NI 43-101 Technical Report on Pre-Feasibility Study

For the Cascabel Project, Imbabura Province, Ecuador
SolGold Plc

Effective Date of the report: 31 December 2023
Report Date: 8 March 2024

Signed by the Qualified Persons:
Gilles Arseneau, Ph.D., P.Geo
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Brian Prosser, PE
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SRK Consulting (Canada) Inc. ▪ CAPR002807 ▪ March 2024

NI 43-101 Technical Report on Pre-Feasibility Study

For the Cascabel Project, Imbabura Province, Ecuador

Prepared for:
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CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Technical Report on Pre-Feasibility Study for the Cascabel project, Imbabura Province, Ecuador" prepared for SolGold PLC (the "Issuer") dated 8 March 2024, with an effective date of 31 December 2023 (the "Technical Report").

I, Dr. Gilles Arseneau, do hereby certify that:

1. I am an associate consultant with with SRK Consulting (Canada) Inc. ("SRK") with an office at Suite #2600 – 320 Granville Street, Vancouver, British Columba, Canada, V6C 1S9.
2. I graduated from the University of New Brunswick (UNB) with a B.Sc. (Geology) in 1979, the University of Western Ontario (UWO) with an M.Sc. (Geology) in 1984, and the Colorado School of Mines (CSM) with a Ph.D. (Geology) in 1995. Aside from the time spent studying at UNB, UWO and CSM, I have practiced my profession continuously since 1979. My relevant experience includes working on similar porphyry copper-gold projects in North and South America. I have over 25 years experience in mineral resource estimation and 10 years working as an exploration geologist supervising exploration and drilling programs.
3. I am a Professional Geoscientist registered as a member, in good standing, with the Association of Professional Engineers & Geoscientists of British Columbia (License #23474).
4. I did visit the Cascabel project site on October 2 and 3, 2023.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for Sections 4 to 12 and Section 14, as well as relevant parts in the Executive Summary, Reliance on Other Experts, Conclusions and Recommendations, Risks and Opportunities, References and Date and Signature of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have not had prior involvement with the subject property.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 8th day of March 2024 in Vancouver, British Columbia Canada.

"original signed"

Dr. Gilles Arseneau, P. Geo.

SRK Consulting (Canada) Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Technical Report on Pre-Feasibility Study for the Cascabel project, Imbabura Province, Ecuador" prepared for SolGold PLC (the "Issuer") dated 8 March 2024, with an effective date of 31 December 2023 (the "Technical Report").

I, Jarek (Jaroslav) Jakubec, residing in North Vancouver, British Columbia, do hereby certify that:

1. I am a Practice Leader / Corporate Consultant (Mining & Geology) with SRK Consulting (Canada) Inc. ("SRK") with an office at Suite #2600 – 320 Granville Street, Vancouver, British Columba, Canada, V6C 1S9.
2. I am a graduate of the Mining University in Ostrava, Czech Republic with a MSc. in Mining Geology (1984). I have practiced my profession continuously since 1984 and I have 40 years' experience in mining. I have been involved in project management, mine design, due diligence studies, geological and geotechnical modeling around the world. I have direct operational experience from caving mines in Canada and have been involved in caving or sublevel caving mines studies in Canada, the United States, Chile, South Africa, Australia, Indonesia, Papua New Guinea, China, Kazakhstan and Mongolia. As a result of my experience and qualifications, I am a Qualified Person as defined in NI 43-101.
3. I am a registered Chartered Engineer (No. 509147) and Fellow of the Institute of Materials, Minerals and Mining in the United Kingdom.
4. I personally inspected the subject concession on October 2 and 3, 2023 and the historic direct precedent concessions from August 13 to 16, 2022.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
6. As a Qualified Person, I am independent of the issuer as defined in Section 1.5 of NI 43-101.
7. I accept professional responsibility for portions of Section 2, Section 15, Sections 16.1, 16.2, 16.3, 16.4, 16.6, 16.7, 16.8, Section 23, as well as relevant parts in the Executive Summary, Reliance on Other Experts, Conclusions and Recommendations, Risks and Opportunities, References and Date and Signature of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have had prior involvement with the subject property in the form of the Issuer's internal technical studies undertaken as a Corporate Consultant with SRK Consulting (Canada) Inc. in August 2022.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 8th day of March 2024 in Vancouver, British Columbia Canada.

"original signed"

Jarek (Jaroslav) Jakubec, C.Eng, FIMMM
Practice Leader / Corporate Consultant (Mining & Geology)
SRK Consulting (Canada) Inc.

CERTIFICATE OF QUALIFIED PERSON

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I, Guy Lauzier, do hereby certify that:

1. I am a Project Engineer with Allnorth Consultants Limited with an office at 1200-1100 Mellville Street, Vancouver, British Columbia V6E 4A6.
2. I am a graduate of McGill University (1979) where I obtained a B. Eng Mining. Aside from the time spent studying at McGill University, I have practiced my profession continuously since 1979. My relevant experience includes mine design, mine construction and operations.
3. I am a Professional Engineer registered with the EGBC (14472), OIQ (34979) and PEO (25782509).
4. I did not visit the Cascabel project site.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for Sections 16.5.1, 16.5.2, 16.5.4, and 16.5.5 of the Technical Report, as well as relevant parts in the Executive Summary, Reliance on Other Experts, Capital and Operating Costs, Conclusions and Recommendations, Risks and Opportunities, References and Date and Signature of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have not had prior involvement with the subject property.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 8th day of March 2024 in Wellington, Ontario, Canada.

"original signed"

Guy Lauzier, P.Eng.
Allnorth Consultants Limited

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Technical Report on Pre-Feasibility Study for the Cascabel project, Imbabura Province, Ecuador" prepared for SolGold PLC (the "Issuer") dated 8 March 2024, with an effective date of 31 December 2023 (the "Technical Report").

I, Brian Prosser, do hereby certify that:

1. I am a Principal Consultant with SRK Consulting (US) with an office at 1625 Shaw Ave., Suite 103, Clovis, California 93611.
2. I am a graduate of Virginia Poly Technic Institute and State University where I obtained a Bachelor of Science degree in Mining Engineering. Aside from the time spent studying at Virginia Poly Technic Institute and State University, I have practiced my profession continuously since 1994. My relevant experience includes the design and evaluation of ventilation designs for subsurface mining, tunneling, and repositories.
3. I am a Professional Engineer licensed with the State of Nevada Board of Professional Engineers and Land Surveyors License Number 15465.
4. I have not visited the Cascabel project site.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for Section 16.5.3, as well as relevant parts in the Executive Summary, Reliance on Other Experts, Capital and Operating Costs, Conclusions and Recommendations, Risks and Opportunities, References and Date and Signature of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have not had prior involvement with the subject property.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 8th day of March 2024 in Clovis, California, United States.

"original signed"

Brian Prosser, PE

Principal Consultant / Practice Leader (Ventilation Group)

SRK Consulting (US)

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Technical Report on Pre-Feasibility Study for the Cascabel project, Imbabura Province, Ecuador" prepared for SolGold PLC (the "Issuer") dated 8 March 2024, with an effective date of 31 December 2023 (the "Technical Report").

I, Ben Adaszynski, do hereby certify that:

1. I am a Manager Project Development with Sedgman with an office at #860 – 625 Howe Street Vancouver BC, Canada V6C 2T6.
2. I am a graduate of the University of British Columbia 2009 where I obtained a Bachelor of Applied Sciences – Chemical Engineering. Aside from the time spent studying at University of British Columbia, I have practiced my profession continuously since 2009. My relevant experience includes direct involvement in all levels of engineering studies from preliminary economic assessment (PEA) to feasibility studies including copper-gold flotation projects. I have been directly involved with metallurgical test work and flowsheet development from preliminary testing through to detailed design and process optimization in North and South America.
3. I am a Professional Engineer registered with the Engineers and Geoscientists British Columbia member number 40359.
4. I did not visit the Cascabel project site.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for Sections 13 and 17, as well as relevant parts in the Executive Summary, Reliance on Other Experts, Capital and Operating Costs, Conclusions and Recommendations, Risks and Opportunities, References and Date and Signature of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have not had prior involvement with the subject property.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 8th day of March 2024 in Vancouver, British Columbia, Canada.

"original signed"

Ben Adaszynski, P. Eng
Sedgman

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
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jdsmining.ca

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Technical Report on Pre-Feasibility Study for the Cascabel project, Imbabura Province, Ecuador" prepared for SolGold PLC (the "Issuer") dated 8 March 2024, with an effective date of 31 December 2023 (the "Technical Report").

I, Richard Boehnke, do hereby certify that:

1. I am a senior engineer with JDS Energy and Mining Inc. (JDS) with an office at 900-999 West hastings Street Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2.
2. I am a graduate of the University of Manitoba where I obtained a B.Sc. in Industrial Engineering., I have practiced my profession continuously since 1989. My relevant experience includes technical, operations and management positions in industrial facilities in Canada for over 15 years, and I have been working in mining related business for over 17 years. I have been an independent consultant for twelve years and have performed project management, field engineering, engineering management, cost estimation, construction management, technical due diligence reviews and technical report writing for mining projects worldwide.
3. I am a Registered Professional Engineer in British Colombia (#30486).
4. I did visit the Cascabel site October 2 and 3, 2023.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for Sections 18.1, 18.2, 18.3, 18.4, 18.7, 18.8, 18.9, 18.10, and 18.11, as well as relevant parts in the Executive Summary, Reliance on Other Experts, Capital and Operating Costs, Conclusions and Recommendations, Risks and Opportunities, References and Date and Signature of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have not had prior involvement with the subject property.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 8th day of March 2024 in Vancouver, BC, Canada.

(Original signed and sealed) "Richard Boehnke, P.Eng."

Richard Boehnke, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Technical Report on Pre-Feasibility Study for the Cascabel project, Imbabura Province, Ecuador" prepared for SolGold PLC (the "Issuer") dated 8 March 2024, with an effective date of 31 December 2023 (the "Technical Report").

I, **Timothy David Rowles**, do hereby certify that:

1. I am the Regional Manager and a Principal Engineer with Knight Piésold Consulting with an office at Level 1, 36 Cordelia Street, Brisbane, QLD 4101, Australia.
2. I graduated from the Royal School of Mines, Imperial College, London with a Bachelor of Science in Environmental Geology in 1996 and from the University of Manchester with a Master Degree in Earth and Environmental Science in 1998. I was awarded a Professional Certificate in Tailings Management by AusIMM in 2021. I have practiced my profession continuously within the mining industry since 1999. My relevant experience includes environmental assessment, engineering design, construction, operation and closure of tailings management systems, water dams and waste dumps, surface water / sediment management systems. This experience includes mine sites in Australasia, Africa, Asia, Europe and South America.
3. I am a current Fellow and Chartered Professional (CP) of the Australian Institute of Mining and Metallurgy (No 227249) in the field of Environmental Engineering. I am a Registered Professional Engineer of Queensland (No 10166). I am a current member of the Australian Institute of Geoscientists (No. 8161) and the Australian National Committee on Large Dams.
4. I have personally visited the Cascabel project site between the February 6 and 8, 2020.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for Sections 18.5 and 18.6, Section 20, as well as relevant parts in the Executive Summary, Reliance on Other Experts, Capital and Operating Costs, Conclusions and Recommendations, Risks and Opportunities, References and Date and Signature of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have had prior involvement with the subject property having been the qualified person for the NI 43-101 Technical Report on Pre-Feasibility Study with an Effective date 31 March 2022.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 8th day of March 2024 in Brisbane, Queensland, Australia.

AusIMM
THE MINERALS INSTITUTE
CHARTERED PROFESSIONAL
ENVIRONMENT
Timothy Rowles

Timothy David Rowles, B.Sc., M.Sc., FAusIMM (CP), RPEQ
Knight Piésold Consulting

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: “NI 43-101 Technical Report on Pre-Feasibility Study for the Cascabel project, Imbabura Province, Ecuador” prepared for SolGold PLC (the “Issuer”) dated 8 March 2024, with an effective date of 31 December 2023 (the “Technical Report”).

I, Carl Kottmeier, P. Eng., MBA, do hereby certify that:

1. I am a Principal Consultant with SRK Consulting (Canada) Inc. (“SRK”) with an office at Suite #2600 – 320 Granville Street, Vancouver, British Columba, Canada, V6C 1S9.
2. I am a graduate with a B.A. Sc. (Applied Science) degree in Engineering from the University of British Columbia in 1989 and an MBA degree from the University of British Columbia in 2003. I have practiced my profession continuously since 1989. I have worked on mining projects in North and South America, Africa and in Europe, and I have extensive experience with base metal and precious metal mining projects such as Cascabel project. My relevant experience includes responsibilities in operations, maintenance, mine design, mine engineering, management, contracting and construction activities.
3. I am a Professional Engineer registered with the Association of Professional Engineers and Geoscientists of British Columbia, license # 18702.
4. I have not visited the Cascabel project site.
5. I have read the definition of “qualified person” set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
6. I am independent of the issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for Section 19, Section 22, Section 24, as well as relevant parts in the Executive Summary, Reliance on Other Experts, Capital and Operating Costs, Conclusions and Recommendations, Risks and Opportunities, References and Date and Signature of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have not had prior involvement with the subject property.
9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portions of the Technical Report not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 8th day of March 2024 in Vancouver, British Columbia Canada.

“original signed”

Carl Kottmeier, P. Eng., MBA.

SRK Consulting (Canada) Inc.

Contents

Tables

Figures

1 Summary

1.1 Introduction

This report has been prepared by SRK Consulting (Canada) Inc. ("SRK") on behalf of SolGold plc ("SolGold", or the "Company"). The purpose of this report is to provide a technical report that documents all supporting work, methods used and results relevant to a pre-feasibility study (PFS) that fulfills the reporting requirements in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101").

1.2 Property Description and Location

The Cascabel property is located within the Imbabura province of northern Ecuador, approximately 100 km north of the capital city of Quito and approximately 50 km north-northwest of the provincial capital, Ibarra.

The northern border of the Project lies approximately 20 km south of the Colombia-Ecuador border, and 75 km southeast of San Lorenzo on Ecuador's Pacific coast.

The Cascabel concession was initially granted over 5,000 hectares, valid for 25 years, and renewable for a further 25 years. On 16 December 2016, as required by the Mining Law due to a change of period to advanced exploration, the Cascabel Concession area underwent a compulsory reduction to cover 4,979 hectares (49.79 km^2) as it does presently. The Cascabel Concession is currently registered as an Exploitation License for metallic minerals under cadastral code 402288. On 10 July 2023, the Ministry of Energy and Mines issued the renewal of the Concession title for 25 years. With this new term, the Concession term lasts until 2048 and it is subject to renewal.

The license area is recorded under both political and geographical datums, being PSAD56-17S and WGS84-UTM-17N.

The Concession is duly registered and in good standing, and there are no liens registered. There is neither any indication of any potential issue that could result in the termination, revocation, or suspension of the Concession, nor any evidence of grounds for the nullification of the ownership of the Concession held by Exploraciones Novomining S.A. (ENSA). SolGold holds a 100% legal and beneficial interest in ENSA.

ENSA holds the necessary licenses, permits and registrations, including all necessary environmental licenses and water permits, to conduct advanced exploration, including early works activities and underground exploration. Such licenses, registrations and permits have been duly granted to and are validly held by ENSA. There are no outstanding agreements or operations that may limit the right of ENSA to conduct mining activities. There are no agreements for operations with artisanal miners.

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

Access to Cascabel is via sealed highways through the closest major centre of Ibarra, located approximately 80 km by road south of the property. The Cascabel concession area contains three small settlements: Santa Cecilia Village, Rocafuerte Office Complex, and the Alpala Base Camp. Four further settlements exist proximal to the Project area at Rocafuerte, San Pedro, Urbina and Cachaco.

The topography of the Project area is moderate to steep, with elevations rising from 750 m to 2,140 m above sea level. The rugged terrain is deeply incised by four large drainage complexes. Vegetation is tropical forest with a well-developed soil horizon that is up to 12 m thick in places.

The climate of the Project area is characterised by humid weather with bi-modal rain seasons that peak in December and March. Total average rainfall for the region is approximately 1,500 mm per annum. Regionally, temperatures remain relatively consistent throughout the year, with average annual temperatures of approximately 17°C, with maxima in excess of 30°C and minima typically around 10°C.

1.4 History

From 1980 to the present, ten different entities have conducted field activities and/or studies over the Cascabel area. Historical exploration of the Project area undertaken from 1980 to May 2012, highlighted widespread geochemical anomalism in stream pan-concentrates, stream sediments and rock chips over a 9 km^2 area in the northern half of the license area.

The Cascabel tenement was granted to Santa Barbara Copper and Gold S.A (SBCG) on 26 April 2010. Santa Barbara Copper and Gold S.A. was renamed to Exploraciones Novomining S.A. on 27 July 2011. On 29 March 2011, Cornerstone Capital Resources Inc. (“Cornerstone”) signed a definitive agreement to acquire Santa Barbara Copper and Gold S.A.

On 24 July 2012, SolGold entered into an Earn-in Agreement with Cornerstone, ENSA and Cornerstone Ecuador S.A. (CESA) (see Section 4.3), which was further supplemented by a binding Term Sheet executed on 24 February 2014.

In May 2012, SolGold assumed the management of the Project and commenced the first systematic exploration program at Cascabel. The surface expression of the Alpala Deposit was discovered the same month during reconnaissance mapping, which located an 80 m wide zone of Cu and Au-bearing, dominantly sheeted, and stockwork porphyry-style quartz veining in Alpala Creek. After follow-up mapping, geochemical and geophysics programs were conducted and other porphyry related stockwork veins were subsequently discovered in the Moran, Tandayama and America creeks.

Rock channel sampling and structural measurements of quartz veins over a 430 m by 200 m area at Alpala provided the geological context for a diamond drilling program.

In August 2013, the Environmental License for the Cascabel concession was approved and on 1 September 2013, drilling of the first hole commenced using a modified man-portable drill rig operated by Hubbard Perforaciones (HP).

The first four holes of the diamond drilling program confirmed surface mineralisation to depths of approximately 200 m. However, the course of the program was modified by the extent and high-grades of chalcopyrite-bearing quartz vein stockworks encountered in Hole 5, which was drilled less than 18 months after the location of surface mineralisation. This fifth drill hole marks the discovery of the high-grade world-class Alpala Deposit, with an overall interval of 1,306 m at 0.62% Cu and 0.54 g/t Au, including 552 m at 1.03% Cu and 1.05 g/t Au from a 778 m downhole depth.

