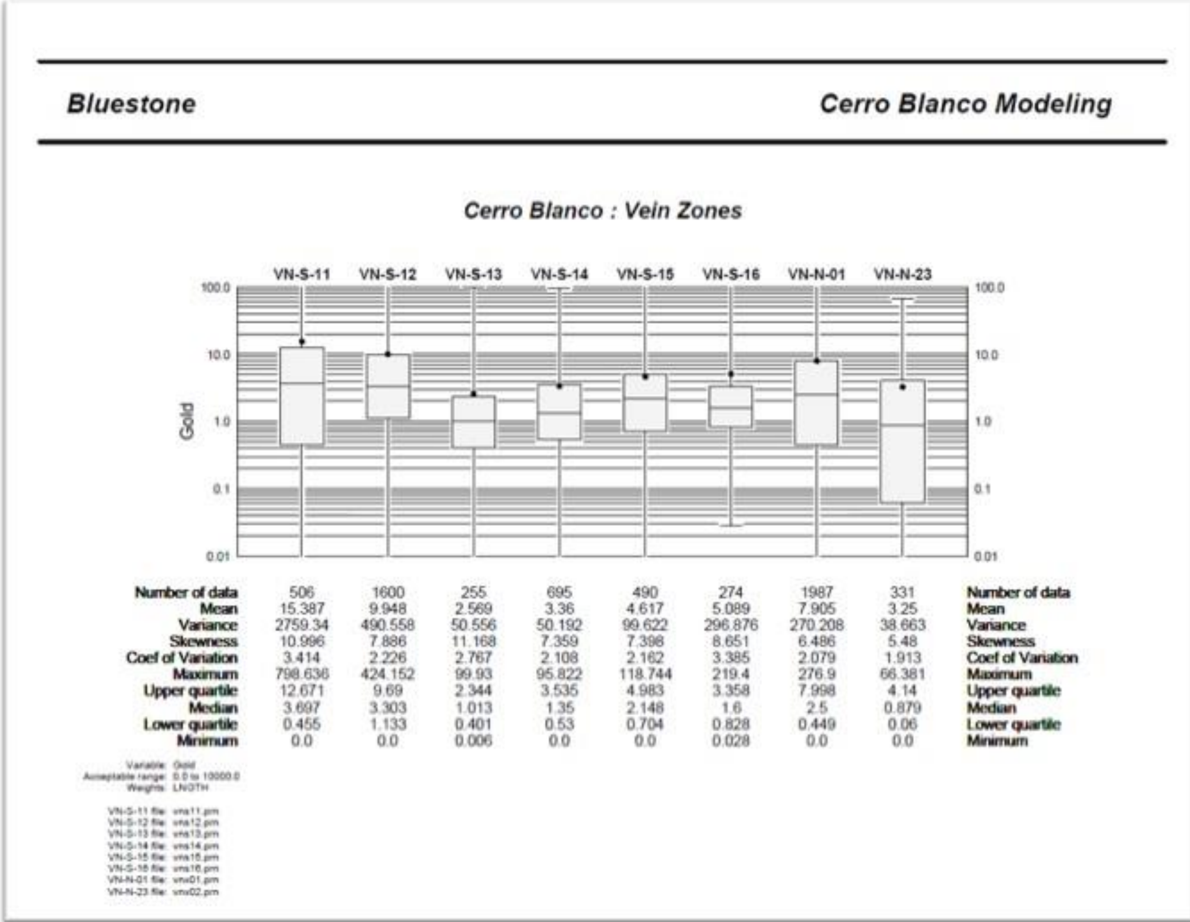

Figure 14-15: Box plot of gold composites for veins

Source: Kirkham, 2025.

Table 14-8: 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.

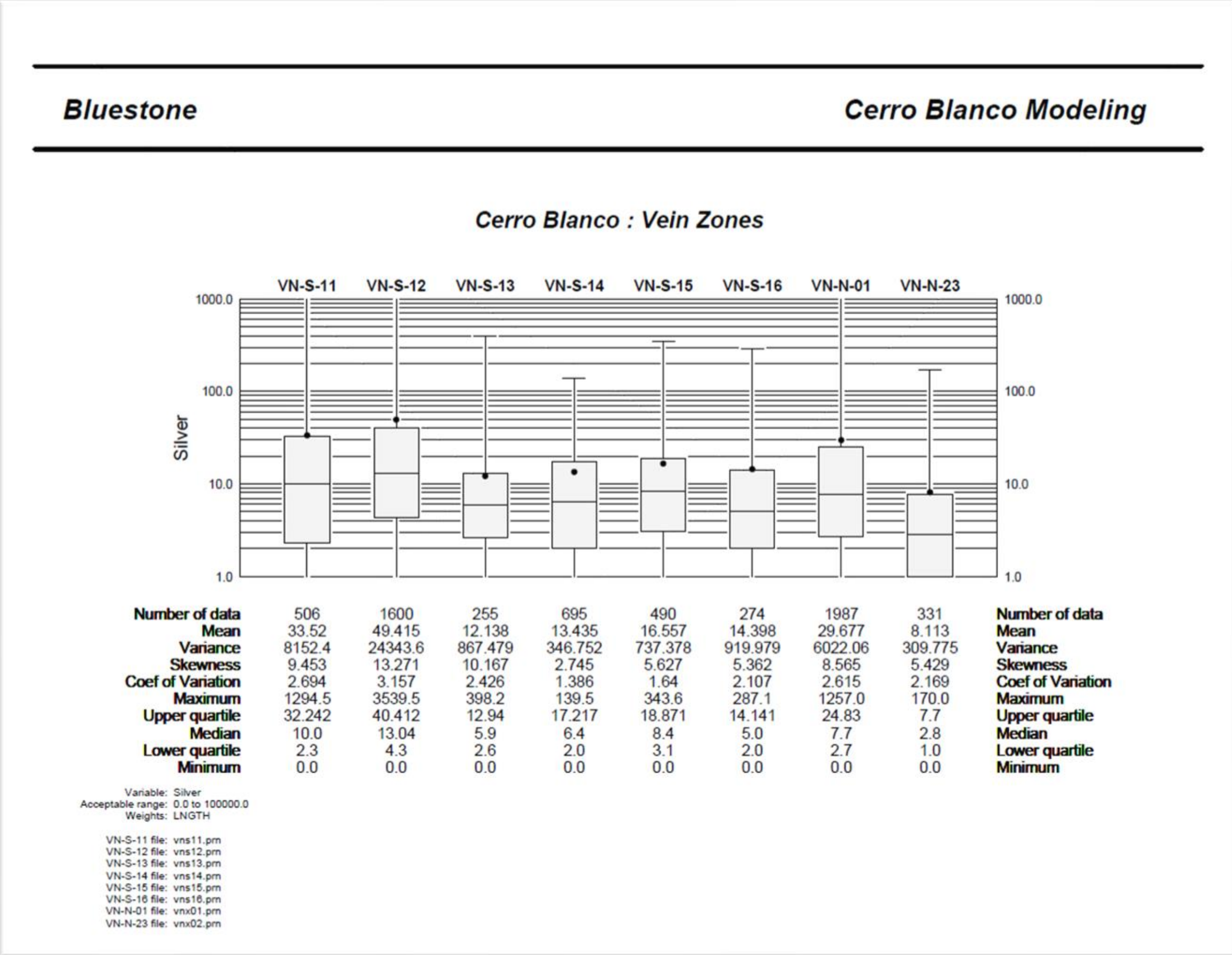

Figure 14-16: Box plot of silver composites for veins

Source: Kirkham, 2025.

Table 14-9: Silver composite statistics weighted by length for veins

Silver (g/t) Composites	South	North
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.

14.5.2 Low-Grade Composite Analysis

Figure 14-17 and Figure 14-18 show the box plots and basic statistics for the grouped (Table 14-10) gold and silver composites, respectively, for the low-grade estimation domains.

Table 14-11 and Table 14-12 show the basicstatistics for the 1.5 m gold and silver composite grades within the low-grade domains, respectively.

Table 14-10: 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 is addressed by a conservative grade limiting and cutting strategy. It is interesting to note that the Salinas (Svc)are markedly higher grade than grades than those analyzed previously, which have increased from 0.19 g/t to 0.32 g/t. This may be primarily attributable to the updated and revised modelling of the Salinas and Sinter units, which was guided by the 2021 drilling program that focussed on delineating and defining the surface Mineral Resources. In addition, the Salinas Group Basal Conglomerate (Scgl) is a significantly higher-grade unit, which has a mean gold grade of 0.55 g/t and has been defined by the updated modelling.

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 approximately a 7:1 ratio Ag:Au.

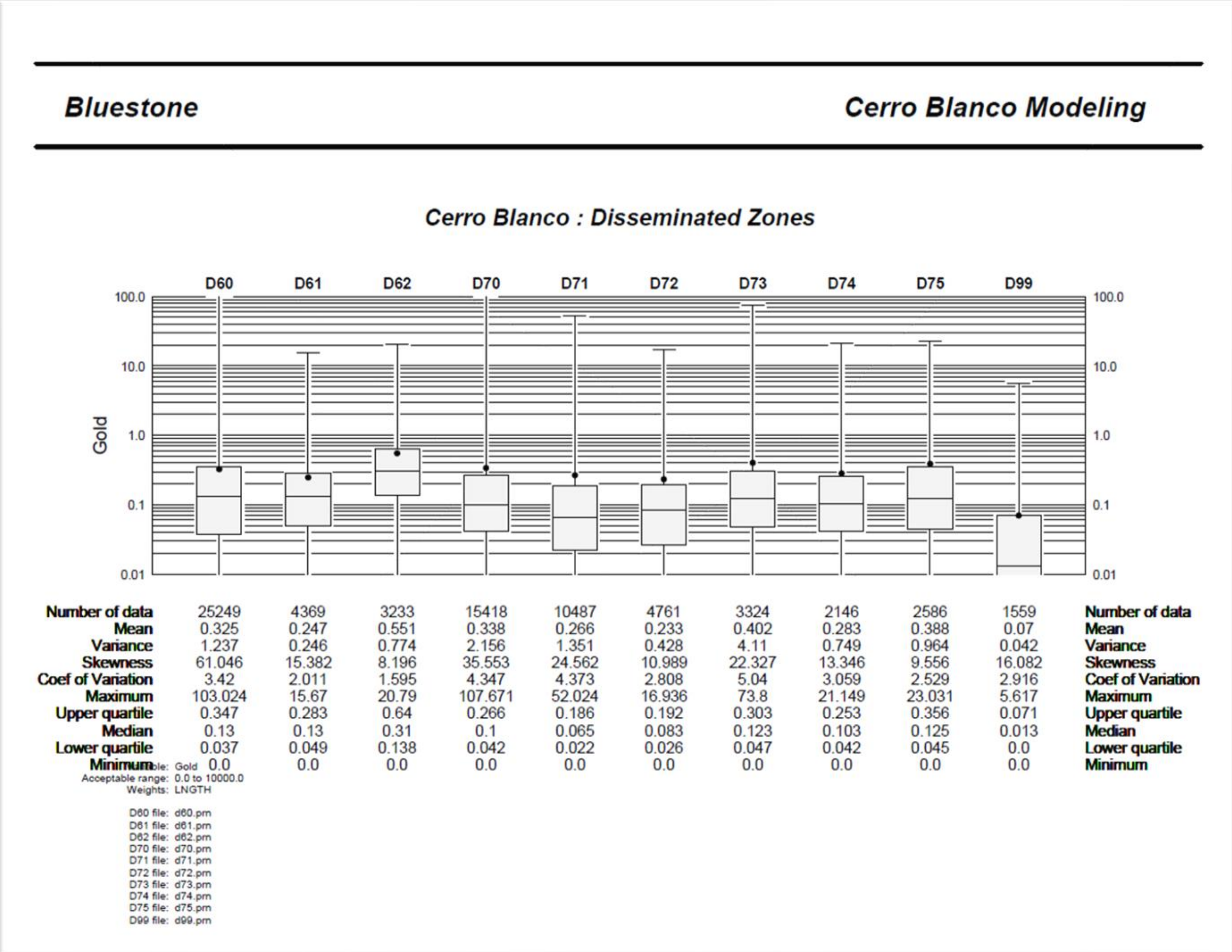

Figure 14-17: Box plot of gold composites for low-grade domains

Source: Kirkham, 2025.

Table 14-11: 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.

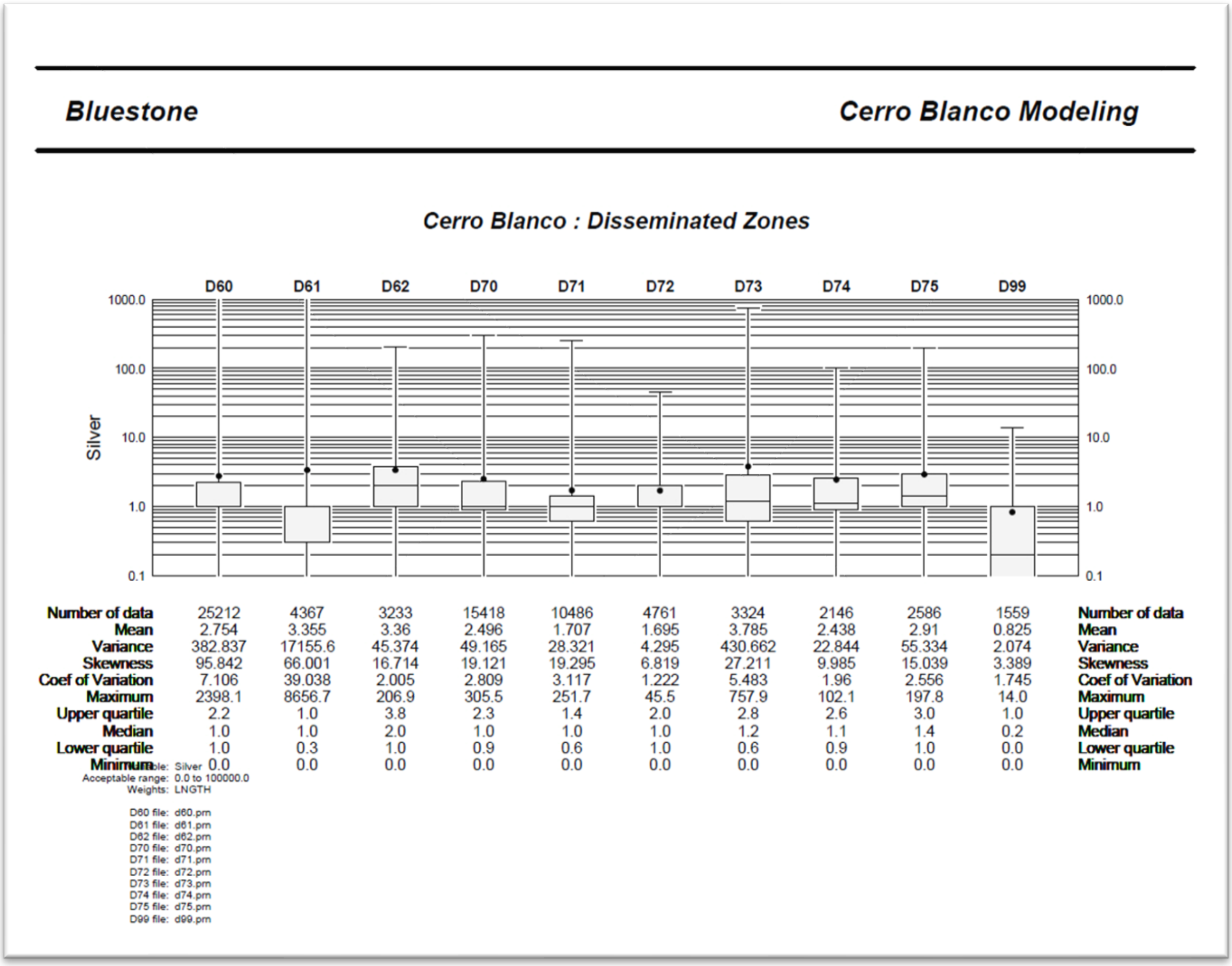

Figure 14-18: Box plot of silver composites for low-grade domains

Source: Kirkham, 2025.

Table 14-12: 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.

14.6 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 14-19 and Figure 14-20 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 14-13 shows the various cut thresholds for the vein groupings and Table 14-14 shows those for the low-grade domains.

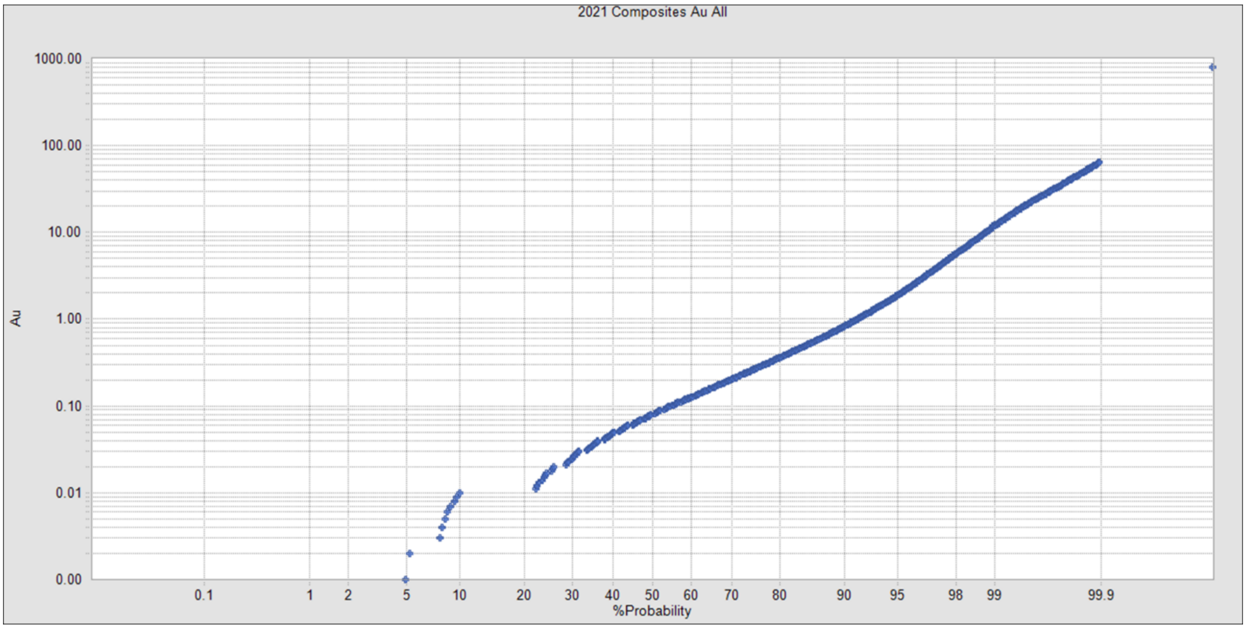

Figure 14-19: Au cumulative frequency plot

Source: Kirkham, 2025.

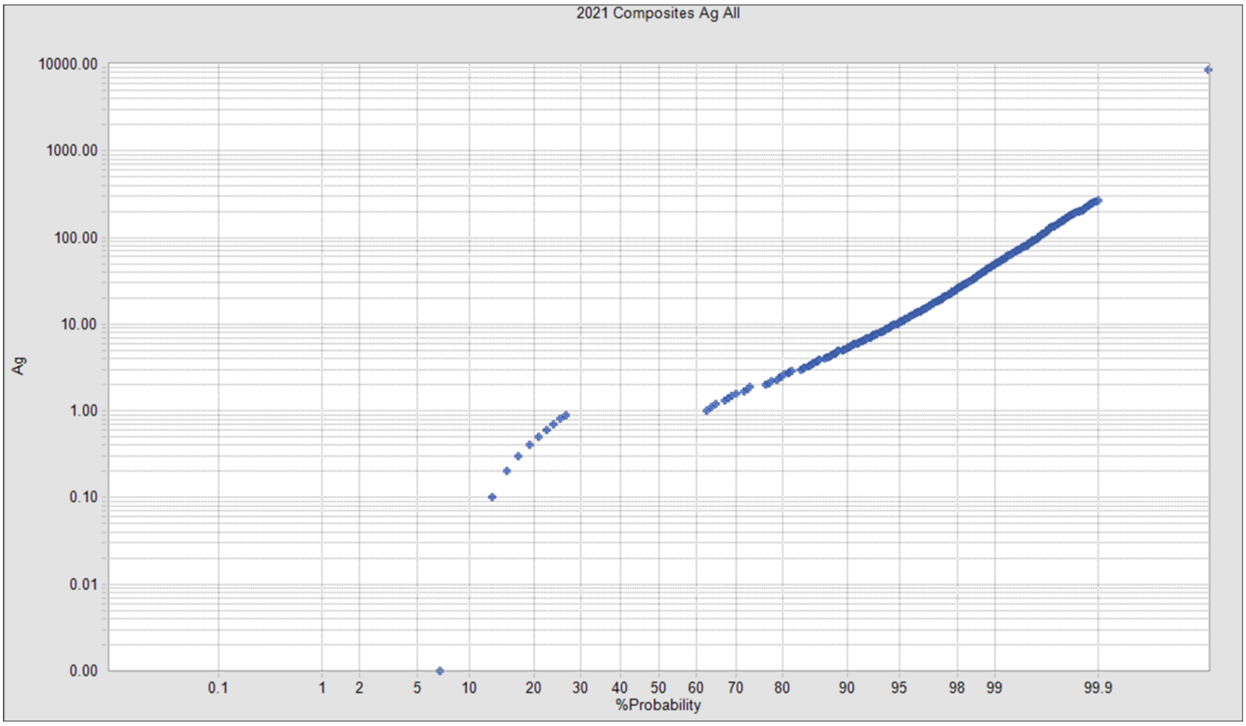

Figure 14-20: Ag cumulative frequency plot

Source: Kirkham, 2025.

Table 14-13: 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 14-14: 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 14-15 and Table 14-16 show 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 14-15: 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 14-16: 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.

14.7 Specific Gravity Estimation

Table 14-17 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 be 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 14-17: SG zone assignments

Lithology Group	Domain / Rock	#	Density (g/mm^3)	Average Density (g/mm^3)
	Ss	27	2.49	
SALINAS	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.

14.8 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 correlogramsfor gold and silver within veins in the south and north zones are shown in Figure 14-21, Figure 14-22 and Figure 14-23 for gold and silver, respectively. These variogram models were used to estimate gold and silver grades using ordinary krigingas the interpolator used to estimate the high-grade veins.

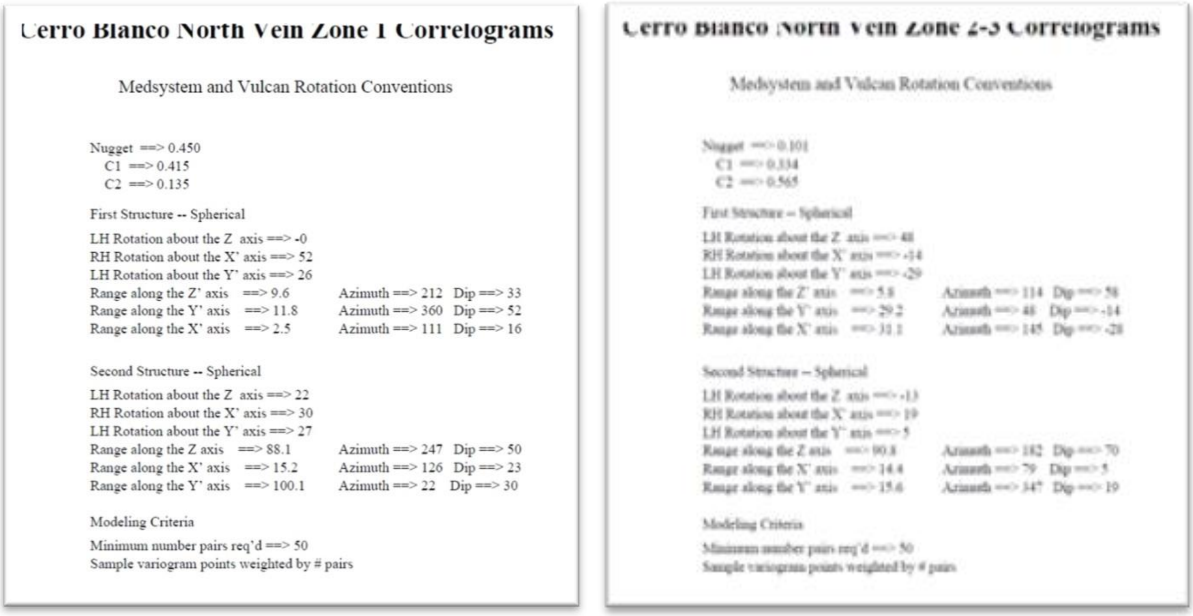

Figure 14-21: Au correlogram models

Source: Kirkham, 2021.

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

Figure 14-22: Ag correlogram models

Source: Kirkham, 2021.

Cerro Blanco North Vein Zone 1 Ag Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.296
C1 ==> 0.596
C2 ==> 0.108

First Structure -- Spherical

LH Rotation about the Z axis ==> 20
RH Rotation about the X' axis ==> 15
LH Rotation about the Y' axis ==> 35
Range along the Z' axis ==> 33.7 Azimuth ==> 270 Dip ==> 52
Range along the Y' axis ==> 4.2 Azimuth ==> 20 Dip ==> 15
Range along the X' axis ==> 3.1 Azimuth ==> 121 Dip ==> 34

Second Structure -- Spherical

LH Rotation about the Z axis ==> 28
RH Rotation about the X' axis ==> 47
LH Rotation about the Y' axis ==> 54
Range along the Z axis ==> 376.0 Azimuth ==> 270 Dip ==> 24
Range along the X' axis ==> 39.8 Azimuth ==> 163 Dip ==> 34
Range along the Y' axis ==> 212.4 Azimuth ==> 28 Dip ==> 47

Modeling Criteria

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

Cerro Blanco North Vein Zone 2-3 Ag Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.271
C1 ==> 0.536
C2 ==> 0.193

First Structure -- Spherical

LH Rotation about the Z axis ==> 33
RH Rotation about the X' axis ==> 87
LH Rotation about the Y' axis ==> -36
Range along the Z' axis ==> 14.2 Azimuth ==> 177 Dip ==> 2
Range along the Y' axis ==> 5.2 Azimuth ==> 33 Dip ==> 87
Range along the X' axis ==> 20.4 Azimuth ==> 87 Dip ==> -2

Second Structure -- Spherical

LH Rotation about the Z axis ==> -20
RH Rotation about the X' axis ==> -43
LH Rotation about the Y' axis ==> -2
Range along the Z axis ==> 242.0 Azimuth ==> 343 Dip ==> 47
Range along the X' axis ==> 46.2 Azimuth ==> 71 Dip ==> -1
Range along the Y' axis ==> 6.3 Azimuth ==> 340 Dip ==> -43

Modeling Criteria

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

Cerro Blanco South Vein Zone 11 Ag Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.203
C1 ==> 0.679
C2 ==> 0.118

First Structure -- Spherical

LH Rotation about the Z axis ==> -2
RH Rotation about the X' axis ==> -9
LH Rotation about the Y' axis ==> 6
Range along the Z' axis ==> 4.0 Azimuth ==> 324 Dip ==> 79
Range along the Y' axis ==> 8.3 Azimuth ==> 358 Dip ==> -9
Range along the X' axis ==> 17.0 Azimuth ==> 87 Dip ==> 6

Second Structure -- Spherical

LH Rotation about the Z axis ==> 13
RH Rotation about the X' axis ==> 1
LH Rotation about the Y' axis ==> 40
Range along the Z axis ==> 315.0 Azimuth ==> 283 Dip ==> 50
Range along the X' axis ==> 10.1 Azimuth ==> 104 Dip ==> 40
Range along the Y' axis ==> 80.0 Azimuth ==> 13 Dip ==> 1

Modeling Criteria

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

Cerro Blanco South Vein Zone 12 Ag Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.450
C1 ==> 0.248
C2 ==> 0.302

First Structure -- Spherical

LH Rotation about the Z axis ==> -51
RH Rotation about the X' axis ==> 56
LH Rotation about the Y' axis ==> 31
Range along the Z' axis ==> 3.4 Azimuth ==> 165 Dip ==> 29
Range along the Y' axis ==> 5.0 Azimuth ==> 309 Dip ==> 56
Range along the X' axis ==> 1.0 Azimuth ==> 66 Dip ==> 17

Second Structure -- Spherical

LH Rotation about the Z axis ==> 10
RH Rotation about the X' axis ==> 34
LH Rotation about the Y' axis ==> 60
Range along the Z axis ==> 82.8 Azimuth ==> 262 Dip ==> 24
Range along the X' axis ==> 9.7 Azimuth ==> 144 Dip ==> 46
Range along the Y' axis ==> 28.9 Azimuth ==> 10 Dip ==> 34

Modeling Criteria

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

Cerro Blanco South Vein Zone 13 Ag Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.113
C1 ==> 0.751
C2 ==> 0.136

First Structure -- Spherical

LH Rotation about the Z axis ==> 44
RH Rotation about the X' axis ==> -16
LH Rotation about the Y' axis ==> 52
Range along the Z' axis ==> 18.3 Azimuth ==> 326 Dip ==> 37
Range along the Y' axis ==> 5.6 Azimuth ==> 44 Dip ==> -16
Range along the X' axis ==> 21.9 Azimuth ==> 115 Dip ==> 49

Second Structure -- Spherical

LH Rotation about the Z axis ==> -52
RH Rotation about the X' axis ==> -2
LH Rotation about the Y' axis ==> 6
Range along the Z axis ==> 45.9 Azimuth ==> 242 Dip ==> 84
Range along the X' axis ==> 14.5 Azimuth ==> 38 Dip ==> 6
Range along the Y' axis ==> 4.9 Azimuth ==> 308 Dip ==> -2

Modeling Criteria

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

Cerro Blanco South Vein Zone 14 Ag Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.350
C1 ==> 0.290
C2 ==> 0.360

First Structure -- Spherical

LH Rotation about the Z axis ==> -0
RH Rotation about the X' axis ==> -3
LH Rotation about the Y' axis ==> 8
Range along the Z' axis ==> 18.7 Azimuth ==> 288 Dip ==> 81
Range along the Y' axis ==> 47.6 Azimuth ==> 360 Dip ==> -3
Range along the X' axis ==> 3.3 Azimuth ==> 89 Dip ==> 8

Second Structure -- Spherical

LH Rotation about the Z axis ==> -37
RH Rotation about the X' axis ==> 15
LH Rotation about the Y' axis ==> 7
Range along the Z axis ==> 106.0 Azimuth ==> 168 Dip ==> 73
Range along the X' axis ==> 8.5 Azimuth ==> 55 Dip ==> 7
Range along the Y' axis ==> 44.6 Azimuth ==> 323 Dip ==> 15

Modeling Criteria

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

Cerro Blanco South Vein Zone 15 Ag Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.314
C1 ==> 0.576
C2 ==> 0.110

First Structure -- Spherical

LH Rotation about the Z axis ==> 5
RH Rotation about the X' axis ==> 35
LH Rotation about the Y' axis ==> 8
Range along the Z' axis ==> 4.2 Azimuth ==> 198 Dip ==> 54
Range along the Y' axis ==> 2.2 Azimuth ==> 5 Dip ==> 35
Range along the X' axis ==> 15.9 Azimuth ==> 100 Dip ==> 6

Second Structure -- Spherical

LH Rotation about the Z axis ==> 55
RH Rotation about the X' axis ==> 21
LH Rotation about the Y' axis ==> -41
Range along the Z axis ==> 332.5 Azimuth ==> 168 Dip ==> 45
Range along the X' axis ==> 20.0 Azimuth ==> 128 Dip ==> -38
Range along the Y' axis ==> 549.1 Azimuth ==> 55 Dip ==> 21

Modeling Criteria

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

Cerro Blanco South Vein Zone 16 Ag Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.210
C1 ==> 0.471
C2 ==> 0.320

First Structure -- Spherical

LH Rotation about the Z axis ==> 42
RH Rotation about the X' axis ==> 13
LH Rotation about the Y' axis ==> 98
Range along the Z' axis ==> 2.9 Azimuth ==> 314 Dip ==> -7
Range along the Y' axis ==> 48.5 Azimuth ==> 42 Dip ==> 13
Range along the X' axis ==> 15.5 Azimuth ==> 252 Dip ==> 75

Second Structure -- Spherical

LH Rotation about the Z axis ==> -16
RH Rotation about the X' axis ==> 22
LH Rotation about the Y' axis ==> 48
Range along the Z axis ==> 33.0 Azimuth ==> 235 Dip ==> 38
Range along the X' axis ==> 51.4 Azimuth ==> 97 Dip ==> 43
Range along the Y' axis ==> 153.4 Azimuth ==> 344 Dip ==> 22

Modeling Criteria

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

Figure 14-23: Ag correlogram models

Source: Kirkham, 2021.

In addition, experimental variograms and variogram models in the form of correlograms were alsogenerated for gold and silver grades within the low-grade domains, namely, Salinas and Mita units. As mentioned above, the definition of nugget value was derivedfrom the downhole variograms. The correlogram models for gold and silver are shown in Table 14-18 and Table 14-19, respectively. These variogram models were used to estimate gold and silver grades using ordinary kriging as the interpolator.

Table 14-18: 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 14-19: 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.

14.9 Block Model Definition

The block model used for estimating the Mineral Resources was defined according to the origin and orientation shownin Figure 14-24 and the limits specified in Figure 14-25.

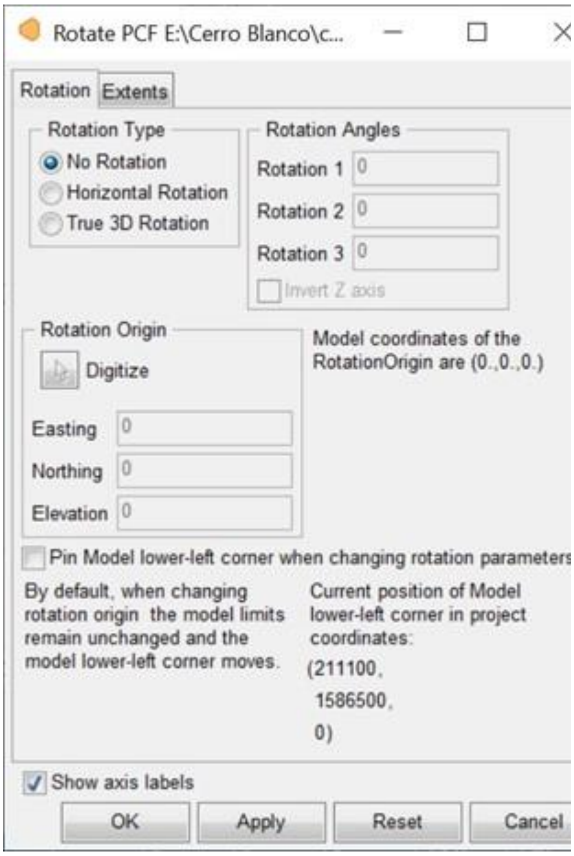

Figure 14-24: Block model origin & orientation

Source: Kirkham, 2025.

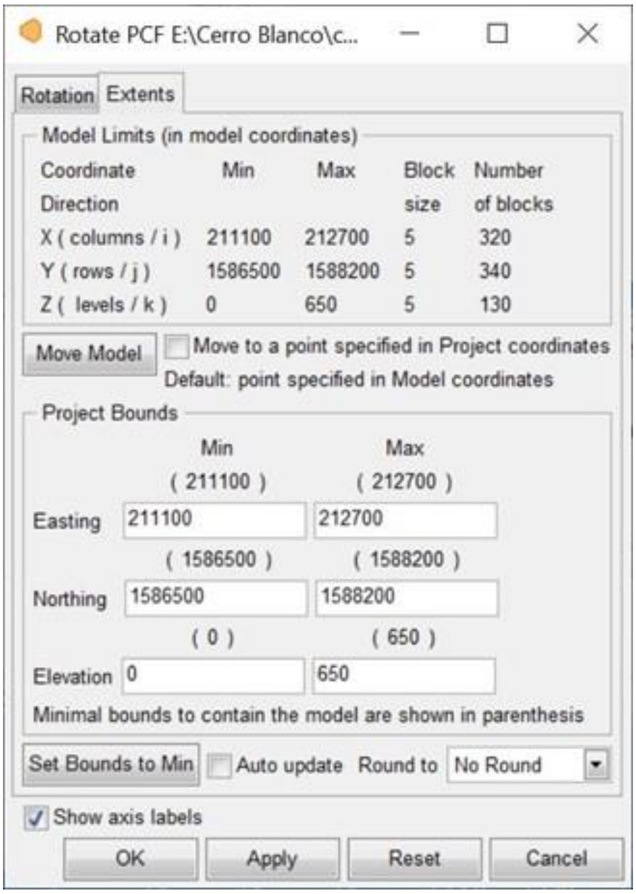

Figure 14-25: Block model extents & dimensions

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.

14.10 Resource Estimation Methodology

The estimation strategy was a two-step process that entailed estimating the high-grade vein component ofeach block and then the low-grade mineralized host rock component. Once completed, the final whole blockgrades 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, a 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 omnidirectional 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 mineralized 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, a maximum of twelve composites, and a maximum of three compositesper 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 omnidirectional to a maximum of 100 m, and hard boundaries were used so that grade is not smeared betweenthe units.