1.5 Geological Setting and Mineralisation

The Cascabel project lies within the Western Tectonic Realm (WTR) of Ecuador and Colombia, which is comprised of three composite terrane assemblages: the Pacific assemblage (PAT), Choco arc (CHO) and Caribbean terranes (CAT). Within the Pacific composite terrane assemblage there are three terranes, from east to west: the Romeral (RO), Dagua-Pinon (DAP) and Gorgona (GOR) terranes.

In the vicinity of the Cascabel project, the principal terrane boundary is the Cauca-Pujili fault system, which forms the suture between the RO and the DAP terranes. The Eocene Alpala Deposit lies in a zone of overlap between the Eocene and Miocene Andean porphyry belts that extend from Colombia through Ecuador and Peru into Chile and Argentina. The basement rocks consist of tholeiitic basalts of the DAP Terrane, an oceanic plateau that is believed to have accreted to South America in the Late Cretaceous.

The major rock types of the Cascabel tenement consist of Cretaceous siltstones and minor sandstones which are unconformably overlain by a Tertiary sequence of andesitic lavas and volcano-sedimentary rocks. A series of hornblende-bearing diorites, quartz diorites and tonalities intruded the volcano-sedimentary sequence as plutons, stocks, and dykes.

Major host rock types of the Tandayama-America (TAM) deposit consist of a sequence of Tertiary volcano- sedimentary and andesitic lavas of the same age as those at the Alpala deposit. This sequence has also been intruded by a series of Middle to Late-Eocene (Bartonian) quartz diorites and diorites that form plutons, stocks, and dykes.

Major host rock types of the deposits consist of gabbroic and basaltic basement rocks, overlain by Cretaceous siltstones and minor sandstones that are unconformably overlain by a sequence of Tertiary volcano-sedimentary and andesitic lavas. The equigranular to sub-porphyritic, hornblende-bearing intrusions at Alpala are narrow and taper upwards.

Mineralisation occurs as a prolate body approximately 2,400 m northwest by 1,200 m northeast and 2,800 m in vertical extent, defined at a Cu equivalent (CuEq) cut-off criteria of greater than 0.15% CuEq and/or greater than 0.55% B-type quartz veins. The main phase of mineralisation was emplaced with the syn-mineralisation QD10 intrusion, resulting in a concentric zone of high-grade mineralisation marked by greater than 10% B-type veins (quartz-magnetite-chalcopyrite). At least two stages of B-type veins have been recognised, B1 and B2, with magnetite more abundant in early B1 veins and chalcopyrite more common in the later B2 veins. B-type veins contain the majority of the Cu and Au in the deposit.

The mineralisation style and trend at Tandayama-America is essentially the same style as that seen in Alpala.

1.6 Deposit Types

The mineralisation observed at the Alpala and the Tandayama-America deposits is considered a classic porphyry Cu-Au system. These mineralised systems are hosted within a linear belt (Andean Porphyry Belt) that extends from southern Chile right through to Ecuador and Colombia to Panama. The Andean Porphyry Belt hosts the largest concentrations of copper in the world, including numerous deposits with active mining operations.

1.7 Exploration

The most recent exploration on the Property has been drilling, which is discussed in the following section. Exploration at Alpala has also consisted of a helicopter-borne magnetics and radiometric survey that was conducted over the entire Cascabel tenement in November 2012, a 3D airborne laser scanning Light Detection and Ranging (Lidar) topographic survey that was completed in November 2018, and photogrammetry topographic surveys were conducted in February 2020.

The remote sensing programs were followed with multi-element grid-based soil geochemistry studies comprising of 3,287 soil grid samples and 550 soil auger samples across 35 km^2. The soil survey indicated several zones of coincident Au, Cu, Mo, and Cu-Zn ratio anomalies across several interpreted porphyry centres, and identified widespread geochemical anomalies, including at least four major porphyry centres characterised by coincident Au, Cu and Mo highs, which consist of the Alpala cluster, Moran, Aguinaga and Tandayama-America.

1.8 Drilling

Since drilling commenced on 1 September 2013, a total of 310,335 m of diamond drilling has been completed at the Cascabel project to date. Drilling includes: 265,224 m at the Alpala Deposit, 36,111 m at Tandayama-America deposit, 8,970 m at the Aguinaga deposit, and 6,774 m of drilling completed on infrastructure, sterilisation and water monitoring.

Existing drilling at the Cascabel project has focussed on delineating copper and gold mineralisation at a cluster of Eocene aged porphyry deposits and prospects. Three significant deposits have been identified thus far at Cascabel, namely the large Alpala porphyry copper-gold-silver deposit, the Tandayama-America porphyry copper-gold deposit, and the Aguinaga porphyry copper-gold deposit.

The initial spread and design of the drill holes at Alpala were impacted by site access and topography. As such, man-portable rigs had to be used which resulted in some low intersection angles with the mineralised deposit, which is common when drilling a steeply dipping porphyry deposit. With the introduction of further drill rigs and the use of the Devico orientation device, SolGold was able to better target and drill the mineralisation at Alpala and Tandayama-America.

SolGold achieved a high degree of control over the complex, multi-contractor drilling programs through the use of their own independent foreman.

From Dr. Arseneau’s review during the technical site visit, the drilling at Alpala has been conducted in a professional manner using industry accepted practices and has produced core of sufficient quality and recovery to be used in Mineral Resource Estimations.

1.9 Sample Preparation, Analyses and Security

Detailed analysis of the results from the quality assurance/quality control (QA/QC) samples have been undertaken by SolGold and independently reviewed by the QPs who have worked on the Alpala project.

SolGold has implemented a QA/QC system that includes standard reference material, blanks, field duplicates and independent third-party umpire laboratory assays.

Based on the review of the QA/QC procedures and results, the QP is of the opinion that the sample preparation, security, and analytical procedures for samples collected at Alpala and the Tandayama-America deposits are adequate for the inclusion in a mineral resource and mineral reserve estimation and is in keeping with best industry practices.

1.10 Data Verification

Dr. Arseneau (QP) undertook a site visit to the Cascabel property on 2-3 October 2023 in relation to the Alpala MRE#4 and TAM#3 mineral resource estimations. The QP concluded that the quality of the geological and assay data collected in relation to the Alpala and TAM deposits, as well as their validation and storage, were aligned with industry accepted practices. The independent samples collected by the QP from the Alpala drill core, and the samples collected from the TAM deposit in 2022, agree very well with the original assay data provided by SolGold. The verification of the assay database against the original assay data sheets provided from the assay laboratory identified no material errors. As such, the QP considered the geological and assay data to be robust and suitable for inclusion in mineral resource and mineral reserve estimations.

1.11 Mineral Processing and Metallurgical Testing

Metallurgical testwork for the Alpala underground deposit commenced in 2014. Testwork was conducted by various independent metallurgical facilities from 2019 to 2023. Tests included mineralogy, material flow, comminution, open and locked cycle flotation, Davis tube magnetic separation, cyanide leaching, and solid-liquid separation.

The proposed process flowsheet has been refined and modified over time, with the current preferred option representing a conventional copper-gold flotation flowsheet with no additional gold cyanidation circuit. The flotation flowsheet consists of a single rougher stage and a multi-stage cleaning circuit to produce a copper-gold-silver concentrate.

Samples selected for the testwork programs considered lithology and alteration descriptions, grade of copper and gold, spatial location, different timing in the mine plan and various geological measurements.

1.12 Mineral Resource Estimates

The current Alpala Mineral Resource update (MRE#4) was estimated from drill holes and rock-saw channel samples that lie within the Alpala block model limits and included 111,435 assays representing 265,225 m of diamond drilling in 185 drill holes, and 696 assays representing 1,441 m of 118 rock-saw channel samples cut from surface rock exposures.

The Tandayama-America (TAM) Mineral Resource (MRE#3) was estimated from drill holes and rock-saw channel samples that lie within the TAM Block Model limits and included 17,574 assays, representing 36,111 m of diamond drilling in 51 drill holes, and 220 assays representing 458 m of 72 rock-saw channel samples cut from surface rock exposures.

Geological and grade domains were prepared by SolGold. Dr. Arseneau reviewed the wireframes with SolGold, made suggestions for improvements where necessary and validated and accepted the geological and grade domains used in the Mineral Resource Estimates for the Cascabel project.

Mineralisation domain wireframes have been interpreted to honour lithological contacts and intrusion geometries and guided locally by structural measurements of B-vein orientations.

The QP undertook Exploratory Data Analysis (EDA) by first investigating the statistics of each individual lithology and grade wireframe, before iteratively generating various groups of lithologies within each grade wireframe and comparing their statistics. L ithology and grade wireframes were combined to form final estimation domains used to prepare the mineral resource estimate.

Mineral resources were classified based on the average distance of multiple drill holes informing the block model. Blocks were coded as Measured mineral resource if estimated by at least three drill holes within an average distance of 80 m from the estimated block. Blocks estimated with at least three drill holes within a 160 m distance were classified as Indicated mineral resource and blocks estimated with at least two drill holes within a 240 m radius were classified as Inferred mineral resource. After estimation, the results were reviewed in plans to assure uniformity within each of the classes, and a smoothing algorithm was run to assure that no isolated mis-classified blocks existed within each of the class domains.

Open pit and underground optimisations were run for mineralised and classified material that was potentially mineable by open pit or underground methods, respectively. The underground optimised shapes (UOS) were then used to report the potion of the Mineral Resource that could potentially be mined from underground block caving methods.

The MRE (MRE#4) for the Alpala deposit was reported in accordance with the Canadian National Instrument 43-101 - Standards of Disclosure for Mineral Projects. The estimation process followed the Canadian Institute of Mining, Metallurgy and Petroleum “Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines” (CIM, 2019). Dr Gilles Arseneau, P.Geo. is the Qualified Person (QP) responsible for the Alpala MRE#4.

Dr Arseneau has estimated that the Alpala porphyry copper-gold-silver deposit contained 3,013 million tonnes grading 0.35% Cu and 0.28 g/t Au in the Measured plus Indicated categories, at a cut-off grade of 0.21% Cu equivalent (CuEq) (Table 1-1). The deposit contains an additional 607 million tonnes grading 0.26% Cu and 0.19 g/t Au in the Inferred category.

Table 1-1: Alpala Mineral Resource statement (effective date 11 November 2023)

Cut-off Grade (CuEq%)	Resource Category	Tonnage (Mt)	Grade				Contained Metal			
			CuEq (%)	Cu (%)	Au (g/t)	Ag (g/t)	CuEq (Mt)	Cu (Mt)	Au (Moz)	Ag (Moz)
0.21	Measured	1,576	0.64	0.43	0.35	1.16	10.0	6.7	17.5	58.6
	Indicated	1,437	0.39	0.28	0.20	0.71	5.6	4.0	9.3	32.7
	Measured + Indicated	**3,013**	**0.52**	**0.35**	**0.28**	**0.94**	**15.6**	**10.7**	**26.8**	**91.3**
	Inferred	607	0.36	0.26	0.19	0.56	2.2	1.5	3.7	11.0

Notes:

1. Dr. Arseneau, P.Geo. Associate Consultant with SRK Consulting (Canada) is responsible for this Mineral Resource statement and is an "Independent Qualified Person" as defined in NI 43-101.
2. Reasonable prospects of eventual economic extraction were assessed by enclosing the mineralised material in the block model estimate in a 3D wireframe shape that was constructed with adherence to a minimum mining unit with geometry appropriate for a block cave.
3. Cut-off grade for the shape was defined as the cut-off grade under a breakeven, eventual economic extraction criterion. The cut-off grade of 0.21% CuEq was calculated using (copper grade (%)) + (gold grade (g/t) x 0.683).
4. All material within this shape was reported in the Mineral Resource statement as block caving is a non-selective method and all material extracted is treated as mill feed.
5. The material inside the shape without a Mineral Resource category was reported as planned dilution.
6. The resulting shape contained planned internal and edge dilution that the QP considers appropriate.
7. Cut-off inputs included:
   a) Metal prices of Cu at $3.60/lb and Au at $1,700/oz
   b) Recoveries of 93% for copper and 83% for gold
   c) Costs including mining, processing and general and administration (G&A) and off-site realisation (TCRC) including royalties.
8. The QP considers that the Mineral Resource has reasonable prospects for eventual economic extraction by an underground mass mining method such as block caving.
9. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
10. Mineral Resources are reported inclusive of those Mineral Resources that were converted to Mineral Reserves.
11. The statement uses the terminology, definitions and guidelines given in the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) as required by NI 43-101.
12. Figures may not sum due to rounding.

The Tandayama-America deposit lies approximately 3 km north of the Alpala deposit. Dr Gilles Arseneau is the QP responsible for the Tandayama-America MRE#3. The Tandayama-America deposits mineral resources are presented in Table 1-2.

Table 1-2: Tandayama-America Mineral Resource Statement (effective date 11 November 2023)

Potential Mining Method	Cut-off Grade (CuEq %)	Resource Category	Tonnage (Mt)	Grade			Contained Metal		
				Cu (%)	Au (g/t)	CuEq (%)	Cu (Mt)	Au (Moz)	CuEq (Mt)
Open Pit	0.16	Indicated	492	0.22	0.20	0.35	1.1	3.1	**1.7**
		Inferred	45	0.18	0.18	0.31	0.1	0.3	**0.1**
Underground	0.19	Indicated	230	0.26	0.18	0.39	0.6	1.3	**0.9**
		Inferred	201	0.21	0.21	0.36	0.4	1.4	**0.7**
Total Indicated			722	0.23	0.19	0.36	1.7	4.5	**2.6**
Total Inferred			247	0.21	0.21	0.35	0.5	1.6	**0.9**

Notes:

1. Dr Gilles Arseneau, P. Geo., Associate Consultant with SRK Consulting (Canada), is responsible for this Mineral Resource statement and is an "Independent Qualified Person" as defined in NI 43-101.
2. Reasonable prospects of eventual economic extraction were assessed by:
   a) First presenting the mineralised material in the block model estimate to a conventional Lersch-Grossman open pit optimisation routine based on a cut-off grade of 0.19% CuEq, and the cost and revenue assumptions listed below. Mineralised material inside the revenue factor one pit and above the cut-off grade were then reported in the "Open pit" section of the Mineral Resource statement.
   b) Subsequently, the remaining material was enclosed in a 3D wireframe shape that was constructed with adherence to a minimum mining unit with geometry appropriate for a block cave.
3. Cut-off grade for the potentially open pit mineable material was 0.16% CuEq. The cut-off grade for the underground shape was defined as the cut-off grade under a breakeven, eventual economic extraction criterion. The cut-off grade of 0.19% CuEq was calculated using (copper grade (%)) + (gold grade (g/t) x 0.683).
4. All material within the underground shape was reported in the "Underground" section of the Mineral Resource statement as block caving is a non-selective method and all material extracted is treated as mill feed.
5. The resulting shape contained planned internal and edge dilution that the QP considers appropriate.
6. Cut-off/Cut-off inputs included:
   a) Metal prices of Cu at $3.60/lb and Au at $1,700/oz
   b) Recoveries of 93% for copper and 83% for gold
   c) Costs including mining, processing and general and administration (G&A) and off-site realisation (TCRC) including royalties.
7. The QP considers that the Mineral Resource has reasonable prospects for eventual economic extraction by open pit or an underground mass mining method such as block caving as presented in the Mineral Resource statement.
8. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
9. Mineral Resources are reported inclusive of those Mineral Resources that were converted to Mineral Reserves.
10. Numbers may not add up due to rounding.

1.13 Metallurgy and Processing

Crushed ore from the underground operation will be conveyed to the surface and stockpiled. Stockpiled material will be reclaimed and ground to a grind size of P_{80} 200 µm through an SAG Mill – Ball Mill – Crusher (SABC) configuration and concentrated by a flotation process. SolGold will target a copper concentrate of 22% Cu by weight. The concentrate will be dewatered in preparation for transport to the port.

By maintaining a copper concentrate target of 22% Cu by weight, fluctuations in ore head grade will mean that recovery rates and concentrate volume will vary by year. In the first 10 years, when head grades are highest, recoveries are expected to average 90.8% for copper and 76.9% for gold. In subsequent years, average recoveries are expected to be 86.8% for copper and 67.0% for gold.

1.14 Mineral Reserve Estimate

The Mineral Reserve for the Cascabel project Alpala underground resource was converted by applying modifying factors to the Mineral Resource Estimate. Only Measured and Indicated categories have been converted to Mineral Reserves, with Inferred categories considered as waste and grades set to zero.

The Mineral Reserve Estimate for the Alpala underground porphyry copper-gold-silver deposit, Cascabel Property, with an effective date of 31 December 2023 (Table 1-3) has been prepared under the supervision of SRK Consulting (Canada) Inc. Corporate Consultant, Jarek Jakubec, C.Eng., FIMMM, who is the Qualified Person responsible for the Mineral Reserve Estimate.

The Mineral Reserve estimation process followed the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" (CIM, 2019). The Mineral Reserve Estimate is stated in accordance with the CIM Definition Standards (CIM, 2014) and Canadian National instrument 43-101 (NI 43-101).

Table 1-3: Alpala Mineral Reserve Estimate (effective date 31 December 2023)

Category	Tonnes (Mt)	Cu Grade (%)	Au Grade (g/t)	Ag Grade (g/t)	Total Cu (Mlbs)	Total Au (koz)	Total Ag (koz)
Proven	457.5	0.64	0.60	1.7	6,475	8,854.5	24,942
Probable	82.2	0.36	0.22	1.2	653	578.7	3,093
Total	**539.7**	**0.60**	**0.54**	**1.6**	**7,128**	**9,433.2**	28,034

Notes:
1. CIM Definition Standards were followed for Mineral Reserves.
2. Mineral Reserves for the Cascabel project have an effective date of December 31, 2023.
3. The Qualified Person responsible for the estimate of Mineral Reserves is Jarek Jakubec, C.Eng., FIMMM.
4. Mineral Reserves reported using long-term metal prices of $1,700/oz Au, $3.60/lb Cu, $19.90/oz Ag.
5. Mineral Reserves are constrained within a block cave design, using the following input parameters: height of draw of 400 m; mixing horizon of 350 m; 15% dilution (at 350 m column height); metallurgical recoveries that range from 68-81% for copper and 85-92% for gold; a footprint development cost of $1,750/m^2; cut-off value of $15.00/t.
6. Units are metric tonnes, metric grams, troy ounces, and imperial pounds. Gold ounces and copper pounds are estimates of in-situ material and do not account for processing losses.
7. Totals may not match due to rounding.

1.15 Mining Method

The mining method selected for maximum extraction of the Alpala deposit is the block caving method. Access will be via twin declines from surface to the extraction level of the block caves. Decline development is estimated to take approximately four years to reach the block cave extraction level at 270 mRL.

Four separate block cave footprints will be developed. Block Cave 1 (footprint area of 67,453 m^2) is estimated to start producing ore in Year 4 of project development, followed by Block Cave 2 (73,980 m^2) in Year 8, Block Cave 3 (93.038 m^2) in Year 15 and Block Cave 4 (103,482 m^2) in Year 18. Primary crushing will occur underground via single-stage jaw crushers; one for each block cave footprint except for Block Cave 4, which will have two.

Total mine life is expected to be 28 years, during which time a total of 539.7 Mt of ore will be extracted, with an average copper grade of 0.60% Cu and an average gold grade of 0.54 g/t Au.

1.16 Recovery Method

The process design basis has been derived from the metallurgical test work results that were provided by SolGold. The Cascabel project will be developed in two phases, with each phase utilising a dedicated processing line suitable for a production rate of 12 Mtpa, recovering copper and gold using a conventional flotation process.

Primary crushing will take place underground and crushed ore will be conveyed to the surface, where a radial stacker will create a separate crushed ore stockpile to feed each processing line. Reclaimed ore from the stockpiles is ground in a SAG Mill – Ball Mill – Crusher (SABC) configuration with 80th percentile data used to size the mills by Sedgman and verified by vendors.

The flotation circuit was sized based on metallurgical testwork, flotation kinetic curves and standard scale-up parameters. Following rougher flotation, rougher concentrate is reground prior to multi-stage cleaner flotation. Rougher and cleaner flotation tailings are thickened individually for separate deposition into the tailings storage facility. The final concentrate is thickened and filtered. The design of the flotation circuit production schedule has been undertaken utilising the recovery and grade equations developed from the testwork completed and target a 22% Cu concentrate grade.

1.17 Project Infrastructure

The Project infrastructure is designed to support the operation of a 12 Mtpa underground mine operation and processing plant, expanding to 24 Mtpa in Phase 2, operating on a 24-hour per day, 7-day per week basis. The Project infrastructure is designed with local conditions and topography in consideration. The infrastructure design is in line with the planned process and mining rates and is appropriate for a greenfield development in a remote area.

To reduce the impact of the Project footprint, some support facilities will be located off-site. The facilities location at the Port area will continue to be evaluated. Borrow pits will be required to provide the materials needed for construction through to mine closure. Water for process demands will be sourced from the Mira River throughout the life of operations.

The required site infrastructure includes, but is not limited to, the following major components:

- Road works
- Waste management (sewage, solid waste, liquid waste)
- Surface water management
- Administration offices and surface facilities, including warehousing and fuel/reagent storage
- Camp – construction and permanent
- Site substation and power distribution, including emergency power generation
- Aggregate borrow pit and stockpiles (topsoil and unsuitable overburden)

The total tailings requiring storage considered in the design was 529 Mt generated from 539 Mt of ore feed. The tailings will comprise 460 Mt of rougher tailings and 69 Mt of cleaner tailings.

The key design objectives for the tailings management system include:

- Eliminate, manage or control environmental, health and safety risks with a zero-harm aspiration
- Design of the tailings management system to meet or exceed the requirements of Ecuadorian and international tailings design guidelines, standards, and regulations
- Permanent, safe and secure containment of all solid waste materials in facilities designed, constructed and operated engineered to international best practice
- Maximise tailings densities using effective tailings deposition strategies
- Minimise the risk of oxidation of potentially acid forming tailings
- Minimise water retained in the tailings facilities
- Allow for effective rehabilitation at cessation of use of the facilities in line with the closure objectives

This study assessed four options for tailings storage, with two tailings storage facilities located on the main concession to provide starter facilities for the project and two larger distal facilities located on the Ecuadorian Coastal Plains approximately 40 km from the processing plant. The preferred option for this study was the facility named the Coastal Planes East TSF. This facility was sized to store the full life of mine tailings in a single facility, with a cross valley downstream constructed embankment and large capacity spillway to manage water. The facility would be constructed of locally sourced borrow material in stages over the life of mine with the embankment having a LOM height of 190 m and a crest length of 3.3 km.

While the Coastal Planes East TSF was the preferred option, the other tailings storage options that were considered could offer benefits in certain scenarios, such as if open pit mining is pursued on the Cascabel concession or if the life of mine throughput for the project increases during future study stages.

1.18 Environment, Permitting and Social Consideration

SolGold has made considerable efforts to undertake environmental studies and community engagement to facilitate the advancement of the Project. Several environmental baseline studies have been initiated in anticipation of eventually permitting operational mine development with an Environmental and Social Impact Assessment (ESIA).

A summary of the major permits required for the construction and operation of the Project is provided in Table 1-4.

Table 1-4: Schedule of major permits for construction and operation

Prior to Construction	
Environmental License (EA) - Exploitation	Ministry of Environment, Water and Ecological Transition (Quito)
Water License - Industrial Use	Ministry of Environment, Water and Ecological Transition (Quito)
Authorisation to Build a Tailings Storage Facility	Ministry of Energy and Mines
Explosive Transportation, Storage, and Use Permits	Armed Forces Department of Arms Control
Quarry / Borrow Permits	Local Government
Fuel Purchase Permit	Regulation and Control Agency of Energy and Non-Renewable Natural Resources
Approval of Camp Specifications	Regulation and Control Agency of Energy and Non-Renewable Natural Resources
Mining Contract	Ministry of Energy and Mines
Prior to Operations	
Registration as a Hazardous Waste Generator	Ministry of Environment, Water and Ecological Transition (Quito)
Possession and Use of Controlled Substances	Technical Secretariat of the National System of Professional Qualifications
Registration of Hazardous Chemical Storage and Use	Ministry of Environment, Water and Ecological Transition (Quito)

To comply with the ESIA submission necessary to obtain mining permits, SolGold will prepare and submit an ESIA with the following components:

- Legal framework
- Detailed description of the Project, including an alternatives analysis
- Determination of the Area of Influence for all planned Project infrastructure on the environmental and social landscape
- Characterisation of the physical and biological baseline condition
- Characterisation of the socio-economic baseline condition
- Characterisation of the archaeological baseline condition
- Identification, prediction, and evaluation of environmental impacts
- Risk assessment
- Forest inventory and economic evaluation
- Citizen Participation Process
- Environmental Management Plan, which includes:
  - Mitigation Plan
  - Waste Management Plan
  - Communication, Training, and Environmental Education Plan
  - Community Relations Plan

– Worker Health and Safety Plan
– Monitoring Plan
– Rescue and Protection Plan (for species of concern that need relocation)
– Closure and Abandonment Plan
– Rehabilitation Plan

The communities closest to the Project are summarised in Table 1-5.

Table 1-5: Communities in the concession area

Community	Characteristics
Santa Cecilia	Located in the centre of the concession, 280 people 74 families 75% of the families works for SolGold
Santa Rita	Located downstream of the Cachaco TSF, 80 people 36 families
Getzsemani	Access road located on Concession 324 people 91 families
Santa Rosa	Located several km downstream of Cachaco TSF, 140 people 42 families
Cachaco	Located near the Rio Mira at the base of the Rio Cachaco watershed, 166 people 65 families
San Pedro	Located downstream in the catchment of the waste rock storage area, 526 people 154 families
Parambas	Located immediately downstream of Parambas on E10, 422 people 132 families

An archaeological study has been completed.

SolGold, through their community consultation program, has additionally identified culturally important sites in the region. Many of these are associated with their importance to the community, including water collection tanks, sports fields, churches and natural areas such as waterfalls.

The Closure Plan will consist of an estimated closure cost, upon which a financial guarantee or insurance policy in favour of the government will be required, which must remain in force until the final closure of operations. The detailed Closure Plan will be developed as part of the ESIA; the approach will be designed to ensure the long-term stability of the site's physical and chemical properties and to return the landscape to its pre-mining capability where possible. Progressive rehabilitation is currently integrated into the exploration phase. It will be an important aspect of concurrent programs during operations to minimise final disturbance areas upon cessation of mining.

1.19 Cost Estimates

The capital cost estimate meets the requirements for a PFS, consistent with AACE® International cost estimating guidelines for a Class 4 estimate. The estimate accuracy range of ± 25% is defined by the level of project definition, the amount of engineering inputs, the time available to prepare the estimate and the amount of project cost data available.

Initial capital costs are estimated to be $1,554 million as summarised in Table 1-6.

Table 1-6: Initial capital cost summary

Area	Initial Capital Cost US$M
Mine	403
Process plant	262
Tailings storage facility	267
Port facility	17
Surface infrastructure	293
Owners costs	92
Contingency	221
Total	**1,554**

Note: Totals do not necessarily equal the sum of the components due to rounding adjustments.

Expansion, sustaining capital and closure costs include the second process plant module, tailings storage facility development, equipment replacement and closure costs. Expansion, sustaining capital and closure costs are estimated to be $2,655 million. The life of mine (LOM) capital cost estimate is summarised in Table 21-3.

Table 1-7: Cascabel project total capital cost summary

Area	Total Capital Cost (US$M)
Pre-Production Capital Cost	1,554
Sustaining/Expansion Capital Cost	2,573
Closure Cost	82
Total Capital Cost	**4,209**

Note: Totals do not necessarily equal the sum of the components due to rounding adjustments.

Overall operating costs are presented in Table 1-8. The process plant comprises two 12 Mtpa module-based concentrators, giving a combined capacity of 24 Mtpa.

Table 1-8: Operating cost summary

Area	LOM Total US$M	Unit Cost US$/t processed
Mine	3,319	6.15
Processing	3,993	7.40
TSF	79	0.15
Port	103	0.19
Surface infrastructure	182	0.34
G&A	551	1.02
Total	**8,227**	**15.24**

Note: Totals do not necessarily equal the sum of the components due to rounding adjustments.

1.20 Market Studies and Economic Analysis

Metallurgical testwork provided for the Cascabel project concentrate indicates that it is a clean, precious metal enriched concentrate containing very low levels of deleterious elements; therefore, it is expected that the Cascabel concentrate will be attractive to global buyers. Numerous commodity traders have expressed interest in the sample Cascabel concentrate; however, no offtake agreements exist at the time of this study.

The Cascabel project generates positive pre- and post-tax financial results. Post-tax IRR is 24% and the post-tax NPV_8 is $3,221 million. Post-tax payback is achieved 4.1 years following the start of production.

2 Introduction

SRK Consulting (Canada) Inc. has compiled this Technical Report (the Report) on "Cascabel Project, Imbabura Province, Ecuador, NI 43-101 Technical Report on Pre-Feasibility Study" (Cascabel Project) on behalf of SolGold Plc (SolGold).

SolGold, through a 100% interest in ENSA, holds 100% of the 50 km^2 Cascabel concession in northern Ecuador. The Project is located at Rocafuerte within the concession in northern Ecuador, an approximately 3-hour drive on a sealed highway north of Quito, and is close to water sources and power supply, and is in relative proximity to port facilities along the Pacific coast.

This report discusses the results of the Cascabel project.

2.1 Terms of Reference

This Report supports SolGold disclosures in the news release dated 16 February 2024, entitled "SolGold plc ("SolGold" or the "Company") Announces Successful Completion of New Cascabel Pre-Feasibility Study with Significantly Reduced Initial Capital Cost and 24% Internal Rate of Return," which has an effective date of 31 December 2023.

All measurement units used in this Report are metric unless otherwise noted, and currency is expressed in United States (US) dollars as identified in the text. All cost estimates are reported in US$.

The Alpala and Tandayama-America deposits, the subject of this NI 43-101 Technical Report, are wholly contained within the Cascabel Property, which is wholly owned by Exploraciones Novomining S.A. (ENSA). SolGold holds a 100% legal and beneficial interest in ENSA.

This report has been produced in accordance with the Standards of Disclosure for Mineral Projects as contained in NI 43-101 and accompanying policies and documents. NI 43-101 utilises the definitions and categories of Mineral Resources and Mineral Reserves as set out in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014).

A draft of this report was provided to SolGold to check for factual accuracy.

2.2 Qualified Persons

The Qualified Persons (QPs) for the "Cascabel Project, Imbabura Province, Ecuador, NI 43-101 Technical Report on Pre-Feasibility Study", which has an effective date of 31 December 2023, and their respective areas of responsibility are detailed in Table 2-1.

Table 2-1: Qualified persons

Category	Name	Responsible Section(s)	Company
Mineral Resource Estimate and Data Verification – Alpala and Tandayama-America Deposits	Gilles Arseneau, Ph.D., P.Geo.	1.1 – 1.10 inclusive; 1.12; 3.0 – all; 4.0 – all; 5.0 – all; 6.0 – all; 7.0 – all; 8.0 – all; 9.0 – all; 10.0 – all; 11.0 – all; 12.0 – all; 14.0 – all; 25.2, 25.3, 25.4, 25.6; 26.4.1; 27; 28	Associate Consultant SRK Consulting (Canada) Inc.
Metallurgy and Process Plant	Ben Adaszynski, P.Eng.	1.11; 1.13; 1.16; 13.0 – all; 17.0 – all; 21 (part); 25.4, 25.9; 26.16.3; 27; 28	Sedgman Canada Ltd.
Mineral Reserve Estimate and Mining (Underground); Other	Jarek Jakubec, C.Eng., FIMMM	1.14; 1.15; 2.0 – all; 15.0 – all; 16.0 – all except 16.5; 21.1.2, 21.2.3; 23.0 – all; 25.7, 25.8; 25.16.2; 26.4.2, 26.4.3; 26 (part); 27; 28	SRK Consulting (Canada) Inc.
Underground Mining Ventilation	Brian Prosser, PE	16.5.3; 21 (part); 27; 28	SRK Consulting (US) Inc.
Underground Mining Infrastructure	Guy Lauzier, P.Eng.	16.5.1, 16.5.2, 16.5.4, 16.5.5; 21 (part); 27; 28	Allnorth Consultants Limited
Surface Infrastructure	Richard Boehnke, P.Eng.	1.17; 18.0 – all except 18.5 and 18.6; 21 (part); 25 (part); 26 (part); 27; 28	JDS Energy & Mining Inc.
Environment, Social, Tailings & Water	Timothy David Rowles, BSc, MSc FAusIMM CP RPEQ	1.18; 18.5, 18.6; 20; 21 (part); 25.11, 25.16.8; 26 (part); 27; 28	Knight Piésold Pty Ltd
Marketing and Financial Evaluation	Carl Kottmeier, P.Eng., MBA	1.19; 1.20; 19.0 – all; 22.0 – all; 24; 25.15; 26.4.9; 27; 28	SRK Consulting (Canada) Inc.

2.3 Personal Inspection of the Cascabel Property

Dr. Arseneau undertook a personal inspection of the Cascabel Property from 2-3 October 2023, in relation to Alpala and Tandayama-America Mineral Resource Estimates, where he visited drill sites, logging stations, sample preparation and storage facilities and interviewed key SolGold personnel. The inspection was conducted with the full support of SolGold and without restriction in any respect.

Mr. Jarek Jakubec, SRK mining lead for the PFS, visited the Cascabel site from 2-3 October 2023. During the site visit, Mr. Jakubec inspected the core and logging facilities, the proposed portal location, and the Alpala Exploration Camp.

Mr. Richard Boehnke visited the site from 2-3 October 2023 in relation to the site facilities and planning for the Cascabel project. Mr. Boehnke visited the proposed plant site, portal location, camp facilities, existing structures, roads, and nearby communities.

Mr. Timothy David Rowles visited the Cascabel site from 6-8 February 2020. Mr. Rowles has been involved with the Cascabel project since January 2020 as the Knight Piésold Project Manager for various engineering and environmental studies undertaken since then. During the site visit, Mr. Rowles visited the proposed plant site, the Parambas dam site, different potential TSF locations, and the Alpala Exploration Camp.

2.4 Effective Date

There are several effective dates pertinent to the Report, as follows:

- Effective date of the Alpala Mineral Resource estimates: 11 November 2023
- Effective date of the Tandayama-America Mineral Resource estimate: 11 November 2023
- Effective date of the Mineral Reserve estimate: 31 December 2023
- Date of the economic analysis that supports Mineral Reserve estimation: 31 December 2023

The overall Report's effective date is 31 December 2023 and is based on the effective date of the Mineral Reserve estimate and economic analysis that supports the Mineral Reserves.

2.5 Information Sources and References

This Report is primarily based on PFS reports and optimisation studies undertaken during a period from February 2020 to March 2022, as listed in Section 27, with additional technical work completed in the areas of mining engineering, metallurgy and processing, tailings management and surface infrastructure.

3 Reliance on Other Experts

3.1 Legal Title

To the extent permitted under NI 43-101, the QPs have relied, in respect of legal aspects, upon the work of the Experts listed below:

"SolGold Plc – Legal Title Opinion on ENSA's Cascabel Mining Tenement", containing information on mineral tenure and status, title issues, royalty obligations, etc." prepared by César Zumarraga, Licensed Lawyer, 1 March 2024. The reliance applies to Section 4 of the technical report.

3.2 Taxation

To the extent permitted under NI 43-101, the QPs have entirely relied upon the information supplied by Tobar ZVS for information related to taxation as applied to the financial model. It is important to note that the taxation assumptions used in the economic model are the same as those used in negotiating the Exploitation Contract with the Ecuadorian government. On 19 July 2023, ENSA and the Ministry of Energy and Mines concluded the pre-contractual negotiation of the Cascabel Mining Exploitation Contract and executed the Final Negotiation Minutes ("Term Sheet").

4 Property Description and Location

4.1 Summary

Exploraciones Novomining S.A. (“ENSA”) is the registered titleholder of the Cascabel Concession, which comprises the Property. The summary details of the Cascabel Property are as shown in Table 4-1 below.

Table 4-1: Property information

Area	Description
Name	Cascabel
Code	402288
Owner	ENSA
Surface	4979 ha (49.79 km^2)
Date Registered	26 July 2023
Term	25 years counted from the registration date of the title
Location	Lita and La Carolina parishes, Ibarra canton, Imbabura Province

Cascabel was duly and validly granted to ENSA in accordance with the applicable laws of Ecuador, and ENSA holds the exclusive right to perform exploration and mining activities free of encumbrances or liens. The Cascabel Property is wholly owned by ENSA. SolGold holds a 100% legal and beneficial interest in ENSA.