14.11 Mineral Resource Classification

Mineral Resources were estimated in conformity with the 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 CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) 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 increaseor decrease in future Mineral 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 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 are 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 should be interpreted and defined to ensure continuity.

Drill hole spacing is sufficient for preliminary geostatistical analysis and evaluating spatial grade variability.The classification of Mineral Resources was based primarily on the 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 the interpretation of the mineralized zones;
- number of composites used to estimate a block;
- number of composites allowed per drill hole;
- distance to the 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 Mineral Resource category to classify the Mineral 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 of 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 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) guidelines. Mineral Resource classification forgold was based primarily on drill hole spacing and on continuity of mineralization. Measured Resources weredefined as blocks within a distance to the 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 amountof drilling.

14.12 Stockpile Resources

Mineralized material from mining activities undertaken to date at Era Dorada, including ramp developmentand 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 14-26.

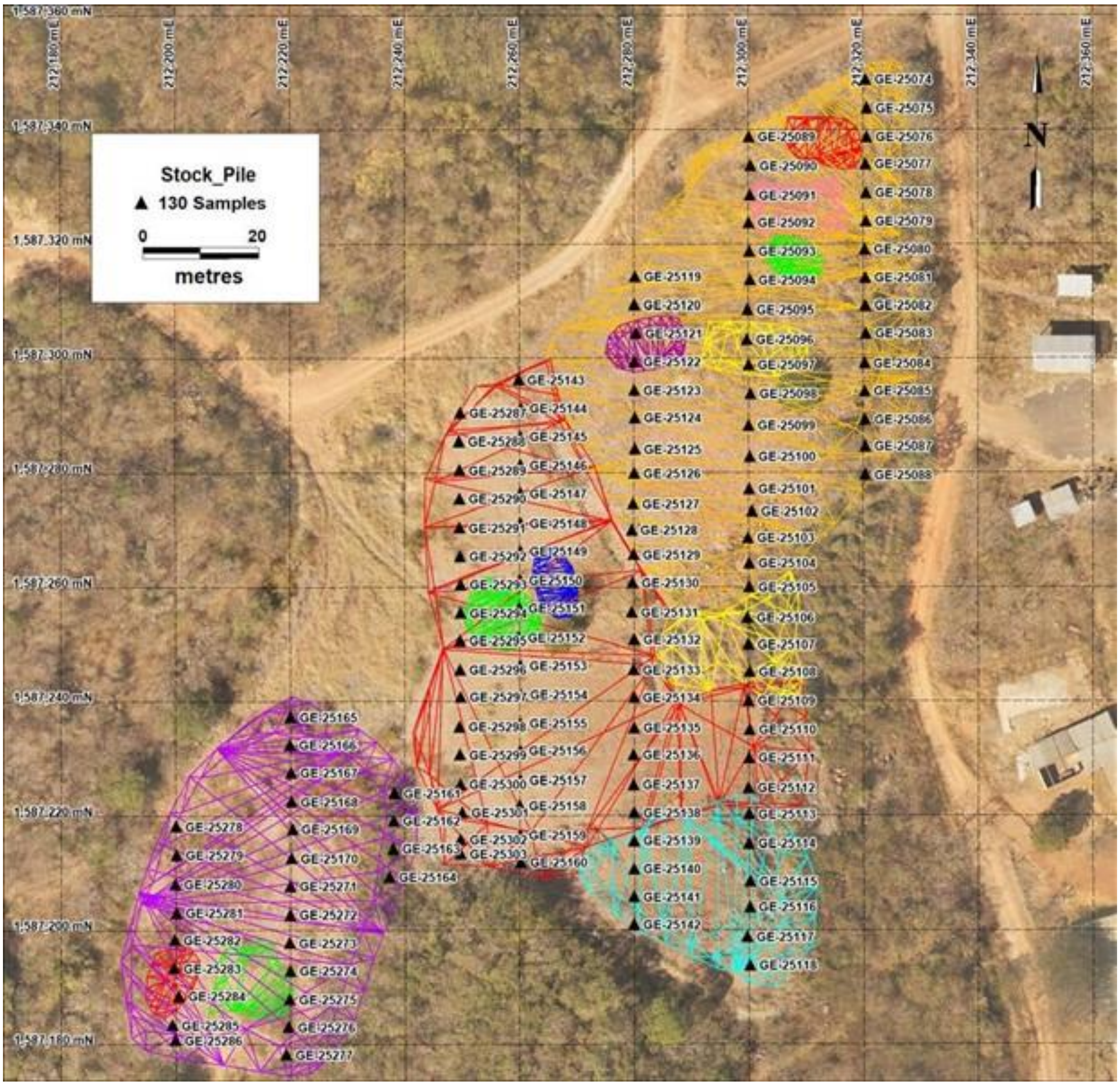

Figure 14-26: Plan view of the stockpile, sample locations & domain solids

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 14-20 shows the volume and tonnage based on an unconsolidated specific gravity of 2.0 g/mm^3, along with gold and silver grades and metal content. These Resources are classified as Measured.

Table 14-20: 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.

14.13 Mineral Resource Estimate

This estimate is based upon the reasonable prospect of eventual economic extraction based on continuity and underground mining shapes, using estimates of operating costs and price assumptions. The “reasonable prospects foreventual economic extraction” were tested using stope optimizations performed using Datamine Studio UG v.2.57™ based on reasonable prospects ofeventual economic assumptions, as shown in Table 14-21.

Metal prices are based on long-term three-year forecast consensus financial institution estimates published by CIBC (Canadian Imperial Bank of Commerce).

Table 14-21: Parameters used for stope optimization and cut-off grade

Parameter	Unit	RPEEE UG Mining Method	
		LH	MCF
Gold price	US$/oz Au	2,500	
Project Parameters			
Process Recovery	%	96.00%	
Payable metal	%	99.92%	
TC/RC	US$/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: GE21, 2025.

Figure 14-27 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.

Table 14-22 shows tonnage and grade in the Era Dorada deposit and includes all domains at a 2.25 Au g/tcut-off grade.

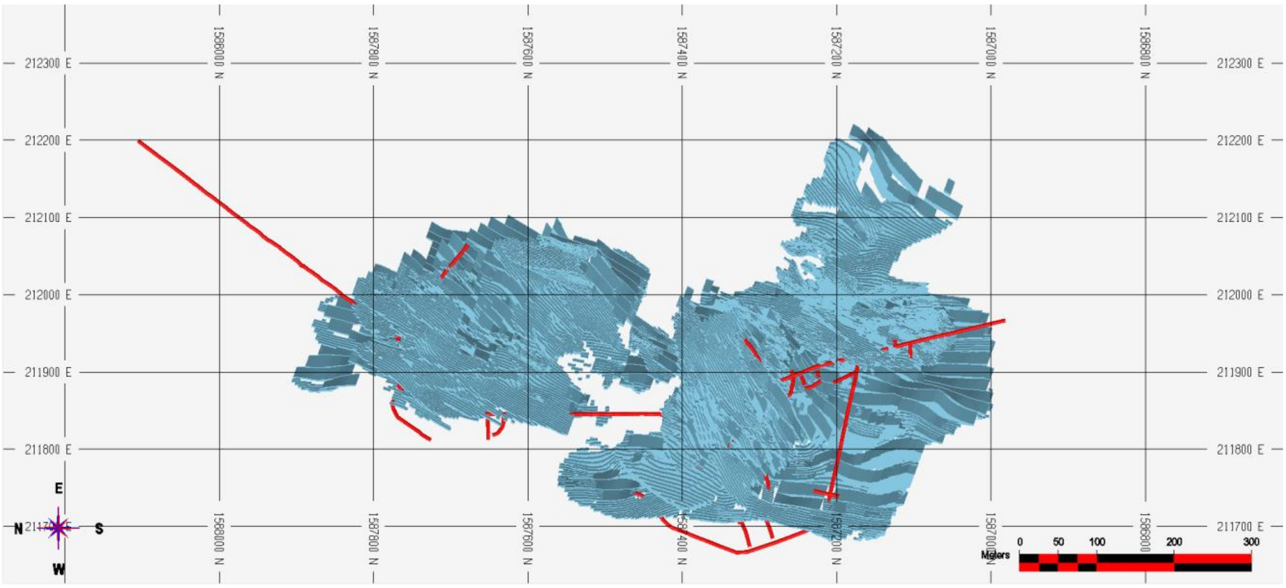

Figure 14-27: Plan view of gold block model with reasonable prospects optimized mine shapes with existing underground ramps

Source: Kirkham, 2025.

Table 14-22: Resource Estimate using 2.25 Au g/t cut-off

Resource Category	Tonnes (kt)	Au Grade (g/t)	Ag Grade (g/t)	Contained Gold (koz)	Contained Silver (koz)
Measured					
Indicated	6,349	9.31	31.54	1,901	6,439
Measured & Indicated	6,349	9.31	31.54	1,901	6,439
Inferred	605	6.02	19.68	117	383

Notes:
1. Mineral Resources are reported in accordance with NI 43-101.
2. Mineral Resource Estimates have been prepared by Garth Kirkham, P.Geo., a Qualified Person as defined by NI 43-101.
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 Au g/t. Cut-off grades are based on 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 NI 43-101. 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 excluding non-mineable material prior to reporting.
7. Metallurgical recoveries are reported as the average over the LoM 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. The effective date of the Mineral Resource Estimate is December 31, 2024.
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 14-28 illustrates a plan view of the 3-D block model for the Resources within the mineralized veins. Figure 14-29 through Figure 14-32 show sectional views of the high-grade veins for gold and silver in the north and south, respectively. Figure 14-33 through Figure 14-36 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.

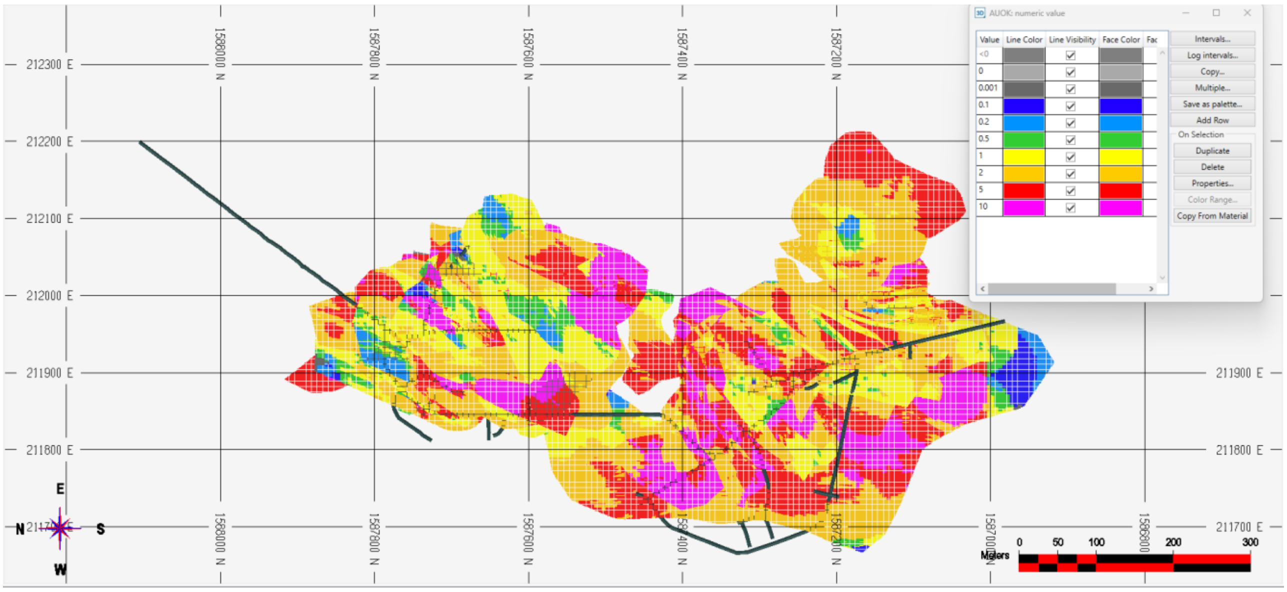

Figure 14-28: Plan view of Au within veins along with existing ramp development

Source: Kirkham, 2025.

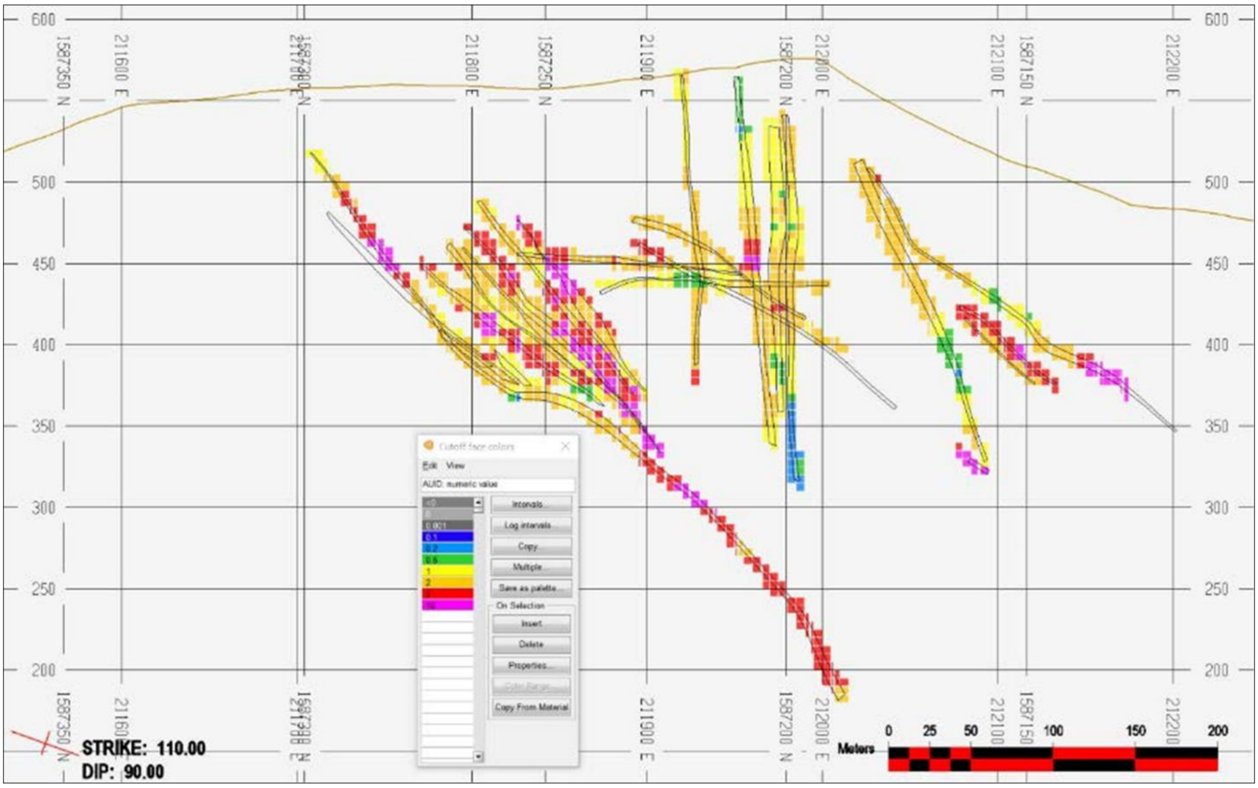

Figure 14-29: Section view of Au south zone veins

Source: Kirkham, 2025.

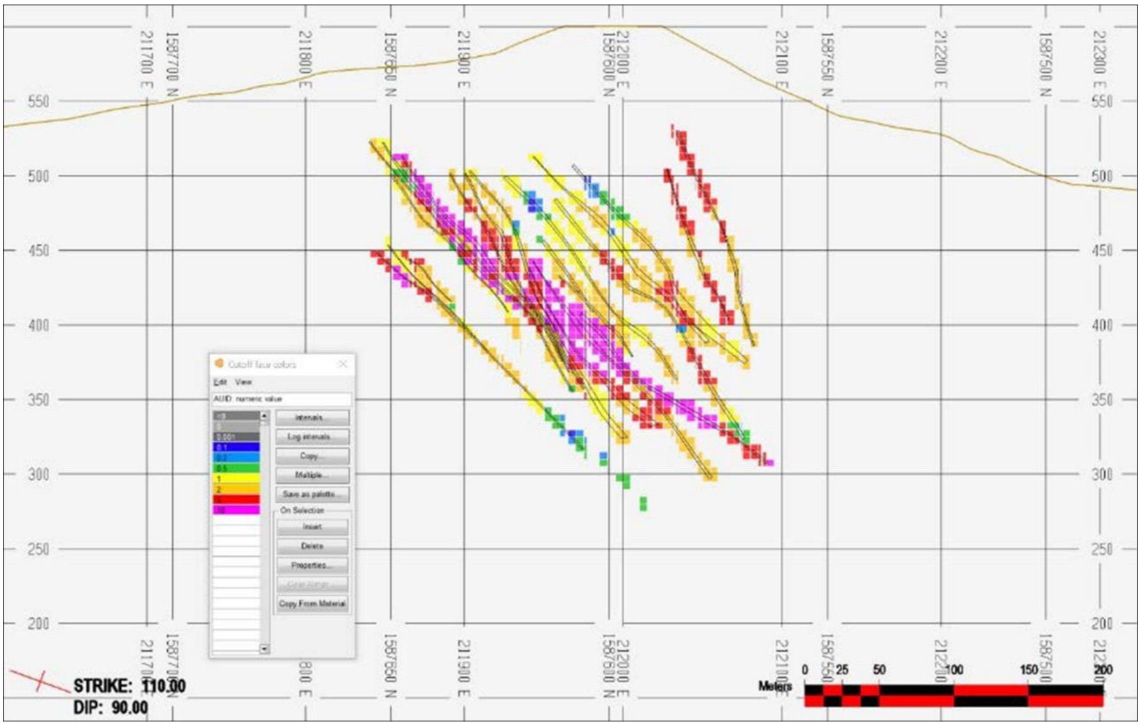

Figure 14-30: Section view of Au block model north zone veins

Source: Kirkham, 2025.

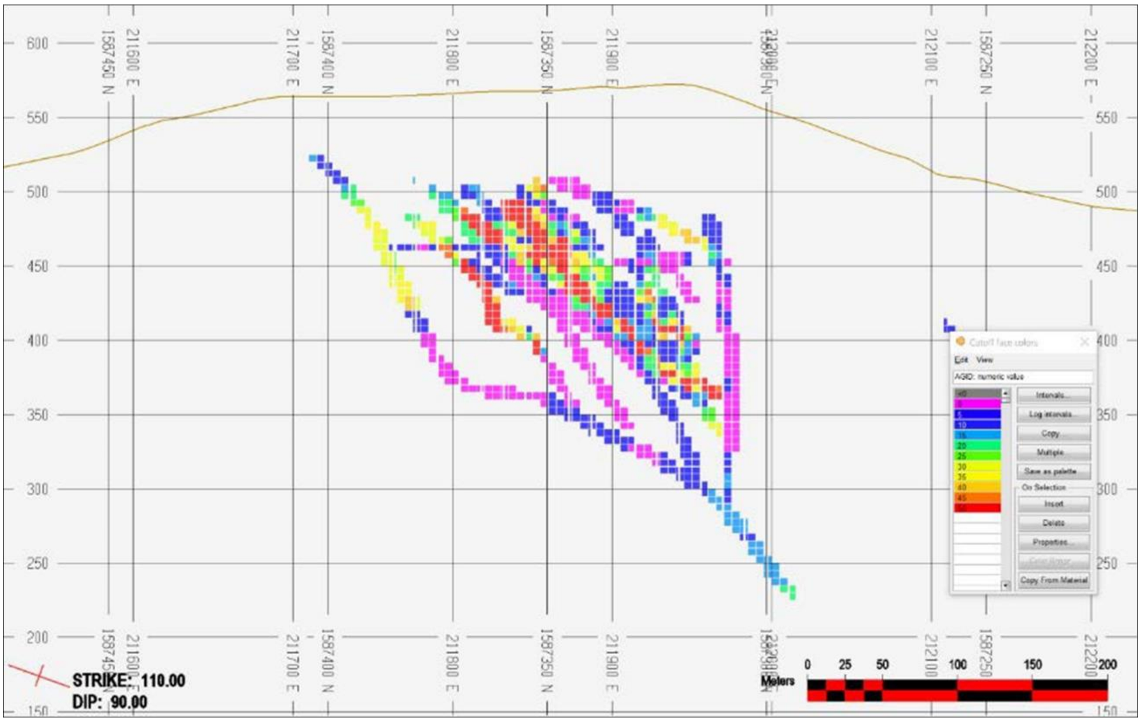

Figure 14-31: Section view of Ag block model south zone veins

Source: Kirkham, 2025.

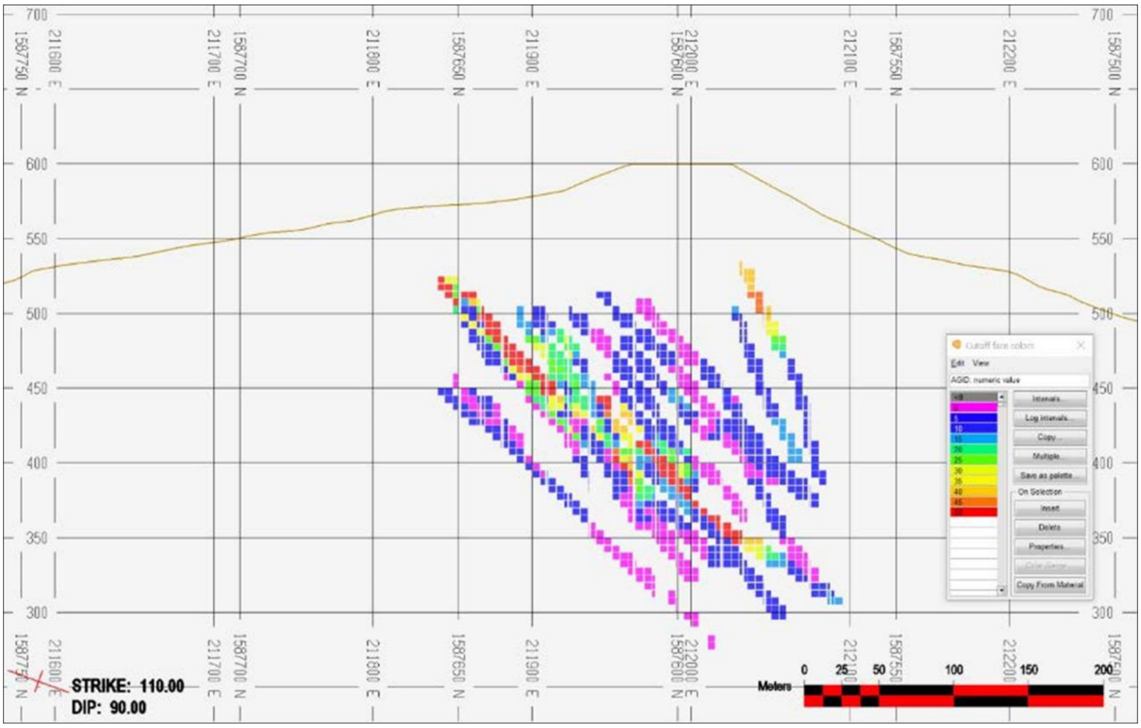

Figure 14-32: Section view of Ag block model north zone veins

Source: Kirkham, 2025.

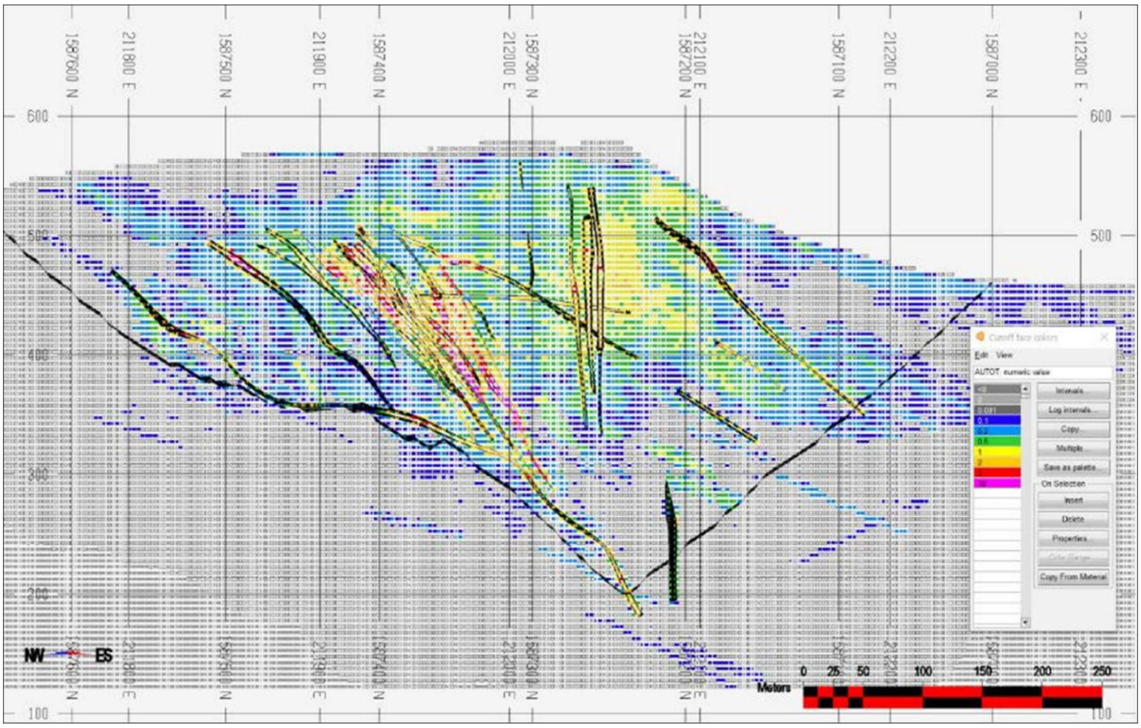

Figure 14-33: **Section view of Au block model south**

Source: Kirkham, 2025.

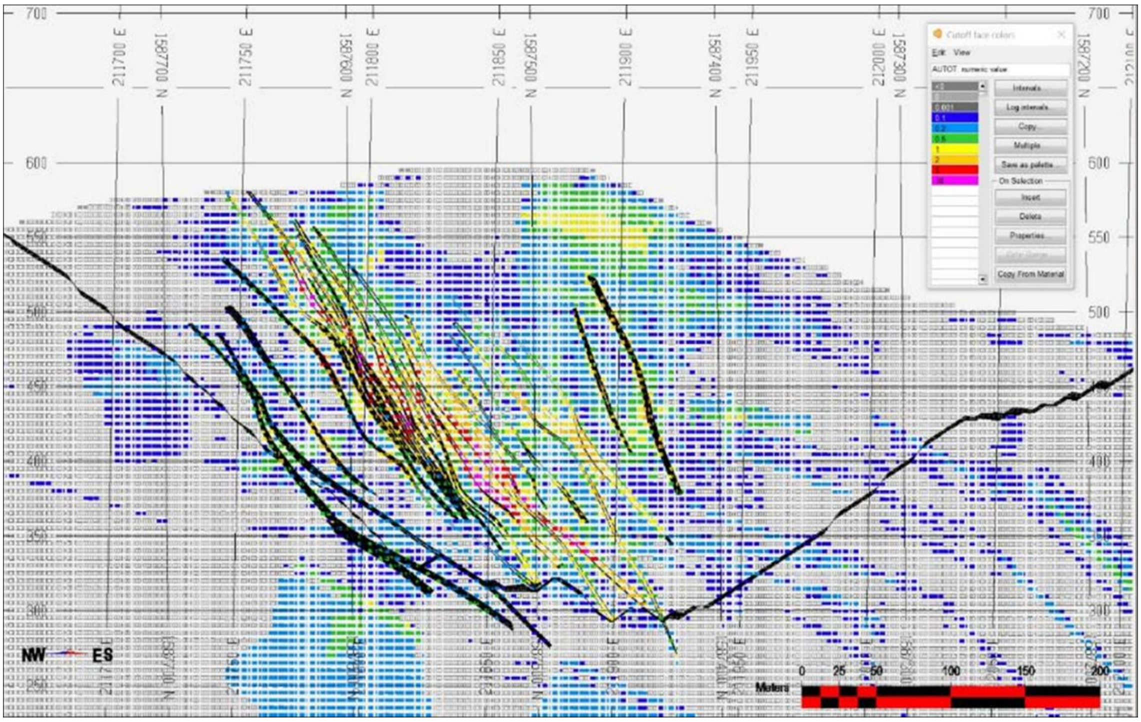

Figure 14-34: Section view of Au block model north

Source: Kirkham, 2025.

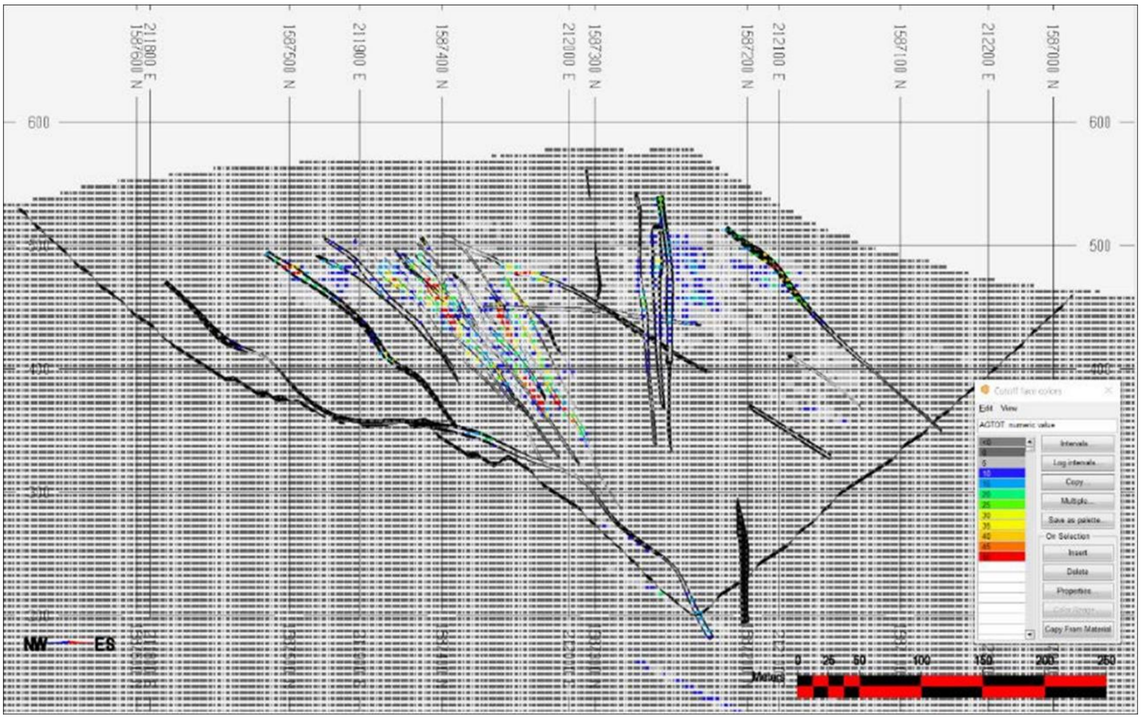

Figure 14-35: Section view of Ag block model north

Source: Kirkham, 2025.

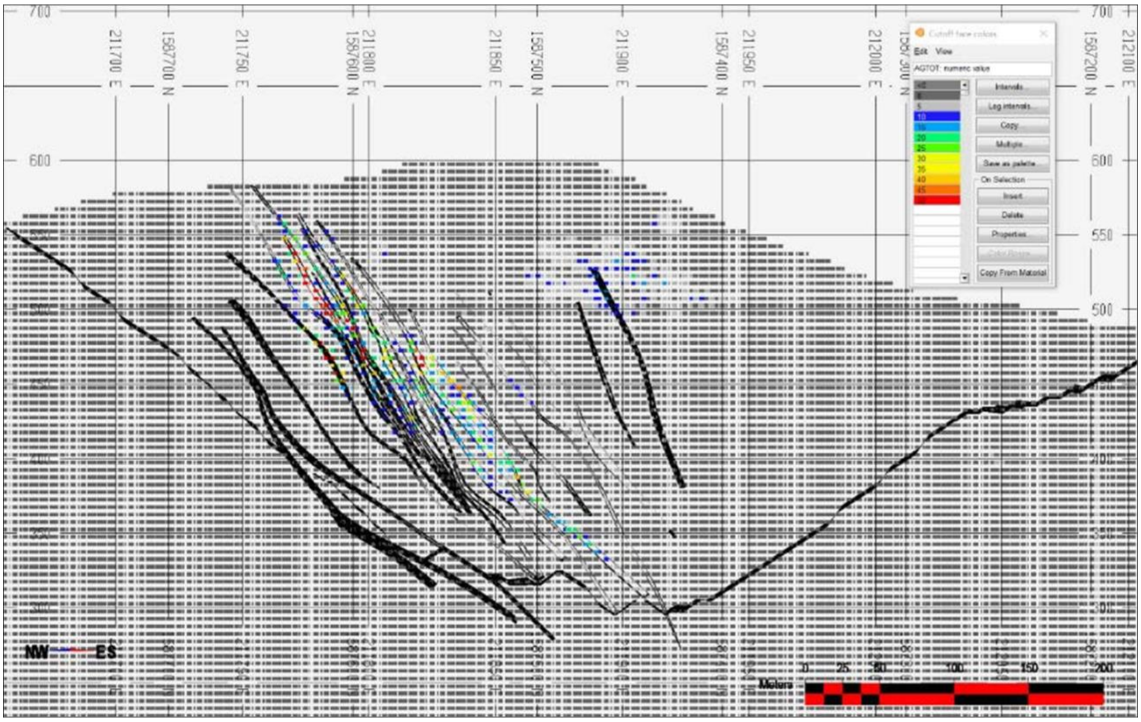

Figure 14-36: Section view of Ag block model south

Source: Kirkham, 2025.

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

The Mineral Resources are sensitive to the selection of cut-off grade. Table 14-23 shows tonnage and gradein the Era Dorada deposit at different gold cut-off grades.

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 14-23: 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**	6,396	9.26	31.39	1,905	6,454
	2.25	6,349	9.31	31.54	1,901	6,439
	2.45	6,289	9.38	31.74	1,897	6,419
	2.5	6,269	9.40	31.81	1,895	6,410
	3	5,969	9.74	32.74	1,868	6,282
	3.5	5,552	10.22	34.11	1,825	6,089
	4	5,087	10.82	35.81	1,769	5,857
Inferred	**2**	623	5.91	19.45	118	389
	2.25	605	6.02	19.68	117	383
	2.45	587	6.13	19.88	116	375
	2.5	580	6.18	19.98	115	372
	3	522	6.56	20.63	110	346
	3.5	446	7.12	21.55	102	309
	4	399	7.53	22.12	96	284

Notes:
1. All Mineral Resources have been estimated in accordance with the Canadian Institute of Mining and Metallurgy and Petroleum (CIM) definitions, with an effective date of December 31, 2020.
2. Mineral Resources reported demonstrate a 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.

14.15 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 nearby composites.
- Check the general drift and the local grade trends compared to the drift and local grade trends of thecomposites.
- Ensure that all blocks in the core of the deposit have been estimated.
- Check that topography has been properly accounted for.
- Check against the 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 highassays.

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 estimatedwas found; it appears that every block grade could be explained as a function of the surrounding compositesand 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 the distance of the first composite to the nearest block and the average distance to blocks for all composites used gives a quantitative measure of confidence that blocksare 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.

14.16 Discussion with Respect to Potential Material Risks to the Resources

There are no known environmental, permitting, legal, taxation, title, socioeconomic, political or other relevant factors that materially affect the Resources.

15 MINERAL RESERVE ESTIMATES

Mineral Resources are not Mineral Reserves and have no demonstrated economic viability. This PEA does not support an estimate of Mineral Reserves since a pre-feasibility or Feasibility Study is required for reporting Mineral Reserve Estimates. This report is based on potentially mineable material (mineable tonnes and/or mineable Resources).

Mineable tonnages were derived from the Resource model described in the previous section. Measured, Indicated, and Inferred Resources were used to establish mineable tonnes.

Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that all or any part of the Mineral Resources or mineable tonnes would be converted into Mineral Reserves.

16 MINING METHODS

16.1 Introduction

The potentially mineable Resources at the Era Dorada deposit will be extracted using underground mining methods, specifically a combination of mechanized long hole stoping (LH) and cut-and-fill (MCF), utilizing both paste and rock backfill. Long hole stoping is expected to account for approximately 77% of total production, while cut-and-fill will contribute the remaining 23%.

Mining method selection was primarily guided by geotechnical rock quality, vein geometry, and orebody continuity. Where geotechnical or geometric conditions necessitated it, mechanized cut-and-fill (MCF) was selected. Otherwise, long hole stoping was preferred due to higher productivity and lower unit mining costs relative to MCF. The proposed mine plan is designed to achieve a targeted production rate of 1,500 t/d, with a total mine life estimated at 17 years.