The concession is registered in the mining register of the Ibarra branch of the Mining Control and Regulation Agency. Such registration shows that the concession is valid and is not subject to any liens or encumbrances. The concession is not in an area protected under the National System of Protected Areas.

SolGold is a registered shareholder with a legal and beneficial 100% interest in ENSA, which holds 100% of the Cascabel concession.

4.2 Location

The Cascabel Property is located within the Imbabura province of northern Ecuador, approximately 100 km north of the capital city of Quito and approximately 50 km north-northwest of the provincial capital, Ibarra (Figure 4-1).

The northern border of the Project lies approximately 20 km south of the Colombia-Ecuador border and 75 km southeast of San Lorenzo, located on Ecuador’s Pacific coast.

Access to the Project is via sealed highways through the closest major centre of Ibarra, located approximately 80 km south of the property. The Cascabel concession area contains three small settlements: Santa Cecilia Village, Rocafuerte Office Complex, and the Alpala Base Camp. Four further settlements exist proximal to the project area at Rocafuerte, San Pedro, Urbina and Cachaco.

Source: This study, 2024

Figure 4-1: Cascabel project location

4.3 Mineral Tenure

SolGold has invested significant resources in exploration, making the area one of the most important assets not only for the Company but for the Ecuadorian State and the communities in the area of influence.

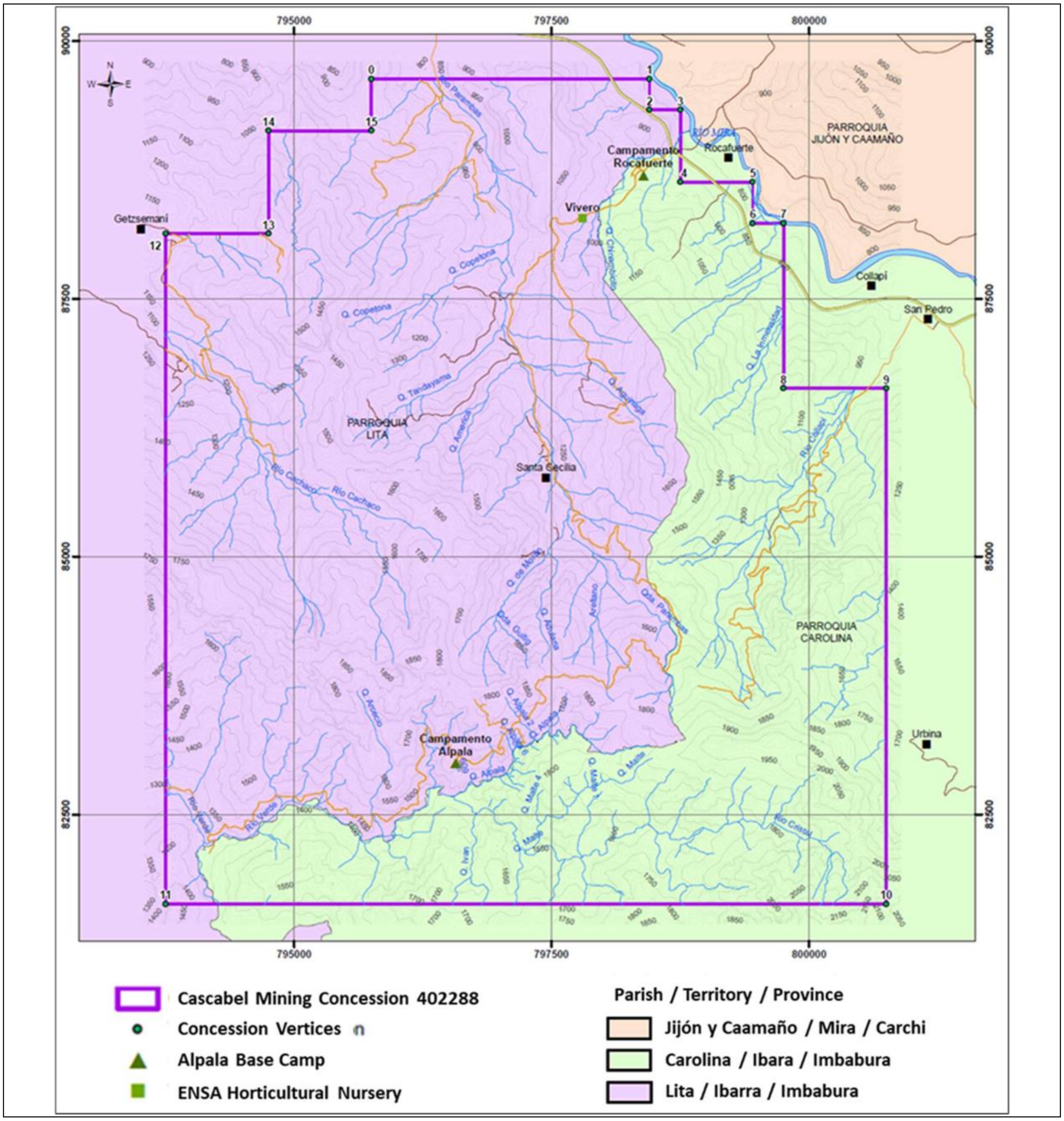
The Cascabel concession was initially granted to Santa Barbara Copper and Gold S.A. on 26 April 2010. Santa Barbara Copper & Gold S.A. reformed its statute, modifying its corporate name to Exploraciones Novomining S.A. (ENSA) on 27 July 2011 and was duly registered on the Commercial Registry on 29 November 2011.

The concession was granted over 5,000 hectares, valid for 25 years and renewable for a further 25 years. Following a compulsory reduction on 16 December 2016 due to a change of period to advanced exploration in compliance with the Mining Law, the Cascabel Concession now covers 4,979 hectares (49.79 km^2). The Cascabel Concession is currently registered as an Exploitation Licence for metallic minerals under cadastral code 402288. On 10 July 2023, the Ministry of Energy and Mines issued the title renewal for 25 years for the Concession. With this, the new term of the Concession expires in 2048.

The license area is recorded under political and geographical datums PSAD56-17S and WGS84- UTM-17N (Table 4-2). The Cascabel concession lies at the confluence of the parishes of Lita, La Carolina (Figure 4-2).

Table 4-2: Cascabel exploration concession boundary coordinates registered under the Ecuador mining cadastre

Vertice ID	DATUM: WGS84-UTM-17N		DATUM: PSAD56-17S	
	X	Y	X	Y
PP	795741.75	89623.32	796000	90000
PP01	798441.75	89623.32	798700	90000
PP02	798441.75	89323.32	798700	89700
PP03	798741.75	89323.32	799000	89700
PP04	798741.75	88623.32	799000	89000
PP05	799441.75	88623.32	799700	89000
PP06	799441.75	88223.32	799700	88600
PP07	799741.75	88223.32	800000	88600
PP08	799741.75	86623.32	800000	87000
PP09	800741.75	86623.32	801000	87000
PP10	800741.75	81623.32	801000	82000
PP11	793741.75	81623.32	794000	82000
PP12	793741.75	88123.32	794000	88500
PP13	794741.75	88123.32	795000	88500
PP14	794741.75	89123.32	795000	89500
PP15	795741.75	89123.32	796000	89500
***Transformation from PSAD56-17N to WGS84-UTM-17N = N-258.25, E-376.68**				



Source: Artica et al., 2022

Figure 4-2: Location and vertices of the Cascabel concession

On 26 April 2010, the Undersecretary of Mines of the Ministry of Non-Renewable Natural Resources issued the title of the exploration concession for metallic minerals called Cascabel (Code 402288) to Santa Barbara Copper and Gold S.A. The Resolution was registered in the Thirty-Second Notary Office of the Quito Canton on 30 April 2010. It was registered in the Mining Registry by the Ibarra Mining Regulation and Control Agency on 7 May 2010, under Registry No. 0022-2010, Directory 024, Volume 001.

On 29 March 2011, Cornerstone Capital Resources Inc. now known as SolGold Canada Inc., signed a definitive agreement to acquire 100% of the shares of Santa Barbara Copper and Gold S.A., which was then a subsidiary of Santa Barbara Resources (TSXV-SBL). Following acquisition of 100% of Santa Barbara Copper and Gold S.A., it was subsequently renamed Exploraciones Novomining S.A. (ENSA).

On 10 April 2012, SolGold entered into a binding Letter of Intent with Cornerstone to earn an 85% interest in ENSA. On 18 February 2013, the Earn-in Agreement was supplemented by a binding Term Sheet between SolGold, Cornerstone, CESA and ENSA. An amended term sheet was executed on 24 February 2014. Having fulfilled the earn-in requirements, SolGold became a registered shareholder with a legal and beneficial 85% interest in ENSA, which holds 100% of the Cascabel concession.

On 20 August 2018, ENSA notified the Zonal Sub-secretariat for Mining North of the Ministry of Mining of the start of the Economic Evaluation period for the Cascabel Deposit, in accordance with articles 27 and 37 of the Mining Law.

On 25 April 2019, the North Zonal Co-ordination (Zones 1, 2 and) of the Ministry of Energy and Non-Renewable Natural Resources issued Resolution No. MERNNR-CZN-2019-0042-RM that contains the Resolution of the Beginning of the Economic Evaluation Period of the Cascabel Deposit. This duly notarised Resolution was registered in the Mining Registry by the Regulatory Agency and Ibarra Mining Control dated 20 June 2019, under the Repertoire Book No.20, Volume No. 1, Item No. 1 and Folio No. 31.

On 30 November 2021, SolGold plc, SolGold Finance AG, and Exploraciones Novomining S.A. and SolGold-Ecuador S.A. signed an Exploration Investment Protection Agreement (IPA) with Ecuador Ministry of Production, Foreign Trade, Investments and Fisheries. The IPA included an intention to invest a total of approximately $430 million over the ten years between 2013 and 2023 in mineral exploration activities in the Cascabel concession in the canton of Ibarra, province of Imbabura, Ecuador.

On 3 January 2022, the Northern Zonal Coordinator of the Ministry of Energy and Non-Renewable Natural Resources issued the Resolution Nº MERNNR-CZN-2022-0001-RM authorising the extension of the economic evaluation of the deposit in Cascabel. The resolution was duly registered in the Mining Registry on 17 March 2022.

On 7 October 2022 SolGold and Cornerstone announced that they had entered into a definitive agreement whereby SolGold would acquire all of the issued and outstanding shares of Cornerstone, other than Cornerstone Shares already held, directly or indirectly, by SolGold, pursuant to a court-approved plan of arrangement.

Effective 24 February 2023, SolGold completed the previously-announced plan of arrangement with Cornerstone Capital Resources Inc. Cornerstone, now known as SolGold Canada Inc. became a wholly-owned subsidiary of SolGold.

On 17 July 2023, The Northern Zonal Coordination of the Ministry of Energy and Mines issued Official Letter No. MEM-CZN-2023-0810-OF containing Resolution No. MEM-CZN -2023-0090-RM for the Term Renewal of the Cascabel Mining Concession (Code 402288) until 2048. The Company may request additional term renewals in the future. The Term Renewal confirms that Cascabel comprises 4,979 contiguous hectares and is a large-scale mining regime in accordance with Ecuador's mining regulations. The mining concession renewal term is 25 years from the registration date of the Resolution in the Mining

Registry under the Agency for the Regulation and Control of Energy and Non-Renewable Natural Resources and subsequent notification to the Northern Zonal Coordination of the Ministry of Energy and Mines.

On 19 July 2023, SolGold announced an agreement with the government of Ecuador on the terms and conditions of the Exploitation Agreement. SolGold completed the contractual negotiations and agreed upon a term sheet in preparation for the execution of the Exploitation Agreement for the Cascabel project. SolGold, through its wholly owned subsidiary in Ecuador, Exploraciones Novomining S.A., has negotiated the right to develop the Cascabel project and produce copper, gold, and silver from the contract area for 33 years, which may be renewed.

On October 26, 2023, the Ministry of Energy and Mines issue the Resolution MEM-CZN-2023-0224-RM declaring the initiation of the exploitation stage of Cascabel mining concession. Such resolution was notarised and registered before the Energy and Non-Renewable Natural Resources Regulation and Control Agency on December 6, 2023.

4.4 Underlying Agreements

As part of the terms of the sale of ENSA by Santa Barbara, Santa Barbara was granted a 2% Net Smelter Return (NSR). The NSR is the gross amount received from the sale of ores, concentrates or precipitates processed from the mine less the fair market costs of smelting, refining, sampling, charges and penalties for treatment and testing and also less the fair market costs of handling, transporting, securing and insuring that material.

Pursuant to the Earn-in Agreement and the Term Sheet, SolGold is entitled to purchase the 1% NSR for $1 million within three months of the completion of a Feasibility Study. SolGold can purchase a further 1% NSR for $3 million within three months of a decision made by the owners to mine the Project. Since the dissolution of Santa Barbara in 2015, the benefit of the NSR has been held by a third-party agent in trust for the benefit of the prior shareholders of Santa Barbara.

As part of the acquisition of Cornerstone, SolGold entered into an Earn-in Agreement with Cornerstone, ENSA and CESA to explore the Cascabel license. On 18 February 2013, the Earn-in Agreement was supplemented by a binding Term Sheet between SolGold, Cornerstone, CESA and ENSA. An amended term sheet was executed on 24 February 2023.

4.5 Licences and Authorisations

Regulatory licenses and authorisations currently in place at the Cascabel Property include:

- Registration of the Cascabel Mining Title for Exploitation phase under Code 402288 by the Ministry of Energy and Mines
- Environmental License under Mining Title Code 402288 (Ministry of the Environment, Ecuador) for the advanced exploration phase of metallic minerals including underground exploration activities
- Water Licensing approved by the National Secretariat of Water of the Republic of Ecuador (SENAGUA). There are two water licences in place.
- Fuel Licensing approved by the Hydrocarbon Regulation and Control Agency (ARCH)

- ANEXO 2.5. Copia de la Declaración Juramentada (ANNEX 2.5 Copy of the Affidavit) in compliance with Article 26 of the Mining Law as a prior administrative authorisation before execution of mining activities
- Certificate of Compliance with Employers Obligations (Ecuadorian Social Security Institute)
- Completed the contractual negotiations and agreed upon a term sheet in preparation for the execution of the Exploitation Agreement for the Cascabel project
- Prior administrative authorisations are in compliance with Article 26 of the Mining Law

SolGold has entered into land access agreements for all areas of the Cascabel property with proprietors and maintains strong working relationships with all stakeholders.

ENSA is a corporation duly incorporated and validly existing under the laws of Ecuador. It is in good standing with respect to all applicable filing and regulatory requirements and has all the legal capacity to conduct its business in Ecuador.

The Concession is duly registered and in good standing; no liens or encumbrances are registered. There is neither any indication of any potential issue that could result in the termination, revocation, or suspension of the Concession nor any evidence of grounds for the nullification of the ownership of the Concession held by ENSA.

ENSA, in accordance with the Mining Law, has complied on a timely basis with its obligations to pay the conservation patent fees and to file annual exploration reports with respect to the Concession.

ENSA holds the necessary licenses, permits and registrations, including all necessary environmental licenses and water permits, to conduct advanced exploration, economic evaluation of the deposit and underground exploration on the Concession for metallic minerals. Such licenses, registrations and permits have been duly granted and are validly held by ENSA. Currently the Concession is in the exploitation stage. There are no outstanding agreements or operations that may limit the right of ENSA to conduct mining activities. There are no agreements for operations with artisanal miners.

Environmental licenses do not have a specific term and are in effect while the licensed activities are developed. Certain annual reporting and auditing obligations apply to maintain the good status of the licenses. The environmental license is in good standing.

ENSA has two permits for industrial water use, conferred by the National Secretariat of Water (SENAGUA) and which are in force and valid. Industrial water use permits are granted for ten renewable years. One of these permits was valid until July 2023; however, it has already been renewed for ten more years, and the other permit is valid until August 2027.

Although the term for mining concessions is 25 years, the Mining Act establishes maximum terms for each mining phase for the industrial mining system. Once a mining concession is granted, concessionaires are subject to the following phases and terms within the exploration phase for medium-scale and large-scale mining:

- Up to 4 years of Initial Exploration
- Up to 4 years of Advanced Exploration

- Up to 2 years of Economic Evaluation of the deposit, which can be extended for an additional 2-year period

Initial exploration commenced upon granting of the concession on 26 April 2010. The Project progressed into the advanced exploration phase on 17 September 2014.

On 28 August 2018, the letter of notification was sent to the applicable Zone Co-ordinator of the Ministry in Ibarra, and the Economic Evaluation phase commenced.

Recently, the Northern Zonal Coordination of the Ministry of Energy and Mines, through Resolution No. MEM-CZN-2023-0224-RM of 26 October 2023, granted ENSA the change from phase to exploitation for the Cascabel concession. Within six months of starting the exploitation phase, large-scale mining concessionaires must sign an exploitation contract with the Ecuadorian government, although negotiations may begin during the economic evaluation phase.

Currently, Exploraciones Novomining S.A. (ENSA) is the registered holder of the mineral tenure of the Property, which is in northern Ecuador. The Alpala resource, the subject of the NI 43-101, is wholly contained within the Cascabel Licence, which is wholly owned by Exploraciones Novomining S.A. (“ENSA”). SolGold holds a 100% legal and beneficial interest in ENSA.

Regulatory licenses and authorisations required for the future entry into the final phase of the Cascabel project include:

- Exploitation Agreement
- Environmental and water licensing for the operation and associated facilities
- Other permits and licenses, including those applicable to construction, hazardous substances, transport, etc.

There are no permanent illegal mining activities within the Cascabel concession. However, there is sporadic entry of artisanal miners panning alluvial gold, and according to the information provided by ENSA, when the presence of such miners is detected, they are asked to leave. In case they do not respond to this petition, there are recourses under Ecuadorian legislation to safeguard ENSA from legal liability in the unlikely event of potential environmental impacts.

There are no outstanding operations or any other kind of agreements that may limit ENSA’s right to conduct mining activities. There are no operation agreements with artisanal miners.

No administrative, legal, or other form of claim or complaint has been made against ENSA by communities and/or Indigenous groups regarding the Cascabel concession.