Indicated and Inferred Mineral Resources were included in the mine design and schedule optimization process supporting this PEA. Inferred Resources are considered too geologically speculative to apply economic parameters to allow their classification as Mineral Reserves, and there is no certainty that any portion of the Inferred Resources will be upgraded to a higher Mineral Resource category. The LoM plan outlined in this PEA is based on a Resource inventory comprising approximately 78% Indicated and 22% Inferred Resources. No Mineral Resources have been classified in the Measured category.

16.2 Deposit Characteristics

High-grade mineralization at the Era Dorada deposit is hosted within laterally stacked, sub-parallel narrow veins that generally strike northeast, with average azimuth ranging from 25° to 50°. Veins dips vary, including both tabular and near-vertical geometries. However, the average dip of high-grade structures is approximately 50° to 55°. The average vein thickness ranges from 2 to 10 m, with an average spacing of 8 m between parallel structures. The potentially mineable Resource ranges from 50 m at the lowest levels to 300 m near the surface. The mineralized system comprises more than 50 modelled veins with variable geometry along both strike and dip.

Mineralization is concentrated within two main zones, the North and South. Both zones exhibit similar vein geometry and spacing. The South Zone contains a greater number of veins by volume and extends to lower elevations compared to the North Zone. The combined strike length of the mineralized zone is approximately 800 m. High-grade mineralization was identified from 540 masl to 180 masl. More than 50% of the total mineralized volume occurs between 400 masl and 480 masl. In general, lower-grade mineralization envelopes the higher-grade lenses.

16.3 Geotechnical Analysis and Recommendations

16.3.1 Rock Mass Characterization

A geotechnical investigation was carried out by Golder in 2011 and 2012 to support underground mine design, during which 16 geotechnical drill holes were logged and sampled for laboratory strength testing. Point load testing was also performed on selected core samples.

In 2018, JDS conducted an independent geotechnical review using the historical data as a baseline. As part of this effort, JDS performed geotechnical face mapping at 15 underground development headings and completed geotechnical logging of two additional Resource drill holes. Oriented core was also collected from five drill holes located in both the North and South Zones.

The geotechnical assessment for this study is supported by the following dataset:

- RQD and core recovery data from the Resource drill hole database;
- detailed logging of 16 geotechnical drill holes from the 2011/2012 campaign;
- over 1,500 point load tests from 43 drill holes (2011/2012);
- laboratory strength testing programs from 2011 and 2018 (UCS, Brazilian tensile strength, and elastic properties);
- oriented core data from five drill holes (2018);
- geotechnical face mapping at 15 underground stations (2018);
- 3D lithologic and structural models developed by SGM (2018).

16.3.2 Geotechnical Domains and Rock Mass Properties

Based on the geologic structural and lithology models and the geotechnical characterization data described above, the deposit was divided by JDS (2018) into three separate geotechnical domains. Each of the domains grouped areas of similar characterized ground conditions and overall rock mass quality, which were then used to develop geotechnical design parameters. Geologic structure and lithology were identified as the dominant factors controlling rock quality domains.

The three geotechnical domains are shown on an E-W cross-section in Figure 16-1 and summarized below. Table 16-1 contains a summary of the key rock mass properties derived from the 2011/2012 geotechnical core logging data. The data in Table 16-1 represent the average value of all the core runs drilled within the respective domains. Local variations will occur, but the values presented are expected to be representative of the overall rock mass behaviour.

1. **Domain 1**: comprises the upper lithological units of the deposit, including the upper lapilli tuff and lower volcanic sediments, both fine-grained clastic rocks characterized by closely spaced fractures. The Salinas Conglomerate, which overlies the South Zone, is also included in this domain due to its intense clay alteration. Additionally, a structurally bounded wedge of poor-quality rock, delimited by the Upper Lapilli Tuff Fault and the Ramp Fault, is part of Domain 1. This fault block has been significantly downthrown, resulting in bedding distortion and intense fracturing. Overall, Domain 1 is of poor geomechanical quality, with heavily fractured rock and relatively low intact rock

strength. The average uniaxial compressive strength (UCS), derived from point load test conversions, is 71 MPa. The mean Rock Mass Rating (RMR76) is 50, classifying the rock mass as "Fair" according to Bieniawski (1976). The corresponding average Q' value is 1.9, placing it in the "Poor" category per Barton's classification (1974). Geotechnical mapping from five underground stations within this domain reported RMR76 values ranging from 42 to 69 (mean of 53 – "Fair") and Q' values from 0.8 to 7.8 (mean of 2.4 – "Poor").

2. **Domain 2:** comprises the middle lithological units of the deposit, including andesite tuff, lower lapilli tuff, and breccias. It also includes two sandstone lenses: one located above the andesite tuff and another below the lower lapilli tuff. Compared to Domain 1, Domain 2 exhibits significantly less fracturing and higher intact rock strength. The average uniaxial compressive strength (UCS), derived from point load test (PLT) conversions, is 78 MPa. The mean Rock Mass Rating (RMR76) is 58, which corresponds to "Fair" geomechanical quality according to the Bieniawski (1976) classification system. The mean Q' value is 4.7, also classifying the domain as "Fair" rock mass quality per Barton's (1974) system. Geotechnical mapping at four underground stations reported RMR76 values ranging from 55 to 81, with a mean of 65—indicating "Good" rock quality. Corresponding Q' values ranged from 3.1 to 33, with a mean value of 6.6, placing the domain within the "Fair" category under the Barton classification.
3. **Domain 3**: comprises the lower stratigraphic units of the deposit, located beneath Domain 2. These include limestone, quartz latite crystal lithic tuff, and conglomerate, as well as a sandstone lens and quartz latite unit situated between the limestone and quartz latite. The Salinas Conglomerate in the South Zone may also be included in Domain 3, where it has undergone strong silicification. This domain is characterized by good geomechanical quality, with significantly lower fracture density and higher intact rock strength. The average uniaxial compressive strength (UCS), based on point load test (PLT) conversions, is 93 MPa, with some laboratory-tested values reaching up to 233 MPa—likely due to samples with exceptionally high silica content. Domain 3 has a mean Rock Mass Rating (RMR76) of 66, corresponding to "Good" rock mass quality according to the Bieniawski (1976) classification. The mean Q' value is 12, which also falls under the "Good" category as per Barton's (1974) classification system. As of the current study, there are no underground exposures in Domain 3 available for geotechnical mapping.

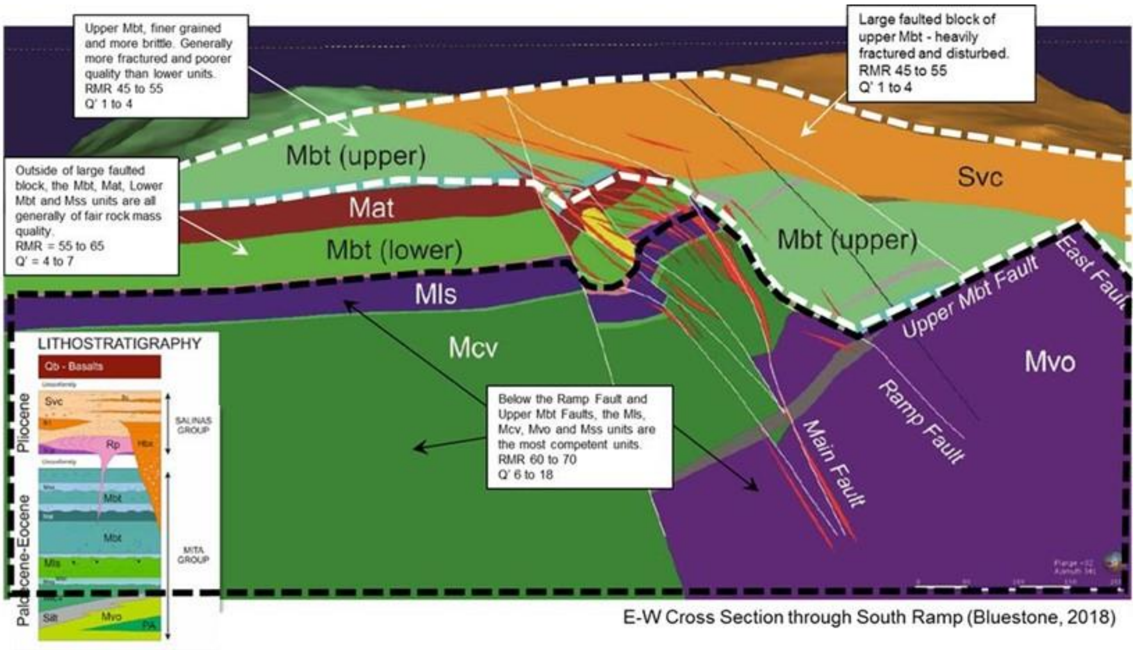

Figure 16-1: Cross-section of geotechnical domain boundaries (looking North)

Source: Bluestone, 2019.

Table 16-1: Mean rock mass properties by domain for 2011/2012 geotechnical core logging data

Domain Nº.	No. of Runs	Weath.[1] (ISRM)	IRS[2] (ISRM)	Recov. (%)	RQD (%)	UCS (MPa)[3]	RMR_{76}	Q'[4]
1	185	W2.6	R3.4	75	29	71	50	1.9
2	625	W2.4	R3.5	89	43	78	58	4.7
3	603	W1.9	R3.8	95	67	93		

Notes:
1 According to ISRM (1978) rock weathering grade index, the results indicate slightly (W2) to moderately weathered (W3) rock.
2 Logged according to ISRM (1978) intact rock strength/hardness system, recorded independent of PLT results. Values indicate medium-strong (R3) to strong (R4).
3 Mean UCS values were calculated using the PLT database and a calculated correlation factor of 21, according to ISRM's (1985) suggested methods.
4 Q' calculated from RMR76' values using Bieniawski's 1989 equation (Q' = e [RMR76 – 44]/9).
Source: Bluestone, 2019.

The 2011/2012 geotechnical logging was done only according to the RMR76 (Bieniawski, 1976) format requiring that a conversion be made to Q' (Barton, 1974), which is necessary for stope and ground support design. An equation developed by Barton (1979) was used to estimate Q' values from the RMR76 values. The conversion Q' to/from RMR using an equation is generally not preferred over collecting the necessary information for each system independently and may not be applicable for all rock masses. However, the equation was validated for the Era Dorada rock mass by comparing the Q' and RMR76 values collected independently by JDS during the underground geotechnical mapping program (Figure 16-2). In addition, JDS geotechnically logged two recent (2018) Resource drill holes and confirmed similar Q' data for Domains 1 and 2 compared to the converted 2011/2012 data.

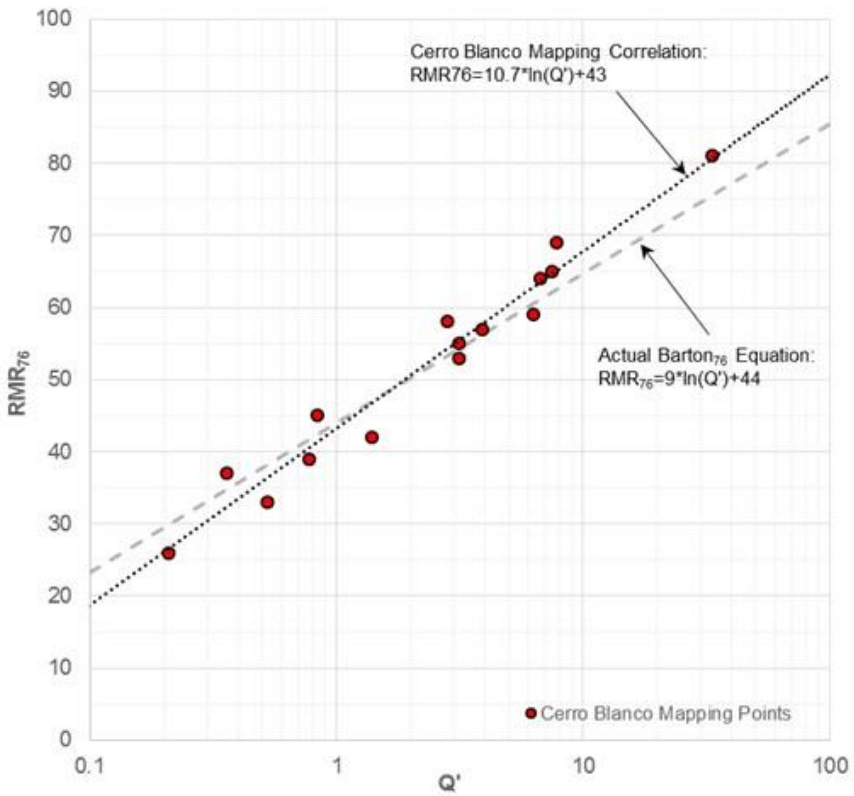

Figure 16-2: JDS (2018) geotechnical mapping Q' values vs. RMR76 values

Source: Bluestone, 2019.

16.3.3 In-situ Stresses

According to JDS, in-situ stresses at the Project have not been directly measured but were estimated using regional geological data and surface topography. Based on the World Stress Map (Heidbach *et al.,* 2016), the nearest stress data, derived from single focal mechanism earthquake events, indicate a strike-slip to the normal faulting regime, with a maximum horizontal stress (σ_1) oriented approximately 345° azimuth. These data were rated as quality 'C' (±25° accuracy).

Given the relatively shallow depth of the planned stoping areas (200–300 m below ground surface), the maximum horizontal stress magnitude is assumed to be low. No underground indicators of high horizontal stress have been observed.

For design purposes, the horizontal-to-vertical stress ratio (σH/σV) is conservatively assumed to range from 1.5 to 2, with σH aligned subparallel to the vein strike. The minimum horizontal stress (σH) is assumed to range from 0.5 to 1 times σV. These assumptions were used in calculating the stress parameter A for the stope design. Additional assessment of horizontal stress conditions may be warranted during operations.

16.3.4 Empirical Stope Design Analysis

Empirical stope design was conducted using stability graph methods, where the stability number (N’), calculated from rock mass quality (Q’) and empirical factors A (stress), B (structure), and C (stope dip), is plotted against the stope face hydraulic radius.

Maximum stope dimensions were estimated using the Potvin (2001) method, considering the expected range of rock mass conditions across the three geotechnical domains. Results were cross-validated using the Trueman & Mawdesley (2003) ‘Stable’ line. Both upper and lower bound estimates of stope geometry and rock mass quality (RMR76 and Q’) were evaluated. A summary of these estimates is provided in Table 16-2.

Table 16-2: Design rock mass quality ranges by geotechnical domain

Domain Nº.	Lower and Upper Ranges		
	RQD	RMR76	Q’1
1	40 to 60	45 to 55	1 to 4
2	60 to 80	55 to 65	4 to 7
3	70 to 100	60 to 70	6 to 18

Note: [1] Q’ calculated from RMR76’ values using Bieniawski’s 1989 equation (Q’ = e [RMR76 – 44]/9).
Source: Bluestone, 2019.

Stability number (N’) calculations were based on the following empirical factors:

- Stress factor (A): assumed as 1, reflecting relatively strong intact rock (UCS 70–100 MPa) and low horizontal stress at shallow depths (200–300 m bgs).
- Joint orientation factor (B): set to 0.3, as dominant discontinuities are sub-parallel to the vein orientation.
- Gravity factor (C): 3.8 for 45° dipping hanging walls and 2.0 for flat backs.

Maximum level spacing was set at 20 m. Hydraulic radii were calculated assuming a stope height of 28 m (true height across a 45° dip).

The geometric inputs for stope design by domain are presented below:

- Domain 1:
  - cut-and-fill stopes up to 4 m high, 5 m wide, and 50 m long.
- Domain 2:
  - longitudinal stopes up to 10 m wide without cable bolting;
  - maximum stable length: 20 m (45° hanging wall);
  - transverse stopes >8 m wide require cable bolts from top cuts.
- Domain 3:
  - longitudinal stopes up to 10 m wide without cable bolting;
  - maximum stable length: 20 m (45° hanging wall);
  - transverse stopes >8 m wide require cable bolts from top cuts.

16.3.5 Estimates of Unplanned Dilution

Unplanned dilution for stope hanging walls and footwalls was estimated using the Equivalent Linear Overbreak/Slough (ELOS) method (Clark, 1998), which relates the stability number (N') to the hydraulic radius of stope faces. Unlike stability charts, the ELOS plot presents contours of expected to overbreak thickness distributed across the stope walls.

The method is not applicable for small hydraulic radii (<4), where dilution is primarily governed by blasting quality. Accordingly, dilution in cut-and-fill stopes within Domain 1 and relevant areas of Domains 2 and 3 was estimated based on anticipated excavation and blasting practices. For long hole stopes in Domains 2 and 3, ELOS-based dilution estimates are summarized in Table 16-3, alongside assumed values for cut-and-fill stopes. Dilution allowances for longitudinal long hole stopes were conservatively increased relative to those for transverse stopes.

Table 16-3: Estimates of unplanned dilution for long hole and cut-and-fill stopes by domain

Mining Type	Stope Wall	Domain Nº.		
		1 (Poor) A	2 (Fair)	3 (Good)
LHOS	Hanging Wall	NA	0.50	0.50
	Footwall	NA	0.40	0.30
Cut and Fill	Hanging Wall	0.50	0.25	0.15
	Footwall	0.50	0.25	0.15

Source: Bluestone, 2019.

16.3.6 Backfill Strength Requirements

Stopes are planned to be backfilled with cemented paste backfill to provide lateral confinement to stope walls and to waste rock pillars between closely spaced veins, particularly in long transverse stopes. The use of cemented backfill also allows the mining of adjacent secondary stopes without leaving rib pillars.

Required uniaxial compressive strengths (UCS) for the paste backfill were determined analytically using a simplified wedge analysis (Mitchell, 1983), assuming a 25 m stope height and a conservative safety factor of 2.0. These values are summarized in Table 16-4. While the safety factor may be reduced to 1.5 or 1.3 during operations, a higher factor is appropriate at the PEA level.

Numerical stress modelling of backfill behaviour may be considered in future stages once site-specific data is available; however, such modelling is not recommended at the current level of study.

Table 16-4: Required UCS for various stope widths

Stope Width (m)	Design Backfill UCS (kPa)
2	75
5	175
10	285
15	375
20	445
25	500

Source: Bluestone, 2019.

16.3.7 Ground Support

Ground support requirements were assessed using the Barton & Grimstad (1994) Q-system, which incorporates rock mass quality (Q') and the Excavation Support Ratio (ESR) to account for the function and expected lifespan of each opening. ESR values of 1.6 were applied to permanent and man-entry ore development, while a value of 3 was used for temporary, non-entry ore headings.

Based on the Q-system, most permanent and temporary ore development openings require only spot bolting to remain stable. However, pattern bolting with welded wire mesh is recommended in all areas where personnel will be working to control loose rock.

Due to concerns regarding long-term resin degradation under sustained high temperatures at Era Dorada, Swellex bolts were selected for permanent development instead of resin-anchored bolts. Groundwater pH ranges from 7.5 to 8.5, indicating minimal corrosion risk for ground support elements.

Recommended support strategies for each excavation type are summarized in Table 16-5 and Table 16-6.

Table 16-5: Ground support recommendations for ore development

Development Type	Design	Domain 1	Domain 2	Domain 3
Temporary/Ore Development Back Bolts: LHOS (4 m x 4 m); and, Shanty Back Cut and Fill (4 m H x 4 to 6 m W).	Bolt Type	Swellex Pm12	Swellex Pm12	Swellex Pm12
	Bolt Diameter (mm)	27.5	27.5	27.5
	Bolt Length (m)	2.1	2.1	2.1
	Bolt Spacing (m)	1.2	1.5	1.5
	WWM Required	# 6-gauge, 10 cm opening size		
Temporary/Ore Development Wall Bolts: LHOS (4 m x 4 m); and, Shanty Back Cut and Fill (4 m H x 4 to 6 m W).	Bolt Type	Split Sets	Split Sets	Split Sets
	Bolt Diameter (mm)	39	39	39
	Bolt Length (m)	1.8	1.8	1.8
	Bolt Spacing (m)	1.2	1.5	1.5
	WWM Required	# 6-gauge, 10 cm WWM to within 1.5 m from the floor		
Temporary/Ore Development 3-Way Intersections: LHOS (4 m x 4 m); and, Shanty Back Cut and Fill (4 m H x 4 to 6 m W).	Bolt Type	Cable Bolts	Cable Bolts	Cable Bolts
	Bolt Diameter (mm)	Single Strand	Single Strand	Single Strand
	Bolt Length (m)	4.0	4.0	4.0
	Bolt Spacing (m)	2.0	2.0	2.0
Temporary/Ore Development 4-Way Intersections: LHOS (4 m x 4 m); and, Shanty Back Cut and Fill (4 m H x 4 to 6 m W).	Bolt Type	Cable Bolts	Cable Bolts	Cable Bolts
	Bolt Diameter (mm)	Single Strand	Single Strand	Single Strand
	Bolt Length (m)	5.0	5.0	5.0
	Bolt Spacing (m)	2.0	2.5	2.5
Temporary/Ore Development Back Bolts: Shanty Back Cut and Fill (4 m H x 6 to 8 m W).	Bolt Type	Cable Bolts	Cable Bolts	Cable Bolts
	Bolt Diameter (mm)	Single Strand	Single Strand	Single Strand
	Bolt Length (m)	4.0	4.0	4.0
	Bolt Spacing (m)	1.2	1.5	1.5
	WWM Required	# 6-gauge, 10 cm	# 6-gauge, 10 cm	# 6-gauge, 10 cm
Temporary/Ore Development Wall Bolts: Shanty Back Cut and Fill (4 m H x 6 to 8 m W).	Bolt Type	Split Sets	Split Sets	Split Sets
	Bolt Diameter (mm)	39	39	39
	Bolt Length (m)	1.8	1.8	1.8
	Bolt Spacing (m)	1.2	1.5	1.5
	WWM Required	# 6-gauge, 10 cm WWM to within 1.5 m from the floor		
Ore Development 3-Way Intersections Shanty Back Cut and Fill (4 m H x 6 to 8 m W).	Bolt Type	Cable Bolts	Cable Bolts	Cable Bolts
	Bolt Diameter (mm)	Single Strand	Single Strand	Single Strand
	Bolt Length (m)	5.0	5.0	5.0
	Bolt Spacing (m)	2.0	2.5	2.5
	Bolt Type	Cable Bolts	Cable Bolts	Cable Bolts
Development Type	Design	Domain 1	Domain 2	Domain 3

Development Type	Design	Domain 1	Domain 2	Domain 3
Ore Development 4-Way Intersections: Shanty Back Cut and Fill (4 m H x 6 to 8 m W).	Bolt Diameter (mm)	Single Strand	Single Strand	Single Strand
	Bolt Length (m)	6.0	6.0	6.0
	Bolt Spacing (m)	2.0	2.5	2.5
Estimates of Shotcrete Required: LHOS (4 m x 4 m); and, Shanty Back Cut and Fill (4 m H x 4 to 8 m W).	Percent Required	5.0	0.0	0.0
	Thickness (cm)	5.0	0.0	0.0
	Max Distance from Floor (m)	0.0	0.0	0.0

Source: Bluestone, 2019.

Table 16-6: Ground support recommendations for permanent development

Development Type	Design	Domain 1	Domain 2	Domain 3
Permanent Development (5 m x 5 m) and Intersection Primary Support	Bolt Type	Swellex Pm12	Swellex Pm12	Swellex Pm12
	Bolt Diameter (mm)	27.5	27.5	27.5
	Bolt Length (m)	2.4	2.4	2.4
	Bolt Spacing (m)	1.2	1.5	1.5
	WWM Required	# 6-gauge, 10 cm WWM to within 1.5 m from the floor		
Permanent Development 3-Way Intersections (Secondary Support)	Bolt Type	Cable Bolts	Cable Bolts	Cable Bolts
	Bolt Diameter (mm)	Single Strand	Single Strand	Single Strand
	Bolt Length (m)	4.0	4.0	4.0
	Bolt Spacing (m)	1.5	1.5	1.5
Permanent Development 4-Way Intersections (Secondary Support)	Bolt Type	Cable Bolts	Cable Bolts	Cable Bolts
	Bolt Diameter (mm)	Single Strand	Single Strand	Single Strand
	Bolt Length (m)	5.0	5.0	5.0
	Bolt Spacing (m)	2.0	2.5	2.5
Estimates of Shotcrete Required	Percent Required	25.0	10.0	5.0
	Thickness (cm)	5.0	5.0	5.0
	Max Distance from Floor (m)	0.0	0.0	0.0

Source: Bluestone, 2019.

16.4 Hydrogeology Analysis and Recommendations

Dewatering of the Era Dorada mine has been a limiting economic factor in mine development because of high-temperature groundwater and thermal gradients. The average static groundwater-level elevation in the Project area is approximately 462 masl. Mine dewatering activities from pumping wells and two in-tunnel sump pumps (combined discharge of about 600 gallons per minute – g/m) have lowered the static groundwater level to approximately 440 m at the mine as of 2013. The existing location of the portals, dewatering wells, and monitoring wells are shown in Figure 16-3.

The current conceptual model suggests that as previous dewatering efforts at Era Dorada increased, groundwater levels declined but remained within the lower Salinas tuff-volcaniclastic sequence and upper Mita Group units. The lower hydraulic heads created by pumping likely enhanced gradient- and thermal-driven recharge through faults and interconnected fractures. There is little information pertaining to the hydrogeology of the faults to determine if they provide higher permeability conduits to flow or act as barriers limiting connection between fault zones and the overall effect on long-term water quality within the area. Regional and intermediate groundwater flow systems between the Ipala Volcano and the Rio Paz shear zone likely also contribute to gradient-driven flow to Era Dorada under pumping conditions. The regional contribution of groundwater is not fully understood and needs to be further evaluated as part of the future dewatering infrastructure installation, as evidence suggests that a broad-relatively flat drawdown cone will result and could impact potential water uses and discharges to the natural environment.

16.4.1 Evaluation of Dewatering Rates and Number of Locations

Historical estimates for mine dewatering have ranged from 2,000 g/m to as high as 10,000 g/m for various mine plans and have advanced as more data becomes available to better document the interconnections of the various fault systems to the regional flow system.

An analysis made by JDS resulted in the northern area of the mine reaching a level of 210 m within five years of mine start-up. In the southern area, the planned mine level is 270 m, which is also attained within five years. The change in planned mining depth reduced the overall dewatering demands at Era Dorada and has allowed focused dewatering strategies to be developed.

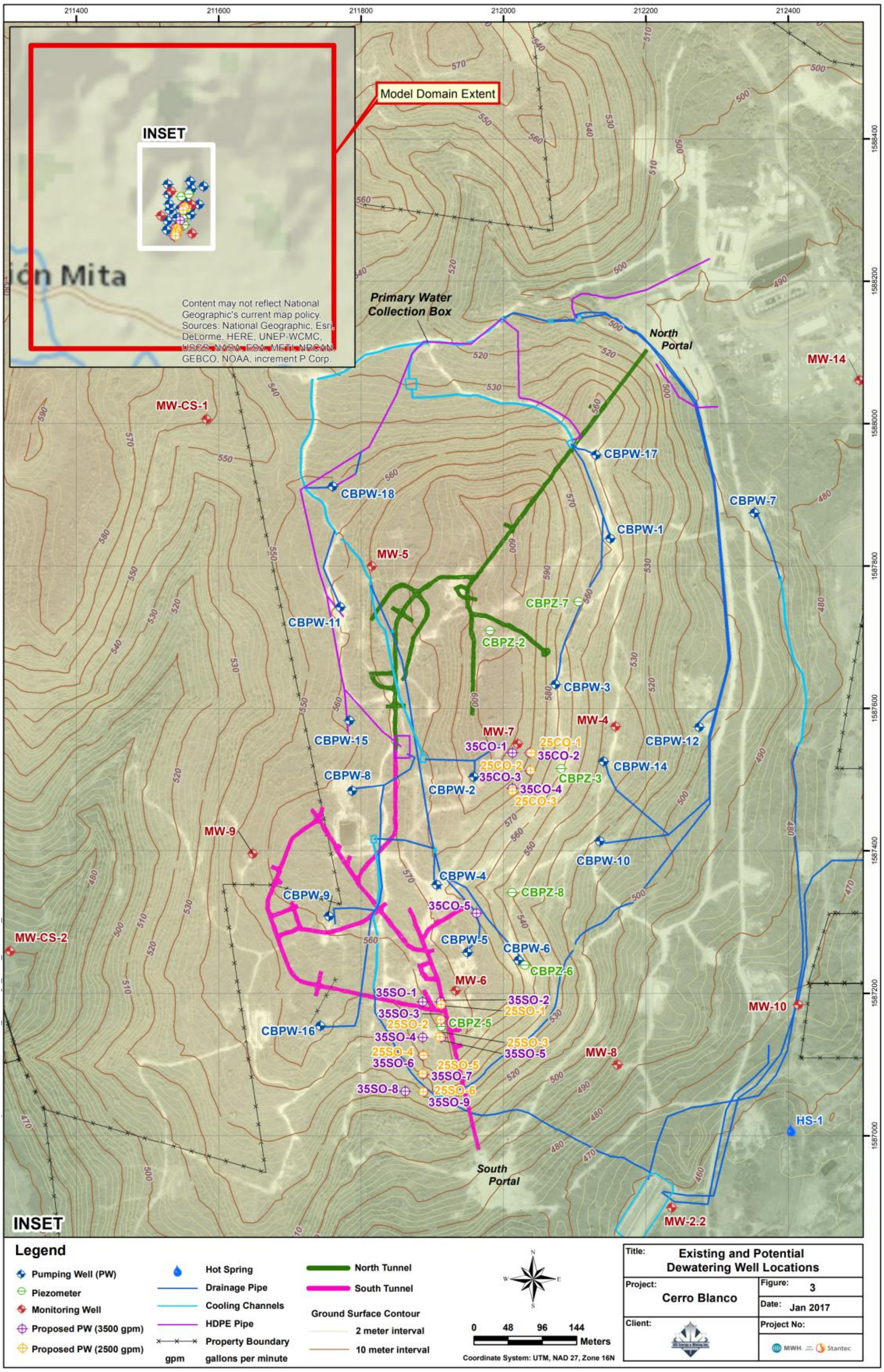

Figure 16-3: Existing location of portals, dewatering wells, monitoring wells and new dewatering well locations

Source: Bluestone, 2017.

As part of the preliminary dewatering evaluation, a simplified numerical groundwater flow model was used to simulate potential impacts associated with groundwater pumping for mine dewatering. The numerical model (MODFLOW) used a single unconfined layer with 462 m of saturated thickness and a grid of 200 rows by 200 columns (40,000 total cells) with a cell size of 25 x 25 x 500 m. The model assumed homogeneous aquifer characteristics, with a hydraulic conductivity of 0.07285 m/day and a storage coefficient of 0.028. These hydrologic parameters are based on the average values derived from testing and analyses conducted previously by MWH (2014). A general head boundary condition was used to simulate flux into and out of the model along all periphery model cells (a total of 796 general head cells). No surface recharge or evapotranspiration was simulated. This model provides a simplified tool for the calculation of dewatering rates and volumes over time and will require further development and calibration to refine dewatering estimates and injection strategies.

Dewatering wells were simulated at combined rates of 2,500 g/m and 3,500 g/m to evaluate the timeframe for dewatering areas around the current mine plan. The location of the dewatering wells for both scenarios is shown in Figure 16-3 (note: 2,500 g/m wells are called the “25” series, and 3,500 g/m wells are called the “35” series). While both dewatering rates, as predicted by the model, reach the desired dewatering levels, the 2,500 g/m rate does not achieve the dewatering in the desired timeframe planned for mine advancement. Therefore, a dewatering rate of 3,500 g/m was selected for a preliminary rate to be used in this PEA. The model-predicted water levels over time are shown in Figure 16-4, with key timeframes for dewatering levels based on the mine plan provided by JDS.

Higher dewatering rates may be required to achieve the 210 masl dewatering level within the required time frame, depending on aquifer storage and regional connection through fault zones. These details are not known at this time and require further assessment and analysis to better confirm the ultimate dewatering requirements. The 3,500 g/m dewatering rate is based on overcoming the existing storage within the aquifer and targeted dewatering along fault zones that provide connection to the regional flow system. If regional influence is significant, dewatering rates could be over twice the focused dewatering rates of 3,500 g/m and would require more wells to achieve dewatering. The increase in dewatering rates would have an effect on the need for additional injection wells and/or water treatment capacity based on the anticipated water quality. This will need to be further evaluated in the next study phase.

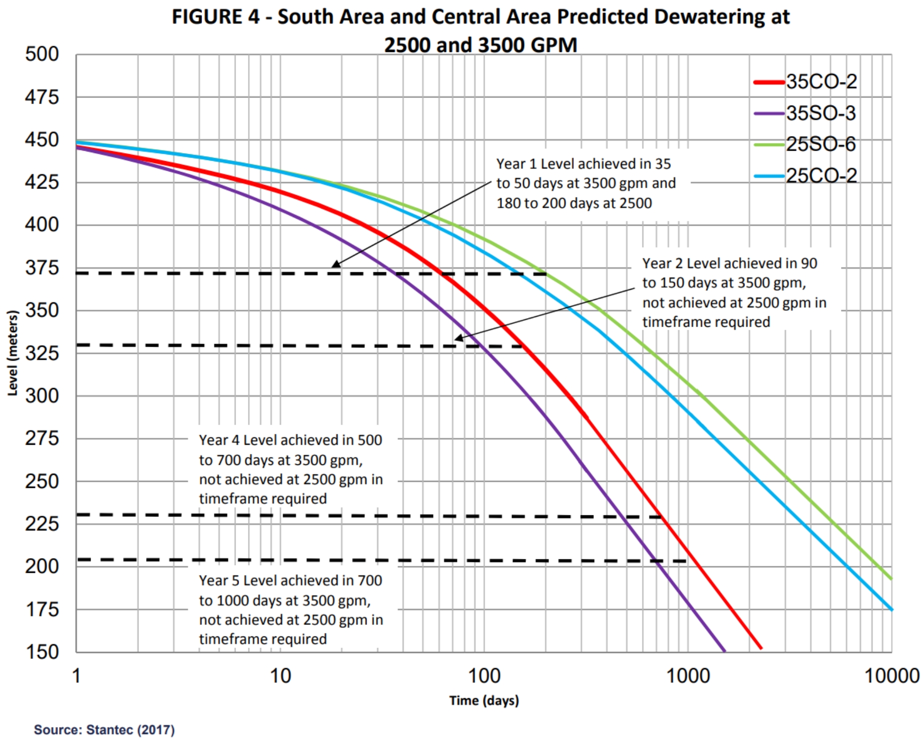

Figure 16-4: Simulation hydrograph – south area and central area predicted dewatering at 2,500 and 3,500 g/m

Source: Bluestone, 2017.

The 3,500 g/m scenario uses 10 existing wells and 14 new dewatering wells, pumping an average of approximately 145 g/m each (see Figure 16-4). The 10 existing wells were selected because of their proximity to the mineralized zones where dewatering is required; however, the current condition of these wells is not known. Further evaluation of the existing capacity and condition is required, and either redevelopment may be required, or if conditions have deteriorated beyond recovery, new wells may be required. The new wells will likely be 12-inch diameter casing in an 18-inch diameter borehole. Telescoping perforations starting at 26 inches will likely be required for the drilling depths. It is recommended that ten piezometers be installed in small diameter boreholes (3.345-inch diameter) and equipped with vibrating wire line transducers and temperature sensors to monitor pressure changes and temperature changes in the rock as part of the dewatering program. With the new dewatering wells, these can be phased over the first two years of planned operations. Year -1 and Year 1 would require six wells installed each, and the remaining two wells could be installed in Year 2.

While the model shows the dewatering being reached within the desired timeframe for mining at a rate of 3,500 g/m, it does not account for the thermal effects that will be at the site, heterogeneities of the aquifer, and faults that may connect the mine site to regional groundwater flow systems, and seasonal variations in groundwater recharge. Observations from the site suggest that the system is highly compartmentalized, which may necessitate the installation of additional wells to achieve the dewatering condition. The degree of compartmentalization will need to be measured with a vibrating wire line piezometer installed in the mine area to assess pressures and temperatures. In addition, a more regional assessment of the potential effects of dewatering on the groundwater Resources of the area will be required as it is expected that a broad and flat drawdown cone will result from the proposed dewatering.

Furthermore, potential changes in water quality over time will need to be better understood as this will impact the need for treatment to meet discharge criteria or for the suitability of the water for reinjection. The water quality assessment will need to consider the potential effects of dewatering on overall water quality and water Resources in the area.

Finally, it is anticipated that precipitation will continue to supply approximately 400 to 600 g/m of water to the shaft, which would need to be dewatered through sumps. Overall, the water modelling suggests that the mine plan should expect to handle (through treatment or injection) a minimum water volume in the range of 4,000 g/m.

Table 16-7: Wells planned and executed

Well	Approx. Pumping Rate to Achieve Planned Dewatering (g/m)	Dewater well Depth for Planning (m)	Dewater Borehole Diameter (inches)	Casing and Screen Diameter (inches)	Easting[1] (m)	Northing (m)
CBPW-2	85	Existing	Existing	Existing	Existing	Existing
CBPW-3	93	Existing	Existing	Existing	Existing	Existing
CBPW-4	141	Existing	Existing	Existing	Existing	Existing
CBPW-5	141	Existing	Existing	Existing	Existing	Existing
CBPW-6	125	Existing	Existing	Existing	Existing	Existing
CBPW-8	146	Existing	Existing	Existing	Existing	Existing
CBPW-9	141	Existing	Existing	Existing	Existing	Existing
CBPW-11	292	Existing	Existing	Existing	Existing	Existing
CBPW-14	141	Existing	Existing	Existing	Existing	Existing
CBPW-15	130	Existing	Existing	Existing	Existing	Existing
35-CO-1	180	450	18	12	212,012.5	1,587,538
35-CO-2	165	450	18	12	212,037	1,587,538
35-CO-3	125	450	18	12	212,037.3	1,587,512
35-CO-4	175	450	18	12	212,012.5	1,587,487
35-CO-5	165	450	18	12	211,962.1	1,587,312
35-SO-1	145	450	18	12	211,886.4	1,587,188
35-SO-2	125	450	18	12	211,911.6	1,587,188
35-SO-3	135	450	18	12	211,912.8	1,587,163
35-SO-4	150	450	18	12	211,886.4	1,587,138
35-SO-5	125	450	18	12	211,912.8	1,587,138
35-SO-6	150	450	18	12	211,886.8	1,587,112
35-SO-7	125	450	18	12	211,887.5	1,587,086
35-SO-8	150	450	18	12	211,862.3	1,587,062
35-SO-9	150	450	18	12	211,888.3	1,587,062
Total	**3,500**	**6,300**				

Note: [1] NAD1927UTMZn16N.
Source: Bluestone, 2017.

16.4.2 Injection Wells

The existing water treatment plant at Era Dorada can treat 1,500 g/m. Considering the dewatering rates are expected to be 3,500 g/m and an additional 500 g/m for mine sump water, total water treatment volumes from dewatering will be approximately 4,000 g/m in the peak dewatering periods. This volume does not consider any contact water that will be managed at the mine site. This means a total of approximately 2,500 g/m of dewatering water will need to be managed by other means than the current water treatment plant. It is recommended that 10 injection wells be installed that are capable of managing approximately 250 g/m each and will be confirmed during future injection well testing. The injection wells would be drilled to a depth of about 150 m using a 12-inch diameter casing in an 18-inch diameter borehole located on the mine property. The preliminary locations of the injectioSeventeen pumpingn wells are shown in Figure 16-5. The injection wells are planned to be located on the south side of the mine so that groundwater withdrawals from dewatering have less impact on the local community and in order to limit pull back of water into the dewatering zone. The injection wells will likely be much shallower than the dewatering wells and will be more focused on the shallow alluvium and upper fractured bedrock. One additional injection well is planned to handle the water from the dry stack tailings facility (DSTF) runoff pond. As with the dewatering wells, injection wells can be phased in, with six planned in Year -1 and five in Year 1. Water quality, especially high iron content in the groundwater discharged to the injection wells, could cause well fouling and present a future issue with the ability to inject water efficiently. A more detailed evaluation of water quality and suitability of the water to meet injection requirements and regulatory approvals should be completed during the next study phase.

Further test work will be carried out in the next phase of engineering to confirm the shallower depth of injection wells will not adversely impact groundwater quality in surrounding communities and the efficiency of the injection system.

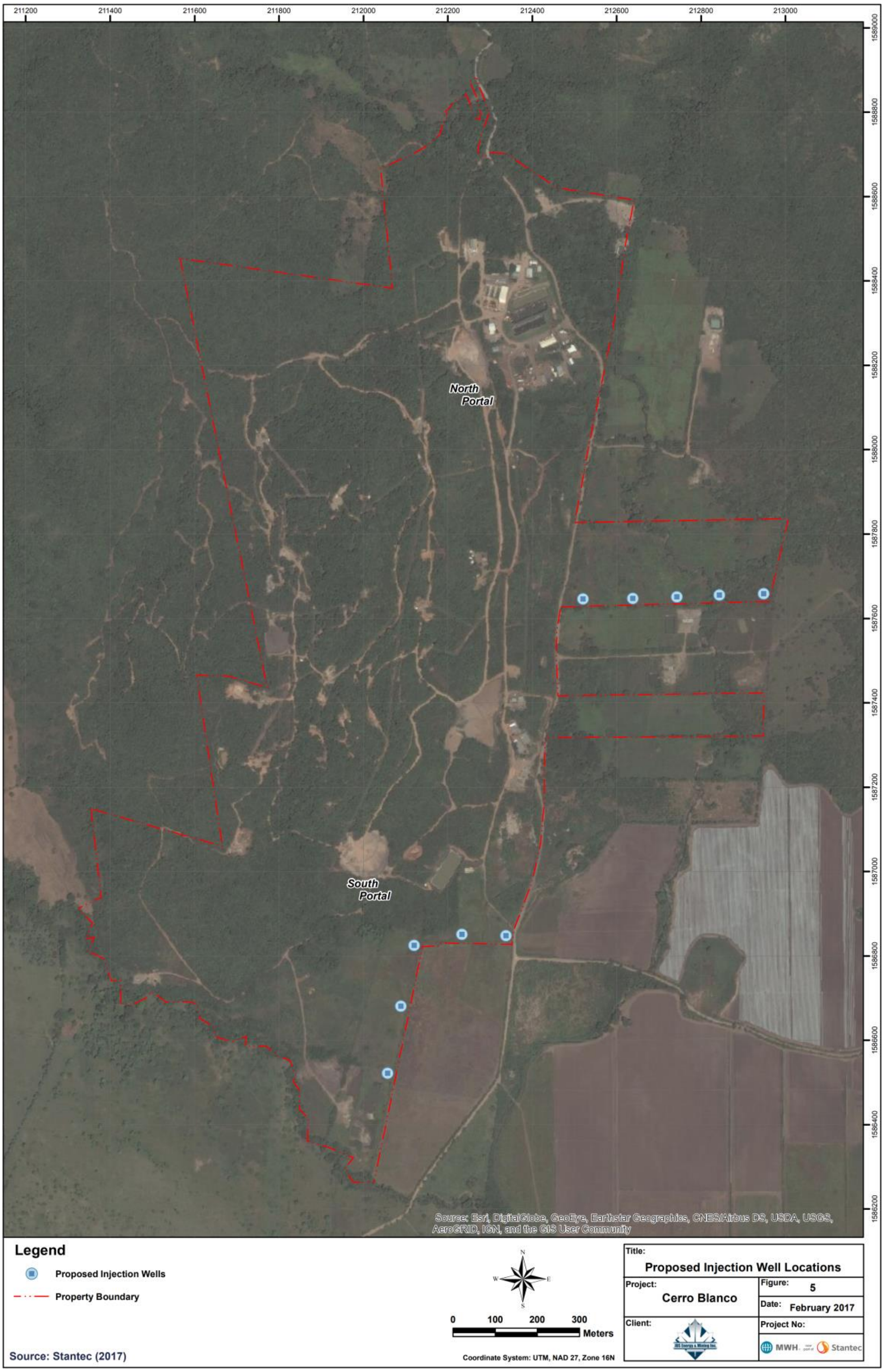

Figure 16-5: Preliminary location of injection wells

Source: Bluestone, 2017.

16.5 Mining Methods

Two mining methods are planned for the Era Dorada deposit: sub-level long hole (LH) and cut-and-fill (MCF). The mine will be divided into mining blocks, each comprising two to three sublevels and operating independently to enhance operational flexibility and maintain production rates.

Sill pillars will be established between blocks to ensure safe working conditions and support high recovery rates. Multiple mining blocks will be mined concurrently, enabling the project to achieve the targeted production rate before the spiral decline reaches its full depth. Each mining block will be extracted using an overhand (bottom-up) sequence. A typical level layout is illustrated in Figure 16-6.

The total average mining dilution assumed for the overall deposit was 17%, based on a combination of empirical stability analysis, ELOS-based overbreak estimates, and operational assumptions related to blasting and excavation practices. This value includes both planned and unplanned dilution across all mining methods and geotechnical domains.

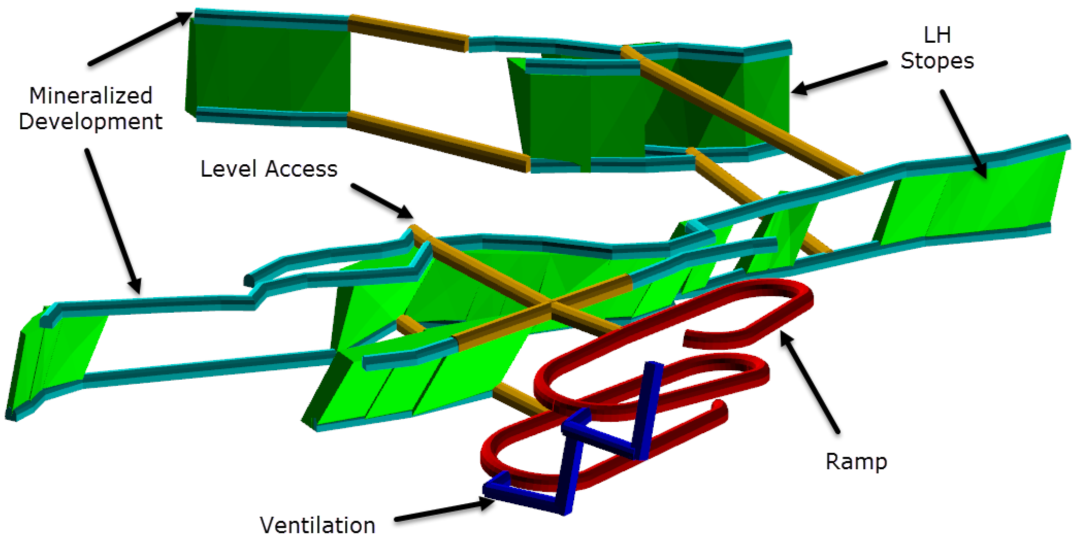

Figure 16-6: Perspective view of a typical mining level

Source: Bluestone, 2017.

16.5.1 Long hole Mining

Sub-level long hole (LH) stoping is the preferred mining method at Era Dorada due to its lower operating cost and higher productivity. It is suitable for steeply dipping, continuous vein geometries in competent ground. LH stoping will be applied where geotechnical and geometric conditions allow for efficient stope design.

Two LH configurations will be used: longitudinal and transverse. Longitudinal stoping will be employed in the thickest and most continuous zones of the deposit. Stopes are designed to be up to 10 m long and 20 m high, with thicknesses ranging from 2 m to 50m. The minimum stope width is 2.0 m before accounting for dilution.

Stopes will be mined using a bottom-up (overhand) retreat sequence toward level access. Each stope is accessed via 5 x 5 m crosscuts above and below. Where stopes exceed 15 m in width, they will be divided into multiple panels with parallel access drifts and backfilled with cemented paste.

At Era Dorada, selected long hole stopes will be backfilled with cemented paste fill, primarily to provide structural confinement in areas where stopes are adjacent or where vein geometry requires support for subsequent mining. In transverse stopes, a primary/secondary mining sequence will be implemented, with primary stopes being backfilled to allow the safe extraction of adjacent secondary stopes without the use of rib pillars. In longitudinal stopes, structural backfill will be placed in all mined-out stopes to ensure stability during extraction. Backfill placement will occur from the top sill using paste lines, ejector trucks, or LHDs, depending on stope access and geometry. However, not all stopes will require backfill, only those where ground conditions, sequencing, or safety considerations necessitate its use.

The total mining dilution for long hole stopes is estimated at 17% by mass, accounting for both primary and secondary stopes. This value is based on an assumed overbreak of 0.30 m on both the hanging wall and footwall. Dilution grades were extracted from the geological block model, and a minimum stope width of 2.0 m was applied prior to dilution. These assumptions were incorporated into the mine plan and economic model to reflect anticipated operating conditions.

Figure 16-7 shows a typical mining sequence for LH at Era Dorada.

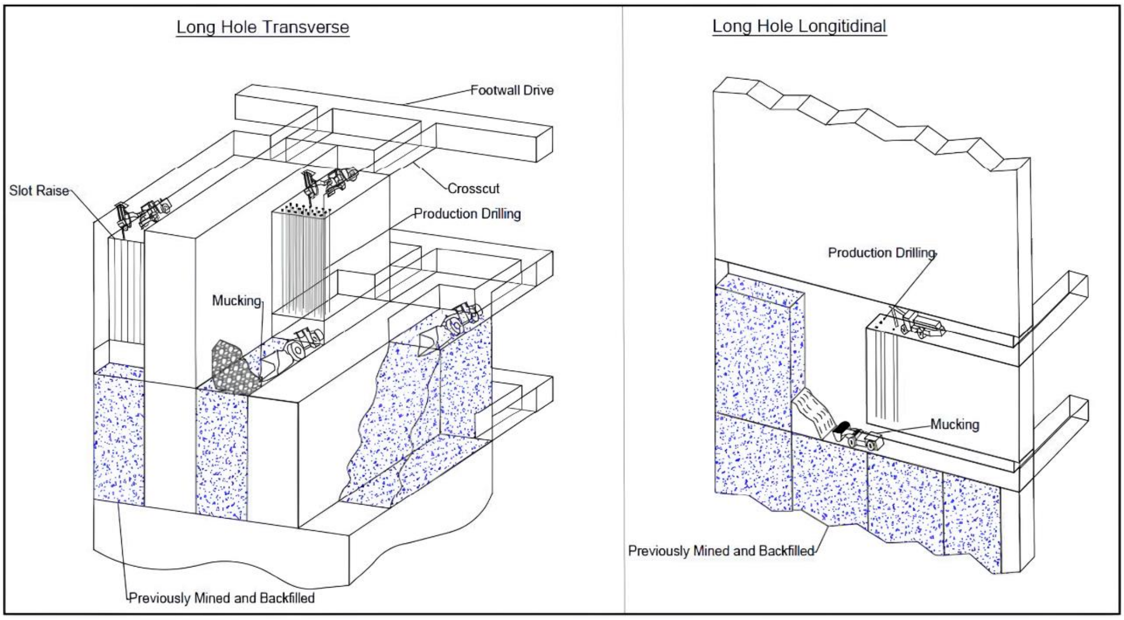

Figure 16-7: Long hole open stoping

Source: Bluestone, 2019.

16.5.2 Mechanized Cut-and-Fill

Overhand mechanized cut-and-fill (MCF) stoping is planned for areas of lower rock quality and/or where the geometry of the orebody is not conducive to long hole (LH) stoping. MCF is a highly selective underground mining method well-suited for narrow, steeply, or shallow dipping high-grade veins in weak ground conditions.

Mining begins at the base of the ore block and progresses upward. Each stope lift is supported temporarily with rock bolts, followed by the placement of a cemented backfill to form a competent working floor for subsequent lifts. The backfill is designed primarily for floor support rather than full structural confinement.

Access between successive MCF lifts is achieved via attack ramps driven at a maximum 15% gradient from the main level access. Within a typical 20-metre level interval, three MCF lifts of 4 m each are planned. The remaining 8 m to the next sublevel are mined in retreat using LH up-holes, avoiding development beneath sill pillars and enhancing miner safety. Wherever possible, internal on-vein ramping is used to reduce waste and lower costs.

A minimum rib pillar spacing of 4.0 m is maintained between adjacent MCF drives. In narrower veins where this spacing is not feasible, a primary/secondary mining sequence is implemented, with primary cuts backfilled using cemented structural fill to enable safe extraction of adjacent secondary cuts.

A schematic of a typical stope development is displayed in Figure 16-8.

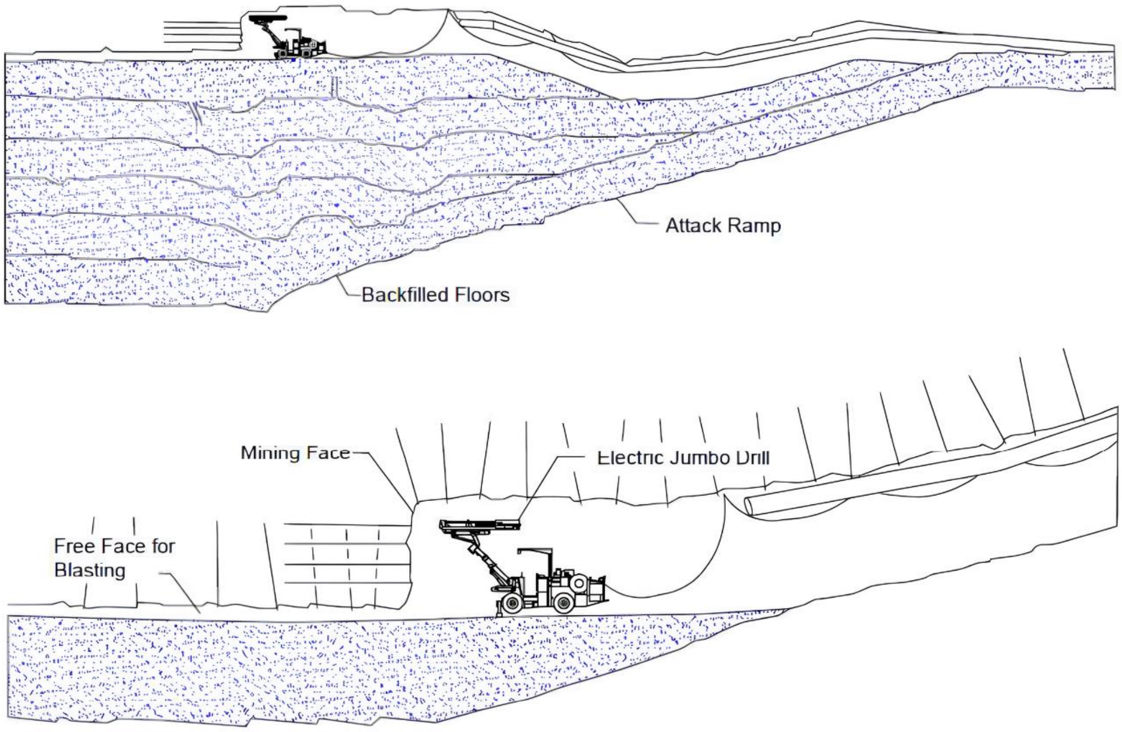

Figure 16-8: Mechanized cut-and-fill

Source: Bluestone, 2019.

16.6 Mine Design

16.6.1 Design and Optimization

Mine planning for the Project was conducted by GE21 using Datamine Studio UG and Mineable Shape Optimizer (MSO) software. A long section of the complete design is shown in Figure 16-9.

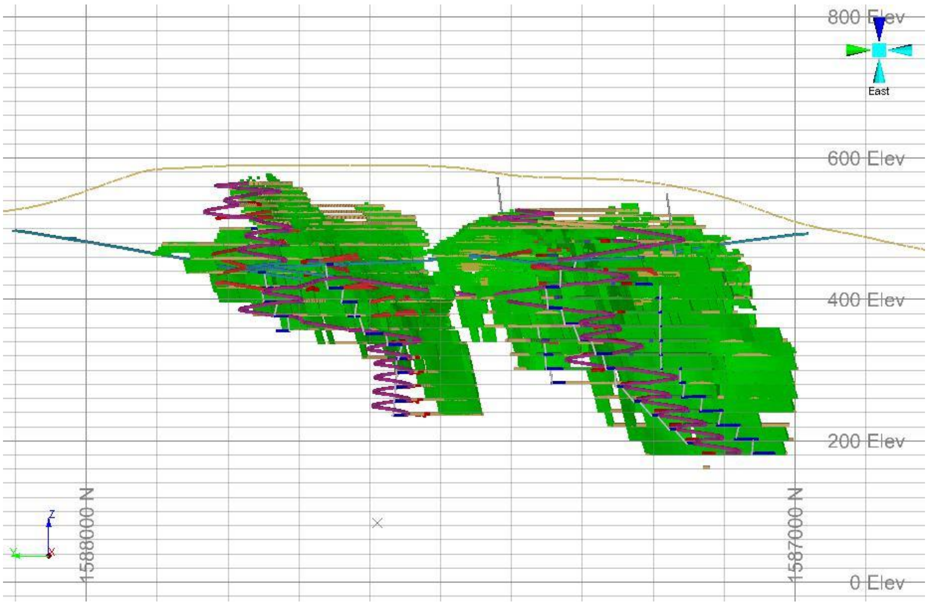

Figure 16-9: Mine long section

Source: GE21, 2025.

Mine design was carried out based on a gold cut-off grade (CoG) calculation specific to each mining method considered within the Resource model. The CoG was determined using estimated gold price, metallurgical recovery, mining, processing, general and administrative (G&A) costs, and applicable royalties. The input parameters used for CoG calculation are summarized (see Table 16-8).

Table 16-8: Cut-off grade calculation inputs

Parameter	Unit	Value	
		LH	MCF
Gold price	US$/oz Au	2,000	
Project Parameters			
Process Recovery	%	96.00%	
Payable metal	%	99.92%	
TC/RC	US$/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 (Underground)	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	2.82	3.07

Source: GE21, 2025.

The MSO software was used to generate optimized stope shapes based on a set of design constraints, including minimum dip angle, stope width, and gold cut-off grade (CoG). Stopes designed for mechanized cut-and-fill (MCF) were located within Geotechnical Domain 1, while long hole (LH) stopes were placed within Domains 2 and 3, where geotechnical and geometric conditions allowed.

Following stope optimization, mine development and access layouts were designed to ensure practical and efficient extraction sequences. Subsequently, the production schedule was optimized using Datamine's Enhanced Production Scheduler (EPS). The scheduler prioritized early access to higher-grade zones while respecting operational constraints such as maximum lateral development advance rates, plant nameplate capacity, paste backfill placement limits, and minimum required backfill cure times.

Mine planning was conducted using a representative stope dimension of 20 m height by 10 m width, with a 2.82 g/t Au cut-off grade for the long hole method, and 4 m high and 5 m width with a 3.07 g/t Au cut-off grade for mechanized cut-and-fill. The full set of parameters used in the selected MSO optimization trial is summarized in Table 16-9.

Table 16-9: Stope optimization parameters

Parameter	Unit	Value
Block Model		April 29, 2025, BM export V2
Cut-off Variable	ppm	AuOK
Stope Orientation Plane		YZ
Framework Bearing	degrees	-20/ 90 Maximum change of 90
Step X	m	10 / 5 (LHS/MCF)
Step Z	m	20 / 4 (LHS/MCF)
Cut-Off	Au g/t	2.82 / 3.07 (LHS/MCF)
Minimum Stope depth	m	2
Wall Dilution	m	1 / 0.3 (LHS/MCF)
Top to Bottom Max Ratio	#	2.25
Max Strike Deviation	degrees	45
Minimum Dip Footwall	degrees	50
Minimum Dip Hanging wall	degrees	130
Max Dip Change between stopes	degrees	45

Source: GE21, 2025.

The stopes resulted from MSO and productive development defined the material to be input to the mining production schedule as "Run of Mine" (ROM). This material includes mineralized material classified as Indicated and Inferred Resources and also the waste within these shapes as diluent material.

The sum of these materials is named "mineable Resources" for this report and resulted in 8.9 Mt @5.01 g/t Au and 17.71 g/t Ag, after applying modifying factors: for operational mining recovery was assumed a value of 95% and for mining dilution 17%.

16.6.2 Access

The Era Dorada deposit will be accessed via two main declines: one servicing the North Zone and another the South Zone. The ramps will provide haulage routes for mineralized material and waste, serve as general access, and work as air-fresh intake paths for mine ventilation.

Previous exploration campaigns have resulted in the development of over 2,700 m of lateral underground workings, including two portals, declines, crosscuts, and vein drifts. These workings are developed at dimensions of 4.5 m wide x 5.0 m high and are equipped with electrical power supply, ventilation infrastructure, and air and water services.

The existing portals were constructed into the hillsides using steel arches, corrugated steel sheeting, and shotcrete. Surface infrastructure at each portal includes supply water tanks, compressed air tanks, and an electrical powerhouse. Although no intake fans or ducts are installed at the portals, the underground ventilation was planned via four existing 3.0 m diameter exhaust raises, each approximately 100 m in length, fitted with square concrete collars extending 1.5 m above ground.

These existing workings will be fully integrated into the proposed mine plan, serving as initial access, ventilation, and production levels. Additional ramps and declines will be developed at a maximum gradient of 15%, with typical dimensions of 5.0 m × 5.0 m to accommodate 30-t haul trucks and temporary 1.4 m diameter ventilation ducts. Separate ramps will be constructed to access the deeper levels of both the North and South Zones.

As the proposed mine does not descend much farther than 400 m below the surface, and mineralization begins near the surface, no shafts were investigated as part of this conceptual study.

Given that the mineralization begins near the surface and extends to a depth of less than 400 m, no vertical shafts were considered in this conceptual study.

16.6.3 Development Types

Spiral ramps will provide access to each production level spaced 20 m vertically apart. The spiral ramps are driven at -15% grade and 5.0 m by 5.0 m, with a maximum curvature radius of 25 m. At each operating level, the spiral ramp will run at 0% grade for 20 m to provide equipment with better visibility and turning abilities on and off the haulage ramp.

Each level is serviced with a footwall drive to provide ventilation, definition drilling, and crosscut development for stopes. Access drifts and footwall drives are developed 5.0 m x 5.0 m to allow truck access and reduce haul distance of LHDs. Footwall drifts are spaced a minimum of 15 m away from LH stopes to prevent stability issues as a result of production blasting.

Transverse LH stopes are accessed by 4.0 m x 4.0 m cross-cuts developed from footwall drives on 7.5 m spacing. Cross-cuts are used to provide a platform for LH production drills, as well as remote mucking access for blasted material.

MCF zones are accessed by attack ramps from footwall drives or the haulage ramp directly. Attack ramps are driven at a maximum 15% grade and will stack vertically to access multiple production levels from a single access point, as shown in Figure 16-5. MCF drifts are driven at 4.0 m x 4.0 m to maintain structural integrity in the lower rock quality areas for which cut-and-fill is targeted.

Ventilation access drifts are driven on each level to ensure fresh and exhaust air raise connections to the stoping levels. The cross-cuts are approximately 4.0 m x 4.0 m.

Remucks are excavated on the main ramp and footwall drives to reduce the development mucking cycle time. A maximum of 150 m separates the remucks, which are typically driven 5.0 m W x 5.0 m H x 12 m l.

The back at the intersection of remucks and the connecting drift will require slashing to 6.4 m H to allow full extension and dumping of the LHD bucket.

Water collection sumps are located on every level adjacent to the exhaust raise and after level intersection in the main ramps. Sumps have been sized at 4.0 m x 4.0 m. Additional cut-outs will be driven beside level sumps to accommodate a portable pumping skid that will collect water from the level sumps pump directly to the main dewatering sumps.

Electric power centers will be located outside the access drift on each level in drifts 4.0 m H x 4.0 m W. Additional power centers will be located adjacent to major power draws, such as main dewatering sumps and cooling machines.

Refuge station cut-outs 4.0 m x 4.0 m will be established on every level adjacent to fresh air raises. Portable refuge chambers will move between these cut-outs as needed, depending on activity within the mine. Refuge chambers will provide sufficient capacity for all persons working in the vicinity.

There is no plan to develop drifts dedicated entirely to diamond drilling. Any definition of diamond drilling will be carried out from footwall drives and level cross-cuts.

A fresh air raise of 3.0 m in diameter will be driven to connect the access drift of each level. Two exhaust raises 3.0 m in diameter will be developed at the extent of footwall drifts on each level where possible.

The raises are driven via a raisebore or long hole machine, depending on height. Fresh air raises will be equipped with ladders for secondary egress. The raises are sequenced in a leapfrog pattern to enable the fresh air to be carried in the direction of the ramp progression. Some return air raises will be equipped with dewatering lines and paste delivery lines as needed to supply each level of the mine.

In general, long-term development will incorporate a 1.0 m radius arched back, while all temporary drifts will be driven with a flat back. In areas of poor ground, it may be required to drive stope sublevels with an arched back, as their life span is generally longer than that of an MCF drift.

Figure 16-10 depicts the various drift dimensions used in the Era Dorada mine plan. Figure 16-11 and Figure 16-12 depict the general arrangement of the mine plan in a long section and plan view.

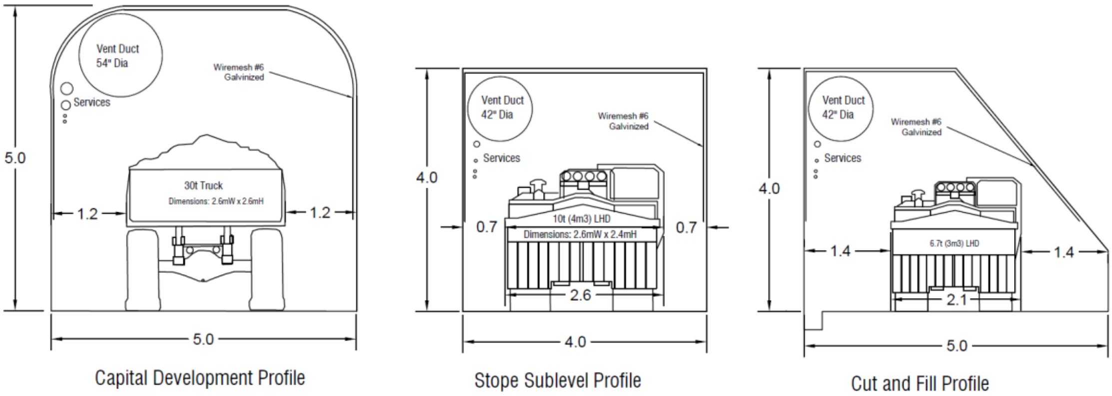

Figure 16-10: Drift profiles

Source: Bluestone, 2019.

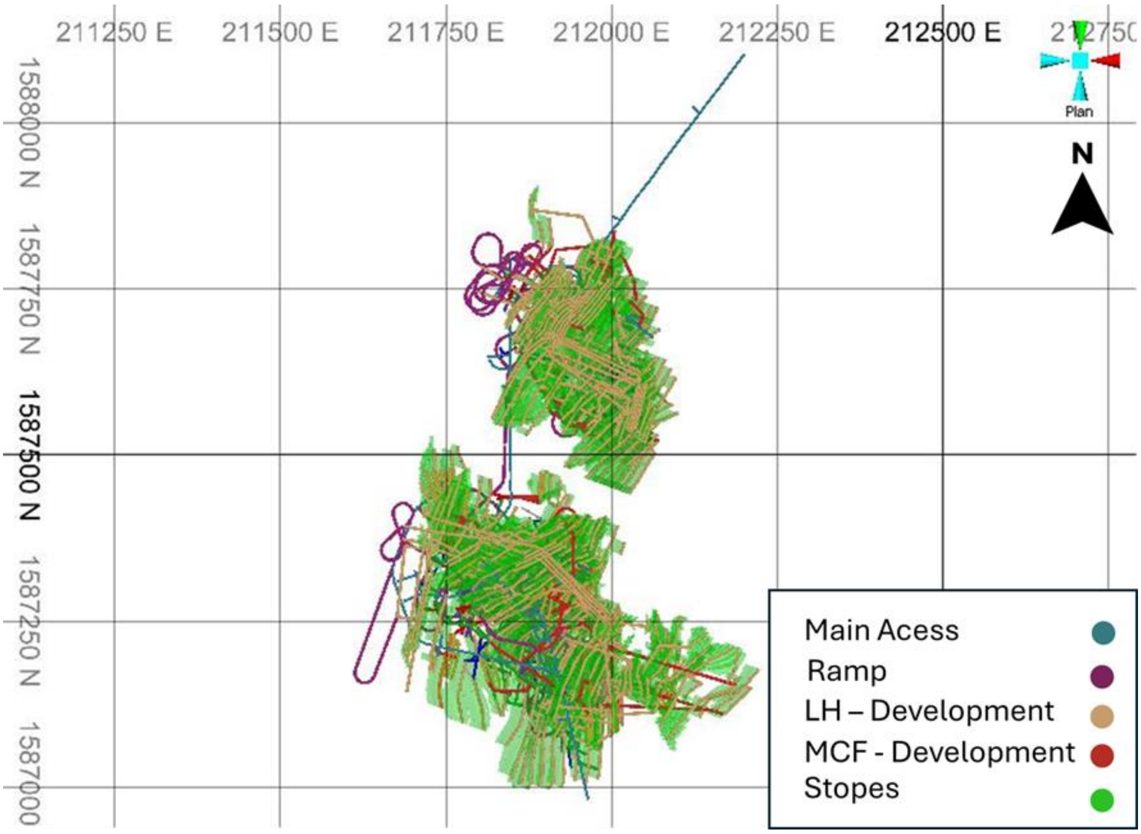

Figure 16-11: Mine design plan view

Source: GE21, 2025.

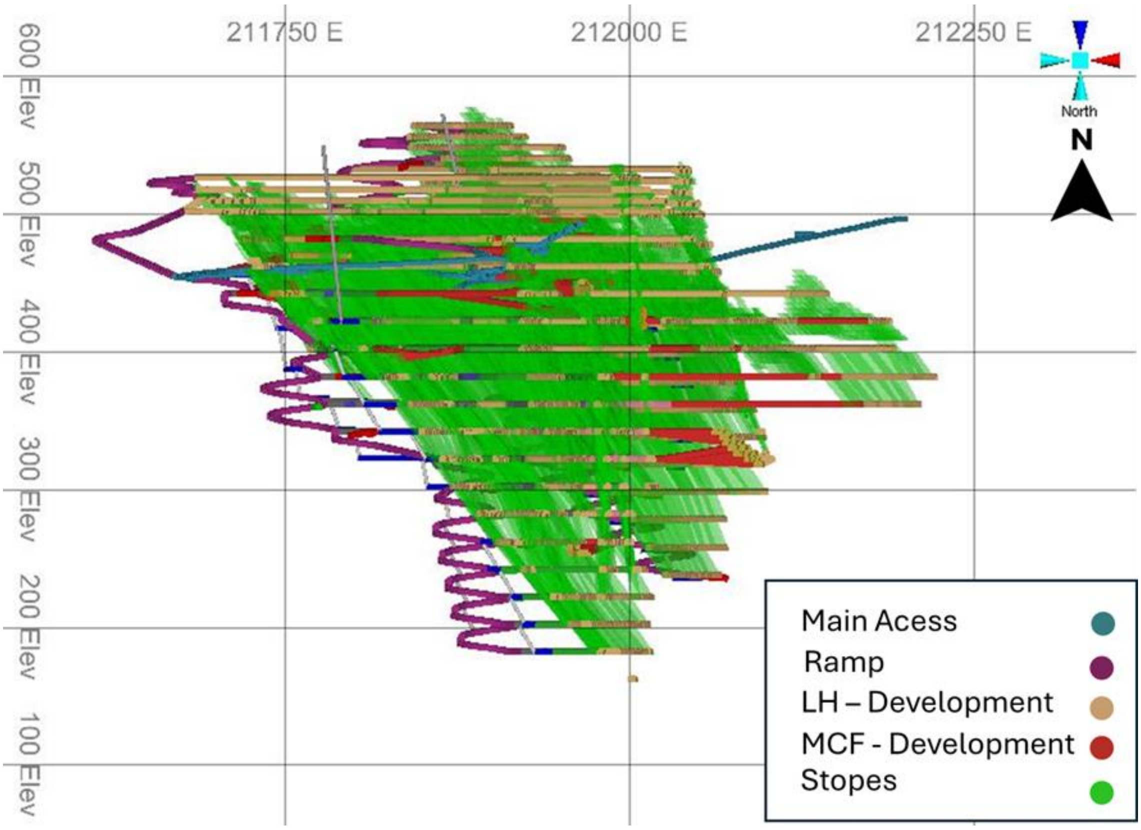

Figure 16-12: Mine design long section (looking Northwest)

Source: GE21, 2025.

The geotechnical review prepared by JDS (2018) highlighted the potential difficulty and increased support requirements involved in creating large open stopes in the weaker ground zones in the Era Dorada Mineral Resource. As a result of this, the mine design has been optimized to restrict LH stoping to Domain 2 and 3, and MCF stopes extract the remaining economic resource from Domain 1 (South of the upper geotech domain boundary line).

16.7 Mine Services

16.7.1 Mine Ventilation

The ventilation system for the Era Dorada operation has been designed to dilute and remove dust, diesel emissions, and blast fumes and provide cooling of the mine workings. The ventilation network was modelled using Ventsim software by JDS 2018 and adapted by GE21 according to the new production plan. The Era Dorada deposit requires additional air to pull away excess heat and control air temperatures.

A total of five ventilation raises are required to ventilate the mine. The Project currently has four ventilation raises; therefore, only one new ventilation raise is required. As the declines are being developed, a series of ventilation drop raises will be developed concurrently. Three ventilation raises are planned to be used as exhaust raises. The remaining two raises are planned to be utilized for fresh air intake, along with the two portal ramps. The return air raises (RAR) will be required to keep air velocity on the ramp at or below 6 m/s. The fresh air raise (FAR) will also act as a means of secondary egress. The new raise will be developed using a raise climber at a 4 m by 4 m profile. Lateral ventilation drifts at a 4 m by 4 m profile will be required to follow the decline and connect the ventilation circuits to the decline and level access.