Administrative permits from ministries and applicable municipalities, for both on-site and off-site locations, required for the final operations phase will depend on the designs to be elaborated in the technical studies. Examples of these include:

- Explosives use and storage
- Special labour shifts
- Permits for construction (building permits) plant sites, access roads, water process plant(s), process water pond(s), accommodation and amenities, office complexes and tailings storage facilities (TSF)

- Right of Way (ROW) permits applicable to pipelines, power transmission lines, TSF, port facilities, water process plant(s), access roads, heavy transport, chemical storage and transportation, and hazardous goods

4.6 Environmental Considerations

Exploration and mining activities in Ecuador are subject to provisions of the Mining Act of 2009 (Ley de Minería, last reformed 2021). According to the Mining Act, the holders of mining licenses must obtain and submit environmental studies to prevent, mitigate, control and repair the environmental and social impact resulting from such activities.

The Ministry of the Environment, Water and Ecological Transition is responsible for the approval of the environmental studies and licenses of the Project. According to the Environmental Regulation for Mining Activities (Reglamento Ambiental para Actividades Mineras), projects, works, or activities within the medium-scale mining and large-scale mining regimes, for their initial exploration phase, will require an environmental registration while for their advanced exploration, exploitation, and subsequent phases, they will require an environmental license. In August 2013, the Ecuadorian Ministry of Environment issued an Environmental License for advanced exploration, including drilling. In October 2023, the Ministry of the Environment granted the update of the environmental license, which includes underground exploration activities within the advanced exploration phase. SolGold has commenced the acquisition of landholdings in the Cascabel Property area in anticipation of infrastructure requirements for project development. As of 31 March 2022, SolGold has purchased 2,492 hectares comprising 80 properties, with negotiations ongoing on other properties.

4.7 Mineral Rights in Ecuador

Mining in Ecuador is mainly governed by the Mining Act (MA), issued on 29 January 2009 and the General Regulation of the Mining Act (GRMA), issued on 16 November 2009, which regulates activity as a whole. The MA and GRMA recognise, regulate, and classify mining activities depending on production levels, namely: large-scale mining; medium-scale mining; small-scale mining; and artisanal mining.

To conduct exploration in Ecuador, a mining concession title must be granted by the Ministry of Energy and Mines and registered at the respective mining registry managed by the Energy and Non-Renewable Natural Resources Regulation and Control Agency (Agencia de Regulacion y Control de Energía y Recursos Naturales No Renovables) (ARCERNNR). The term of a mining license is 25 years and is renewable for similar periods upon request by the license holder. Once the concession title has been granted, exploration may be conducted for a four-year term, identified as the initial exploration period, and governed by Article 37, with such term initiated from the date the prior administrative authorisations indicated in Article 26 are obtained.

The license holder is entitled to request a further 4-year period from the Ministry of Energy and Mines under Article 37 to proceed with advanced exploration. At this point, part of the exploration license will be relinquished (minimum 1 hectare), although there is no legislated minimum area to be dropped. The Ministry will process this application provided the company meets the minimum investment commitment during the initial exploration stage.

Other aspects of the Mining Act which are considered to be pertinent are described as follows:

Regarding taxation and royalties - The holder of the licence is subject to other taxes, payments and contributions, such as:

- Income Tax – 25% of taxable profits. However, this rate will decrease by 5% when the Company completes the Complementary Investment Protection Agreement for Cascabel.
- Labour Profit-Sharing – 15% (12% to the State and 3% to employees in the case of large-scale mining, and 10% to the State and 5% to employees in the case of medium-scale mining; 5% to the State and 10% to employees in the case of small-scale mining)
- Value Added Tax – 12%
- Municipal taxes and contributions, social security contributions
- Annual conservation patent fee that the license holder shall pay by March 31 each year for each mining hectare. This equates to 2.5% of the government-mandated basic salary, currently $460/ha for the 2024 fiscal year for the mining license for the initial exploration period. The patent fee doubles to 5% of the basic salary per hectare for the advanced exploration and economic evaluation periods and doubles again to 10% during the exploitation phase of the mining license.

In addition to the taxes outlined above, the license holder must pay the State a royalty between 3% to 8% of the value of the NSR for Au, Ag and Cu for large-scale mining.

The Ecuadorian government has various taxes, duties and levies that may or may not apply to future mining operations depending on the mining exploitation contract to be established between SolGold and the Ecuadorian government, the Investment Protection Agreement to be negotiated between the Ministry of Production and SolGold, and the laws in force at that time. These are addressed in Section 22.

Regarding surface rights - The holder of a mining license has a right to request easements over the surface land to duly exercise its mining rights. The rights emanating from this easement include, among others, the right to occupy certain areas for construction required for mining activities and rights related to waterways, railways, landing strips, ramps, transport belts, and electrical installations. The easement must be registered in the mining registry managed by ARCERNNR.

The surface landowner is entitled to receive payment from the holder of the mining license for the easement granted. In some instances, the easement rights, including terms and conditions, are expressly agreed to in contracts executed between the license holder and the surface landowner. If no agreement is reached, ARCERNNR may order the creation of the easement and determine the mandatory payments due to the landowner.

In August 2013, the Ecuadorian Ministry of Environment, Water and Ecological Transition issued an Environmental License for advanced exploration, including drilling. In October 2023, the Ministry of the Environment granted the update of the environmental license, which includes underground exploration activities within the advanced exploration phase.

On July 26, 2013, the National Water Secretariat resolved to grant ENSA the right to exploit the waters of the Mira River Hydrographic Demarcation to be used during the execution of the advanced mining exploration period in the Cascabel project. The water concession has a renewable validity of 10 years, which has been renewed.

On 7 August 2017, the National Water Secretariat granted a second concession for the Use and Consumption of Industrial Water for the Mira Hydrographic Demarcation for the Cascabel project for advanced exploration activities. The combined concessions allow extraction from a maximum of 14 collection points or water sources (water collection points are included for use in advanced exploration activities and use in camps); and an authorised flow rate of 1.5 L/s for each collection point.

The QP understands that SolGold holds all required permits and easements to operate across the Cascabel Property.

4.8 Comments on Section 4

The Cascabel Property has in place the necessary regulatory licenses and authorisations required for its current status as an Advanced Exploration Project, including early works and underground exploration. Furthermore, with the ongoing support of the community and government organisations, future license and authorisation requirements to advance the Project are expected to continue to be successfully obtained.

The Cascabel mining concession has an environmental license for the development of advanced exploration activities. In October 2023, the Ministry of the Environment granted the update of the environmental license, which includes underground exploration activities within the advanced exploration phase. Furthermore, with the ongoing support of the community and government organisations, it is expected that in the future, the requirements for obtaining a new environmental license will be successfully met, allowing progress with the development of the Project in its exploitation and benefit phases.

In the opinion of the QP:

- Information provided by experts contacted by SolGold on the mining tenure held by ENSA in the Project area supports that the Cascabel project has a valid title that is sufficient to support the declaration of Mineral Resources and Mineral Reserves.
- Additional negotiations and permits are required for the proposed operation (see discussion in Section 20), with key areas of focus being:
  - Exploitation Agreement
  - Development Stage Exploration Permit for Early Works
  - Amended Investment Protection Agreement
  - Permitting/Environmental and Social Impact Assessment and License

To the extent known to the QP, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the Project that are not discussed in this Report.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The following Section is taken from Artica et al. (2022) with minor modifications where required.

5.1 Summary

Cascabel is located in northern Ecuador, approximately a three-hour drive north of Quito, the capital city of Ecuador. Access is via paved highways through the closest major city of Ibarra, located approximately 90 km south of the property. The Cascabel concession area contains three small settlements: Santa Cecilia Village, Rocafuerte Office Complex, and the Alpala Base Camp. Four further settlements exist near the Project area at Rocafuerte, San Pedro, Urbina and Cachaco.

The topography of the Project area is moderate to steep, with elevations rising from 750 m to 2,140 m above sea level. The rugged terrain is deeply incised by four large drainage complexes. Vegetation is tropical forest with a well-developed soil horizon up to 12 m thick in places.

The climate of the Project area is characterised by humid weather, with bi-modal rain seasons that peak in December and March. The total average rainfall for the region is approximately 1,500 mm per annum. Regionally, temperatures remain relatively consistent throughout the year, with average annual temperatures of approximately 17°C, with a maximum in excess of 30°C and a minimum typically around 10°C.

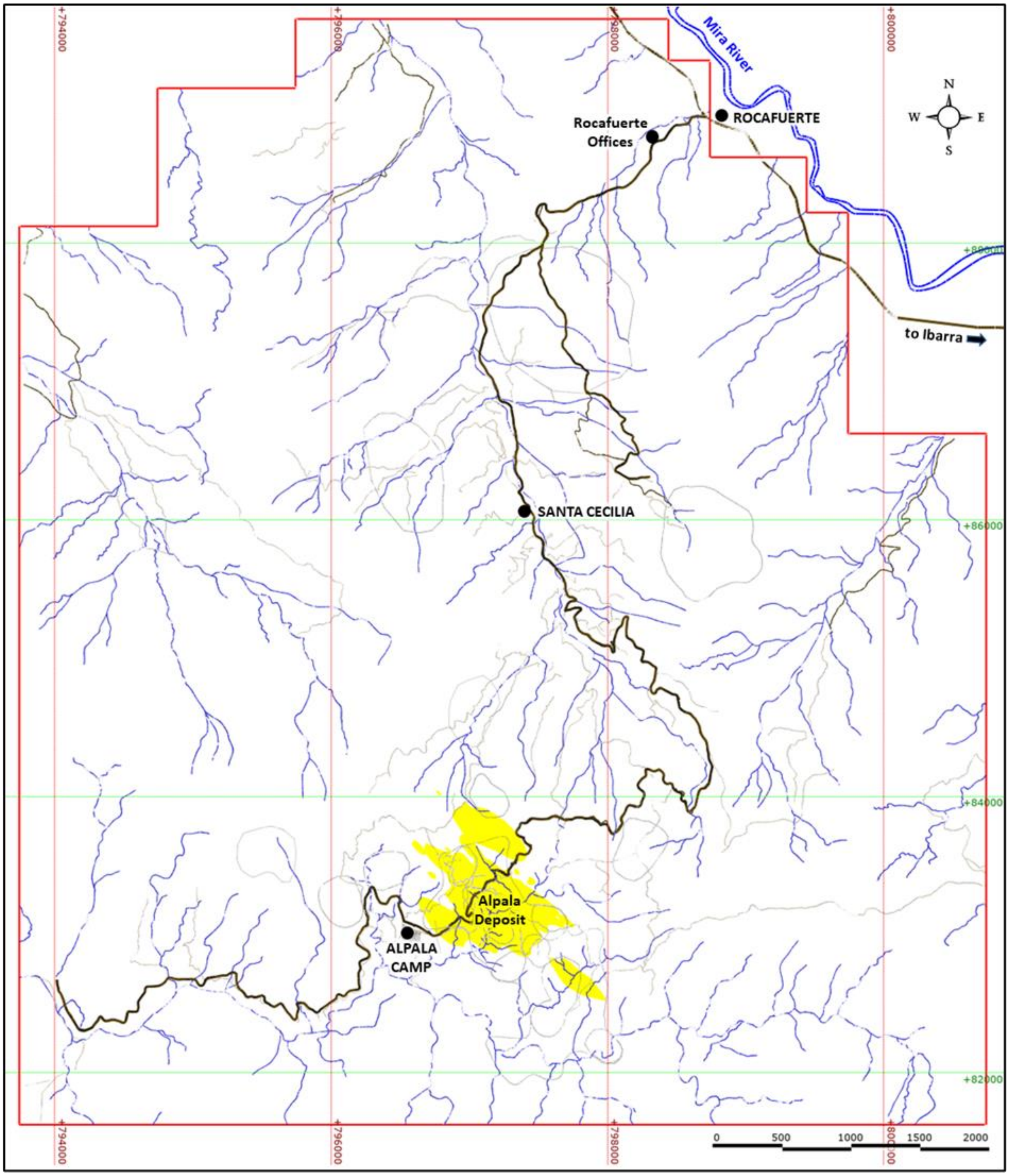
5.2 Accessibility

International flights regularly arrive and depart from Mariscal Sucre International Airport, 18 km east of Quito, from major carriers, including KLM, Delta, American Airlines, Iberia, United and Copa Airlines.

The Cascabel project is easily accessible from Quito via the paved multi-lane E35 Pan-American North highway to Ibarra (approximately 100 km) and connected to the northern margin of the concession area near Rocafuerte Village (approximately 80 km) via the paved two-lane E10 Salinas-San Lorenzo highway that runs along the Rio Mira valley. Driving time to the Project offices at Rocafuerte is approximately 3 hours.

Within the Cascabel concession area, access to the Rocafuerte Office Complex, the Alpala and Tandayama-America deposits, and Alpala Base Camp is via Carmen Road, a maintained two-lane dirt road from Rocafuerte village, through Santa Cecilia village (Figure 5-1).

The main exploration prospects within Cascabel are accessible via a series of maintained single-lane dirt roads, single-lane four-wheel drive tracks and hiking trails off Carmen Road.



Source: Artica et al., 2022

Figure 5-1: Major access roads (black) and tracks (grey) within the Cascabel concession area, highlighting the footprint of the Alpala deposit (yellow)

5.3 Local Resources and Infrastructure

The property is largely undeveloped, containing only three small settlements: Santa Cecilia Village, Rocafuerte Base Camp and Alpala Camp. Three further settlements lie proximal to the Project area at San Pedro, Urbina and Cachaco.

SolGold is continually developing its social and community programs around Cascabel to maximise employment for Ecuadorians, create, support, and endorse local Ecuadorian businesses, and provide financial assistance and support for secondary and tertiary educational institutions (including universities and technical colleges). This strategy is also being applied to SolGold's regional exploration activity areas.

Skilled and unskilled labourers, operators and helpers are readily available in the local area. The majority of local labourers, operators, and helpers are drawn from these settlements, and a number of training programs are in place for the advancement of local assistants. Professional labour is sourced from all over Ecuador, as well as a range of globally situated professionals and consultants. Other services and goods are often procured from the surrounding settlements, with further options available at Ibarra or Quito.

Infrastructure in the region and throughout Ecuador is generally of a high standard, with excellent road access, power, and water sources readily available in the local area. A two-lane sealed highway (E10) connecting the cities of Ibarra and San Lorenzo runs along the northern margin of the property, and a further multi-lane highway (E15) provides a link further south to the port city of Esmeraldas. A multi-lane highway (Pan-American E35) links Ibarra and the capital Quito.

Power generation in Ecuador is dominated by hydroelectricity, with 38 of the 74 power plants across the country being hydroelectric. Installed nominal power exceeds 5,000 MW, while total installed nominal power exceeds 8,900 MW. In the last decade, a number of new large hydroelectric dams have come online in Ecuador. These projects have greatly increased Ecuador's generating capacity to the extent that almost 90% of electricity is now hydropower. Due to the weather phenomenon "El Niño," Ecuador experiences droughts every three to five years that can impact hydropower production. As a result, the current government recently passed new energy legislation striving to create more renewable capacity outside of the hydro area. In addition, the government is vigorously promoting private participation in power projects by providing incentives and preferential rates.

The construction of new electrical substations near Ibarra in Northern Ecuador provides potential access to hydroelectric power for the provision of power needs to the Cascabel project.

SolGold focuses its operations on being safe, dependable and environmentally responsible and maintains close relationships with its local communities.

Ecuador is evolving rapidly in terms of infrastructure and development, with 10 renewed and 21 operating national airports, four of which are international airports. Ecuador has more than 966 km of state railways, linking highlands and coastal regions and mainly used for tourism purposes. The port system comprises seven state-owned ports and ten private docks, specialised in general cargo and oil. The three major ports include Guayaquil, Manta, and Bolívar. The Ministry of Transport and Public Works contributes to national development by formulating policies, regulations, plans, programs, and projects to ensure a National Intermodal and Multimodal Transport based on international quality transport network standards, aligned with economic, social, and environmental guidelines and the national development plan.

5.4 Climate

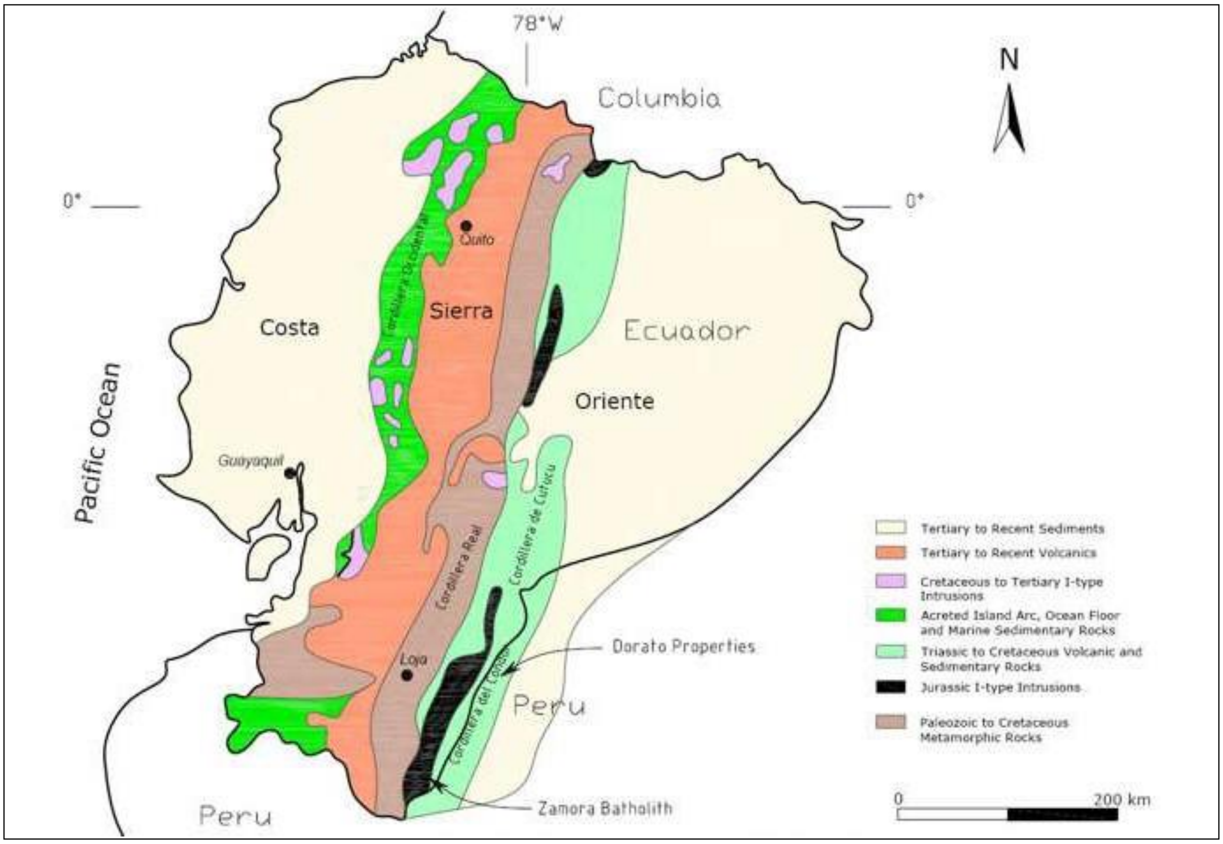
Based on long-term data from regional stations operated by the National Weather and Hydrology Institute (Instituto Nacional de Meteorología e Hidrología, or INAMHI), the climate of the Project area is characterised by humid weather, with a bi-modal rainy season, having peaks in December and March, each with rainfall more than 200 mm on average. The total average rainfall for the region is approximately 1500 mm. The Alpala camp receives substantially higher rainfall than Rocafuerte due to the orographic effect of its mountainous location. The driest month is July, with an average of less than 30 mm of rain.