Minimum airflow requirements were based on the expected diesel emissions of the underground mining fleet required at peak mine production. Additional airflow is used underground for general cooling. The power rating of each piece of equipment was determined, and the utilization factors representing the equipment in use at any time were applied to estimate the amount of air required. Equipment specified for the site has undergone testing by either MSHA or CANMET to determine the ventilation requirements to dilute the engine emissions to a safe working level. The volume of air required for ventilating the diesel emissions is 165 m^3/s. An additional 25 m^3/s is used for cooling, and with a 10% safety factor, the final airflow requirements for the mine were calculated at 205 m^3/s.

Auxiliary fans will be used to ventilate the advancing development and active production levels. Fresh air will be sourced from the FAR and distributed using the auxiliary fans through ventilation ducting to the active mine areas.

In order to control the underground ventilation temperatures, underground mining equipment will be purchased with sealed and air-conditioned cabs. When working at the active face, portable spot coolers will be used. A chiller plant located on the surface will pump cold water through insulated pipes to the spot coolers used at the active faces underground to cool the air to approximately 28° C.

16.7.2 Water Supply

Service water will be required mainly for drilling, dust suppression and washing of development faces. Water will be supplied from a service water tank located close to the portal and will be gravity-fed to the underground work areas via 100 mm diameter pipelines. Pressure reduction valves will be installed along the decline as required. The service water tank will be refilled with cooled underground mine water or externally sourced water.

16.7.3 Dewatering

Inflows into the underground workings were estimated at 25 l/s for Year 1 through 3 and 38 l/s for Year 4 onwards to the end of the mine life.

The mine dewatering system is designed as two standalone systems, the northern system and the southern system. Both systems have been designed to accommodate a peak flow of 30 l/s and will use a combination of 4-inch piping and 4-inch drill holes drilled between levels to transport water. A summary of the dewatering system is summarized. The location of the sumps and pumps are shown in Table 16-10.

Table 16-10: Underground dewatering system

Sump No.	Section	Elevation (masl)	Pumps to	Pump (kw)	Booster (kw)
1	Northem	412	Surface via North Ramp	104	56
2	Northem	420	N Main Sump (up and around)	43	
3	Northem	360	N Main Sump	43	
4	Northem	275	360 Sump	104	56
5	Southern	440	Surface via south Ramp	104	
6	Southern	420	South Main	43	
7	Southern	330	S 420	56	
8	Southern	330	S 390	104	
9	Southern	250	S 330	104	56
10	Southern	210	S 250	104	

Source: GE21, 2025.

16.8 Unit Operations

16.8.1 Drilling

Development headings are planned to be driven with electro-hydraulic two-boom jumbos. Blast holes with 48 mm diameter will be drilled to a depth of 4.88 m. The advance per round is assumed to be 4.4 m. It is envisioned that one jumbo could drill between two to three rounds per shift.

Production drilling for the long hole stopes will be performed by long hole drills. Blast holes with 89 mm diameter will be drilled in a fan pattern from the overcut to the undercut.

16.8.2 Blasting

Development rounds will be charged by an explosives and ANFO loader. Lifter holes will be loaded with bulk emulsion. Blasting is planned to be initiated by non-electric (NONEL) detonators.

For long hole production blasting, bulk emulsion will be used together with NONEL detonators and Pentex boosters.

16.8.3 Ground Support

After mucking and scaling is complete, ground support will be installed by a mechanized scissor bolter. Typical ground support in access development is planned to consist of 2.4 m long resin rebar bolts in the back and in the walls at a spacing ranging from 1.5 x 1.5 m in moderate and poor ground. Welded wire mesh will be installed in all ground conditions. The anticipated breakdown between good, moderate, and poor is 25%, 50%, and 25%, respectively. In intersections, 3.0 m bolts will be used for deep-ground support.

It was assumed that 25% of the development would be in poor ground conditions, which would require shotcrete. A shotcrete machine will be used to apply shotcrete at 60 mm thickness.

16.8.4 Mucking

Blasted material from development headings will be mucked with a 6.0 yd^3 (10 t) LHD directly to a haul truck or to a remuck bay. Broken material from long hole stopes will be mucked by remote control LHD.

16.8.5 Hauling

30 t haul trucks will drive on the decline to surface, where they will dump the material on mineralized material or waste stockpiles in close proximity to the portal.

Haulage profiles for all production levels were generated to calculate equipment hours for the fleet.

16.8.6 Backfill

The selected mining methods require the placement of backfill for full extraction of the mineralized zones. Stopes require the use of cemented paste backfill to provide stability to exposed backfill walls when mining the adjacent stopes. The use of paste backfill will also minimize the storage requirements for process plant tailings on the surface. The paste will be mixed at a paste plant and pumped through pipelines underground to the stopes. A cement content of 8% was assumed for the cemented paste fill of primary stopes. Due to the high mica content of the mineralized material, 46% of the paste recipe will be crushed and screened waste rock, and 46% will be tailings from the plant. Further test work will be required to determine the optimum cement content, curing time and achievable backfill strength.

Underground development waste may be used for un-cemented backfill in attack ramps and remote stopes to minimize waste haulage to the surface.

16.9 Mine Equipment

The mobile equipment fleet to support the mining operation is summarized in Table 16-11.

Table 16-11: Mobile equipment fleet

Equipment	Avg	Peak	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17
Truck (30t/14.5 m^3)	3	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-
LHD (4.5t/2.0 m^3)	2	3	3	2	3	3	3	2	3	3	2	2	2	2	2	2	2	2	2	-
LHD (6.7t/3.0 m^3)	2	3	3	2	3	3	3	2	3	3	2	2	2	2	2	2	2	2	2	-
LHD (10t/4.0 m^3)	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Jumbo - 1 Boom	2	3	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Jumbo - 2 Boom	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Long hole Drill - Top Hammer	1	2	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Long hole Drill - ITH	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Bolter	3	4	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Exploration Drill	1	1	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Small Explosives Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Large Explosives Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Scissor Lift	1	2	1	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	1	1
Personnel Carrier	1	2	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2

Equipment	Avg	Peak	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17
Fuel / Lube Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Boom Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Electrician Truck	1	1	0	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	1	1
Utility Vehicle	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Backhoe	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Telehandler	0	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0	0	0	0	0
Mechanics Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Supervisor Truck	4	4	3	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4

Source: GE21, 2025.

16.10 Mine Personnel

The underground mine is planned to operate on two 12-hour shifts (day shift/night shift), 365 days per year, with four crews on rotation. Two crews will be on-site at any time, with the other crews off-site on break. Both hourly mining and maintenance personnel and salaried supervisors and technical staff will work on the same four days on- four days off-rotation.

Hourly personnel were estimated based on development and production rates, operation productivity and maintenance requirements.

Underground mining personnel requirements are summarized in Table 16-12.

Table 16-12: Underground mine operations personnel

Position	Avg. Quantity	Hourly/Salary
Mine Management		
Mine Manager	1	Salary
Mining Superintendent	1	Salary
Maintenance Superintendent	1	Salary
Technical Services Superintendent	1	Salary
Mine Foreman	1	Salary
Mine Clerk	1	Salary
Mining Operations (Production)		
Shift Supervisor	4	Salary
Blasting Supervisor	4	Salary
Trainer	10	Hourly
Blaster	7	Hourly
Development Services/Shotcrete	6	Hourly
Waste Development Miner	7	Hourly
LH Production Miner	7	Hourly
Scooptram Operator (Large)	19	Hourly
Haul Truck Operator	18	Hourly
Bolter Operator	7	Hourly
Jackleg/Stoper Miner	8	Hourly
Grader Operator	4	Hourly
Mining Operations (Services)		
Paste Plant Operators	8	Hourly
Backfill Miner	8	Hourly
Backfill Helper	8	Hourly
Mine Electrician	8	Hourly
Mine Maintenance		
Maintenance Supervisor	1	Hourly
Maintenance Planner	1	Hourly
Heavy Equipment Mechanic	16	Hourly

Position	Avg. Quantity	Hourly/Salary
Mechanic Helper	4	Hourly
Welder	8	Hourly
Electric/Hydraulic Mechanic	4	Hourly
Mining Technical Services		
Senior Mine Engineer	1	Salary
Geotechnical Engineer	1	Salary
Chief Geologist	1	Salary
Ventilation Engineer	1	Salary
Mine Surveyor	2	Salary
Surveyor Helper	2	Salary
Geologist	4	Salary
Sampler	4	Salary
Short-Term Mine Planner	1	Salary
Project Engineer	1	Salary
Long-Term Mine Planner	1	Salary
Technician	2	Salary
Total Underground	**194**	

Source: Bluestone, 2017.

16.11 Mine Production Schedule

An underground mine production rate of 1,500 tpd was assumed for this conceptual study, which applied indexes considered appropriate to the high degree of mechanization and productivities of the selected stoping methods and available working faces and/or stopes.

Table 16-13 shows the mine production schedule.

Table 16-13: Mine production schedule

	Item	Unit	Total	Y-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Material	**Stopes***	**kt**	**8,136.47**	**-**	475.07	468.42	438.79	442.33	426.62	452.20	481.77	476.41	518.71	540.35	538.33	547.50	547.50	547.50	547.50	547.50	139.95
	Des. Prod	**kt**	**763.49**	**-**	72.43	79.08	108.71	105.17	120.88	95.30	65.73	71.09	28.79	7.15	9.17	-	-	-	-	-	-
	ROM	**kt**	**8,899.95**	**-**	**547.50**	**547.50**	**547.50**	**547.50**	**547.50**	**547.50**	**547.50**	**547.50**	**547.50**	**547.50**	**547.50**	**547.50**	**547.50**	**547.50**	**547.50**	**547.50**	**139.95**
	Prod/day	**tpd**	-	-	1,500.00	1,500.00	1,500.00	1,500.00	1,500.00	1,500.00	1,500.00	1,500.00	1,500.00	1,500.00	1,500.00	1,500.00	1,500.00	1,500.00	1,500.00	1,500.00	1,500.00
	Des. N. Prod	**kt**	**1,966.10**	150.00	172.00	134.03	119.22	127.57	117.45	97.21	97.21	116.53	134.28	150.46	155.01	164.12	104.58	115.47	10.97	-	-
	Mass Mov Total	**kt**	**10,866.17**	**150.00**	**719.50**	**681.53**	**666.72**	**675.07**	**664.95**	**644.71**	**644.71**	**664.03**	**681.78**	**697.96**	**702.51**	**711.62**	**652.08**	**662.97**	**558.47**	**547.50**	**139.95**
Gold (Au)	**Stope***	**g/t**	**4.99**	**-**	5.49	4.95	4.80	4.81	4.69	5.40	4.60	4.88	4.73	4.64	4.64	4.69	5.26	5.50	5.49	5.15	4.89
	Des Prod	**g/t**	**5.29**	**-**	5.87	5.95	5.46	5.20	4.14	5.31	4.95	5.42	6.68	7.97	3.96	-	-	-	-	-	-
	ROM	**g/t**	5.01	-	5.54	5.09	4.93	4.88	4.57	5.38	4.65	4.95	4.83	4.68	4.63	4.69	5.26	5.50	5.49	5.15	4.89
	Stope*	**koz**	**1,304.66**	**-**	83.90	74.54	67.76	68.39	64.37	78.50	71.31	74.75	78.86	80.61	80.31	82.59	92.68	96.74	96.72	90.64	21.99
	Des Prod	**koz**	**129.86**	**-**	13.66	15.14	19.09	17.59	16.09	16.28	10.46	12.38	6.19	1.83	1.17	-	-	-	-	-	-
	Total	**koz**	**1,434.52**	**-**	**97.56**	**89.68**	**86.84**	**85.98**	**80.46**	**94.78**	**81.77**	**87.13**	**85.05**	**82.44**	**81.47**	**82.59**	**92.68**	**96.74**	**96.72**	**90.64**	**21.99**
Silver (Ag)	**Stope***	**g/t**	**17.48**	**-**	25.27	23.79	28.27	20.37	18.17	25.91	19.61	22.22	16.85	13.49	13.49	13.30	11.12	10.24	10.25	14.34	17.44
	Des Prod	**g/t**	**20.18**	**-**	26.53	24.21	26.41	29.19	20.55	14.07	12.98	8.60	7.26	10.56	5.73	-	-	-	-	-	-
	ROM	**g/t**	**17.71**	**-**	**25.43**	**23.85**	**27.90**	**22.06**	**18.70**	**23.85**	**18.81**	**20.45**	**16.35**	**13.46**	**13.36**	**13.30**	**11.12**	**10.24**	**10.25**	**14.34**	**17.44**
	Stope*	**koz**	**4,573.07**	**-**	385.90	358.30	398.85	289.66	249.29	376.68	303.70	340.35	281.01	234.44	233.56	234.07	195.69	180.23	180.44	252.41	78.50
	Des Prod	**koz**	**495.24**	**-**	61.77	61.55	92.31	98.71	79.88	43.10	27.43	19.65	6.72	2.43	1.69	-	-	-	-	-	-
	Total	**koz**	**5,068.31**	**-**	**447.67**	**419.85**	**491.16**	**388.37**	**329.17**	**419.78**	**331.13**	**359.99**	**287.73**	**236.87**	**235.25**	**234.07**	**195.69**	**180.23**	**180.44**	**252.41**	**78.50**
Developme	**Lateral**	**m**	**33,491.24**	3,306.00	3,204.40	2,448.91	2,134.00	2,259.66	2,077.70	1,713.78	1,713.78	1,990.22	2,272.36	2,560.19	2,592.19	2,609.03	1,565.42	1,043.61	-		
	Vertical	**m**	**3,138.68**	**-**	-	48.10	87.13	117.09	110.47	97.24	97.24	180.87	229.46	242.99	295.83	401.50	467.73	401.64	361.40		
	Operational	m	**38,257.66**	**-**	2,045.60	2,752.99	3,028.88	2,873.25	3,061.83	3,438.99	3,438.99	3,078.90	2,748.18	2,446.81	2,361.98	2,192.32	2,246.31	2,043.57	499.05		
	Total	**m**	**74,887.58**	**3,306.00**	**5,250.00**	**5,250.00**	**5,250.00**	**5,250.00**	**5,250.00**	**5,250.00**	**5,250.00**	**5,250.00**	**5,250.00**	**5,250.00**	**5,250.00**	**5,202.85**	**4,279.46**	**3,488.81**	**860.46**	**-**	**-**
Plant	**ROM processed**	**kt**	**8,899.95**	**-**	547.50	547.50	547.50	547.50	547.50	547.50	547.50	547.50	547.50	547.50	547.50	547.50	547.50	547.50	547.50	547.50	139.95
	Au recov.**	**koz**	**1,377.14**	**-**	93.66	86.09	83.37	82.54	77.24	90.99	78.50	83.64	81.65	79.14	78.21	79.28	88.97	92.87	92.85	87.01	21.11
	Ag recov.**	**koz**	**4,308.07**	**-**	380.52	356.87	417.49	330.12	279.80	356.81	281.46	305.99	244.57	201.34	199.96	198.96	166.34	153.20	153.37	214.55	66.72
	Backfill Placed	**kt**	**2,847.99**	**-**	175.20	175.20	175.20	175.20	175.20	175.20	175.20	175.20	175.20	175.20	175.20	175.20	175.20	175.20	175.20	175.20	44.79

Notes: *Material applying mining recovery and/or dilution.
Source: GE21, 2025.

During mine operation, two zones, North and South, each with multiple mining horizons, will be in production simultaneously. The underground mine life is estimated at 17 years of production.

17 RECOVERY METHODS

The process flowsheet for the Project was based on the conclusions previously described in Section 13. Results from test programs were used to develop the corresponding process design criteria, mechanical equipment list, flowsheets and operating costs.

The process plant will include:

- Multi-staged crushing.
- Two-staged grinding circuit.
- Gravity concentration and intensive leaching (ILR).
- Cyanide leaching and carbon adsorption using carbon-in-pulp (CIP).
- Cyanide destruction, dewatering, storage of tailings, dry stacking or underground deposition as paste.
- Carbon acid wash, elution and regeneration.
- Electrowinning and refining.

The main design criteria adopted in designing the processing circuit are listed as follows:

- Nominal processing rate of 1,000 tpd, which is equivalent to 0.34 Mtpa.
- Crusher circuit operational performance (product of availability by utilization) – OP: 65%.
- Grinding and extraction circuit OP: 92%.
- Filtration circuit OP: 92%.
- Grinding circuit product size: P_{80} of 0.053 mm.

17.1 Description of the Process Plant

The selected metallurgical process flowsheet for the industrial processing of the Project comprised the following circuits:

- Crushing of run of mine (ROM) ore.
- Crushed ore storage and reclaim.
- Grinding circuit with ball mills and hydrocyclones.
- Gravity concentration circuit, including a scalp screen, a centrifugal concentrator, and an intensive leaching reactor.
- Trash screening.
- Pre-leaching thickener.
- Pre-oxidation in an agitated tank sparged with oxygen to oxidize the slurry prior to leaching.
- Cyanide Leaching in agitated leach tanks for providing 36-hour residence time to leach gold and silver into solution.
- Carbon in Pulp in CIP tanks to adsorb gold and silver cyanide complexes onto the pores of activated carbon.

- Activated carbon acid wash, Carbon Elution and Regeneration – Acid wash of carbon to remove inorganic foulants, elution (strip) of carbon to produce a precious gold and silver-rich solution, and thermal regeneration of carbon to remove organic foulants.
- Gold and Silver Refining by electrowinning (sludge production), filtration, drying and refining to produce gold and silver doré.
- Carbon safety screening.
- Neutralization of residual cyanide present in tailings (SO_2/Air method).
- Final Tailings Dewatering in a thickener followed by a filter plant to reduce final tailings moisture to 18.6%, therefore adequate for dry stacking or paste backfill.
- Water recirculation system.
- Reagent storage, preparation, and dosage systems.

The overall process flowsheet is presented in Figure 17-1.

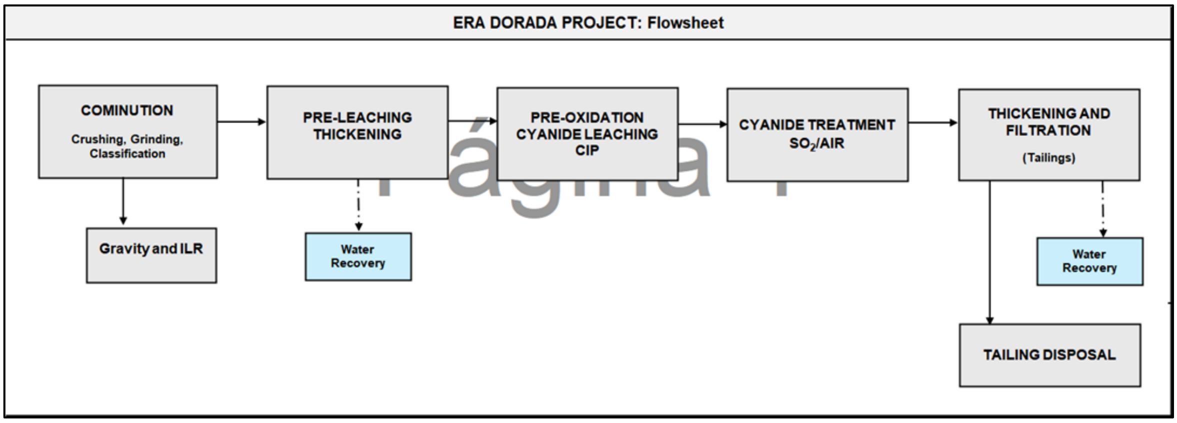

Figure 17-1: Overall process flowsheet – Era Dorada Project

Source: Author, 2025.

17.2 Design Criteria

Key process design criteria are summarized in Table 17-1.

Table 17-1: Key process design criteria

Processing Stage	Unit	Nominal Value
General		
Plant Daily Throughput	tpd	1000
Plant Operational Performance	%	92
Overall Au Recovery	%	96
Overall Ag Recovery	%	85
Crushing		
Operational Performance	%	65
Grinding		
Bond Ball Mill Work Index (design)	kWh/t	19.9
Bond Abrasion Index	g	0.24
Classification Equipment	-	Hydrocyclones
Final Target Product Size (P_{80})	mm	0.053
Gravity Concentration		
Concentrator Type	-	Semicontinuous Batch Centrifugal
Number of Units	-	1

Processing Stage	Unit	Nominal Value
Feed Source	-	Cyclone Underflow
Recovery Method	-	Intensive Leach Reactor (ILR)
Pre-Leach Thickening		
Thickener Underflow Concentration of Solids	% w/w	50
Leaching		
Pre-Oxidation	Y / N	Yes
Pre-Oxidation Residence Time	h	2
Dissolved Oxygen Target (DO)	mg/l	<20
Leach Residence Time	h	36
Sodium Cyanide Consumption	kg/t	0.3
Lime Consumption	kg/t	1.71
CIP		
CIP Residence Time	h	6
Carbon Concentration	mg/l	50
Carbon Loading	g Au / t carbon	2,500
Carbon Processing		
Acid Wash Type	-	Hydrochloric Acid
Elution Operating Temperature	ºC	140
Elution Operating Pressure	kPa	350 to 500
Smelting Furnace Type	-	Electric Induction Furnace
Carbon Consumption Rate	kg / t Carbon Stripped	30
Cyanide Destruction		
Feed Solution, CN_{WAD}	mg/l	191
Discharge Solution, CN_{WAD}	mg/l	< 1.0
SO_2 Consumption	g / g CN_{WAD}	4
Lime Consumption	g / g CN_{WAD}	0.8
$CuSO_4$-$5H_2O$ Concentration	mg/l	25
Tailings Management		
Disposal Type	-	Dry stack/Paste
Final Moisture Content	%	18.6

Source: Bluestone, 2019.

17.3 Process Plant Description

17.3.1 Crushing

The underground Run of Mine (ROM) will be directed to an industrial crushing plant, whose product will be conveyed to a dedicated storage bin designed for a 24-hour live storage capacity.

17.3.2 Grinding

The grinding circuit will process a nominal throughput of 1,000 tpd (fresh feed) for 0.053 mm (P_{80}) ground product. A gravity concentration circuit will be installed in the grinding circuit.

17.3.3 Gravity Concentration and Intensive Leaching

A fraction of the hydrocyclone nest combined underflow will be directed to the gravity concentrator scalp screen. The scalp screen will remove oversize particles prior to gravity concentration. The screen undersize will feed a semi-continuous batch gravity concentrator.

The gravity concentrate will be collected in the storage cone and subsequently leached in an intensive cyanidation (leaching) reactor (ILR) circuit. The ILR pregnant solution will be pumped to the ILR pregnant solution tank, the latter located in the gold room.

17.3.4 Pre-Leach Thickening

Hydrocyclone overflow will be directed to a vibrating trash screen for the removal of trash material. Oversize material will discharge into a trash bin, while screen undersize will flow by gravity to pre-leach thickener. Flocculant solutions will be added to the thickener feed to enhance thickening to a nominal product of 50% w/w solids. The thickener underflow will be pumped to the pre-oxidation tank, while the thickener overflow will flow by gravity into the process water tank for recirculating in the grinding circuit.

17.3.5 Pre-Oxidation

Pre-leach thickener underflow will be pumped to a pre-oxidation circuit prior to leaching. Oxygen will be sparged into the bottom of the agitated tank, and the slurry will be conditioned for 2 hours to oxidize sulphide minerals.

Pre-oxidation 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 consumption in the cyanide destruction circuit.

17.3.6 Leaching

The leach circuit will be designed to provide 36-hour residence time. The lime slurry will be added to the first 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 solutions will be added to the circuit at a rate of up to 0.30 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 the solution.

Slurry from the leach circuit will then flow by gravity to the CIP circuit for carbon adsorption.

17.3.7 Carbon in Pulp – CIP

The leached slurry will flow through CIP tanks for adsorbing gold-cyanide and silver-cyanide complexes onto the pores of activated carbon at an average carbon of 50 g/l to maximize adsorption.

As the slurry proceeds through the circuit, metal values in the solution will progressively decrease. The carbon will be transferred countercurrent to the slurry flow to maximize precious metal recovery. 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, will be located in the first CIP tank, interacting with the highest concentrations of gold and silver.

The tailings stream from the last CIP tank will flow onto a stationary safety screen to capture any carbon particles not captured in the CIP circuit. The safety screen undersize will then be pumped to the cyanide destruction circuit.

17.3.8 Carbon Acid Wash, Elution and Regeneration (Carbon Processing)

The carbon processing plant will process the loaded carbon, producing gold and silver doré.

17.3.8.1 Carbon Acid Wash

Loaded carbon from the CIP circuit will flow by gravity into an acid wash vessel constructed of fibre-reinforced plastic. The carbon will be treated with a circulating 3% hydrochloric acid (HCL) solution to remove calcium deposits, magnesium, sodium salts, silica, 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 regeneration circuit using a horizontal electric kiln.

After the acid wash cycle, the carbon will be directed to the elution vessel using water. Under normal operation, only one acid wash and elution cycle will take place per day.

17.3.8.2 Elution (Carbon Stripping)

The carbon stripping (elution) process will use the barren strip 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 solution exiting the electrowinning cells will be circulated back to the barren solution tank for reuse.

During the strip cycle, a solution containing approximately 1% sodium hydroxide and 0.1% NaCN at a temperature of 140 °C will be pumped up through the strip. The solution exiting the top of the vessel will be cooled down to below its boiling point by a recovery heat exchanger. The heat from the outgoing solution will be transferred to the incoming cold, barren solution prior to passing through the solution heater.

17.3.8.3 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 loss during the adsorption and desorption processes.

A horizontal electric kiln equipped with a residual heat dryer will be utilized to treat the carbon. The regenerated carbon from the kiln will flow by gravity into the carbon quench tank, where it will be cooled by fresh water and/or carbon fines water before being pumped back to the CIP circuit.

To compensate for carbon losses from attrition and impact, fresh carbon will be added to the carbon attrition tank and mixed with fresh water to activate the carbon pores. The fresh carbon will then drain into the carbon quench tank and combine with the regenerated carbon discharging from the kiln.

17.3.9 Electrowinning and Refining

Pregnant solutions derived both from the strip circuit and ILR will be pumped to the refinery for electrowinning, therefore resulting in gold and silver sludge.

Pregnant solution will be pumped through electrowinning cells, where gold and silver will plate on the stainless-steel cathodes, while the barren solution will flow into the barren return tank before being pumped back to the barren solution tank for reuse. The sludge will then be filtered, dried and refined in an electric induction furnace, producing gold and silver doré bars.

17.3.10 Cyanide Destruction

The cyanide destruction circuit will consist of mechanically agitated tanks. Cyanide will be destroyed using the SO_2/Air process. Treated slurry from the circuit will then be pumped into the final tailings thickener. The cyanide destruction circuit will treat CIP tailings slurry and process spills from various contained areas, as well as process bleeding streams.

Oxygen will be sparged from near the bottom of the tanks, under the agitator impeller. If necessary, lime slurry will be added to maintain the optimum pH of 8.0–8.5. Copper sulphate ($CuSO_4$) will also be added as a catalyst, maintaining a 25 mg/l concentration in solution. A sodium metabisulphite (SMBS) solution, at a rate of up to 789 g/t, will be added into the system as the source of SO_2. The system is designed to reduce the CN_{WAD} concentration to below 1.0 mg/l.

17.3.11 Tailing Thickening and Filtering Circuit

Tailings resulting from the Detox circuit at a solids concentration of 50% w/w will be pumped to a thickener, where flocculant will be added.

The thickener underflow will be pumped to the filtering circuit to reduce the cake to a moisture content of 18.6% (dry basis). Filtering and thickening water will be recirculated within the processing plant, whereas the filtered product will be transferred to the disposal system.

17.4 Reagent Handling, Storage and Preparation System

Reagents consumed within the plant will be prepared on-site and distributed via dedicated reagent handling systems. These reagents include:

- Sodium cyanide (NaCN).
- Lime.

- Lead nitrate (Pb_2NO_3).
- Hydrochloric acid (HCl).
- Caustic soda (NaOH).
- Copper sulphate ($CuSO_4$).
- Sodium metabisulphite (SMBS).
- Flocculant.
- Activated carbon.
- Antiscalant.

Reagents will be received and stored in appropriate facilities. Each reagent will be prepared in accordance with occupational/environmental safety standards, preventing incompatible reagents from mixing. Storage tanks will be equipped with level indicators, instrumentation, and alarms to ensure spills do not occur during normal operation. Appropriate ventilation, fire and safety protection, eyewash stations, and Material Safety Data Sheet (MSDS) stations will be located throughout the facilities.

The reagents will be delivered to the thickener, leach, CIP, acid wash, elution, and cyanide destruction circuits. Dosages will be controlled by flow meters and manual control valves. The capacity of the storage tanks will be designed to handle one day of production.

Table 17-2 summarizes the reagents used in the process plant and the respective estimated daily consumption rates.

Table 17-2: Reagent consumption

Reagent	Delivering Form	Daily Usage
NaCN	1 t bags (dry)	520 kg/d
Lime	2 t bags (dry)	2.6 tpd
Pb_2NO_3	50 kg bags (dry)	313 kg/d
HCL	208 l drums (liquid)	592 kg/d
NaOH	50 kg bags (dry)	184 kg/d
$CuSO_4$	50 kg bags (dry)	127 kg/d
SMBS	500 kg bags (dry)	1.35 tpd
Antiscalant	50 kg barrels	41 kg/d
Flocculant	25 kg bags (dry)	79 kg/d
Activated Carbon	50 kg bags (dry)	120 kg/d

Source: Bluestone, 2019.

17.5 Utilities and Water

17.5.1 Air Supply / Oxygen

An instrument and plant air system with four compressors and associated dryers, filters, and receivers will be installed in a compressor room, the latter located inside the plant building.

Oxygen will be used in pre-oxidation, leach, CIP and cyanide destruction circuits and will be supplied by generation systems.

17.5.2 Water Supply

Overflow water resulting from the pre-leach and tailings thickeners, as well as from the filters, will be used as process water, mainly in the grinding circuit, to dilute slurry to the required densities. Treated water will supply process make-up water, gland water, reagent make-up water and cooling water services in the elution circuit.

18 PROJECT INFRASTRUCTURE

18.1 General

The Project infrastructure is designed to support the operation of a 1,000 tpd underground mine and processing plant, operating on a 24-hour per day, 7-day per week basis. Support facilities have been designed to suit local conditions and topography. The main infrastructure components include the following:

- 5 km new site access road, including a 110 m long bridge.
- 8.2 km new 69 kV power line.
- On-site substation (69 kV to 13.8 kV).
- Water management facilities, including a flood protection levee, diversion channel, ditches and collection ponds.
- Process plant site pad and associated buildings.
- Primary crusher pad.
- Emergency power genset.
- Communications system upgrade.
- Rehabilitation of five existing dewatering wells.
- Construction of eight new dewatering wells.
- Construction of nine new reinjection wells.
- Reagent warehouse and storage facilities.
- Truck shop (existing facility to be used in pre-production, new shop to be constructed in Operating Year 1).
- Fresh / Fire water tank.
- Process water tank.
- Upgrade fuel storage facility.
- New helipad.
- Upgrade the septic system for an upgrade for sewage management.
- Solid waste disposal facility.
- Dry stack tailings facility (DSTF).
- Temporary waste rock storage facility.
- 1.0 km North and South portal connector haul road.
- On-site access roads for plant and facilities.

Additional security facilities, including site access control station.

18.2 General Site Layout

The proposed site layout has been designed to support mining and plant operations while minimizing environmental and community impacts, reducing construction costs, ensuring secure access, and optimizing operational efficiency.

Existing infrastructure will be used to the greatest extent possible to reduce capital costs and construction timelines.

Several site facilities already established at the Project will remain in use during both construction and operations. These include administrative and technical offices, modular geology and environmental units and a new assay laboratory equipped with assets from the former Marlin Mine.

Security infrastructure, a first aid and emergency response center, and warehouse and maintenance shops are also in place, supporting key functions such as logistics, safety, equipment servicing, and sample processing. Together, these facilities provide adequate support for mining, processing, environmental, and administrative activities across the life of the project.

The Project's overall layout is provided in Figure 18-1. The plant site and the main infrastructure facilities arrangement are presented in Figure 18-2.

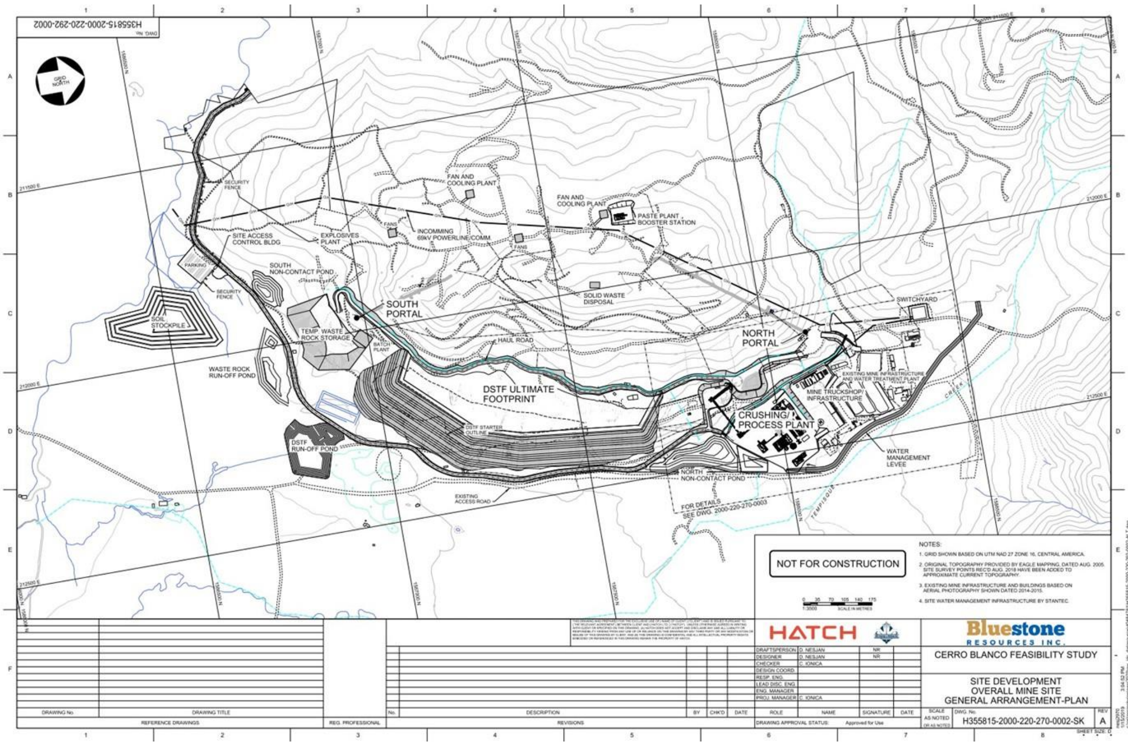

Figure 18-1: Overall mine site

Source: Bluestone, 2019.

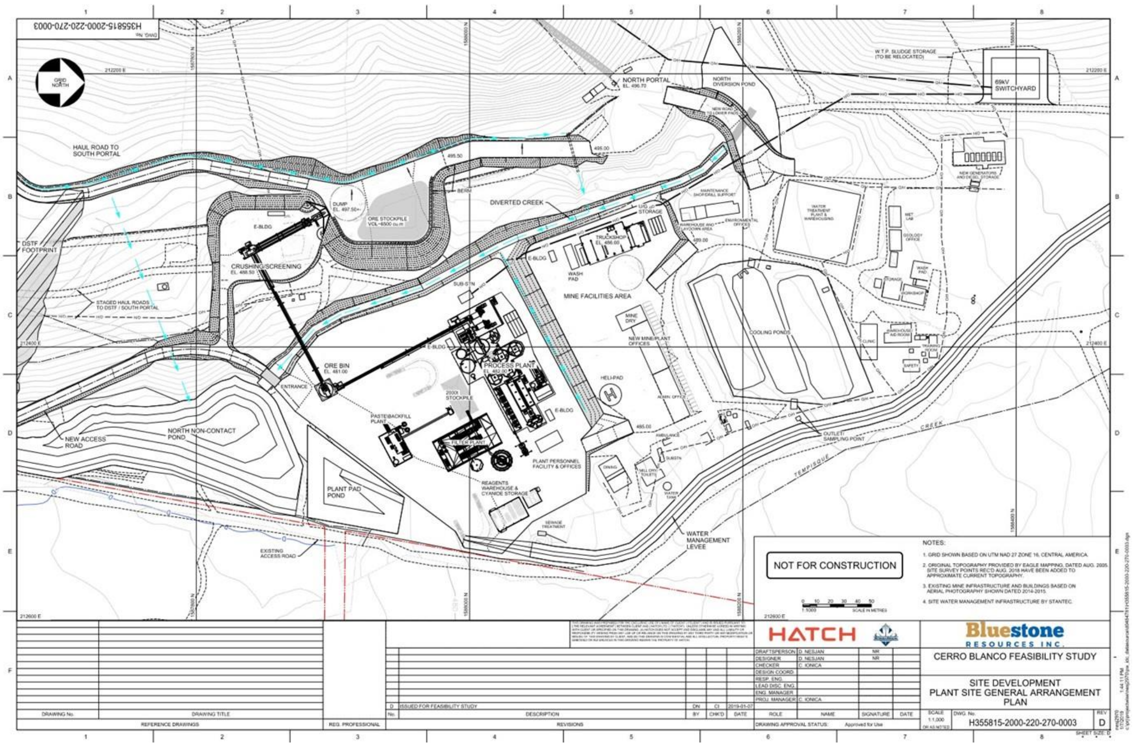

Figure 18-2: Plant site general arrangement

Source: Bluestone, 2019.