The climate in the mountainous regions of Ecuador is typically cooler than coastal parts of the country due to the altitude. The Alpala camp lies at approximately 1750 mRL, and nightly temperatures can drop below 9°C. Rocafuerte lies at approximately 800 m RL and is often significantly warmer than Alpala.

Regionally, temperatures do not fluctuate greatly throughout the year. The average annual temperature is approximately 17°C, with maxima in excess of 30°C and minima typically around 10°C.

5.5 Physiography

Ecuador comprises three central physical regions: The Costa (Pacific Coastal Region), the Sierra (Andes Region), and the Oriente (eastern Amazon Region). A central graben called the inter-Andean graben effectively divides the Sierra region into the Cordillera Real (Eastern Cordillera) and the Cordillera Occidental (Western Cordillera) (Figure 5-2).



Source: Belik., 2008

Figure 5-2: Topographic map of Ecuador

The Cascabel Property is located within Ecuador's tropical savannah climate zone on the lower western foothills of the Cordillera Occidental. The topography of the Project area is moderate to steep, with elevations of 750 m in the valley bottom to 2,200 m in the higher exploration zones, incised by dendritic drainage complexes within the tributary watersheds of the Mira River basin (Figure 5-3).

Source: Artica et al., 2022

Figure 5-3: Typical landscape in the Cascabel project area

The Property is characterised as having a patchwork of remnant mature tropical forest interspersed with disturbed forest and cleared/agricultural land. The Cotacachi Cayapas Ecological Reserve is the nearest protected area, approximately 20 km southwest of the Cascabel concession. With its borders well outside the Mira River catchment, the Cotacachi Cayapas Ecological Reserve watershed is well outside the Project's area of influence.

The rugged terrain is incised by four large drainage complexes. Vegetation is tropical forest with a well-developed soil horizon of up to 12 m thick in parts.

5.6 Location of Mine Facilities

The Cascabel Property has suitable areas for the location of mine facilities, including tailings storage facilities, processing plant, process water ponds, office and accommodation complexes, excess rock disposal, and suitable heavy vehicle access routes; they have been evaluated as part of the study.

5.7 Comments on Section 5

In the opinion of the QP:

- There is sufficient area within the Project to host an underground mining operation, including mine and plant infrastructure, waste rock, and tailings storage facilities. The existing local and planned infrastructure, staff availability, power, and water are adequate to support the proposed mining operation.
- Current surface rights are adequate to support the Project, the development and operations of the Project.
- It is expected that any future mining operations will be able to be conducted year-round.

6 History

6.1 Summary

From 1980 to the present, ten different entities have conducted field activities and/or studies over the Cascabel area. Historical exploration of the Project area, undertaken from 1980 to May 2012, highlighted widespread geochemical anomalism in stream pan-concentrates, stream sediments and rock chips over a nine-square kilometre area in the northern half of the license area Previous explorers focused on the source of Au, Cu, Pb and Zn in stream sediments, which led to the location of gold-bearing, polymetallic base-metal sulphide quartz veins in streams that flank the northern periphery of the Alpala Deposit.

The Cascabel tenement was granted to Santa Barbara Copper and Gold S.A (SBCG) on 26 April 2010. Santa Barbara Copper and Gold S.A. was renamed Exploraciones Novomining S.A. on 27 July 2011. On 29 March 2011, Cornerstone signed a definitive agreement to acquire Santa Barbara Copper and Gold S.A.

On 24 July 2012, SolGold entered into an Earn-in Agreement with Cornerstone, ENSA, and CESA (see section 4.3), which was further supplemented by a binding Term Sheet executed on 24 February 2014.

In May 2012, SolGold assumed the management of the Project and commenced the first systematic exploration program at Cascabel. The surface expression of the Alpala Deposit was discovered the same month during reconnaissance mapping, which located an 80 m wide zone of Cu and Au bearing, dominantly sheeted and stockwork porphyry-style quartz veining in Alpala Creek. After follow-up mapping, geochemical and geophysics programs were conducted, and other porphyry-related stockwork veins were subsequently discovered in the Moran, Tandayama and America Creeks.

Rock channel sampling and structural measurements of quartz veins over a 430 m by 200 m area at Alpala provided the geological context for a diamond drilling program.

In August 2013, the Environmental License for the Cascabel concession was approved. On 1 September 2013, drilling of the first hole commenced using a modified man-portable drill rig operated by Hubbard Perforaciones (HP).

The first four holes of the drill program confirmed surface mineralisation to depths of approximately 200 m. However, the course of the program was modified by the extent and high grades of chalcopyrite-bearing quartz vein stockworks encountered in Hole 5, which was drilled less than 18 months after the location of surface mineralisation. This fifth drill hole marks the discovery of the high-grade, world-class Alpala Deposit, with an overall interval of 1,306 m at 0.62% Cu and 0.54 g/t Au, including 552 m at 1.03% Cu and 1.05 g/t Au from a 778 m downhole depth.

6.2 Previous Work (1980-2012)

The earliest documented exploration in the Cascabel Property area includes work conducted by the Directorate General of Geology and Mines (DGGM) from 1980 to 1984 and a cooperative agreement with the Belgium Mission from 1984 to 1985 (Gilbertson, 2017). This work identified quartz veins, stockworks and disseminated sulphides at Parambas Creek in the southern part of the current tenement. An agreement between Rio Tinto Zinc (RTZ) and the Ecuadorian Government in 1986 facilitated the

Inductively Coupled Plasma (ICP) analysis of rock samples from outcrops within the area. Still, the focus of this work was west of Junin, southwest of Cascabel. Lumina Gold Corp, formerly Odin Mining and Exploration Ltd, undertook limited stream sediment sampling in the license area between 1988 and 1991, which generated Ag, Cu, Pb and Zn anomalies. However, Odin relinquished the Cascabel tenement back to the Ecuadorian Government.

The Ecuadorian Mining Development and Environmental Control Project (1998 to 2000) completed 1:50,000-scale geological mapping and stream-sediment sampling over much of the Western Cordillera with the assistance of the British Geological Survey. This work identified Au-Ag-Cu-Pb-Zn-bearing, epithermal-type quartz veins hosted by propylitic and clay-silica-altered volcanic rocks in the vicinity of Cascabel, including outcrops in the Parambas Creek.

ENSA was granted the current Cascabel license area along with other concessions in 2008. Subsequent prospecting, stream sediment, and rock sampling generated anomalous results in Au, Ag, Cu, Pb, and Zn. Cornerstone acquired the Project from Santa Barbara Resources Limited through the purchase of ENSA in February 2011. Prospecting, reconnaissance mapping and a stream sediment survey in June and July 2011 delineated Cu-Au-Mo and Pb-Zn-As rock chip anomalies Cu-Mo-Au stream sediment anomalies. A central 4 km by 5 km area of interest was identified around porphyry-style outcrops in Moran Creek. Gold anomalous rock samples, containing greater than 0.1 g/t Au to greater than 1 g/t Au, were collected in Cachaco Creek and Parambas Creek from outcrops that are located less than one to three kilometres from what became the discovery outcrop in Alpala Creek.

A summary of the Project ownership and key activities is provided in Table 6-1.

Noroccidente Project (1980-1984)

The first exploration undertaken at the Cascabel project, and surrounding areas was through an initiative of the Directorate General of Geology and Mines (DGGM) called the Noroccidente project. The Project targeted mineral occurrences in the northern provinces of Carchi, part of Esmeraldas and Imbabura. Work involved 1:500,000 scale regional geology mapping and the collection of 822 stream sediment samples, which were analysed by atomic absorption spectroscopy (AAS) for Au, Ag, Cu, Zn and Pb. This work identified 10 anomalies, including the Junín Cu-Mo porphyry mineral property owned by Empresa Nacional Minera del Ecuador (Enami) in JV with Codelco (Chile) through the Llurimagua Joint Venture.

The National Government of Ecuador signed a technical assistance agreement with the Government of Belgium to undertake exploration work over each of the anomalies detected, including the Parambas River (partly within the Cascabel License), and to expand regional exploration.

Belgian Co-operation Project (1984-1985)

A cooperative agreement between the Belgian Mission and the Ecuadorian Institute of Mining (INEMIN, ex-DGGM) resulted in geological, geochemical, and geophysical investigations being carried out for VMS (Volcanogenic-massive sulphide) and porphyry-style mineralisation. This exploration covered the Cascabel license and surrounding areas. Stockworks, veins and disseminated sulphides and sulphosalts were discovered on several sites. Only the Junín, Parambas (Cascabel) and Zarapullo occurrences were deemed to have economic potential.

Table 6-1: Cascabel ownership history and key activities

Period	Company	Major Activities Advancing Cascabel Project
1980-1984	INEMIN (Ecuadorian Institute of Mining)	Noroccidente Project, including geological mapping and stream sediment sampling
1984-1985	INEMIN & BEM	Preliminary field inspections only
1986	INEMIN (ex-DGGM) & Rio Tinto Zinc Corp.	Western Cordillera I; analyses of samples from stream anomalies and extensions identified in previous INMEM/BEM study
1988-1991	Lumina Gold (ODIN Mining & Exploration)	Preliminary stream sediment sampling
1991-1997	Japan International Cooperation Agency	Reported discovery of mineralised porphyries that intrude the Apuela batholith at Junin (Llurimagua), approximately 60 km SW of Cascabel
1998-2000	Government of Ecuador (PRODEMINCA)	Western Cordillera II: Definition of geochemical provinces within the Western Cordillera, including identification of polymetallic base metal sulphide veins at Moran Creek in Cascabel
1998-2000	INEMIN	EMDEC Project (Ecuadorian Mining Development & Environmental Control Project)
2008-2011	Santa Barbara Copper and Gold (SBCG)	26 Apr 2010: Tenement granted to SBCG
		Stream sediment, rock chip sampling and prospecting identified widespread geochemical anomalism
2011-2012	Cornerstone Capital Resources (CCR)	29 Mar 2011: CCR acquires 100% of SBCG
		27 Jul 2011: SBCG corporate name is modified to ENSA
		Stream sediment, pan-conc., and rock chip sampling (93 samples)
2012-2022	SolGold Plc (SolGold)	12 April 2012: SolGold enters a binding Letter of Intent with CCR to earn 85% interest in ENSA. Cascabel license 100% owned by ENSA
		3 Jan 2018: Alpala Maiden Mineral Resource Estimate (MRE)
		3 Jan 2019: Alpala MRE#2
		7 May 2019: Preliminary Economic Assessment (PEA) (Amended 19 Nov 2020)
		7 Apr 2020: Alpala MRE#3 (Amended 29 Sep 2020)
		31 March 2022: SolGold releases Prefeasibility study for the Cascabel project

Western Cordillera Agreement (1986)

Through an agreement between INEMIN and RTZ, selected stream sediment samples were collected from previously determined anomalies at Parambas and Morán Rivers using ICP for 29 elements. The samples were historically collected during exploration projects sponsored by the United Nations, with technical assistance from the United Kingdom and cooperation with Belgium. Some exploration and sampling to the west of Junín, outside the current Cascabel license area, was undertaken, where additional samples were collected for analysis by RTZ. A database was compiled containing 9,120 samples.

Lumina Gold Corp (Formally Odin Mining and Exploration Ltd). (1988-1991)

Lumina Gold Corp conducted limited stream sediment sampling in the Cascabel license area and surrounding areas. Anomalous Cu, Pb, Zn, and Ag results were obtained in an area controlled by mainly propylitic alteration. Despite this, Odin did not continue its work and the license was returned to the Ecuadorian State.

Japan International Co-operation Agency (1991-1997)

Detailed exploration studies were conducted by the Japan International Co-operation Agency - Metal Mining Agency of Japan (JICA-MMAJ) in the Junín area proximal to, but not including, the Cascabel license, resulting in the discovery of porphyries that intrude the Apuela batholith. They concluded that the mineralisation is associated with zones of sericitic alteration and facies of granodioritic porphyries. Using the available geological data, preliminary mineral resource estimates were made for the Junín project (more recently known as the Llurimagua project), which estimated 982 Mt at 0.89% Cu, 0.04% Mo and 0.01 g/t Au, with a 0.4% Cu cut-off grade (Gribble, 2004).

The mineral resource estimate is a historical estimate; the QP has no information on the parameters or key assumptions used to prepare the estimate. The estimate is only presented here for historical completeness. It is relevant because it shows that the Cascabel project has been the subject of advanced exploration since 2004. The historical estimate does not use the resource categories as outlined in NI 43-101. The QP has not done the work to validate or verify the historical estimate, and it is superseded by the estimate presented in Section 14 of this report and should not be relied upon.

Western Cordillera Agreement (1998-2000)

Under the Mining Development and Environmental Control Project (PRODEMINCA) along the Western Cordillera, the Government of Ecuador collected 15,175 stream sediment samples. Samples were analysed by ICP for 38 elements. Spatial and geochemical analysis of this data led to the definition of geochemical provinces within the Western Cordillera. Within the framework of this project, the Parambas sector, which contains the Cascabel license, was considered a Cu-Pb-Zn-Ag-Au epithermal deposit, consisting of irregular veins in an area of propylitic alteration and mineralised silicification. The mineralisation may be related to the volcanic activity of the San Juan de Lachas Unit (Boland et al., 2000).

Santa Barbara Copper and Gold S.A. (2008-2011)

In 2008, Santa Bárbara Copper and Gold S.A. (SBCG) applied for the Cascabel license along with other licenses from the Ecuadorian State. SBCG submitted an environmental impact study to the Ministry of the Environment. Stream silt surveys and additional prospecting identified widespread geochemical anomalism.

Cornerstone Capital Resources Inc (2011-2012)

On 29 March 2011, Cornerstone Capital Resources Inc. signed a definitive agreement to acquire 100% of the shares of SBCG, a subsidiary of then Santa Barbara Resources Limited (TSXV-SBL). SBCG was subsequently renamed Exploraciones Novomining S.A. (ENSA). ENSA was granted the current Cascabel license area along with other concessions in 2008. Subsequent prospecting, stream sediment and rock sampling generated results for Au, Ag, Cu, Pb and Zn.

Prospecting, reconnaissance mapping and a stream sediment survey in June and July 2011 delineated Cu-Au-Mo and Pb-Zn-As rock chip anomalies and Cu-Mo-Au stream sediment anomalies. A central 4 km by 5 km area of interest was identified around porphyry-style outcrops along Moran Creek. Gold anomalous rock samples, containing greater than 0.1 g/t Au to greater than 1.0 g/t Au, were collected in Cachaco Creek and Parambas Creek from outcrops that are located less than one to three kilometres from what became the discovery outcrop in Alpala Creek.

In April 2012, SolGold entered into a binding Letter of Intent with Cornerstone to earn an 85% interest in ENSA. SolGold now holds an 85% registered and beneficial interest in ENSA. The Cascabel license is 100% owned by ENSA. Upon signing the agreement, SolGold assumed management of the Project.

6.3 Recent Work (from 2012)

NOTE: This section includes several previously disclosed mineral resource and mineral reserves statements for the Cascabel and Tandayama-America deposits. While all the estimates are reliable and relevant and use categories of mineral resources and mineral reserves as set out in Section 1.2 and 1.3 of NI 43-101, they are not considered current as they don't consider all of the current drilling information. The estimates are only presented here for historical completeness. The estimates have all been superseded by the estimates presented in Section 14 of this report. The assumptions, parameters and methods used to prepare each estimate is discussed after each table reporting the results.

In May 2012, SolGold commenced the first systematic exploration program at Cascabel. The surface expression of the Alpala Deposit was discovered the same month during reconnaissance mapping, which located an 80 m wide zone of Cu and Au bearing, dominantly sheeted and stockwork porphyry-style quartz veining in Alpala Creek. After follow-up mapping, geochemical and geophysics programs were conducted, and other porphyry-related stockwork veins were subsequently discovered in the Moran, Tandayama and America Creeks.

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Maiden Mineral Resource Estimate – Alpala Deposit (MRE#1)

In December 2017, a maiden Mineral Resource Estimate (MRE#1) for the Alpala Porphyry Copper-Gold-Silver Deposit was announced, comprising 430 Mt at 0.8% CuEq in the Indicated category, and 650 Mt at 0.6% CuEq in the Inferred category (at 0.3% CuEq cut-off), for a contained metal content of 2.3 Mt Cu and 6.0 Moz Au in the Indicated category, and 2.9 Mt Cu and 6.3 Moz Au in the Inferred category (at 0.3% CuEq cut-off) (Gilbertson et al, 2017). This work is summarised in the NI 43-101 Report entitled “A Technical Report on the Maiden Mineral Resource Estimate for the Alpala Deposit, Ecuador” (Gilbertson, J., Pittuck, M., & Willis, J. 2017.).

Table 6-2: Alpala Maiden Mineral Resource (MRE#1), effective date 18 December 2017

	Resource Category	Tonnage (Mt)	Grade			Contained Metal		
			Cu (%)	Au (g/t)	CuEq (%)	Cu (Mt)	Au (Moz)	CuEq (Mt)
>1.1% CuEq	Indicated	70	1.1	1.3	1.8	0.7	2.8	1.2
	Inferred	50	1.1	1.3	1.8	0.5	1.9	0.8
0.9 -1.1% CuEq	Indicated	50	0.7	0.5	1.0	0.3	0.9	0.5
	Inferred	50	0.7	0.5	1.0	0.4	0.9	0.5
0.3 – 0.9% CuEq	Indicated	310	0.4	0.2	0.5	1.2	2.3	1.6
	Inferred	550	0.4	0.2	0.5	2.0	3.5	2.6
Total >0.3% CuEq	Indicated	430	0.5	0.4	0.8	2.3	6.0	3.4
	Inferred	650	0.4	0.3	0.6	2.9	6.3	4.0

Notes:
1. Mr. Martin Pittuck, CEng, MIMMM, FGS, was responsible for this Mineral Resource estimate and was an "independent qualified person" as such term is defined in NI 43-101.
2. The Mineral Resource was reported using a cut-off grade of 0.3% CuEq calculated using [copper grade (%)] + [gold grade (g/t) x 0.6].
3. The Mineral Resource was considered to have reasonable potential for eventual economic extraction by underground mass mining, such as block caving.
4. Mineral Resources were not Mineral Reserves and do not have demonstrated economic viability.
5. The statement used the terminology, definitions and guidelines given in the CIM Standards on Mineral Resources and Mineral Reserves (May 2014) as required by NI 43-101.
6. The MRE was reported on 100 percent basis.

Updated Mineral Resource Estimate – Alpala Deposit (MRE#2)

In December 2018, an updated Mineral Resource Update (MRE#2) was announced, comprising 2,050 Mt at 0.60% CuEq in the Indicated category and 900 Mt at 0.35% CuEq in the Inferred category (at 0.2% CuEq cut-off), for a contained metal content of 8.4 Mt Cu and 19.4 Moz Au in the Indicated category, and 2.5 Mt Cu and 3.8 Moz Au in the Inferred category (at 0.2% CuEq cut-off). This work is summarised in the NI 43-101 Report entitled, “A technical report on an updated mineral resource estimate for the Alpala deposit, Cascabel Project, Northern Ecuador.” (Gilbertson, J., & Pittuck, M. 2018).

Table 6-3: Alpala Mineral Resource update (MRE#2)

Resource Category	Tonnage (Mt)	Grade			Contained Metal		
		Cu (%)	Au (g/t)	CuEq (%)	Cu (Mt)	Au (Moz)	CuEq (Mt)
Indicated	2,050	0.41	0.29	0.60	8.4	19.4	12.2
Inferred	900	0.27	0.13	0.35	2.5	3.8	3.2

Notes:

1. Mr. Martin Pittuck, CEng, MIMMM, FGS, was responsible for this Mineral Resource statement and was an "independent qualified person" as such term is defined in NI 43-101.
2. Mineral Resource was reported using a cut-off grade of 0.2% CuEq calculated using [copper grade (%)] + [gold grade (g/t) x 0.63].
3. Mineral Resource was considered to have reasonable prospects for eventual economic extraction by underground mass mining such as block caving.
4. Mineral Resources were not Mineral Reserves and do not have demonstrated economic viability.
5. The statement used the terminology, definitions and guidelines given in the CIM Standards on Mineral Resources and Mineral Reserves (May 2014) as required by NI 43-101.
6. The MRE was reported on 100 percent basis.

Updated Mineral Resource Estimate – Alpala Deposit (MRE#3)

On 7 April 2020, SolGold announced a further Mineral Resource Estimate Update (MRE#3). The MRE (MRE#3) for the Alpala and Tandayama-America (TAM) deposits were reported in accordance with the Canadian National Instrument 43-101 (NI 43-101) Standards for Disclosure for Mineral Projects using mineral resource categories as outlined in NI 43-101. The mineral resources are discussed in Section 14 of this report under Previous Mineral Resources. MRE#3 is superseded by MRE#4, which is presented in Section 14 of this report.

The MRE#3 update followed a further 83,650 m of infill drilling since the previous Mineral Resource Estimate (MRE#2); this update delivered the conversion of considerable tonnages into the Measured Resource category, plus the addition of 1.6 Mt Cu, 2.5 Moz Au, and 92.2 Moz Ag (not previously estimated) to the Measured plus Indicated Mineral Resources at the Alpala Deposit. Highlights of the Alpala MRE#3 Mineral Resource at 0.21% CuEq cut-off grade included:

- A Mineral Resource of 2,663 Mt at 0.53% CuEq for 9.9 Mt Cu, 21.7 Moz Au and 92.2 Moz Ag in the Measured plus Indicated categories (Table 64)
- A Mineral Resource of 544 Mt at 0.31% CuEq for 1.3 Mt Cu, 1.9 Moz Au and 10.6 Moz Ag in the Inferred category, including:
    - A High-grade core of 442 Mt at 1.40% CuEq for 3.8 Mt Cu, 12.3 Moz Au and 33.3 Moz Ag in the Measured plus Indicated categories

Table 6-4: Alpala Mineral Resource update (MRE#3)

Cut-off Grade	Mineral Resource Category	Mt	Grade				Contained Metal			
			CuEq (%)	Cu (%)	Au (g/t)	Ag (ppm)	CuEq (Mt)	Cu (Mt)	Au (Moz)	Ag (Moz)
0.21	Measured	1,192	0.72	0.48	0.39	1.37	8.6	5.7	15.0	52.4
	Indicated	1,470	0.37	0.28	0.14	0.84	5.5	4.2	6.6	39.8
	Measured + Indicated	**2,663**	**0.53**	**0.37**	**0.25**	**1.08**	**14.0**	**9.9**	**21.7**	**92.2**
	Inferred	544	0.31	0.24	0.11	0.61	1.7	1.3	1.9	10.6
	Planned dilution	*5*	*0.00*	*0.00*	*0.00*	*0.00*	*0.0*	*0.0*	*0.0*	*0.0*

Notes:
1. Mrs. Cecilia Artica, SME Registered Member, Principal Geology Consultant of Mining Plus, is responsible for this Mineral Resource statement and is an "independent Qualified Person" as such term is defined in NI 43-101.
2. The Mineral Resource is reported using a cut-off grade of 0.21% CuEq calculated using [copper grade (%)] + [gold grade (g/t) x 0.613] as discussed below.
3. The Mineral Resource is considered to have reasonable prospects for eventual economic extraction by underground mass mining such as block caving.
4. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
5. The statement uses the terminology, definitions and guidelines given in the CIM Standards on Mineral Resources and Mineral Reserves (May 2014) as required by NI 43-101.
6. MRE is reported on a 100 percent basis within an optimised shape as described below.
7. Figures may not compute due to rounding.

The data cut-off for the MRE#3 totaled 227,961.6 m of diamond drilling at the Cascabel project, with 217,225.9 m completed at the Alpala deposit, 7,258.7 m completed at the Aguinaga deposit, and 3,477.0 m completed on sterilisation drilling and water monitoring wells.

The cut-off grade used for reporting was based on recent third-party metal price research, forecasts of copper and gold prices, and an operating cost structure for the Alpala deposit reported in the Preliminary Economic Assessment (PEA) filed on 19 November 2019. Operating costs included mining, processing and general and administration (G&A). Net Smelter Return (NSR) included metallurgical recoveries and off-site realisation (TCRC), including royalties. Metal prices used were $3.40/lb for Cu and $1,400/oz for Au.

Maiden Mineral Resource Estimate - Tandayama-America (TAM) Deposit (TAM MRE#1)

On 19 October 2021, SolGold announced the maiden Mineral Resource Estimate for the Tandayama-America Deposit at Cascabel (TAM MRE#1). This work was detailed in the report filed on 29 January 2021, entitled “Cascabel Property NI 43-101 Technical Report, Alpala Porphyry Copper-Gold-Silver Deposit – Mineral Resource Estimation, January 2021”. The TAM deposit lies approximately 3 km north of the Alpala deposit within the Cascabel concession,

The TAM MRE#1 contained 0.53 Mt Cu and 1.20 Moz Au in the Indicated category, plus 197.0 Mt at 0.39% CuEq containing 0.52 Mt Cu and 1.24 Moz Au in the Inferred category. Potentially open pittable Mineral Resources comprise 201.0 Mt at 0.33% CuEq in the Indicated category, plus 61.8 Mt at 0.44% CuEq in the Inferred category, at a cut-off grade of 0.16% CuEq.

Table 6-5: TAM Maiden Mineral Resource (TAM MRE#1)

Potential Mining Method	Cut-off Grade (CuEq %)	Resource Category	Tonnage (Mt)	Grade			Contained Metal		
				Cu (%)	Au (g/t)	CuEq (%)	Cu (Mt)	Au (Moz)	CuEq (Mt)
Open Pit	0.16	Indicated	201	0.22	0.16	0.33	0.45	1.06	0.66
		Inferred	61.8	0.25	0.3	0.44	0.16	0.59	0.27
Underground	0.28	Indicated	32	0.26	0.14	0.35	0.08	0.14	0.11
		Inferred	135.2	0.27	0.15	0.37	0.37	0.65	0.5
Total Indicated			**233**	**0.23**	**0.16**	**0.33**	**0.53**	**1.2**	**0.77**
Total Inferred			**197**	**0.27**	**0.2**	**0.39**	**0.52**	**1.24**	**0.77**

Notes:

1. Dr. Andrew Fowler, MAusIMM CP(Geo), Principal Geology Consultant of Mining Plus, is responsible for this Mineral Resource statement and is an "independent Qualified Person" as such a term is defined in NI 43-101.
2. The Mineral Resource is reported using cut-off grades that are applied according to the mining method, where 0.16% CuEq applies to potentially open-pittable material and 0.28% CuEq applies to material potentially mineable by underground bulk mining methods. Copper equivalency is discussed in detail in “Reasonable Prospects for Eventual Economic Extraction”.
3. The Mineral Resource is considered to have reasonable prospects for eventual economic extraction by open pit or underground bulk mining, such as block caving, as described below.
4. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
5. The statement uses the terminology, definitions and guidelines given in the CIM Standards on Mineral Resources and Mineral Reserves (May 2014) as required by NI 43-101.
6. The underground portion of the Mineral Resource is reported on 100 percent basis within an optimised shape as described below.
7. Figures may not compute due to rounding.

Potentially open pittable Mineral Resources include a higher-grade near-surface zone containing 10.6 Mt at 0.41% CuEq and 5.2 Mt at 0.45% CuEq. Mineral Resources potentially mineable by underground bulk mining methods comprise 32.0 Mt at 0.35% CuEq in the Indicated category, plus 135.2 Mt at 0.37% CuEq in the Inferred category, at a cut-off grade of 0.28% CuEq.

A data cut-off was applied to the TAM dataset for the purposes of Mineral Resource Estimation. The TAM maiden MRE dataset comprised 17,535 m of diamond drilling from holes 1-23, 458 m of surface rock-saw channel sampling from 72 outcrops, and 14,566 m of final assay results from holes 1-18.

Updated Mineral Resource Estimate - Tandayama-America (TAM) Deposit (TAM MRE#2)

On 26 May 2022, SolGold announced an updated Mineral Resource Estimate for the Tandayama-America Deposit at Cascabel (TAM MRE#2). The Total Mineral Resource at the TAM deposit is updated to 528.5 Mt at 0.36% CuEq for 1.27 Mt Cu and 3.16 Moz Au in the Measured plus Indicated categories, equating to an increase in the contained metal of approximately 0.74 Mt Cu and 1.96 Moz Au compared to the maiden MRE announced on 19 October 2021.

The potentially open pittable Mineral Resource comprises 17.8 Mt at 0.30% CuEq in the Measured category, 338.7 Mt at 0.36% CuEq in the Indicated category, plus 35.7 Mt at 0.36% CuEq in the Inferred category, at a cut-off grade of 0.16% CuEq.

The Mineral Resource potentially mineable by underground bulk mining methods comprises 172.0 Mt at 0.35% CuEq in the Indicated category, plus 69.4 Mt at 0.36% CuEq in the Inferred category at a cut-off grade of 0.28% CuEq.

Table 6-6: TAM Maiden Mineral Resource (TAM MRE#1)

Potential Mining Method	Cut-off Grade (CuEq %)	Resource Category	Tonnage (Mt)	Grade			Contained Metal		
				Cu (%)	Au (g/t)	CuEq (%)	Cu (Mt)	Au (Moz)	CuEq (Mt)
Open Pit	0.16	Measured	17.8	0.20	0.16	0.30	0.04	0.09	0.05
		Indicated	338.7	0.23	0.21	0.36	0.78	2.28	1.23
		Inferred	35.7	0.22	0.23	0.36	0.08	0.26	0.13
Underground	0.28	Indicated	172.0	0.26	0.14	0.35	0.45	0.78	0.60
		Inferred	69.4	0.26	0.16	0.36	0.18	0.36	0.25
Total Measured + Indicated			**528.5**	**0.24**	**0.19**	**0.36**	**1.27**	**3.16**	**1.89**
Total Inferred			**105.1**	**0.24**	**0.18**	**0.36**	**0.26**	**0.62**	**0.38**

Notes:

1. Dr. Andrew Fowler, MAusIMM CP(Geo), Principal Geology Consultant of Mining Plus, is responsible for this Mineral Resource statement and is an "independent Qualified Person" as such term is defined in NI 43-101. Reasonable prospects of eventual economic extraction were assessed by:
    a) First presenting the mineralised material in the block model estimate to a conventional Lersch-Grossman open pit optimisation routine based on a cut-off grade of 0.16% CuEq, and the cost and revenue assumptions listed below. Mineralised material inside the revenue factor one pit and above the cut-off grade were then reported in the “Open pit” section of the Mineral Resource statement.
    b) Subsequently, the remaining material was enclosed in a 3D wireframe shape that was constructed with adherence to a minimum mining unit with geometry appropriate for a block cave.
2. Cut-off grade for the potentially open pittable material was 0.16% CuEq, calculated using (copper grade (%)) + (gold grade (g/t) x 0.632).
3. Cut-off grade for the underground shape was defined as the shut-off grade under a breakeven, eventual economic extraction criteria. The shut-off grade of 0.28% CuEq was calculated using (copper grade (%)) + (gold grade (g/t) x 0.654).
4. All material within the underground shape was reported in the “Underground” section of the Mineral Resource statement, as block caving is a non-selective method, and all material extracted is treated as mill feed.
5. The resulting shape contained planned internal and edge dilution that the QP considers appropriate.
6. Cut-off/Shut-off inputs included:
    a) Metal prices of Cu at $3.30/lb and Au at $1,700/oz
    b) Recoveries of copper at 84.4% and gold at 65%
    c) Costs including mining, processing and general and administration (G&A)
    d) Off-site realisation (TCRC), including royalties
7. Cut-off/Shut-off inputs excluded:
    a) Capital costs (non-mining, access and footprint establishment)
    b) Unplanned dilution
    c) The time value of money
8. The QP considers that the Mineral Resource has reasonable prospects for eventual economic extraction by open pit or an underground mass mining method such as block caving, as presented in the Mineral Resource statement.
9. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
10. No Mineral Reserves exist for the Property at the time of reporting. Nevertheless, the Mineral Resources are reported inclusive of those Mineral Resources that may be converted to Mineral Reserves in the future.
11. The statement uses the terminology, definitions and guidelines given in the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) as required by NI 43-101.
12. Figures may not sum due to rounding.

The TAM MRE#2 Mineral Resource update was estimated from 14,806 assays, 14,586 assays from diamond drill core samples, and 220 assays from rock-saw channel samples cut from surface outcrops. Drill core samples were obtained from 26,631.6 m of drilling from 40 diamond drill holes. Surface rock-saw channel samples were obtained from 458 m of channel intervals from 72 surface channels.

2022 Pre-Feasibility Study

On 31 March 2022, SolGold released a PFS for the Cascabel project (Artica et al., 2022). This report was prepared as National Instrument 43-101 Technical Report for SolGold plc and Cornerstone Capital Resources Inc by qualified persons employed by Mining Plus Pty Ltd, Knight Piésold Pty Ltd, Wood Australia Pty Ltd and Wood Mackenzie plc (the Consultants). The report included an updated mineral resource for the Alpala and Tandayama-America (TAM) deposits and the first-time disclosure of mineral reserves for the Alpala deposit.

The Mineral Reserve Estimate for the Alpala Underground Porphyry Copper-Gold-Silver Deposit, Cascabel Property, with an effective date of 31 March 2022, was prepared by Mining Plus Pty Ltd (Mining Plus). Mining Plus employee Aaron Spong is the Qualified Person responsible for the Reserve Estimate. The Reserve estimation process followed the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) “Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines” (CIM, 2019).

The Mineral Reserve Estimate was stated in accordance with the CIM Definition Standards (CIM, 2014) and Canadian National Instrument 43-101 (NI 43-101). The Reserve has been estimated for a block caving method. It considers the effect of mixing indicated material with dilution from low-grade or barren material originating from within the caved zone and the overlying cave backs. As stated above, only Measured and Indicated categories have been considered for the reserve, with Inferred categories considered as waste and grades set to zero.

Table 6-7: Cascabel project pre-feasibility study, 31 March 2022; Alpala Mineral Reserves Statement

Classification	Tonnes	Copper Grade (%)	Gold Grade (g/t)	Silver Grade (g/t)	Copper Metal (kt)	Gold Metal (koz)	Silver Metal (Moz)
Probable	558	0.58	0.52	1.65	3.26	9.37	30
Total	**558**	**0.58**	**0.52**	**1.65**	**3.26**	**9.37**	**30**

Notes:
1. Effective date of the Mineral Reserves is 31 March 2022.
2. Only Measured and Indicated Mineral Resources were converted to Probable Mineral Reserves.
3. Mineral Reserves reported above are not additive to the Mineral Resource and are quoted on a 100% project basis.
4. The Mineral Reserve is based on the March 2020 Mineral Resource.
5. Totals may not match due to rounding.
6. The Mineral Reserve Estimate has been reported to conform with the CIM Definition Standards on Mineral Resources and Mineral Reserves (CIM, 2014).
7. The Mineral Reserve Estimate for Alpala was independently verified by Aaron Spong FAusIMM.