18.3 Site Access Road

Road access to the site is currently through Asunción Mita by gravel road, crossing the river Grande de Mita using a 27-tonnes-bridge capacity. This access is not suitable to support mining construction and operations.

Road access to the site is currently via Asunción Mita along a gravel road, which crosses the Grande de Mita River using a bridge with a 27-t capacity. This existing route is inadequate to support the heavy equipment loads required for mine construction and long-term operations.

To address this limitation, a new access route has been planned to accommodate heavy haul traffic. The proposed road will extend 5.5 km and connect directly to the Pan-American Highway (CA-1), approximately 3 km north of Asunción Mita. As part of this upgrade, a new 80-metre-long bridge will be constructed over the El Achotal River to ensure reliable and safe access to the project site during both the construction and operational phases.

18.4 Security

A new access gate and guard facility will be installed at the main site entrance, including a barrier and fenced gate for preliminary screening of all incoming traffic. A site access control building and parking lot will be located nearby, where personnel access will be managed, and only approved vehicles may proceed beyond this point. The project area will be enclosed by a combination of chain-link fencing with barbed or razor wire and barbed wire livestock fencing in remote areas. Additional fencing will be installed around sensitive infrastructure such as the warehouse, refinery sections, and substations. Security will be provided by a team of uniformed and plain-clothes personnel who will monitor the main gate, conduct roving patrols, and oversee security within the refinery and general site. Security will be intensified during the construction phase, including increased staffing and coordination with contractors and local law enforcement Mine Offices.

18.5 Power Supply and Distribution

Electrical power for the project will be sourced from the Energuate Barranca Honda Substation, located approximately 1 km south of Asunción Mita. An 8.2 km long, 69 kV single-circuit overhead transmission line will be constructed. The estimated total operating load for the site, including dewatering and reinjection systems, is 14.3 MW, with a total connected load of 25.0 MW.

The 69 kV will be steped down power to 13.8 kV through transformer circuits for primary on-site power delivery to key areas, including the mine, crushing plant, process plant, paste plant, cooling systems, and well fields. Existing 4.16 kV distribution lines are designed to handle 13.8 kV and will be reused where possible. However, a new 13.8 kV overhead distribution line will be built to supply power to the mine portal, mill, and related buildings. Secondary power distribution includes 4.16 kV (medium voltage) for large equipment and 480 V (low voltage) for smaller loads. Area substations will step down power accordingly, using transformers sized based on projected loads.

18.5.1 Emergency Power

The existing on-site power plant is planned to be repurposed as the emergency power facility, using diesel generators relocated from the former Marlin Mine. Generator outputs include 4.16 kV units and 600 V units, the latter of which will be stepped up to 4.16 kV via internal transformers. The standby power system is designed to supply critical loads in the event of a grid power failure, ensuring controlled shutdown of the process plant, continued operation of essential underground fans and pumps, and provision of minimum emergency power. The total estimated emergency demand is slightly under 7 MVA for both surface and underground installations.

18.5.2 Construction Power

The existing standalone generators will supply the power required during the construction phase. Temporary construction generators will be used where required to provide power to remote locations or where distribution from the existing generators is not practical.

18.6 Process Plant

The proposed process plant will occupy a footprint of approximately 150 m by 70 m. It will house grinding equipment, leaching and CCD tanks, a Merrill-Crowe circuit, filtration and detox systems, a gold-silver gold room, reagent preparation facilities, a dry stack tailings filtration plant, and electrical rooms. Enclosed structures will be provided for the milling circuit, Merrill-Crowe system, refinery, electrical rooms, and reagent preparation facilities. Open-air installations will include the leaching and CCD tanks, as well as the cyanide destruction area.

Electrical equipment buildings, including MCCs and operator control rooms for the crusher, process plant, tailings filtration, and associated areas, will be built. These units will be compliant with all local electrical and fire safety codes. Where feasible, MCCs and control gear will be pre-installed at the factory to minimize on-site work.

18.7 Dewatering and Reinjection

There are currently 10 existing dewatering wells on site that are suitable for reuse. To effectively manage groundwater inflows, 14 new dewatering wells—each 450 m deep and 12 inches in diameter—will be installed. These are expected to handle a peak surface dewatering rate of approximately 795 m^3/h, supplemented by 114 m^3/h from underground sumps. To accommodate the increased flow, a new cooling pond will be constructed downstream of the existing south cooling pond. The site's water treatment plant is permitted to treat 341 m^3/h; excess water will be managed via reinjection through 11 newly constructed wells, each 150 m deep and capable of receiving 57 m^3/h. One reinjection well will serve as contingency capacity for extreme storm events (1-in-100-year).

18.8 Truck shop, Warehouse, Mine Dry and Administration Buildings

The mine truck shop and maintenance facility will be located near the mine administration building and the North Portal, providing easy access from both the mine and the plant site. The maintenance facility will include a warehouse for parts and spares. The truck shop will contain three vehicle service bays, a general shop and weld bay, and an oil change/lubrication bay. An outdoor wash bay will also be provided. An office area within the truck shop will accommodate the mine maintenance supervisor and planner.

A mobile equipment parts and spares warehouse will be integrated into this facility. The building will be a steel structure with metal cladding and a concrete slab-on-grade. A 10-t service crane will be installed over the service bays.

18.9 On-Site Water Tanks

A new dual-purpose fresh / fire water tank will be erected with a capacity of 640,000 l. Internal risers on all non-firewater suction lines will ensure a minimum firewater reserve of 470,000 l, allowing for approximately two hours of firefighting capability.

A new process water tank with 170,000 l capacity will also be erected adjacent to the fresh/fire water tank to service the process plant.

18.10 Bulk Fuel Storage and Delivery

An existing diesel fuel storage facility is already installed at the site for the existing power generators. It consists of two 37,500 l tanks within a concrete containment area. The existing fuel storage facility will be expanded by installing an additional 37,500 l diesel tank and increasing the containment area to accommodate the additional tank.

This expanded fuel storage facility will service the underground mining and site surface fleet with a capacity for approximately 14 days of mobile equipment operations or two days running all critical support loads.

18.11 Haul Roads

The existing North and South portal access roads will be significantly upgraded to accommodate increased traffic. Both roads will be increased to 22 m wide and the north access road will be extended to provide access to the dry stack tailing facility.

Various temporary construction access roads will be made or modified from existing roads for temporary construction laydown facilities, the staged DSTF construction, and construction access, where required.

18.12 Communications / IT

The existing communications tower currently provides sufficient access to stream data to and from the site and will service the site during the initial construction period. A new fibre optic cable will be installed as part of the 69 kV powerline.

The site communications will be distributed by fibre optic cable among the site facilities with the power transmission infrastructure. The underground mine will use a dedicated communications system. Mobile equipment and security will also use handheld radios for communications.

18.13 First Aid / Emergency Services

A qualified nurse or first aid attendant will be available on site. The first aid clinic, currently under construction, will be located adjacent to the administration building and will support future operations. The facility includes a 100 m² training room for emergency response and a dedicated storage area for medical equipment and supplies.

An ambulance and a fire truck will be stationed in covered parking stalls near the process plant, ready for immediate deployment. All relevant buildings will be equipped with smoke, carbon monoxide, and heat detectors, as well as appropriate chemical fire extinguishers.

18.14 Explosives Storage and Magazines

The existing explosives magazine, located west of the South Portal on the southwestern side of the property, will continue to be used throughout the life of mine. The facility has adequate capacity to support approximately 50 days of operations, with storage for up to 75,000 kg of explosives (3,000 bags at 25 kg each) and 10,000 detonators. Monthly deliveries will be scheduled according to operational demand. The storage structure is built of concrete and reinforced concrete blocks and is surrounded by protective earthen and rock berms to comply with safety standards.

18.15 Sewage Treatment

Sewage water will be handled by standard septic tank collection systems using natural breakdown bioreactors prior to discharge. The sanitary waste from buildings at the plant and main infrastructure site will drain to a buried septic system area below the process area. An existing bio-reactor tank is already installed and connected with buried sewer pipe for the existing site facilities buildings. An additional unit will be installed and connected to the new facilities that are being added to the project.

Sewage will be treated, separated, and the liquid discharged. Water for septic operation and wash use will be made up from the raw water supply.

18.16 Surface Water Management

Surface water infrastructure at the site is designed to separate and manage “contact” and “non-contact” water, minimizing the potential for contamination during mine operations. Contact water—originating from areas such as the process plant and DSTF where filtered tailings are handled—is either reused in the plant or treated at the water treatment plant (WTP). Non-contact runoff is directed to designated discharge points with sediment control capacity. Stormwater is managed through 13 lined channels routed to seven ponding areas, which include both contact and non-contact ponds. The system incorporates reinforced channels, culverts, bridges, and energy dissipation structures to prevent erosion and manage flow during storm events. Contact water ponds are sized for 100-year, 24-hour storm retention and were further evaluated using a site-wide water balance based on climate data from 1970 to 2017.

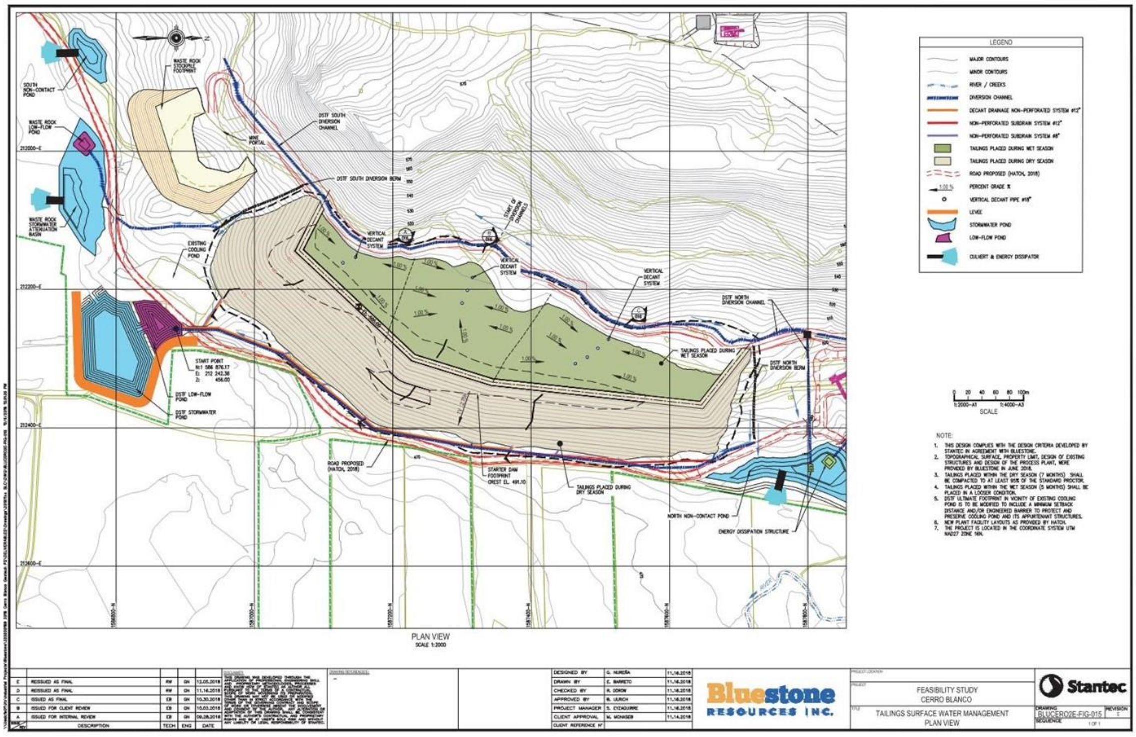

Figure 18-3: Stormwater management infrastructure surrounding the DSTF

Source: Bluestone, 2019.

18.17 Fresh Water Supply

Mine water will be the primary source of fresh water for the process plant, dust control, and treated water use for the site personnel facilities. The mine water will be treated in the existing water treatment plant prior to use as the site's fresh water supply. On-site office facilities and staff/contractor dining facilities will use the treated water for washing, laundry, and bathing. Drinking water and cooking water will be provided by purchasing potable bottled water from a local vendor.

18.18 Water Treatment Infrastructure

The existing water treatment plant (WTP) at Era Dorada is permitted to treat and discharge up to approximately 341 m³/h. The plant is designed to remove arsenic via co-precipitation with ferric salts, using a treatment sequence that includes chemical oxidation, pH adjustment, ferric dosing, and solids separation. It also receives up to ≈ 61 m³/h of process water bleed from the tailing's thickener overflow. The facility has modular capacity, allowing for future expansion if permit limits are increased.

To address potential mercury and copper concentrations in effluents, proposed process modifications include the use of a sulphur-based reagent effective in removing divalent heavy metals. The treated and cooled mine water is also used as process and utility water and will be stored in a dual-purpose raw/fire water tank located near the process plant. As gravity pressure is insufficient due to elevation differences, pumps will be installed to ensure adequate pressure throughout the distribution system. The plant also plays a key role in thermal regulation by cooling mine water via a series of ponds prior to reuse or discharge.

The Project incorporates a water management strategy that allows for the reuse of treated water from the water treatment plant (WTP) in addition to groundwater sources. Treated water is reused in the process plant and for operational services, while excess volumes are managed according to environmental discharge and reinjection plans.

18.19 Tailings Management Facility

18.19.1 Dry Stack Tailings Facility

The Project will utilize a Dry Stack Tailings Facility (DSTF) for tailings management, incorporating filtered tailings transport and placement to minimize environmental impacts and enhance long-term stability. The DSTF will accommodate approximately 3 Mt of tailings over the mine life, corresponding to a required volume of 1.9 Mm^3, based on a weighted average dry density of 1.59 t/m^3. The facility is configured as a centerline-raised embankment, starting with a rockfill starter dam and built progressively with compacted filtered tailings.

Tailings placement follows a seasonal deposition strategy: during the wet season, the material will be deposited loosely in the western section, shaped to direct runoff into decant structures; in the dry season, tailings will be compacted in horizontal lifts in the eastern section to provide structural support and storage capacity. Transport to the DSTF will be carried out by a mine fleet of haul trucks. Initial capital works include the construction of the impoundment area, underdrain systems, geotextile lining, and reclaim ponds, as well as the installation of the mechanical and electrical systems required to recirculate water back to the process plant. The general layout of the DSTF is illustrated in Figure 18-4.

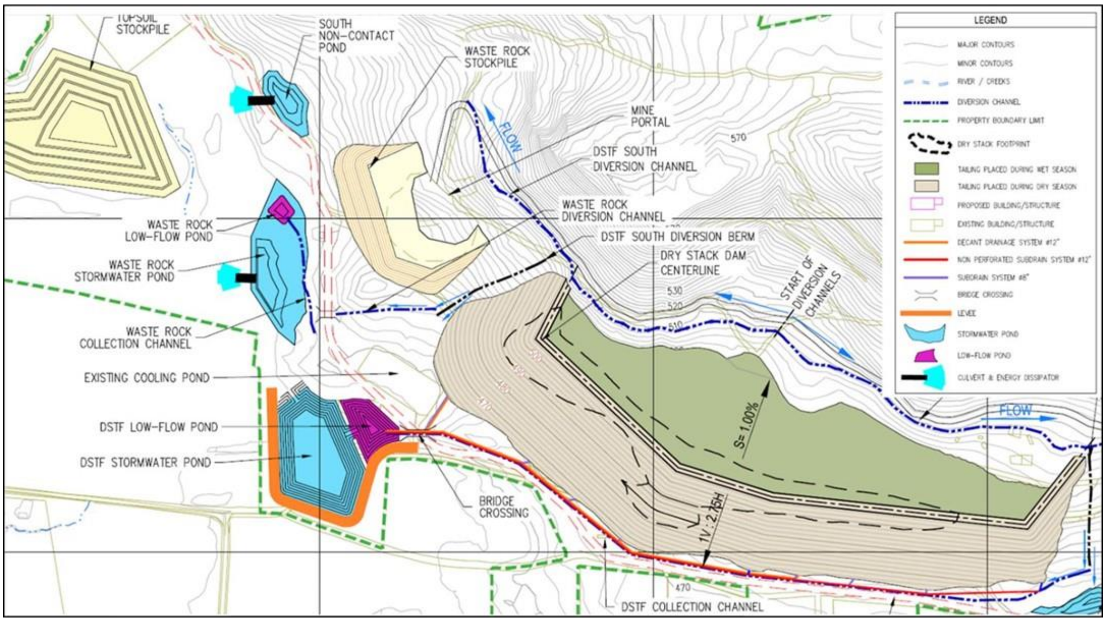

Figure 18-4: DSTF seasonal material placement plan

Source: Bluestone, 2019.

18.19.2 Subsurface Preparation

A series of geotechnical investigations were conducted to support the design of the DSTF. These included test pits, boreholes, Standard Penetration Tests (SPT), and in-situ permeability testing. The 2018 campaign by Stantec, complemented by prior work from Golder (2012–2013), provided a comprehensive understanding of the foundation conditions at the proposed DSTF site. The subsurface profile generally consists of colluvial and alluvial soils over residual materials derived from sedimentary and volcanic rocks. Bedrock was not encountered within the depth range investigated (up to 30.3 m).

This site characterization allowed for the development of foundation design criteria, material suitability assessments, and embankment performance evaluations under both static and seismic loading. The data were used to confirm the feasibility of the selected location and inform the layout and sequencing of construction activities.

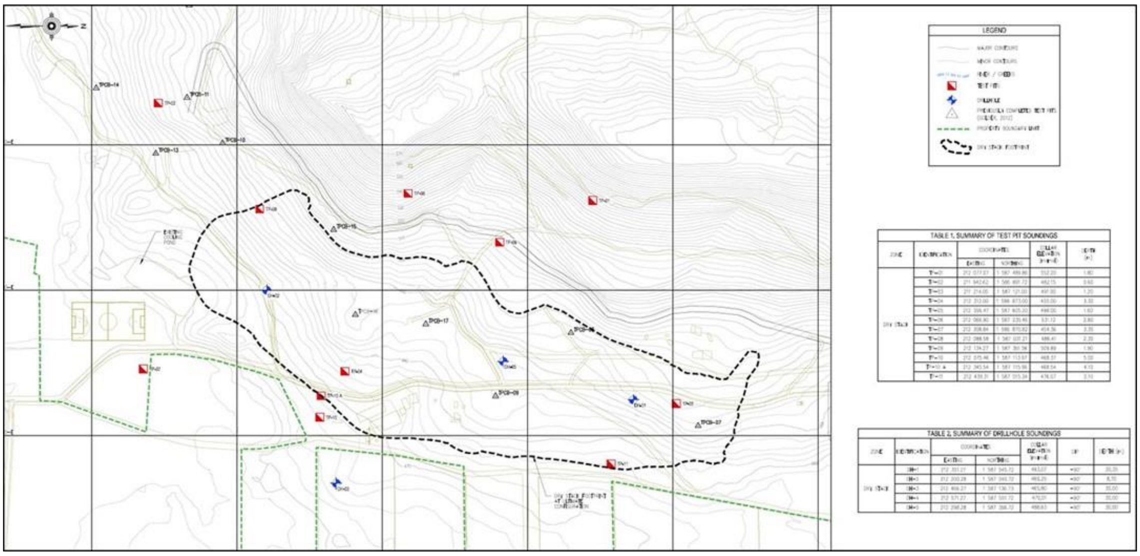

Figure 18-5: DSTF geotechnical site investigation plan

Source: Bluestone, 2019.

18.19.3 Seepage Collection System

To ensure long-term performance and reduce pore pressure within the tailings mass, a dedicated seepage collection system has been incorporated into the DSTF design. The system includes a foundation-level underdrain network and perforated vertical decant towers designed to intercept percolated water and surface runoff. Water collected from both systems will flow to a low flow reclaim pond, with overflow capacity directed to a stormwater pond for exceptional rainfall events.

Each pond is equipped with a sump and pump system to return water to the process plant for reuse. The construction of drainage infrastructure will be staged in coordination with tailings deposition, ensuring operational readiness as the facility expands. This approach supports the project’s zero-discharge water management strategy and enhances geotechnical stability over time.

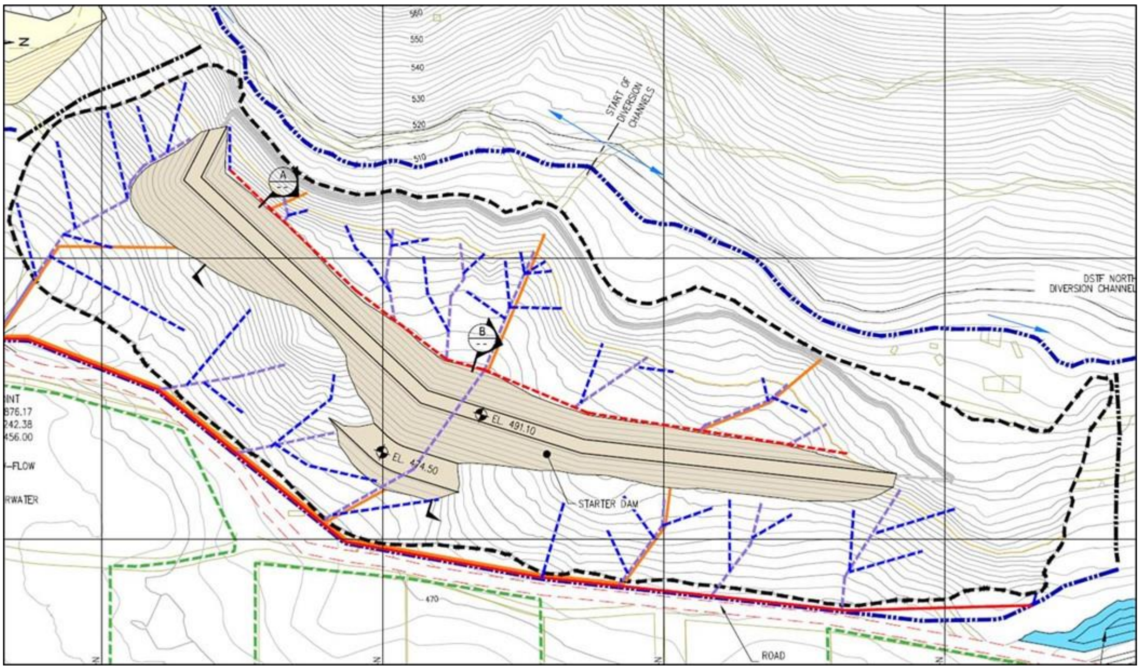

Figure 18-6: DSTF underdrain plan showing starter dam

Source: Bluestone, 2019.

18.20 Waste Rock Facility

The planned Waste Rock Facility (WRF) is located southwest of the DSTF, near the south portal, and is designed to temporarily store up to 120,000 m³ of waste rock generated during mining. Bluestone intends to use most of this material in the underground backfill program, resulting in minimal long-term accumulation. While geochemical testing is ongoing to confirm the potential for acid generation (PAG), current assumptions consider low risk due to limited exposure time. Field humidity cell tests on representative samples are recommended prior to detailed engineering.

Although no targeted site investigation was conducted beneath the WRF footprint, general test pits in the area indicate a subsurface profile of sands and gravels, considered typical and adequate for feasibility-level design. Observations of existing waste rock at the south portal confirm that materials are hard and durable. Further geotechnical and geochemical studies will be required in subsequent design phases to validate long-term stability and environmental compliance.

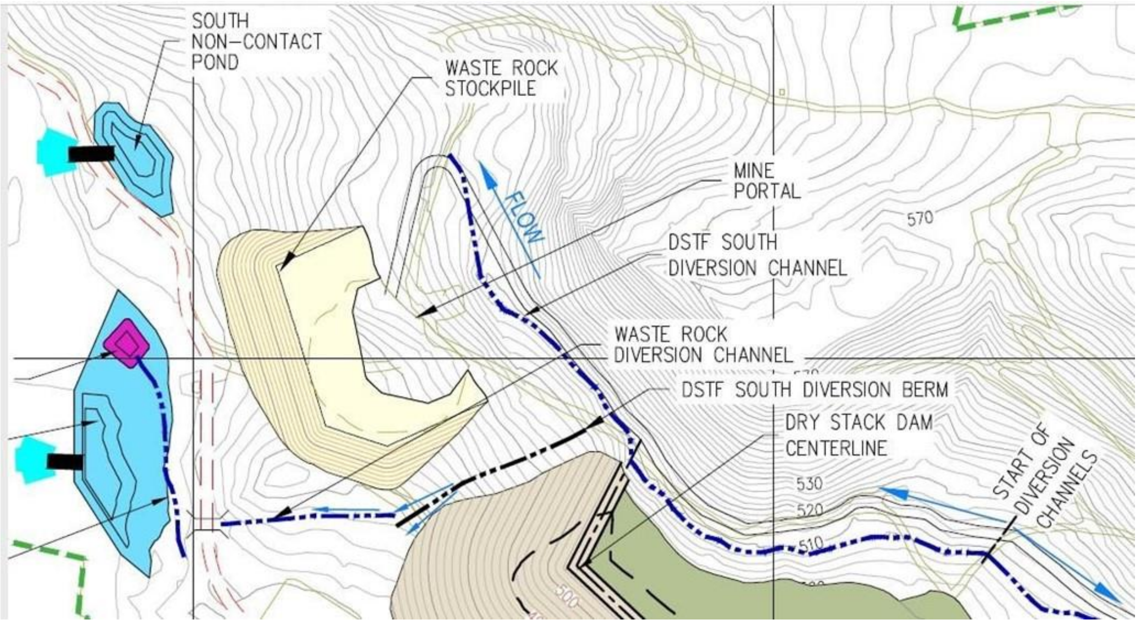

Figure 18-7: WRF general configuration plan

Source: Bluestone, 2019.

19 MARKET STUDIES AND CONTRACTS

19.1 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 behaviour, 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 19-1).

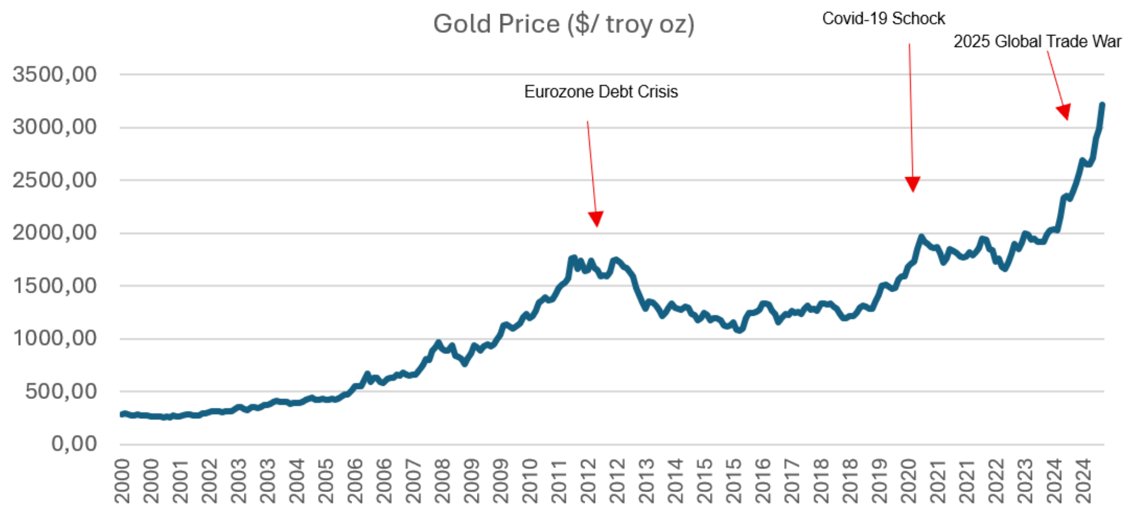

Figure 19-1: Gold price behaviour since 2000

Source: World Bank Group, 2025.

19.1.1 Gold Price

Mineral Resources have been modelled at a gold price of US$ 2,000/troy oz. Project economics have also been assessed at a base case gold price of US$ 2,389/troy 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 25.

19.2 Silver Market

Relative to global markets such as 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 19-2).

Figure 19-2: Silver price behaviour since 2000

Source: World Bank Group, 2025.

19.2.1 Silver Price

Although silver tonnage has not been explicitly modelled within the Mineral Resource Estimate, project economics were assessed using a silver price of US$ 28.44/troy oz, based on the long-term consensus forecast from over 20 investment banks.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACTS

20.1 Introduction

Aura's review shows that the Project has all necessary permits to proceed with the development of the underground mine and construction of the process facilities, subjected to the future operation to adhere to the conditions of the existing permits. The review shows that the Project has been operating so far under high levels of environmental and CSR standards and has been maintaining a comprehensive Permits Register, which shows that all applicable permit commitments have been fulfilled over time.

20.2 Environmental Impact Assessment and Permitting

There are various environmental studies and ongoing monitoring activities which have been performed at the Era Dorada site since the start of the project. An Environmental Impact Assessment (EIA) was submitted and approved by the Ministerio de Ambiente y Recursos Naturales (MARN) in 2007. However, some components of the mine design have changed since that time and specific permit amendments are required. Additionally, it is important to highlight that new baseline environmental and social studies are required for the power line.

The approved EIA from 2007 included a basic Environmental Management Plan (EMP), Social Management Plan (SMP) and Conceptual Mine Closure Plan, which have been reviewed and updated during the Feasibility Study (Bluestone, 2019) to account for current international good practices and the updated project design. Over the next project phase, those plans will be updated to reflect optimization and further development.

20.2.1 EIA Areas of Influence

In 2007, the approved EIA was prepared based on three specific areas, as shown in Figure 20-1. The approved EIA has defined the direct area of influence within an irregular polygon of 235,452 m^2 in area. This includes the underground mine, the processing plant and its surrounding service buildings. The indirect area of influence includes exploration areas in addition to the direct area of influence, covering a total area of 7,050,000 m^2. An external area of influence is also considered; this includes the surrounding areas and the following seven communities (total area of 5,500 ha):

- Caserlo La Lima;
- Trapiche Vargas;
- El Cerron;
- El Tule;
- Las Animas;
- San Rafael Cerro Blanco; and

- Municipality of Asunción Mita.

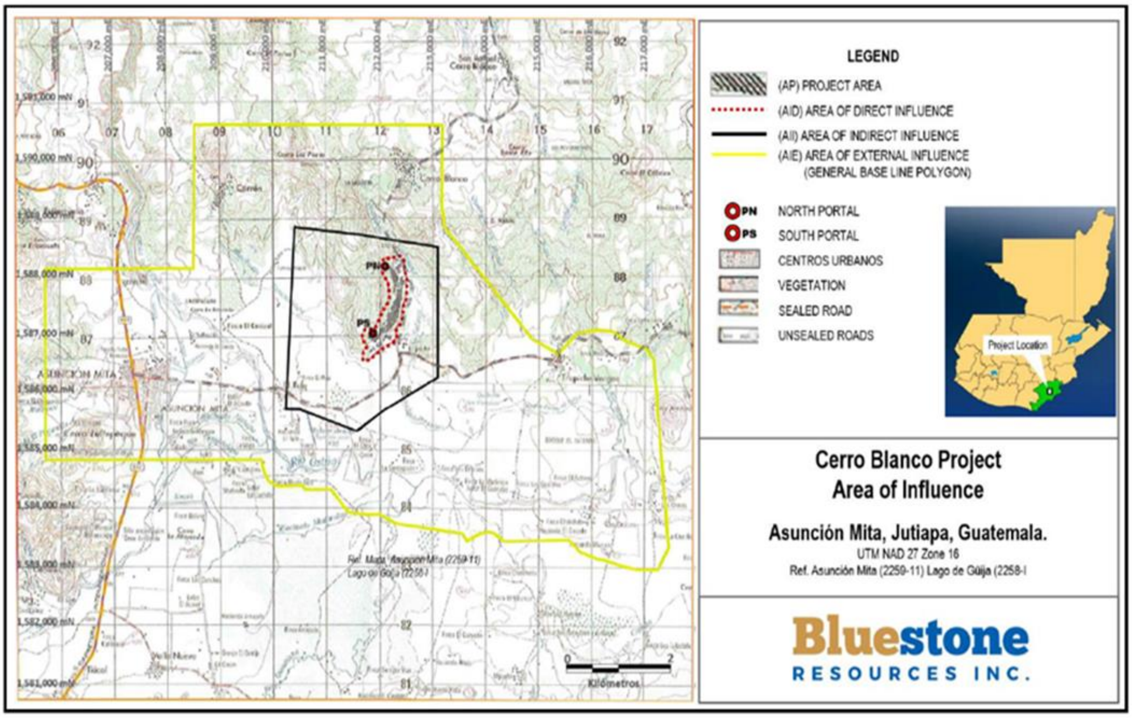

Figure 20-1: EIA areas of influence

Source: Bluestone, 2019

20.2.2 Permitting

As already mentioned, since the design has been updated and optimized, an amendment of the 2007 EIA and specific permits will be required for approval to be aligned with the updated project design.

The power line is not covered by any previous studies or permits, therefore requiring new baseline studies, EIA, and permit applications to be submitted to MARN for approval, with input from the following Guatemalan authorities: Ministerio de Energía y Mineria (MEM), Consejo Nacional de Areas Protegidas (CONAP), Instituto Nacional de Bosques (INAB), Ministerio e Salud y Asistencia social (Ministry of Health & Social Assistance), and the local municipality of Asunción Mita. The anticipated duration for completion of baseline studies, submittal/approval of EIA, and issue of permits is 8-10 months.

Table 20-1 provides a summary of main permit amendments and new permits required, while Table 20-2 summarizes the ongoing permits and current status of each.

Table 20-1: Main permit amendments & new permit required

Project Component	Action Required
Water Management (Injection of Mine Water// Increase of water discharge flow)	EIA Amendment
DSTF Optimization (Changes from 2007 EIA Design)	EIA Amendment
Change of project footprint or approved design (Non authorized activities)	EIA Amendment
New Power Line	New EIA and Permit
Export Permit	New Permit

Source: Bluestone, 2019 (updated 2025).

Table 20-2: Current permits

License / Permit	Resolution / Date of Issue	Expiration Date
Mining	Resolution No.1942 MEM	2032
Tracking and Surveillance Licence Category A	2613-2007/ECM/LP	2028
EIA approval	2613-2007/ECM/LP MARN	The duration of the Project life
Export Permit	DGLEX-07-2018 MEM	Valid from April 25, 2018, until April 25, 2019
Discharge Abatement Cerro Blanco Project and Environmental Management Plan - Category B2	511-2011/DIGARN/ECM/caml MARN	2028
Property Registry	October 31, 2007	2032
Cerro Blanco Building Permit Municipality As. Mita - The difference between the previous value and the current value must be paid	December 29, 2007	Indefinite
Forestry License #1 (East Zone)	No. 40-2205-155-1.6-2007	In process of renewal
Forestry License #2 (West Zone)	No. 40-2205-035-1.1.5.2020	2031
WTP Handling and disposal of sludge	Resolution 00244-2016-DIGARN/FACD/gamc MARN	2028
Amendment Handling and disposal of sludge	Resolution 03749-2019 - DIGARN/MOCMD/RJOP	2027
Medical Clinic	Sanitary License 14047 Ministry of Health and Social Assistance June 14th, 2016	2026
Resolution: no pre-Hispanic or paleontological remains in the Project area	Opinion No. 002/mc.2008 Department of Pre-Hispanic and Colonial Monuments.	Indefinite
Diesel Tank Operating License, Own Consumption	Lic No. 0627	2029
License for operation and management of Explosives	1942	Undefined
Other resolutions of environmental documents from previously acquired commitments	2007	Undefined

Source: Bluestone, 2019 (updated 2025).

20.3 Water Resources

20.3.1 Water Quality

The environmental monitoring program involves continuous sampling of surface and groundwater. Monitoring at the discharge point shows that all parameters meet the criteria set by MARN and EPA guidelines for ore mining effluent. The study conducted by Consultoria y Tecnologia Ambiental, S.A. (CTA) in 2010 summarizes water quality monitoring results from 2002 to 2007. The findings include:

- pH levels between 6.3 and 8.5;
- surface water temperatures between 26 °C and 27 °C, and groundwater temperatures between 33 °C and 72 °C;
- conductivity in surface water between 226 and 693 µS/cm, and groundwater between 2,235 and 3,962 µS/cm.

20.3.2 Water Management

The Project's water management infrastructure includes a Water Treatment Plant (WTP), pipelines, settling ponds, channels, ponds, and groundwater wells. Regular monitoring of surface and groundwater has been conducted for the past 10 years. The 2007 Environmental Impact

Assessment (EIA) indicates naturally occurring metals like Aluminum, Arsenic, Iron, and Manganese in the region’s water.

Installed in 2011, the WTP is capable of treating up to 1,500 gallons per minute (gpm), discharging the treated water into Quebrada Tempisque. It removes arsenic using co-precipitation with ferric salt, involving chemical oxidation, pH adjustment, ferric iron addition, and solids separation. Sludge generated from the process is disposed of in lined trenches, as required by Resolution No. 00244-2016- DIGARN/FACD/gamc MARN and will be backfilled into underground facilities at the end of mining operations.

A monitoring and sampling program has been in place since 2011, with monthly compliance reports approved by MARN. There have been no incidents of non-compliance, and the program will continue during the project’s operational phase.

- Surface Water Management: Runoff is classified as “contact” or “non-contact” water. Contact water undergoes treatment before reuse or discharge, while non-contact water is diverted and monitored.
- Groundwater Management: Dewatering of the mine is achieved through surface wells and underground sumps. Treated water is either reused or discharged into Quebrada Tempisque. Reinjection wells are used to manage groundwater.

20.4 Waste Rock and Tailings Management

20.4.1 Waste Rock

The waste rock facility is described in Section 18.20. Temporary storage of waste rock occurs for a maximum of one year before being deposited back underground as cemented rock fill (CRF) or loose aggregate fill.

Based on historical geochemistry test work and waste rock exposure, time will be limited, it was thus assumed that any potential acid generation will not have sufficient time to occur. The current design includes a pond that collects run-off water from the waste rock facility. A water quality monitoring program will be implemented to evaluate groundwater and run-off water quality prior to discharge. The implementation of environmental monitoring and controls will be focused on ensuring that potential acid generation does not occur.

20.4.2 Tailings

Tailings generated from the process plant will be dewatered through filtration before disposal in the Dry Stack Tailings Facility (DSTF), described in Section 18.19.1. A separate contact water management system is designed to collect all run-off from the DSTF to prevent potential surface water contamination. Contact water will be pumped to the WTP for treatment prior to discharge. The facility is designed to prevent environmental contamination by collecting runoff water for treatment.

Based on studies of geochemistry characterization and consistent with the approved EIA, tailings are considered non-acid generating (NAG). The next section provides a summary of the geochemical tests performed for the Project.

20.4.3 Geochemistry Testwork

For the Feasibility Study (FS), as carried out in 2019, new tailings samples underwent geochemical testing, and existing data on ore and tailings geochemistry were reviewed.

Historical and FS testing results are summarized below:

20.4.3.1 Previous Test Work

A 2006 Water Management Consultants (WMC) report details Preliminary and Phase I characterizations. Waste rock and ore samples from five major ore veins (S1A, S1B, S2, S3, North zone) underwent Acid Base Accounting (ABA) testing (65 samples) and leach extractions (17 samples). Net Neutralization Potential (NNP) values indicated 27 Non-Acid-Generating (NAG) samples, 27 uncertain, and nine potentially acid-generating (PAG). Using Neutralization Potential Ratio -NPR, Phase I data (n=42) showed 48% NAG, 36% PAG and 17 % uncertain samples.

Ore samples tested by WMC, which would become tailings, likely retain their original geochemical characteristics post-milling. WMC reported a wide NPR range for ore (0.01 to 1163) with a geometric mean of 3.4, suggesting that tailings would likely be NAG.

The 2012 DSTF feasibility-level design and cost estimate report by Golder described geochemical testing for a single tailings sample (run in duplicate) using static and kinetic methods. Results indicated the sample was NAG with sufficient carbonate (calcite) to neutralize any acid from residual sulphide minerals. No evidence of metal leaching was found under aggressive or ambient conditions (SPLP testing). Thus, the evaluated tailings sample showed no potential for acid drainage or metal leaching.

20.4.3.2 Feasibility Study (2019)

In June of 2018, a tailings sample (DS-32-0261 Tailings) was generated by blending various ore types from different locations around the mine and sent to Maxxam Analytics in Burnaby, British Columbia. The sample was subjected to ABA tests to assess the potential for acid rock drainage and Shake Flask Extraction (SFE) to assess the potential for metal leaching.

The results were similar to historical testing campaigns and showed an abundance of acid-neutralizing potential (ANP) compared to the acid-generating potential (AGP). With a neutralization potential ratio (NPR) value of 4.8 and a net neutralization potential (NNP) of 41, the tailings were classified as non-acid generating.

Regarding the potential for metal leaching, Mercury is the only constituent leached at a concentration at the limit identified by the International Finance Corporation (IFC). All other metals that might cause an impact on the environment if discharged in surface water or groundwater are below both the IFC and limits identified by the U.S. Environmental Protection Agency (EPA).

20.5 Solid Waste Management

20.5.1 Non-Hazardous Solid Waste

The project proposes continued landfill disposal for inert, non-hazardous solid waste generated during construction and operations. Waste management practices prioritize valorization, recycling, and off-site reuse. The existing landfill will be expanded to accommodate construction waste and remain in use throughout operations, with final reclamation at mine closure. A temporary facility currently stores a Water Treatment Plant (WTP) sludge, which will eventually be disposed of underground. A new temporary storage site will be built near the landfill for ongoing WTP sludge management. The landfill, located on a hilltop outside catchment areas, avoids the need for water diversion. Waste is managed in trenches excavated and compacted by dozers, with each trench backfilled and covered with at least 1.5 metres of material before a new trench is created.

20.5.2 Solid Hazardous Waste

The anticipated hazardous waste primarily includes waste oils, process reagents, and laboratory chemicals. Waste oils will be incinerated or recycled by the supplier. Most reagents and chemicals will be disposed of within the process, with the remainder recycled by the supplier.

Cyanide containers and other reagent containers will be washed with fresh water in contained areas, complying with the International Cyanide Code standards. Neutralized products and containers will be disposed of or recycled according to local regulations. Laboratory fire assay wastes, which may contain small amounts of lead and any lead-contaminated dust, will be disposed of in accordance with local regulations. Hazardous material spill clean-ups will be prioritized, involving excavation, neutralization, and disposal of contaminated soils either on-site or at a licensed facility.

20.6 Flora and Fauna

Baseline studies have recorded the region’s biodiversity since 2007. Ongoing monitoring indicates minimal impact from the project. The local ecosystem consists of subtropical and tropical dry forests, supporting diverse plant species. Wildlife monitoring shows stable populations of birds, reptiles, and aquatic fauna.

Based on a specific Flora and Fauna Management Plan, to protect both flora and fauna environments and species, preventive conservation measures, including habitat relocation for threatened species, such as orchids, tillandsias, cactus or pitayas, have been implemented.

20.7 Cultural and Archeological Resources

A dedicated onsite team monitors the potential impact on cultural and archeological artifacts. Pre-construction inspections and external expert consultations ensure compliance. To date, no significant historical artifacts have been identified within the project's direct area of influence.

20.8 Environmental Monitoring

The Project is currently operating and reporting on a comprehensive environmental monitoring network consisting of 26 monitoring stations for water quality within and outside of the Project boundaries. There are also nine stations for monitoring air quality.

As part of the commitments under the approved EIA, the Project performs monthly monitoring to evaluate the water quality, air quality and noise levels in the direct and indirect areas of influence. Monthly and annual reports are prepared and presented to the Authorities (MARN, Ministerio de Energía y Mina – MEN, Ministerio de Salud de Jutiapa and Ministerio de Ambiente de Jutiapa) to report on the results of monitoring.

20.9 Environmental Management Plan

The Environmental Management Plan (EMP) has been updated using lessons learned from a decade of onsite environmental data collection. The plan is aligned with regulatory requirements and international best practices. It integrates corporate health, safety, and environmental programs, including emergency response strategies.

20.10 Social Management

Aura prioritizes strong community relationships. The Project retains a comprehensive database of community engagement activities and sustainability initiatives.

The Social Baseline Study (SBS) included in the EIA was prepared in 2006 using information extracted from the 2002 national census. It was updated in 2018 with the most recent Guatemalan official data at that time and information collected from a census conducted in the rural area, along with numerous face-to-face meetings with representatives of local organizations.

The scope of the study covered the town of Asunción Mita (capital of the municipality), also referred to as the "Urban Area," plus six rural villages included in the external area of influence and nine rural villages located near Lake Güija (collectively referred to as the "Rural Area"). The Lake Güija villages were included in the SBS update to gain a better understanding of the project perception outside of the external area of influence.

In Asunción Mita, a total of 28 registered small local community organizations, called Consejos Comunitarios de Desarrollo (COCODEs), manage public budgets and formulate projects to meet community needs. During the SBS update, meetings were held with representatives from the 20 active COCODEs in the urban area.

Era Dorada team members are actively involved with local organizations and communities to inform the population project activities and development strategies. In order to respond to the concern around the lack of specific knowledge regarding the Project, COCODE representatives and the public were invited to visit the project site, which proved to be very successful.

With respect to the surrounding villages, the majority have small populations (i.e. less than 400), except for Trapiche Vargas (566) and San Joaquín (745). The population characteristics of the villages are generally in line with those in the urban area.

To manage social issues, dedicated software has been implemented as a tool to register, monitor and report all aspects of the social management program. This has improved access to information for the onsite team and has resulted in more efficient monitoring and reporting activities.

The updated Social Management Plan (SMP) incorporates IFC performance standards and includes mechanisms for communication, grievance handling, conjuncture monitoring, local relationships, corporate social alignment, social impact evaluation, continuous training and community involvement. A Social Monitoring Committee (SMC) is in the process of being implemented to ensure transparency.

20.11 Mine Closure

The approved EIA includes a conceptual mine closure plan, which was further refined and is based on long-term monitoring of water quality and ecosystem restoration, using native plant species in revegetation.

Total closure costs are estimated at MUS$ 17.2.

The main requirements of the updated closure plan are summarized below:

20.11.1 Underground Mine

- Progressive underground backfilling of waste rock and tailings.
- Removal of all underground equipment.
- Portal and vent raises will be blocked with concrete and/or steel plugs.
- All infrastructure at portal pads will be removed and concrete pads will be covered with locally sourced fill and indigenous vegetation.

20.11.2 Process Plant

- Adequate cleaning of infrastructure and drainage of piping before demolition.
- Recycling, reuse, and reclamation of materials will be evaluated prior to the closure phase to avoid disposal in landfills.

- Concrete foundations will remain in place and be covered with locally sourced fill and indigenous vegetation. The surface will be graded to prevent water accumulation.

20.11.3 Administration Offices and Ancillary Buildings

- Administration offices and all other buildings (including the explosives storage warehouse) will be decommissioned and demolished.
- Cleaning and decontamination procedures will include equipment/waste management disposal/recycling prior to decommissioning.
- Concrete foundations will remain in place and be covered with locally sourced fill and indigenous vegetation. The surface will be graded to prevent water accumulation.

20.11.4 Dry Stack Tailings Facility (DSTF)

- Once mining operations have ceased, the DSTF will be closed, covered with locally sourced fill and revegetated with appropriate native species.
- All surface piping, mechanical equipment and electrical services associated with the DSTF will be decommissioned and disposed of.
- Soils around the facility will be tested for potential contamination.
- Final reclaimed profile will respect the site-specific landform objectives.

20.11.5 Waste Rock Facility

- All waste rock will be hauled and stored underground mine throughout the mining operations.
- Natural soil cover will be put in place and vegetation with indigenous species will be planted over the impacted area.

The Dry Stacking Tailings Facility (DSTF) will be constructed continuously over the life of the mine using the downstream construction method, so concurrent reclamation will not be possible. At the end of operations, exposed portions of the decant piping will be dismantled and the decant pipes will be plugged below the final surface.

The surface of the DSTF will be contoured so that it will shed precipitation rather than impound it. Topsoil that is stockpiled from the DSTF footprint during construction will be spread over the surface of the DSTF. Native grass seed mixture will be planted to reduce erosion.

20.12 Potential Risks and Mitigation Actions

20.12.1 Permitting

- Potential Risk: Delays may occur, resulting in increased duration of the assumed project development schedule due to new EIAs and permits needed for the power line and approval of permit amendments for injection of mine water and/or new permits.
- Mitigation Action: Proceed with the application of permit amendments and new EIAs/permits immediately to mitigate potential schedule impact. Continued discussions with local regulatory bodies are required to ensure the avoidance of unforeseen delays in permitting.

20.12.2 Tailings and Waste Rock

- Potential Risk: Tailings and waste rock were assumed to be Non-Acid- Generating (NAG) based on test work completed to date and the limited exposure time at the surface for waste rock. Additional test work is required before detailed engineering to confirm this assumption. If the classification is changed to Potentially-Acid Generating (PAG), the design will need to be updated accordingly, which could result in increased capital costs.
- Mitigation Action: The costs of geochemical testing should be included in the project budget and testing should be performed prior to detailed engineering.

20.12.3 Socio-Political

- Socio-Political Risk: Although the local community is favourable to the development of Era Dorada as an underground mine, there is a potential risk of socio-political opposition to mine development, which could adversely impact the project development schedule.
- Mitigation Action: The development of close relationships with the local communities, landowners and government, along with implementation of the Environmental Management Plan (EMP) and Social Management Plan (SMP), is required to engage the people with the project, as well as consistent monitoring of the main stakeholders.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate

LoM Project capital costs total $417 M, consisting of the following distinct phases:

- Pre-production capital costs – includes all costs to develop the property to a 1,500 tpd production. Initial capital costs total MUS$ 263.6 and are expended over a pre-production period on engineering, construction and commissioning activities.
- Sustaining capital costs – includes all costs related to the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations. Sustaining capital costs total MUS$ 136.2 and are expended in operating Years 1 through 16.
- Closure costs – includes all costs related to the closure, reclamation, and ongoing monitoring of the mine, post operations. Closure costs total MUS$ 17.2 and are primarily incurred in Year 1, with costs extending into Year 17 for ongoing monitoring activities.

The capital cost estimate was compiled using a combination of database costs and factors; the overall cost estimate was benchmarked against similar operations. Table 21-1 presents the capital estimate summary for initial and sustaining capital costs with no escalation.

21.1.1 Capital Cost Summary

Table 21-1: Capital cost summary

WBS DESCRIPTION	Pre-Production Cost (MUS$)	Sustaining Cost/Closure (MUS$)	Project Total Cost (MUS$)
Infrastructure	8.2	8.4	16.6
Power and Electrical	17.0	-	16.7
Water Management	16.1	24.7	40.8
Surface Operations	14.7	1.7	16.4
Mining	63.3	80.2	143.6
Process Plant	49.7	16.7	66.4
Construction Indirect	38.0	4.5	42.6
General Services – Owner's Costs	21.3	-	21.3
Logistics/ Taxes/ Insurance	9.0	-	9.0
Pre-production, Start-up & Commissioning	5.0	-	5.0
Contingency	21.2	-	21.9
Closure Costs	-	17.2	17.2
Total	**263.5**	**153.4**	**416.9**

Source: GE21, 2025.

Figure 21-1 and Figure 21-2 present the capital cost distribution for the pre-production and sustaining phases. As is typical with underground operations, the majority of sustaining capital costs relate to underground lateral and vertical development. In addition, due to the geothermal nature of the Project, the sustaining capital costs include a significant amount of reinjection well drilling and dewatering wells.

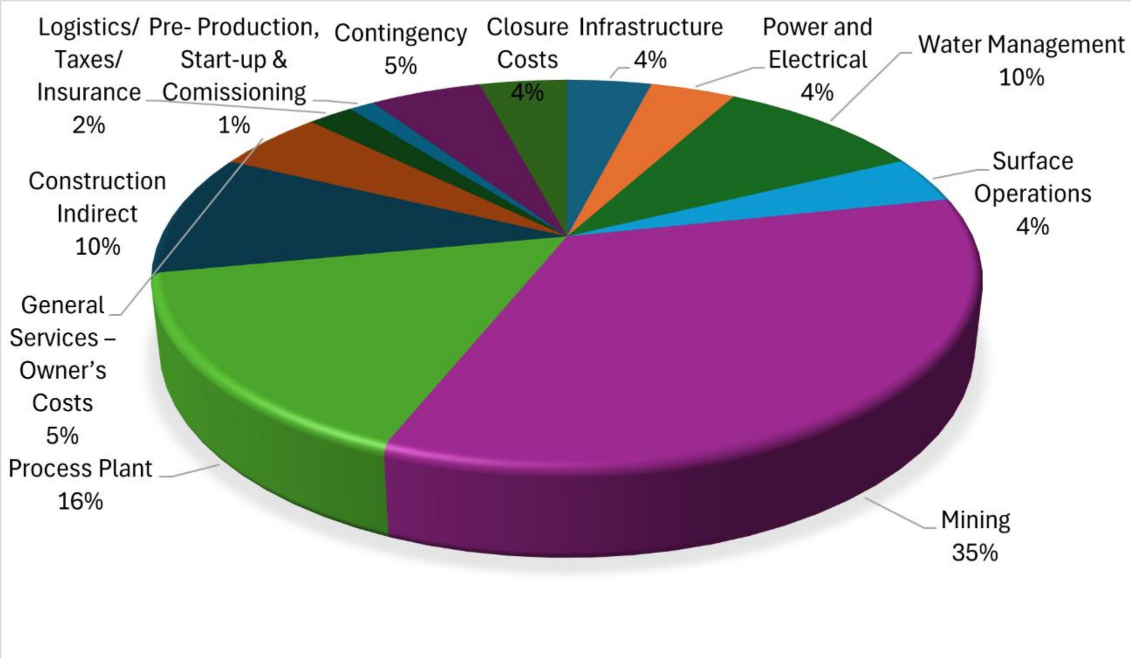

Figure 21-1: Distribution of initial capital cost

Source: GE21, 2025.

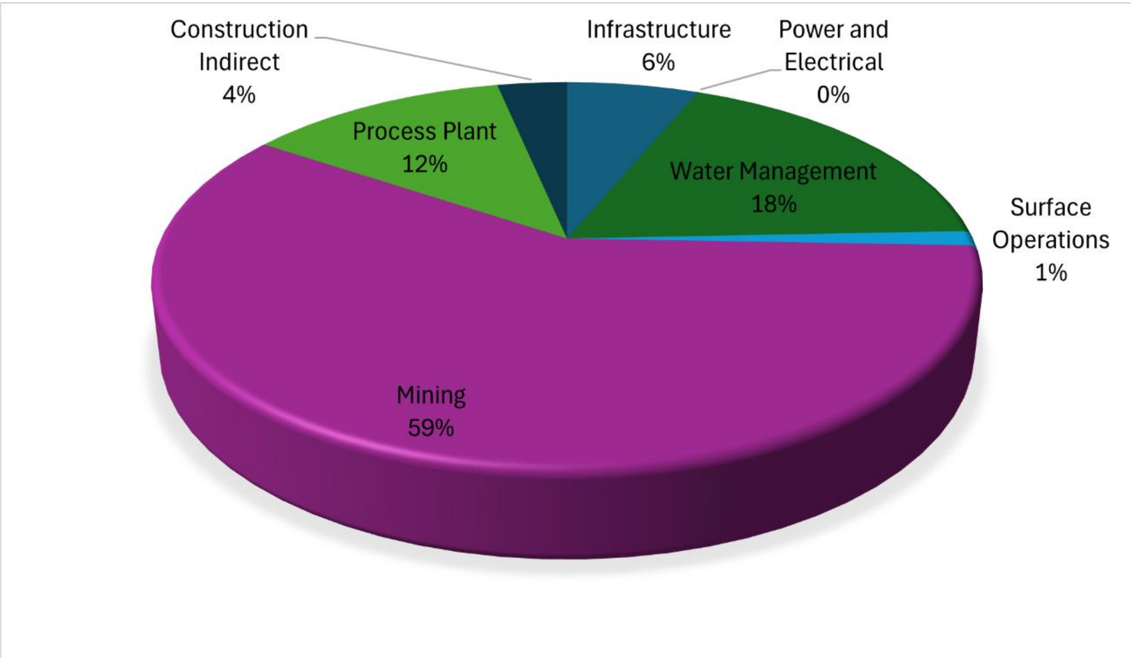

Figure 21-2: Distribution of sustaining capital cost

Source: GE21, 2025.

21.1.2 Capital Cost Profile

All capital costs for the Project have been distributed against the development schedule in order to support the economic cash flow model. Figure 21-3 presents an annual life of mine capital cost profile, including closure years (Years 20-23).

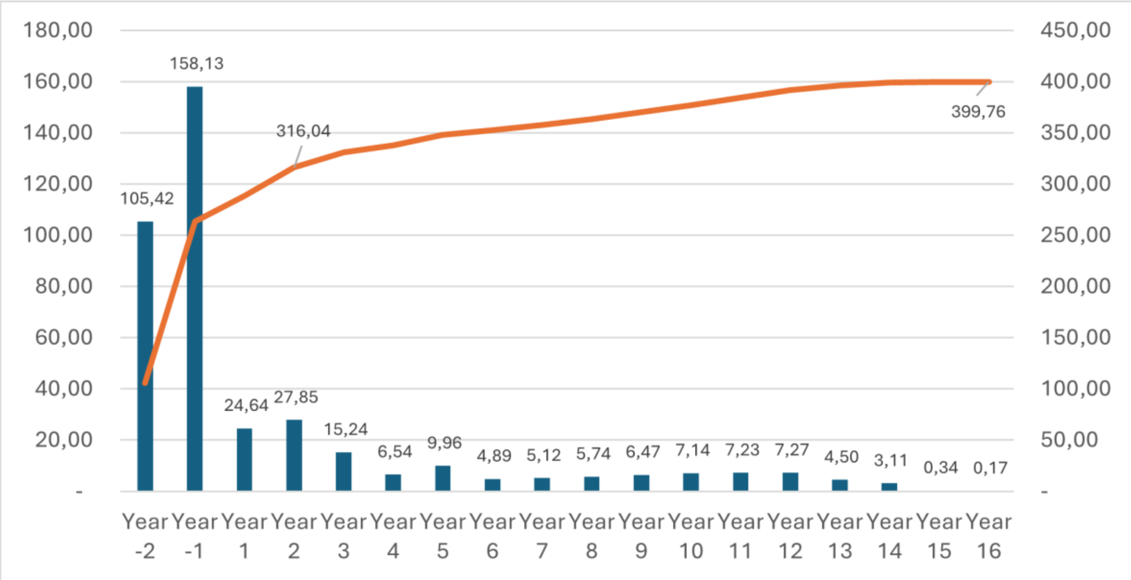

Figure 21-3: Capital cost profile

Source: GE21, 2025.

21.1.3 Key Estimation Assumptions

The following key assumptions were made during the development of the capital estimate:

- Underground development activities will be performed by the owner's forces and
- All surface construction (including earthworks) will be performed by contractors.

21.1.4 Key Estimation Parameters

The following key parameters apply to the capital estimates:

- Estimate Class: The capital cost estimates are considered Class 5 estimates (-30%/+50%). The overall Project definition is estimated to be 5%.
- Estimate Base Date: The base date of the estimate is May 2025. No escalation has been applied to the capital cost estimate for costs occurring in the future.
- Units of Measure: The International System of Units (SI) is used throughout the capital estimate.
- Currency: All capital costs are expressed in United States Dollars (US$). Portions of the estimate were estimated in Canadian Dollars (CA$) and converted to US$ at a rate of CA$1.00: US$0.78.

21.1.5 Basis of Estimate

21.1.5.1 Mine Capital Cost

Capital cost estimates are based on a combination of budgetary quotes from equipment suppliers, in-house cost databases and similar mines in Guatemala. Table 21-2 summarizes the underground mine capital cost estimate.

Table 21-2: Mine capital cost

WBS DESCRIPTION	Pre-Production Cost (MUS$)	Sustaining Cost/Closure (MUS$)	Project Total Cost (MUS$)
Capital Development	20.5	29.7	50.2
Underground Mobile Equipment	0.2	20.6	20.8
Ventilation	3.6	3.0	6.6
Water Management	0.6	0.6	1.3
Fixed Infrastructure	0.6	0.6	1.1
Material Handling	0.6	3.0	3.5
Electrical and Automation	2.6	1.6	4.2
Technical and Safety	1.4	1.2	2.6
Mining - Total	30.1	60.3	90.4

Source: GE21, 2025.

21.1.5.1.1 Capital Development

Capital development includes the labour, fuel, equipment usage, power, and consumables costs for lateral and vertical development required for underground access to stopes, and underground infrastructure.

- Lateral development fuel, equipment usage, power, and consumables requirements were developed based on the mine plan requirements. The manufacturer’s database equipment usage rates were applied to the required operating hours.
- Lateral development labour requirements were determined by the required equipment fleet in operation. Supervision and support services were prorated to the development costs based on the mix of underground activities occurring.

21.1.5.1.2 Underground Mobile Equipment

Underground mining equipment quantities and costs were determined through the buildup of mine plan quantities and associated equipment utilization requirements. Budgetary quotes were received and applied to the required quantities. The mining fleet is assumed to be provided by a contractor up to the end of Year 2, after which the Owner purchases new mobile equipment and takes control of mining development.

21.1.5.1.3 Underground Infrastructure

Design requirements for underground infrastructure were determined from design calculations for ventilation, dewatering, and material handling. Budgetary quotations or database costs were used for major infrastructure components. Allowances have been made for miscellaneous items, such as initial Personal Protective Equipment (PPE), radios, water supply, refuge stations, and geotechnical investigations. Acquisition of underground infrastructure is timed to support the mine plan requirements.

21.1.5.1.4 Capitalized Production Costs

Capitalized production costs are defined as mine operating expenses (operating development, mineralized material extraction, mine maintenance, and mine general costs) incurred prior to the introduction of feed to the processing facilities and the commencement of

Project revenues. They are included as a pre-production capital cost. Capitalized production costs are included in the asset value of the mine development and are depreciated over the mine life within the financial model.

21.1.5.1.5 Contractor Development

Contractor mine development is based on bid proposals received from Latin American contractors working in similar locations and production rates. The unit cost of development and stoping is based on unit costs for mine activities, including drill, blast, muck, haul, and support. A nominal KUS$ 250.00 was assumed for contractor mobilization and demobilization, including the cost to set up and take down any additional site facilities to store and maintain equipment. Table 21-3 outlines contractor rates used for capital development.

Table 21-3: Contractor capital development rates

Activity	Profile	Unit Price
Drift Rehabilitation	5.0 m x 5.0 m	$ 510/m
Capital Lateral Development	5.0 m x 5.0 m	$ 3,080/m
Capital Vertical Development	4.0 m x 4.0 m	$ 1,860/m
Capital Raisebore	3.0 m diameter	$ 2,150/m

Source: GE21, 2025.

21.1.5.2 Surface Construction Cost

Surface construction costs include site development, mineral processing plant, tailings management facility, and on-site and off-site infrastructure. These cost estimates are primarily based on material and equipment costs from material take-offs and detailed equipment lists. Pricing for main equipment and bulk materials was primarily estimated from similar projects, with some factors applied for minor cost elements.

Table 21-4 presents a summary basis of estimates for the various commodity types within the surface construction estimates. Growth factors were included above the neat material take-off quantities for all areas.

Table 21-4: Surface construction basis of estimate

Commodity	Basis
Access Roads	Material take-offs are developed based on general arrangements by local contractors.
Bulk Earthworks	Model volumes from preliminary 3D grading model. Database unit rates for bulk excavation and fill. Material take-offs for surface drainage, water management ponds and temporary roads from general drawings.
Concrete and Structural Steel	Material take-offs estimated and factored costs applied from similar projects.
Buildings and warehouses	Buildings according to general arrangements with factored costs for overall building structures.
Mechanical / Electrical Equipment	Mechanical/Electrical equipment based on a database with factored costs applied from similar projects.
Piping	Material take-offs are estimated for major pipelines and small-bore piping. Factored costs applied from similar projects.
Dewatering and Injection Wells	Number of wells based on dewatering and injection plan. Factored costs applied from similar projects.
Power Transmission Line and Major Sub-stations	Estimated based on general arrangements and site layouts. Factored costs applied from similar projects.

Source: GE21, 2025.

21.1.5.2.1 Surface Construction Sustaining Capital

Sustaining capital costs include the following: dewatering and injection wells.

Dry stack storage facility earthworks quantities were developed from engineering drawings by the design engineer. Database unit rates benchmarked against projects recently constructed in the region have been applied to the engineered quantities.

It is assumed that DSF construction will be performed by a contractor during both the construction and operations phases.

21.1.6 Indirect Cost Estimate

Indirect costs are classified as costs not directly accountable to a specific cost object. Table 21-5 presents the subjects and basis for the indirect costs within the capital estimate.

Table 21-5: Indirect cost basis of estimate

Commodity	Basis
On-Site Contract Services	Heavy Lift Crane Services based on estimated durations and historical rates for crane services.
Contractor Field Indirects	Estimated by first principles and including the following items:
	Time-based cost allowance for general construction site services (temporary power, heating and hoarding, contractor support, etc.) applied against the surface construction schedule.
	Construction offices and wash car facilities.
	Safety training, tools and equipment.
	Environmental cost.
	Materials Management and Warehouse Operations
	Site Maintenance and Temporary Services
	Surveying and Quality Assurance
	Communications
	Contractor facilities and related cost.
	Construction team facilities, fuel
Freight and Logistics	Factor (10%) for freight and logistics related to the materials and equipment required for the crushing plant, mineral processing plant, and on-site and off-site infrastructure. Factor excludes mining equipment as prices are FOB site.
Vendor Representatives	Estimated by first principles, assessing the equipment supply packages and vendor services hours required for commissioning equipment.
Capital Spares	Based on material take-offs from similar projects.
Start-up and Commissioning	Included under EPCM (personnel) and Owner's team costs (material and consumables).
First Fills	Based on requirements determined by engineering and database pricing.
Detailed Engineering and Procurement	Factor applied against direct and indirect hours for engineering management, detailed design, drawings, and major equipment procurement.
Project and Construction Management	Staffing plan built up against the development schedule for project management, health and safety, construction management, field engineering, project controls, and contract administration. Costs are based on similar projects.

Source: GE21, 2025.

21.1.7 Owner Cost Estimate

Owner's costs are capitalized in the initial capital costs during the construction phase. Owner's costs for the project start in Month 10 of Year -2 of the CAPEX cash flows. Any Owner's costs prior to this are assumed to be within the Owner's approved budget expenses and are considered sunk costs.

21.1.7.1 Process Plant Operations

The following processing-related costs are included in the initial capital:

- Management, technical, operations, and maintenance labour employed during the construction phase
- First fills of consumables and reagents, and initial consumption during process commissioning to initiate operations, and
- Energy costs for power consumed during process commissioning and start-up activities.

21.1.7.2 Water Treatment Plant Operation

The following cost elements are included in the initial capital costs for operation of the water treatment plant:

- Technical and operations labour
- Maintenance and parts
- Power consumption, and
- Reagents, consumables, and third-party services.

21.1.7.3 Dewatering Wells Operations

The following cost elements are included in the initial capital costs for the operations of dewatering wells:

- Supervision of technical and operations labour
- Maintenance
- Power consumption, and
- Reagents, consumables, and third-party services.

21.1.7.4 Pre-Production G&A – Labour

Costs for general and administrative labour are included for the following sectors:

- Business Services
- General management, sustainability, including:
    - Community relations
    - Health and Safety
    - Environmental
- Human resources training
- Procurement Logistics
- Security, and
- Site services and facilities maintenance.

Costs associated with the following activities are included within the Sustainability category:

- Submit permit amendments for the injection of mine water and application for new EIA/permits for the access road and powerline.

- Complete land agreement negotiations for the access road and power line right-of-ways and injection well pads.

21.1.7.5 Pre-Production G&A – Equipment

Costs for owner-site support equipment usage are included for the following sectors:

- Site Services
- Warehouse/material management
- Security
- Health, Safety, and Environment, and
- Administration/management.

21.1.7.6 Pre-Production G&A – Expenses and Service

Costs for general and administrative expenses and fees are included for the following sectors:

- Health, safety and medical supplies
- Staff safety equipment
- Environmental services, fees, and outside laboratory costs
- Human resources (training, recruitment)
- Construction insurance
- Community relations and programs
- Legal and regulatory, including property tax
- External consulting IT and communications
- Site office costs
- Office lease and services for Guatemala City
- Waste disposal
- Existing infrastructure, power and maintenance

21.1.8 Closure Cost Estimate

Closure costs have been estimated based on the typical closure, reclamation, and monitoring activities for an underground mine. Activities include:

- Removal of all surface infrastructure and buildings
- Closure and capping of the DSTF
- Removal of all fixed underground equipment
- Closure of the underground mine portals
- Power transmission line and substation removal
- Re-vegetation and seeding, and
- Ongoing site monitoring

The majority of closure costs are incurred immediately following the completion of operations (20). Monitoring activities are anticipated to extend to Year 24.

21.1.9 Contingency

Contingency has been applied to the estimate on a line-by-line basis as a deterministic allowance by assessing the level of confidence of the scope definition, supply cost and installation cost and then applying an appropriate weighting to each of the three inputs. The overall recommended pre-production contingency resulted in approximately 12% of direct, indirect, and Owner's costs.

21.1.10 Capital Estimate Exclusions

- The following items have been excluded from the capital cost estimate:
- Working capital (included in the financial model)
- Financing costs
- Currency fluctuations
- Lost time due to severe weather conditions beyond those expected in the region
- Lost time due to force majeure
- Additional costs for accelerated or decelerated deliveries of equipment, materials or services resulting from a change in the Project schedule
- Warehouse inventories, other than those supplied in initial fills, capital spares, or commissioning spares
- Any Project sunk costs (studies, exploration programs, etc.)
- Value added tax (VAT)
- Closure bonding
- Escalation cost

21.2 Operating Cost Estimate

The operating cost estimate in this study includes the costs to mine and process the mineralized material to produce doré, along with site services to maintain the site and general and administrative expenses (G&A). These items total the Project operating costs and are summarized in Table 21-1. The target accuracy of the operating cost is -30/+50%. The operating cost estimate is broken into four major sections:

- Underground mining
- Processing
- Site Services
- General and Administrative (G&A)

The total operating unit cost is estimated to be US$ 170/t processed. Average annual, total LoM and unit operating cost estimates are summarized in Table 21-6. The unit rates in this table include tonnes mined during pre-production. Figure 21-4 illustrates the operating cost distribution.

Operating costs are expressed in USD. No allowance for inflation has been applied.

Table 21-6: Breakdown of estimated operating costs

Operating Costs	Avg Annual (M$)	$/t processed	LoM (M$)
Mining	38.70	100	890.00
Processing	12.38	32	284.80
Site Services	6.97	18	160.20
G&A	7.74	20	178.00
Total	**65.78**	**170**	**1,513.00**

Source: GE21, 2025.

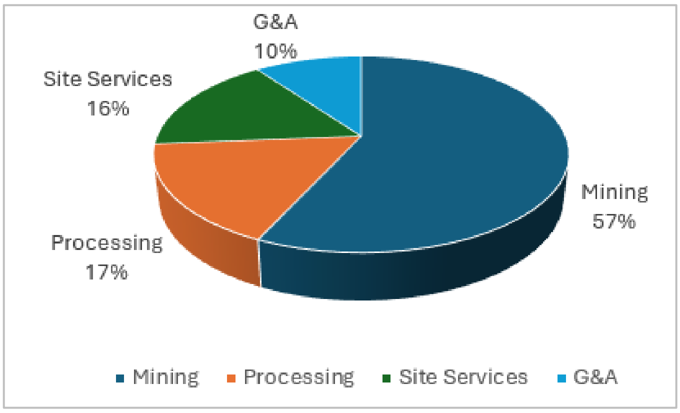

Figure 21-4: Operating cost distribution

Source: GE21, 2025.

The main operating cost component assumptions are shown in Table 21-7.

Table 21-7: Main OPEX component assumptions

Item	Unit	Value
Electrical Power Cost	$/kWh	0.06
Overall Power Consumption (all facilities)	kWh/t processed	215
Diesel Cost (delivered)	$/litre	0.79
LoM Average Operations Workforce	employees	584

Source: Bluestone, 2019.

21.2.1 Operation Labour

This section provides an overview of the total workforce, and the methods used to compile the labour rates. Table 21-8 summarizes the total planned workforce during Project operations.

Table 21-8: Main OPEX component assumptions

Operating Costs	Construction	Operations
Mining	193	329
Processing	97	97
Site Services	64	69
G&A	120	105
Total	**474**	**600**

Source: Bluestone, 2019.

Labour is a significant portion of annual operating costs. Labour rates include base wage and allowances for overtime, night shift, insurance, tax, and benefits. Labour burdens were assembled using first principles and ranged from 34 to 42%. The following items are included in the burdened labour rates:

- unscheduled overtime at 10%;
- social security at 12.67%;
- yearly Christmas bonus at 1 month's salary per year;
- yearly bonus of 14 at 1 month's salary per year;
- vacation pay at 4%;
- statutory holiday pay at 3%;
- yearly insurance payments of $ 54.00/employee/month; and
- Monthly savings funds at 13%.

21.2.2 Mining Operating Cost Estimate

Mining operating costs listed in Table 21-9 are averaged over the life of the mine. Mine operating costs have been built up by using a combination of first-principle engineering and equivalent Project scaling.

Mine operating unit costs are summarized below in Table 21-9 and include:

- Waste development – costs related to the drilling, blasting, mucking, and hauling of non-capital development
- Production – costs related to the drilling, blasting, mucking, and hauling of ore for both long hole and cut and fill stoping
- Backfill – costs related to CRF and paste backfill operations, including the CRF and paste plants
- Mine Maintenance – maintenance labour costs that support all other sectors
- Mine General – costs related to mine support activities, such as technical services, shared infrastructure, support equipment, and definition drilling

Table 21-9: Underground mine operating costs

Mining Category	Unit Cost ($/t processed)	LoM Cost (MUS$)
Lateral Waste Development	6.27	55.78
Long Hole Stoping	21.95	195.37
Cut and Fill Stoping	25.52	227.12
Backfill	24.92	221.80
Mine Maintenance	3.88	34.53
Mine General	17.46	155.40
Total	100.00	890.00

Source: GE21, 2025.

21.2.3 Processing Operating Cost

Process operating costs were developed using labour rates based on operating mines in the area and sufficient personnel to operate the process plant, factored maintenance costs, budget quotes for consumables and a factored power requirement. Process operating costs are summarized below in Table 21-10. Costs are subdivided into operating categories.

Table 21-10: Process operating costs

Mining Category	Unit Cost ($/t processed)	LoM Cost (MUS$)
Labour	10.53	93.69
Power	6.81	60.58
Maintenance and Consumables	14.66	130.53
Total Processing OPEX	32.00	284.80

Source: GE21, 2025.

Process labour includes the burden for salaried and hourly employees to account for in-country benefits, training, production bonuses and potential ex-patriot benefits & costs.

Equipment maintenance was calculated by applying a factor of 4% to major process equipment costs.

Power costs were calculated from the total installed power, assuming $ 0.06/kWh.

21.2.4 General and Administration Operating Cost Estimate

General and Administration costs include all on-site activities, including but not limited to dry stack tailings (DST) haulage, personal protection services, water treatment plant operation, site services equipment and labour, office operating costs and associated labour. The summary of costs is shown in Table 21-11, averaged over the life of the mine.

Table 21-11: General and administration (G&A) operating cost summary

Sector	Average US$ M/year	LoM MUS$	US$/t processed
G&A Labour Total	**4.18**	**100.20**	**11.26**
Business Services	0.85	20.43	2.30
General Management	0.50	12.11	1.36
Sustainability	1.26	30.27	3.40
Human Resources	0.27	6.51	0.73
Purchasing and Logistics	0.20	4.84	0.54
Security	0.52	12.56	1.41
G&A Service Staff	0.30	7.27	0.82
Project Team	0.26	6.21	0.70
G&A Services and Expenses Total	**3.24**	**77.80**	**8.74**
Accommodations	0.18	4.24	0.48
Heath and Safety, Medical, and First Aid	0.22	5.30	0.60
Environmental	0.44	10.60	1.19
Human Resources	0.63	15.14	1.70
Insurance and Legal	0.84	20.13	2.26
External Consulting	0.47	11.20	1.26
IT and Communications	0.16	3.78	0.43
Office and Miscellaneous Costs	0.01	0.30	0.03
Satellite Office	0.11	2.57	0.29
Employee Travel	0.20	4.84	0.54
Total G&A	**7.42**	**178.00**	**20.00**

Source: GE21, 2025.

21.2.4.1 G&A Labour Requirements

Table 21-12 lists the site supervision and support personnel requirements and costs.

Table 21-12: G&A labour requirements & costs

Labour	Salary	Loaded Pay (US$)	Quantity
Mine/General Manager	Salary	243,459	1
Site Administrator	Salary	12,956	1
Accounting & Taxes Manager	Salary	20,340	1
Accounting/Payroll Coordinator	Salary	15,417	2
Human Resources Manager	Salary	22,802	1
Human Resources Clerk	Salary	12,956	1
Trainer	Salary	15,417	2
Community Relations Manager	Salary	74,493	1
Community Relations Coordinator	Salary	15,417	1
IT/Telecom. Technician	Salary	22,802	1
Procurement/Contracts/Logistics Manager	Salary	25,263	1
Procurement/Contracts Agent	Salary	20,340	1
Warehouse Operators	Salary	15,417	1
Multi-Equipment Operators	Salary	15,417	2
Health, Safety, and Security Manager	Salary	49,878	1
Health & Safety Coordinator	Salary	22,802	2
First Aid Attendant/Nurse	Salary	7,591	4
Environmental Manager	Salary	49,878	1
Environmental Technician	Salary	15,417	1
Environmental Coordinator	Salary	25,263	2
Protective Services Officials	Salary	12,956	30
Site Services Foreman	Salary	37,571	2
Carpenters	Salary	15,417	1
WTP Operator	Salary	15,417	4
Multi-Equipment Operator	Salary	15,417	4
Skilled Labourers	Salary	11,725	2
Total G&A Labour			**71**

Source: Bluestone, 2019.

22 ECONOMIC ANALYSIS

The economic analysis for the Project is based on Mineral Resource Estimates, including the annual mine production schedule previously outlined in this report. As required under NI 43-101, the results of this analysis should not be interpreted as demonstrating the economic viability of the project.

The outcome of the economic analysis is subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those. The information on which this analysis is based is listed below:

- Mineral Resource Estimates
- Assumed fixed exchange rate
- Proposed mine production plan
- Projected mining and processing recovery rates
- Fixed installed processing plant capacity
- Assumptions on closure costs
- Assumptions on environmental, licensing, and social risks
- Changes in production costs relative to the assumptions

This analysis does not rely on:

- Unrecognized environmental risks
- Unanticipated recovery expenses
- Different geotechnical and/or hydrogeological considerations during mining
- Unexpected variations in the quantity of mineralized material, grade, metallurgical recovery efficiency, and plant recovery efficiency
- Accidents, labour disputes, and other mining industry risks
- Changes in tax rates
- Assumptions of commercial discounts that are not foreseen in the financial analysis

Based on the Cash Flow Model results, the Era Dorada Project has a project-level after-tax NPV of MUS$ 485.5 at a 5% discount rate, an after-tax IRR of 23.8%, and a payback period of 3.75 years. Project results are summarized in Table 22-1.

Table 22-1: Summary of key financial results

Description	Unit	Value
Price Au[1]	$/ozt	$ 2,409.68
Price Ag[1]	$/ozt	$ 28.64
Au recovery	kozt	$ 1,377.14
Ag recovery	kozt	$ 4,308.07
Gold Equivalent - GEO (kozt)	kozt	$ 1,426.88
NPV - After-tax	MUS$	485.49
IRR - After-tax	%	23.8%
After-tax Payback[2]	years	3.75
NPV - Pre-tax @5%	MUS$	680.39
IRR - Pre-tax	%	30.4%

Description	Unit	Value
Pre-tax Payback[2]	years	3.35
Life of Mine (LoM)	years	17.00
Average Annual Production (Au)	kozt	$ 81.01
Average Annual Production (Ag)	kozt	$ 253.42
Average Annual Production (Au Eq)	kozt	$ 83.93
Total Oz Payable Au	kozt	$ 1,376.04
Total Oz Payable Ag	kozt	$ 4,286.53
Closure Cost (MUS$)	MUS$	$ 17.2
Royalties	MUS$	$ 242.07
OPEX	MUS$	$ 1,512.99
Initial Capital	MUS$	$ 263.55
Sustaining and Closure Capital	MUS$	$ 153.41
Average Cash Cost	MUS$/oz	$ 1,072.40
Average All in Sustaining Cost	$/oz	$ 1,179.91

Notes:
1. Price resulting from the weighted average of forecast gold and silver prices from 2028 onwards.
2. Payback period post-construction.
Source: GE21, 2025.

22.1 Methodology

An economic model was developed to estimate the Project's post-tax annual cash flow and sensitivity analysis based on an assumed discount rate of 5%. Capital and operating cost estimates were summarized in Section 21 of this report. The economic analysis was performed without inflation.

This analysis was conducted with the following assumptions:

- Year -1 corresponds to the pre-production phase
- Price inflation and escalation factors are ignored (constant-dollar basis)
- Results are based on 100% equity capital
- Project revenue is derived from selling a basket list of graphite production
- All production is sold in the year of production

22.2 Gold and Silver Prices

The silver and gold prices used for the economic evaluation are 28.44 US$/oz and 2,389 US$/oz, respectively, based on the long-term consensus forecast from over 20 investment banks.

22.3 Mine Production

The annual production figures were derived from the mine plan presented in Section 16. Over the life of the mine, a total of 9 Mt of ore is mined.

22.4 Plant Production

The silver and gold production plan was estimated considering the recoveries described in Section 14. The process plant is designed based on the following criteria:

- 1,500 tpd ore production;

- 85% average silver recovery;
- 96 % average gold recovery;
- estimated LoM gold production of 1.4 Moz and silver production of 4.3 Moz.

22.5 Revenue

Annual revenue is calculated by applying the estimated silver and gold prices to the annual payable metal for each operating year. Gross revenue represents the total value of payable silver and gold. Net Smelter Return (NSR) revenue accounts for associated selling costs, while Net Revenue further accounts for royalties payable, as shown in Table 22-2.

Table 22-2: Revenue composition

Description	LoM (MUS$)
Gold Gross Revenue	$ 3,315.80
Silver Gross Revenue	$ 122.78
Total Gross Revenues	$ 3,438.59
Smelting and Refining	$ 17.19
Total NSR Revenues	$ 3,421.39
Goldcorp Royalty	$ 35.92
Newmont Royalty	$ 34.21
Guatemalan Government Royalty	$ 171.93
Total Net Revenues	**$ 3,179.33**

Source: GE21, 2025.

22.6 Total Operating Cost

The average total unit operating cost over the life of the mine is estimated at $170 t of ore processed. A detail of the operating cost is shown in Section 21 and summarized in Table 22-3.

Table 22-3: Detailed operating costs

Description	LoM (MUS$)	$/t ROM	$/ozt GEO
Mining	$ 890.00	$ 100.00	$ 623.73
Processing	$ 284.80	$ 32.00	$ 199.59
Site Service	$ 160.20	$ 18.00	$ 112.27
SG&A	$ 178.00	$ 20.00	$ 124.75
Total Operating Costs	**$ 1,512.99**	**$ 170.00**	**$ 1,060.35**

Source: GE21, 2025.

22.7 Royalty Rights

Royalties in mining are financial compensation paid to the State for the right to exploit Mineral Resources, contributing to the redistribution of the benefits of mining activities. In Guatemala, the royalties due for mineral extraction amount to 5% of gross revenue. Additionally, a 1.05% rate on the Net Smelter Return is considered for royalty payments to Goldcorp Royalty.

22.8 Capital Expenditure

22.8.1 Initial Capital

The financial analysis for the Era Dorada Project was prepared under the assumption of 100% equity financing for initial capital expenditures. The total initial capital cost estimate is MUS$ 263.5 and includes expenditures related to:

- pre-stripping;
- power supply and electrical infrastructure;
- water management systems;
- surface infrastructure;
- mine equipment and development;
- process plant construction;
- indirect costs;
- general services;
- owner's costs, logistics, taxes, and insurance.

The estimate also includes costs associated with process plant operations during pre-production, start-up, and commissioning, as well as an appropriate contingency allowance.

22.8.2 Sustaining Capital

Sustaining capital is estimated at MUS$ 136.00 and includes the renewal of the mining fleet, water management, process plant maintenance, surface operations and indirect constructions and infrastructure costs.

22.8.3 Remediation and Closure Capital

The Project includes a provision of MUS$ 17.19 to be accumulated over the LoM through the allocation of 0.5% of gross revenue. This amount is considered appropriate given the scale of the project.

22.9 Total All-in Sustaining Cost

The average total All-In Sustaining cost over the LoM is estimated at $1,179 per ounce of payable gold equivalent, as shown in Table 22-4.

Table 22-4: All in sustaining costs composition

Description	LoM (MUS$)	$/t ROM	$/ozt GEO
Mining	$ 890.00	$ 100.00	$ 623.73
Processing	$ 284.80	$ 32.00	$ 199.59
Site Service	$ 160.20	$ 18.00	$ 112.27
SG&A	$ 178.00	$ 20.00	$ 124.75
Selling Costs and Royalties	$ 17.19	$ 1.93	$ 12.05
Sustaining and Closure Capital	$ 153.41	$ 17.24	$ 107.51
Total All-in Sustaining Costs	$ 1,683.59	189.17	$ 1,179.91

Source: GE21, 2025

22.10 Working Capital

For the purposes of estimating working capital requirements, the following assumptions were adopted for the financial analysis in Table 22-5.

Table 22-5: Working capital periods

Working Capital Component	Days
Average collection period	30
Average inventory turnover period	30
Average payment period	30

Source: GE21, 2025.

22.11 Depreciation

Depreciation of capital assets has been estimated at 10% annually for the purpose of simplifying the analysis. Fiscal reserves related to exploration and Resource development have been excluded from the depreciation calculation. No salvage value has been applied to capital items, as any salvage proceeds are treated as taxable income.

22.12 Exchange Rate Forecast

The exchange rate was defined based on parameters adopted in international projects, not using values projected by any financial institution. The exchange rate used was Q7.75/USD.

22.12.1 Income Tax

Income tax applies to the profits earned by companies and other legal entities. It is calculated based on the accounting results determined at the end of a reporting period, such as a quarter or a fiscal year. In Guatemala, companies are taxed on their gross profits at a rate of 25% of taxable income.

22.13 Discounted Cash Flow

A simplified discounted cash flow was developed to evaluate the Project based on economic-financial parameters, mining schedule results, and capital, sustaining and operating cost estimates, and is presented in Table 22-6.

Table 22-6: Simplified discounted cash flow

Cash Flow	Un.	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Gold Equivalent Ounce	ktoz	1,426.88	-	-	97.91	90.25	88.25	86.39	80.50	95.15	81.77	87.20	84.48	81.46	80.52	81.58	90.87	94.61	94.60	89.48	21.88	
PriceAu	US$/toz	2,409.68	-	-	2,693.00	2,389.00	2,389.00	2,389.00	2,389.00	2,389.00	2,389.00	2,389.00	2,389.00	2,389.00	2,389.00	2,389.00	2,389.00	2,389.00	2,389.00	2,389.00	2,389.00	
Gross Revenue	MUS$	3,438.59	-	-	263.67	215.60	210.82	206.38	192.30	227.30	195.35	208.33	201.81	194.61	192.36	194.88	217.08	226.03	225.99	213.78	52.28	
Freight & Refining	MUS$	17.19	-	-	1.32	1.08	1.05	1.03	0.96	1.14	0.98	1.04	1.01	0.97	0.96	0.97	1.09	1.13	1.13	1.07	0.26	
Net Smelter Return	MUS$	3,421.39	-	-	262.35	214.52	209.77	205.35	191.34	226.17	194.37	207.29	200.81	193.64	191.40	193.91	216.00	224.90	224.86	212.71	52.01	
Royalties	MUS$	242.07	-	-	18.56	15.18	14.84	14.53	13.54	16.00	13.75	14.67	14.21	13.70	13.54	13.72	15.28	15.91	15.91	15.05	3.68	
Tax Sales	MUS$	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
Net Revenue	MUS$	3,179.33	-	-	243.79	199.35	194.92	190.82	177.81	210.16	180.62	192.62	186.60	179.94	177.86	180.19	200.72	208.98	208.95	197.66	48.33	
Total opex	MUS$	1,512.99	-	-	93.08	93.08	93.08	93.08	93.08	93.08	93.08	93.08	93.08	93.08	93.08	93.08	93.08	93.08	93.08	93.08	23.79	
Cash Cost	**US$/oz**	**1,072.40**	**-**	**-**	**964.08**	**1,043.27**	**1,066.67**	**1,089.36**	**1,168.22**	**990.18**	**1,150.21**	**1,079.29**	**1,113.73**	**1,154.49**	**1,167.86**	**1,152.91**	**1,036.23**	**995.71**	**995.86**	**1,052.07**	**1,099.24**	
All in Sustaining Costs	**US$/oz**	**1,179.91**	**-**	**-**	**1,215.77**	**1,351.83**	**1,239.35**	**1,165.06**	**1,291.94**	**1,041.61**	**1,212.88**	**1,145.06**	**1,190.31**	**1,242.17**	**1,257.62**	**1,242.06**	**1,085.75**	**1,028.60**	**1,061.24**	**1,117.37**	**1,358.52**	
Gross Profit	MUS$	1,666.33	-	-	150.72	106.27	101.85	97.74	84.73	117.09	87.54	99.54	93.52	86.87	84.79	87.12	107.64	115.91	115.88	104.59	24.54	
Depreciation	MUS$	373.38	-	-	26.35	28.82	31.60	33.13	33.78	34.78	35.27	35.78	36.35	26.46	11.36	9.62	7.56	6.49	6.14	5.18	4.71	
EBIT (US$)	MUS$	1,292.95	-	-	124.36	77.45	70.25	64.62	50.95	82.31	52.28	63.77	57.17	60.41	73.43	77.50	100.08	109.42	109.73	99.40	19.83	
Income Tax	MUS$	323.24	-	-	31.09	19.36	17.56	16.15	12.74	20.58	13.07	15.94	14.29	15.10	18.36	19.37	25.02	27.36	27.43	24.85	4.96	
Operational profit(US$)	MUS$	969.72	-	-	93.27	58.09	52.68	48.46	38.21	61.73	39.21	47.82	42.88	45.31	55.07	58.12	75.06	82.07	82.30	74.55	14.88	
(=) EBIT	MUS$	1,292.95	-	-	124.36	77.45	70.25	64.62	50.95	82.31	52.28	63.77	57.17	60.41	73.43	77.50	100.08	109.42	109.73	99.40	19.83	
Depreciation	MUS$	373.38	-	-	26.35	28.82	31.60	33.13	33.78	34.78	35.27	35.78	36.35	26.46	11.36	9.62	7.56	6.49	6.14	5.18	4.71	
(=) EBITDA	MUS$	1,666.33	-	-	150.72	106.27	101.85	97.74	84.73	117.09	87.54	99.54	93.52	86.87	84.79	87.12	107.64	115.91	115.88	104.59	24.54	
(-) Capex	MUS$	399.76	105.42	158.13	24.64	27.85	15.24	6.54	9.96	4.89	5.12	5.74	6.47	7.14	7.23	7.27	4.50	3.11	0.34	0.17	-	
(+) Residual Value	MUS$	26.38	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	26.38	
(+-) Working Capital	MUS$	-	-	-	22.01	0.38	-0.22	-3.33	-0.80	2.04	-1.84	0.76	-0.36	2.48	-0.44	0.11	1.24	0.49	-0.01	-0.73	-16.29	-5.50
(-)ARO	MUS$	17.19	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5.85	5.67	5.67	-
(-) Income Tax	MUS$	323.24	-	-	31.09	19.36	17.56	16.15	12.74	20.58	13.07	15.94	14.29	15.10	18.36	19.37	25.02	27.36	27.43	24.85	4.96	-
(=) Post-tax Cash Flow	MUS$	952.52	-105.42	-158.13	72.97	58.69	69.27	78.38	62.83	89.58	71.19	77.11	73.13	62.14	59.65	60.36	76.88	84.95	82.27	74.62	56.58	5.50
(=) Post-Tax Acumulated Cash Flow	MUS$	7,345.55	-105.42	-263.55	-190.58	-131.89	-62.63	15.75	78.59	168.16	239.35	316.45	389.58	451.72	511.37	571.73	648.61	733.55	815.82	890.44	947.02	952.52
(=) Pre-Tax Cash Flow	MUS$	1,275.76	-105.42	-158.13	104.06	78.05	86.83	94.53	75.57	110.15	84.26	93.05	87.42	77.24	78.00	79.73	101.90	112.30	109.70	99.47	61.54	5.50
(=) Pre-tax Accumulated Cash Flow	MUS$	10,578.08	-105.42	-263.55	-159.49	-81.44	5.39	99.92	175.49	285.64	369.90	462.95	550.37	627.61	705.61	785.35	887.25	999.55	1,109.25	1,208.72	1,270.26	1,275.76

Source: GE21, 2025.

The economic results are presented in Table 22-7.

Table 22-7: Simplified discounted cash flow results

Discount Rate (%)	5%
NPV – After Tax (MUS$)	485.49
IRR – After Tax (%)	23.8%
Payback - After Tax (years)	3.75

Source: GE21, 2025.

22.14 Sensitivity Analysis

Based on the result, a sensitivity analysis was conducted to observe the Project's value and profitability behaviour in response to:

- Selling price variation
- CAPEX variation
- Mining OPEX variation
- Freight / Refining costs variation

The sensitivity analysis results are presented in Figure 22-1.

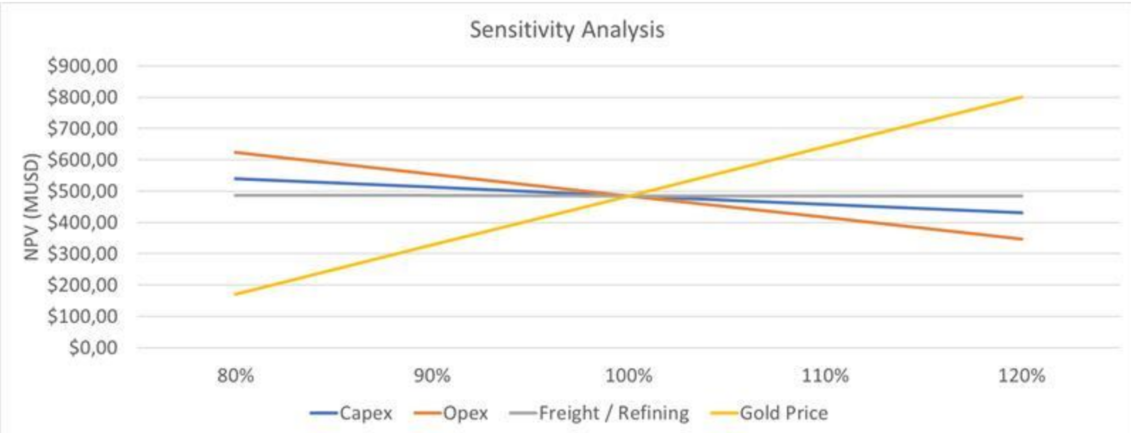

Figure 22-1: Sensitivity analysis

Source: GE21, 2025.

A sensitivity analysis was conducted on the applied discount rate and the equivalent gold price. The results are presented in Table 22-8.

Table 22-8: Sensitivity analysis results

NPV (MMUS$)		**Gold Price (%)**				
		BEP	**R.P. – 10%**	**R.P.**	**R.P. +10%**	**R.P. +20%**
$ 485.49		$ 1,928	$ 2,169	$ 2,410	$ 2,651	$ 2,892
Discount Rate	5%	0	328	485	643	800
	10%	(75)	141	245	348	452
	15%	(110)	41	113	185	257
	20%	(127)	(16)	37	90	143
	24%	(133)	(43)	0	43	86

Source: GE21, 21025.

23 ADJACENT PROPERTIES

There are no adjacent properties relevant to the scope of this report.

24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information relative to the scope of this report.

25 INTERPRETATION AND CONCLUSIONS

25.1 Geology & Mineral Resources

Era Dorada is a classic hot springs-related, low-sulphidation 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 Au g/t and 52 Ag g/t). Bonanza gold grades are associated with ginguru banding and carbonate replacement textures. Sulphide 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 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 the discovery of the deposit, with grades typically ranging from 0.3 to 1.5 Au g/t. 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 (totalling 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 veins.

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 to have a reasonable prospect of economic extraction. Therefore, improved and refined geological models of the lithological units were 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-metre 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 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 at a 2.25 g/t Au/t cut-off grade.

Table 25-1: Resource Estimate using 2.25 Au g/t Cut-off

Resource Category	Tonnes (kt)	Au Grade (g/t)	Ag Grade (g/t)	Contained Gold (koz)	Contained Silver (koz)
Measured					
Indicated	6,349	9.31	31.54	1,901	6,439
Measured & Indicated	6,349	9.31	31.54	1,901	6,439
Inferred	605	6.02	19.68	117	383

Notes:
The Mineral Resource statement is subject to the following:
1. Mineral Resources are reported in accordance with NI 43-101.
2. Mineral Resource Estimates have been prepared by Garth Kirkham, P.Geo., a Qualified Person as defined by NI 43-101.
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 Au g/t. Cut-off grades are based on 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 NI 43-101. 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 excluding non-mineable material prior to reporting.
7. Metallurgical recoveries are reported as the average over the LoM 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. The effective date of the Mineral Resource Estimate is December 31, 2024.
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 the North Ramp entrance. Table 25-2 shows the volume and tonnage based on an unconsolidated specific gravity of 2.0 g/cm^{3,} along with gold and silver grades and metal content. These Resources are classified as Measured.

Table 25-2: 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.

25.1.1 Risks

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.

25.2 Mineral Processing and Metallurgical Testing and Processing and Recovery Methods

The metallurgical test work campaigns resulted in an adequate database for estimating overall gold and silver recoveries. The same campaigns adequately supported the selected flowsheet configuration, the latter strictly following standard practices of similar industrial circuits.

The stipulated capacity of the processing plant for the Project is 1000 tpd for a ground product with a P_{80} of 0.053 mm.

The selected method for cyanide neutralization (SO_2/air) resulted in adequate performance according to environmental regulatory specifications.

25.3 Mining Methods, Infrastructure, Capital and Operating Costs

The Project outlines a conceptual mine plan involving the extraction of 8.9 Mt of ROM over a 17-year LoM, with a production rate of 1,500 tpd. The selected underground mining methods are suitable for ensuring stable and consistent mill feed throughout the mine life.

The project features a comprehensive and integrated infrastructure plan that includes new access roads, power supply systems, water management facilities, a process plant, and tailings and waste rock storage. Existing support infrastructure will be leveraged, while new installations will address essential gaps in utility access, safety, and environmental control.

Strong emphasis has been placed on environmental sustainability, with features such as a zero-discharge water strategy, a dry stack tailings facility, stormwater control systems, and reinjection wells. Emergency services, communications, and workforce facilities are also well-planned, aligning with the best practices in mine development.

The total LoM capital cost is estimated at MUS$ 417, comprising:

- pre-production capital of MUS$ 263.6 (23-month period);
- sustaining capital of MUS$ 136.2 (over 17 years); and
- closure capital of MUS$ 17.2, mainly in Year 14.

The cost estimate is a Class 5 estimate (-30% / +50%) with a 12% contingency, excluding working capital, VAT, escalation, and financing. It is based on budgetary quotes, benchmarks from Latin American projects, and internal cost databases.

Operating costs were derived using first principles and local benchmarks. Processing, site services, and general & administrative (G&A) costs were carefully broken down and included labour, power, consumables, and maintenance.

25.3.1 Risks

The most significant potential risks associated with the Project consist of the hot water management that will be encountered in the mine dewatering effort and socio-political resistance to the development of the planned mine in Guatemala. The latter is a common risk to most mining projects and can be mitigated, at least to some degree, with adequate planning and proactive management. The risk associated with water management is not entirely unknown due to the presence of existing dewatering wells and continued dewatering, treatment and discharge of underground water.

It is important to note that the current mine plan is based on a Resource model composed exclusively of Indicated and Inferred Resources, and Inferred Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. As such, there is a significant degree of uncertainty associated with the tonnages and grades used in the sequencing.

The cost of grid power is based on a market survey and not an actual power supply agreement. A higher power cost would result in increased operating costs.

Although the local community is favourable to the development of Era Dorada as an underground mine, there is a potential risk of socio-political opposition to mine development, which could adversely impact the project development schedule.

The ability to achieve the estimated CAPEX and OPEX costs is an important element of Project success. If OPEX increases, then the NSR cut-off would increase, and all else being equal, the size of the mineable Resource would reduce, yielding fewer mineable tonnes.

25.4 Environmental Studies, Permitting, and Social or Community Impacts

The project is fully permitted and has an approved EIA in place for the underground mine; however, permit amendments are required for some of the proposed modifications, including increased processing rate and reinjection of mine water. New EIAs and permits are also needed for the power line. Potential delays in the approval of permit amendments and/or new permits could result in an increased duration of the assumed project development schedule.

Tailings and waste rock are assumed to be Non-Acid-Generating (NAG) based on test work completed to date and the limited exposure time at the surface for waste rock. Additional test work is required prior to detailed engineering to confirm this assumption. If the classification is changed to Potentially-Acid Generating (PAG), the design will need to be updated accordingly.

Although the local community is favourable to the development of Era Dorada as an underground mine, there is a potential risk of socio-political opposition to mine development, which could adversely impact the project development schedule.

25.5 Discounted Cash Flow

The economic analysis demonstrates that the Project has robust economic potential under the developed scenario. At an average gold price of US$ 2,389/ozt, the after-tax NPV (5%) is estimated at MUS$ 485.49, an IRR of 23.8% and a payback of 3.75 years. These results indicate a financially attractive opportunity, supporting continued advancement of the Project.

26 RECOMMENDATIONS

26.1 Exploration, Geology & Resources

Additional drilling will increase Resources and improve the understanding and modelling 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 brownfield exploration program along the trend of the main deposit is recommended to explore additional gold and silver Resources that could potentially extend the project's life.

26.2 Mineral Processing and Metallurgical Testing and Processing and Recovery Methods

Based on the metallurgical test work program and the selected process route, it is recommended:

- To evaluate the best configuration for the comminution circuit – Primary crushing followed by a Single Stage SAG milling (SSSAG), or multi-staged crushing followed by a two-stage ball milling.
- To perform a trade-off study comparing the CIL circuit with CIP and CIP pumpcell, all based on costs, inventory carbon and other parameters.
- To increase the residence time for the cyanide destruction circuit to ensure the minimum residence time for coping with situations of reduced tank operation.

26.3 Mining Methods, Infrastructure, Capital and Operating Costs

It is recommended for the mining methods and mining planning:

- Optimize the Project schedule to prioritize higher-grade zones during the initial years of operation, thereby enhancing early revenue generation.
- Evaluate alternative production scenarios involving variable feed rates throughout the LoM to improve project flexibility and economic performance.
- Conduct a Pre-Feasibility Study (PFS) or Feasibility Study (FS) for Mineral Reserve certification, considering potential variations in mining methods and/or stope geometry to identify opportunities for improved Resource recovery and economic efficiency.
- Implementation of power generation for the cooling of the mine water.
- Mining Study detailing mining dilution for both mining methods.

- Detailed groundwater and dewatering control along LoM.
- Develop a detailed mining operating plan that respects all the mining activities, accounting project restrictions, equipment productivities and limitations.
- Complete detailed engineering for the site infrastructure, ensuring optimization of costs, constructability, and operational integration.
- Submit permitting documentation and ensure all facilities are compliant with local, national, and international environmental standards and regulations.
- Detailed geochemical testing for waste rock and tailings to confirm long-term environmental stability and support facility design.
- Maintain proactive communication with local communities and stakeholders to support social license and minimize construction-related disruptions.
- Implement a robust risk mitigation plan for infrastructure development, including contingency planning for stormwater events, equipment delays, and logistics challenges.
- Undertake a comprehensive technical and economic evaluation of the dewatering system to identify opportunities for cost reduction and efficiency improvements.
- Refine cost estimates to Class 5 level or higher, incorporating detailed engineering, contractor bids, and updated procurement quotes to improve accuracy and reduce contingency requirements.
- Evaluate project economics under different gold price scenarios, inflation rates, and cost escalations to test project resilience and identify key cost drivers.
- Use the current capital structure and cost estimates to support investment discussions, including potential financing, offtake agreements, or joint venture opportunities.
- Incorporate local tax regimes, VAT recoverability, depreciation schedules, and financing structures to derive a complete economic picture for stakeholders.
- Ensure that projected expenditures for G&A, environmental compliance, and social responsibility are transparently communicated and aligned with local expectations.
- Evaluate alternative technologies, energy-saving strategies, and hydrological modelling to minimize the operational impact of dewatering on OPEX.
- Implement a continuous monitoring strategy for gold price fluctuations, with regular updates to the economic model to assess impacts on Net Present Value (NPV), Internal Rate of Return (IRR), and payback period.
- Perform updated sensitivity analyses at key decision points to evaluate the project's resilience under various pricing scenarios.

26.4 Environmental Studies, Permitting, and Social or Community Impacts

Continuous efforts in obtaining the environmental permit amendments for groundwater injection and new EIA/permits for the power line, while advancing key activities that will reduce and de-risk the project execution schedule.

Costs for additional geochemical testing should be included in the budget and should be carried out prior to detailed engineering.

Continue to monitor and update stakeholder engagements through the site Community Relations team. The development of close relationships with the local communities, landowners and government, along with the implementation of the Environmental Management Plan (EMP) and Social Management Plan (SMP), is required.

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APPENDIX A – CERTIFICATE OF QUALIFIED PERSON

I, **Porfirio Cabaleiro Rodriguez**, FAIG (#3708), as an author of the independent technical report titled “NI 43-101 Preliminary Economic Assessment Era Dorada Gold Project, Jutiapa, Guatemala”, prepared on behalf of Aura Mineral Inc., do hereby certify that, dated June 27, 2025, with an effective date of December 31, 2024:

1. I am a Mining Engineer and Director for GE21 Consultoria Mineral Ltda., which is located on 3130 Afonso Pena Ave, 3130, 9th floor, Savassi, Belo Horizonte, MG, Brazil, 30130-910.
2. I am a graduate of the Federal University of Minas Gerais, located in Belo Horizonte, Brazil, and hold a Bachelor of Science Degree in Mining Engineering (1978). I have practised my profession continuously since 1979.
3. I am a Fellow of the Australian Institute of Geoscientists (FAIG #3708).
4. I am a professional Mining Engineer, with more than 40 years’ relevant experience in Mineral Resource and Mineral Reserves estimation, which includes numerous mineral properties in Brazil, including iron ore and manganese properties.
5. I have read the definition of qualified person (QP) set out in “National Instrument 43-101 – Standards of Disclosure for Mineral Projects” (NI 43-101) and certify that, by reason of my education, affiliation with a professional association as defined in NI 43-101, and past relevant work experience, I fulfill the requirements to be a QP for the purposes of NI 43-101.
6. I have supervised the preparation of the hole Technical Report, and I am responsible for Sections 2, 3, 4, 5, 6, 15, 16, 19, 21, 22, 23, and its corresponding parts within Sections 1, 25 and 26.
7. As of the date of the Report, I have not visited the Project site.
8. As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I have authored and am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I have no personal knowledge, as of the date of this certificate, of any material fact or material change which is not reflected in this Technical Report.
10. I am independent of the Issuer, applying all the tests in section 1.5 of NI 43-101.
11. I have read NI 43-101 and Form 43-101F1 – Technical Report, and the Technical Report has been prepared in compliance with such instrument and form.

Brazil, June 27, 2025.

<Signed and sealed in the original>

Porfirio Cabaleiro Rodriguez

I, **Homero Delboni Jr.**, AusIMM (#112813), as an author of the independent technical report titled "NI 43-101 Preliminary Economic Assessment Era Dorada Gold Project, Jutiapa, Guatemala", prepared on behalf of Aura Mineral Inc., do hereby certify that, dated June 27, 2025, with an effective date of December 31, 2024:

1. I'm a Mining Engineer and Minerals Processing, Ph.D in Minerals Processing, with over 40 years of experience in the mining industry.
2. I am a registered professional member of the Australasian Institute of Mining and Metallurgy (AusIMM #112813).
3. I meet all the education, work experience and logging requirements of a "qualified person" as defined in Section 1.1 of NI 43-101.
4. As of the date of the Report, I have not visited the Project site.
5. I am responsible for Sections 13, 17, 20, and co-responsible for Sections 1, 25 and 26 of the Report.
6. As of the effective date of the Report, to the best of my knowledge, information and belief, the sections of which I am the author and for which I am responsible contain all the scientific and technical information that must be disclosed so the Report is not misleading.
7. I am independent of the Issuer and confirm that I have applied all the tests required by Section 1.5 of NI 43-101.
8. I have read NI 43-101 and Form 43-101F1 – Technical Report, and I believe the Report has been prepared in accordance with those documents.

Brazil, June 27, 2025.

<Signed and sealed in original>

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Homero Delboni Jr.

I, **Garth Kirkham**, EGBC (#30043), as an author of the independent technical report titled "NI 43-101 Preliminary Economic Assessment Era Dorada Gold Project, Jutiapa, Guatemala", prepared on behalf of Aura Mineral Inc., do hereby certify that, dated June 27, 2025, with an effective date of December 31, 2024:

1. I am a P.Geo. and Principal of Kirkham Geosystems Ltd.
2. I am a geophysicist and geologist (EGBC #30043) and have over 30 years of experience in experience supplying 3D geoscience modelling, geological and geophysical consulting services to the mining, environmental and oil & gas industries.
3. I have sufficient and relevant experience in mineral deposit geology and Resource estimation to be considered a "qualified person" as defined in Section 1.1 of NI 43-101.
4. I have visited the Project site on May 8, 2017; September 21-22, 2017; April 24-28, 2018; February 16-22, 2020, and January 10-15, 2021.
5. I am responsible for Sections 7, 8, 9, 10, 11, 12, 14, 24, and co-responsible for Sections 1, 25 and 26 of the Report.
6. As of the effective date of the Report, to the best of my knowledge, information and belief, the sections of which I am the author and for which I am responsible contain all the scientific and technical information that must be disclosed so the Report is not misleading.
7. I am independent of the Issuer and confirm that I have applied all the tests required by Section 1.5 of NI 43-101.
8. I have read NI 43-101 and Form 43-101F1 – Technical Report, and I believe the Report has been prepared in accordance with those documents.

United States, June 27, 2025.

<Signed and sealed in original>

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Garth Kirkham