Mining Plus utilised empirical and analytical analyses of geotechnical data provided by SolGold to determine the geotechnical characteristics of various lithological and alteration domains within the deposit. The geotechnical properties informed the:

- Caveability and critical span/hydraulic radius (HR) required for caving
- Stability of long-term excavations
- Primary and secondary cave fragmentation
- Stand-offs required for underground infrastructure
- Surface subsidence zone

Based on the compilation and analyses of all available hydrogeologic data and the mine plans for underground infrastructure (including ramps, drifts, and shafts) and block caving, Itasca developed a conceptual hydrogeologic model and the three-dimensional numerical groundwater flow model. The numerical groundwater flow model was calibrated to the groundwater levels measured during hydraulic testing performed in boreholes drilled across the Project. The calibrated groundwater flow model was used to predict the seepage to the underground workings and the effects of mining on baseflow and groundwater levels during a 38-year underground mining period (8 years for underground infrastructure development prior to commencing block caving and 30 years of block caving operations).

Mine design parameters assumed a 700 m max height of draw. The development rate for the development of the decline (drill & blast) was set at 180 m/month. The cave propagation (draw) rate was assumed to be 60 m/yr. The mine production rate was set at 25 Mtpa. The extraction level layout utilised for the study was of “El Teniente” type with cross-cuts spaced at 32 m centres and draw points spaced at 20 m.

The economic criteria for the Cascabel project PFS Mineral Reserves Statement were as follows:

- Metal prices are $3.30/lb Cu and $1,700/oz Au, with 8% and 5% refining charges, respectively
- Average LOM concentrate grade was 28% Cu and 17 g/t Au
- Metallurgical recovery of 85% for copper and 70% gold
- NSR/grade units of $53.4/% Cu and $35.10 g/t Au

7 Geological Setting and Mineralisation

7.1 Tectonic Framework

The Cascabel project lies within the Western Tectonic Realm (WTR) of Ecuador and Colombia, according to Cedial et al. (2003), and within the Cordillera Occidental of Northern Ecuador. The Western Tectonic Realm of Ecuador and Colombia comprises the three composite terrane assemblages: the Pacific assemblage (PAT), Choco arc (CHO) and Caribbean terranes (CAT). Within the Pacific composite terrane assemblage (PAT), there are three terranes, from east to west: the Romeral (RO), Dagua-Pinon (DAP) and Gorgona (GOR) terranes.

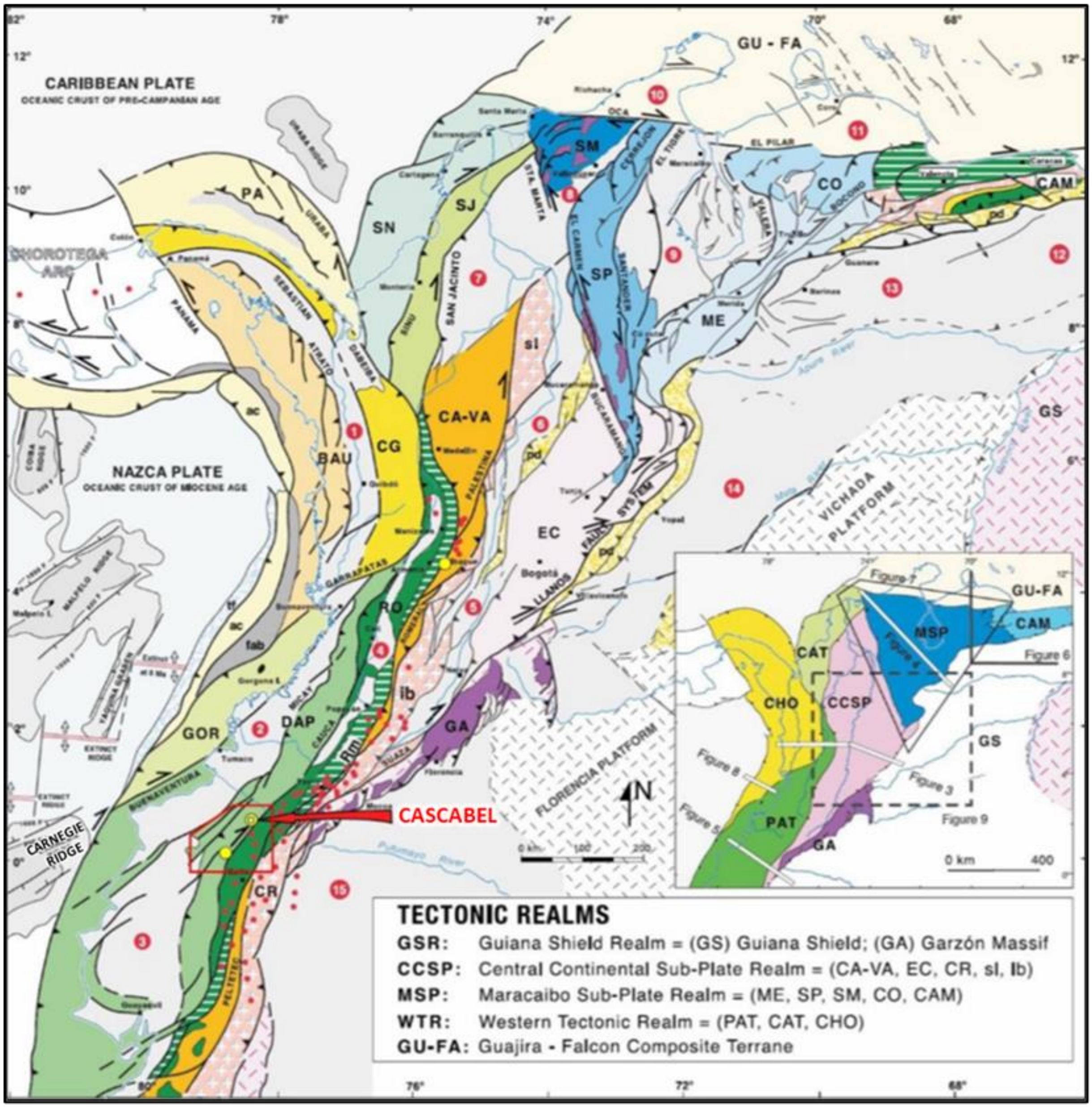
The DAP is correlated with the Pinon and Macuchi terranes of Western Ecuador (Figure 7-1).

The tectonic building blocks that comprise this northwest margin of South America are bound by north-northeast trending crustal-scale faults or sutures. Strike-slip structures are the dominant structural pattern. In the vicinity of the Cascabel project, the principal terrane boundary is the Cauca-Pujili fault system, which forms the suture between the RO and the DAP terranes. This is a significant fault system which, in detail, comprises several strands, several of which pass near or through the Cascabel project, such as the Toachi Fault Zone (TFZ). The presence of ophiolite along the extension of this fault zone, such as the Pujili Fault southwest of Quito, attests to its role as a major terrane suture. In northern Ecuador, the Cauca Fault is referred to as the Pujili Fault.

In addition to the long history of transpressive compression in the Cascabel region as recorded by the docking of the RO, DAP and GOR terranes, a more recent cause for ongoing tectonic compression (an important requirement for forming porphyry systems) is the shallow buoyant subduction of the Carnegie Ridge (post 8 Ma), whose eastward subducted projection is interpreted to extend beneath the Ecuador-Colombia border and underlie the area of the Cascabel project (Bourdon et al., 2003).

The leading edge of the subducted portion of the Carnegie Ridge lies beneath the northernmost portion of Ecuador, including Cascabel. This buoyant, aseismic submarine ridge is inferred to have commenced subduction beneath Ecuador in the Late Miocene (Gutscher et al.,1999), which is after the formation of the Eocene Alpala Deposit.

This magmatism in northern Ecuador and southern Colombia is characterised by the lack of a well-developed arc and erratic pluton distribution. This suggests a low-angle subduction environment, conducive to compression and porphyry mineralisation. There is a general eastward migration of magmatic focus from the DAP terrane to the RO terrane, suggesting a final approach of the Gorgona oceanic plateau as subduction progressively shallowed due to the increasingly buoyant nature of crust entering the subduction zone.



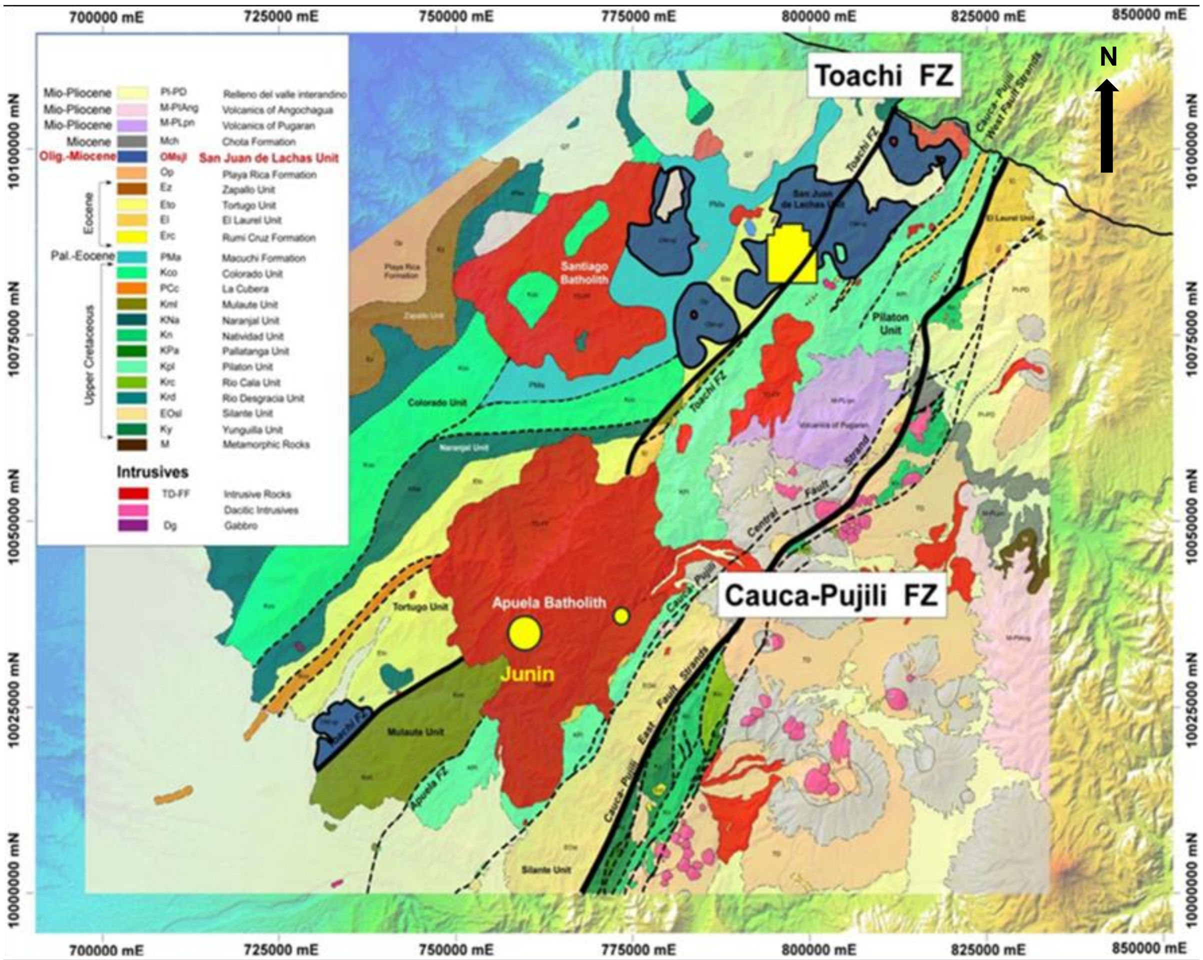
Source: Cedial et al., 2003
Note: Showing Carnegie Ridge and the Cascabel Deposit Area
Figure 7-1: Regional tectonic elements of northern Ecuador and Colombia

7.2 Regional Geology

The Eocene Alpala Deposit lies in a zone of overlap between the Eocene and Miocene Andean porphyry belts that extend from Colombia through Ecuador and Peru into Chile and Argentina. The basement rocks consist of tholeiitic basalts of the Dagua-Piñon Terrane, an oceanic plateau believed to have accreted to South America in the Late Cretaceous. The magmatism in northern Ecuador and southern Colombia is characterised by the lack of a well-developed arc and erratic pluton distribution. This suggests a low-angle subduction environment, conducive to compression and porphyry mineralisation.

Submarine arc volcanism deposited the volcano-sedimentary rock sequence of the Macuchi Formation during the Palaeocene through the Eocene, followed by the sub-aerial deposition of volcanic and volcaniclastic rocks of the San Juan de Lachas Formation during the Oligocene to mid-Miocene. Late Eocene to Miocene age plutons and stocks of hornblende-bearing diorite, quartz diorite and tonalite form major intrusive complexes known as the Santiago batholith (Eocene) and Apuela batholith (Miocene). The Alpuela batholith hosts the Late Miocene Llurimagua (Junin) Cu-Mo porphyry deposit (Gribble, 2004).

The TFZ is a major north-northeast trending structure that separates Eocene magmatism to the west from the Miocene magmatism to the east. The TFZ cuts through the Macuchi and San Juan de Lachas Formations and juxtaposes these sequences against Cretaceous sedimentary rock units (Figure 7-2).



Source: modified from Boland et al., 2000
Note: Showing location of the Cascabel concession (yellow polygon), Alpala Cu-Au-Ag deposit and Llurimagua (Junin) Cu-Mo deposit

Figure 7-2: Cascabel regional geology

The Cascabel project lies along the western foothills of the Western Cordillera. The Caucha-Pujili Fault zone is defined by the series of sub-parallel structures located midway between Otavalo and the Apuela Batholith. The TFZ is a significant structure that is sub-parallel to the Cauca-Pujili Fault Zone and cuts through both the Macuchi and San Juan de Lachas Formations and juxtaposes these sequences against

Cretaceous sedimentary rock units. The mapped extension of the northeast-trending TFZ, to the northeast of the Apuela Batholith, runs through the southeast corner of the Cascabel concession.

The Apuela Batholith sits astride the TFZ and has likely been intruded along the fault plane. This structure is consequently inferred to penetrate to or near the base of the crust, facilitating mid to upper-crustal emplacement of batholiths, including the Apuela Batholith.

The Apuela Batholith comprises a nested series of intrusions that include quartz porphyry, granodioritic porphyry and diorite porphyry, all of which are different intrusive facies of the larger composite batholith.

Situated at the margins of a large regional batholith and centred at the confluence of a prominent northeast-trending fault zone, the location of the Alpala Porphyry Copper-Gold-Silver Deposit, is controlled by the first-order tectonic structure, the TFZ, with a northwest trending second order tectonic structure, typical of the location of a number of porphyry deposits globally.

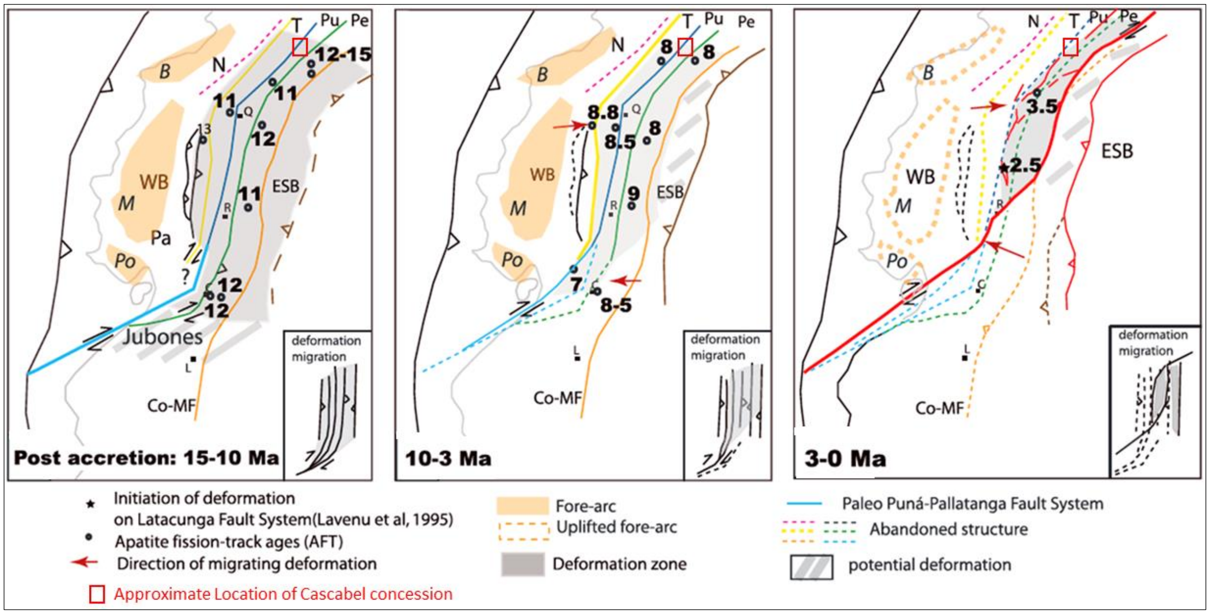
The geology of the Cascabel project on the northwest side of the TFZ comprises a series of relatively small to modest-size stocks. These small stocks, together with the abundance of Tertiary andesitic volcanic and volcaniclastic rocks, suggest limited amounts of erosion since intrusion emplacement. In contrast, to the southeast of the TFZ, the distribution of the intrusions is more extensive. There is an abundance of Cretaceous sedimentary rocks, which suggests a deeper level of erosion than to the northwest.

7.3 Regional Structural Framework

The Alpala Deposit is interpreted to have been broadly controlled by primary northeast-southwest to north northeast-south southwest structural systems with reverse dextral strike slip kinematics and secondary northwest-southeast normal fault systems.

The Cenozoic tectonics of Ecuador and Colombia involved successive oblique collisions of allochthonous marine arcs, leading to shortening strains and formation of dextral strike slip faults. Oblique collision along the plate margin has partitioned into a plate margin parallel component (northwest-southeast) and a plate margin normal component (northeast-southwest). Tectonic reconstructions of Ecuador and Colombia show relative plate motions, collisions of allochthonous terranes, and the creation of strike-slip bounded blocks (Cediel et al., 2003).

At approximately 39 Ma, when the Alpala Deposit formed, plate convergence was likely to have been occurring at a highly oblique angle. Since the mid-Miocene, plate convergence has established an overall oblique contractional strain in northern Ecuador, and the net result is that northeast striking faults such as the TFZ may have either dextral strike slip shortening displacements or simply have oblique slip (Alvarado et al., 2016) (Figure 7-3).



Source: Alvardo et al., 2016

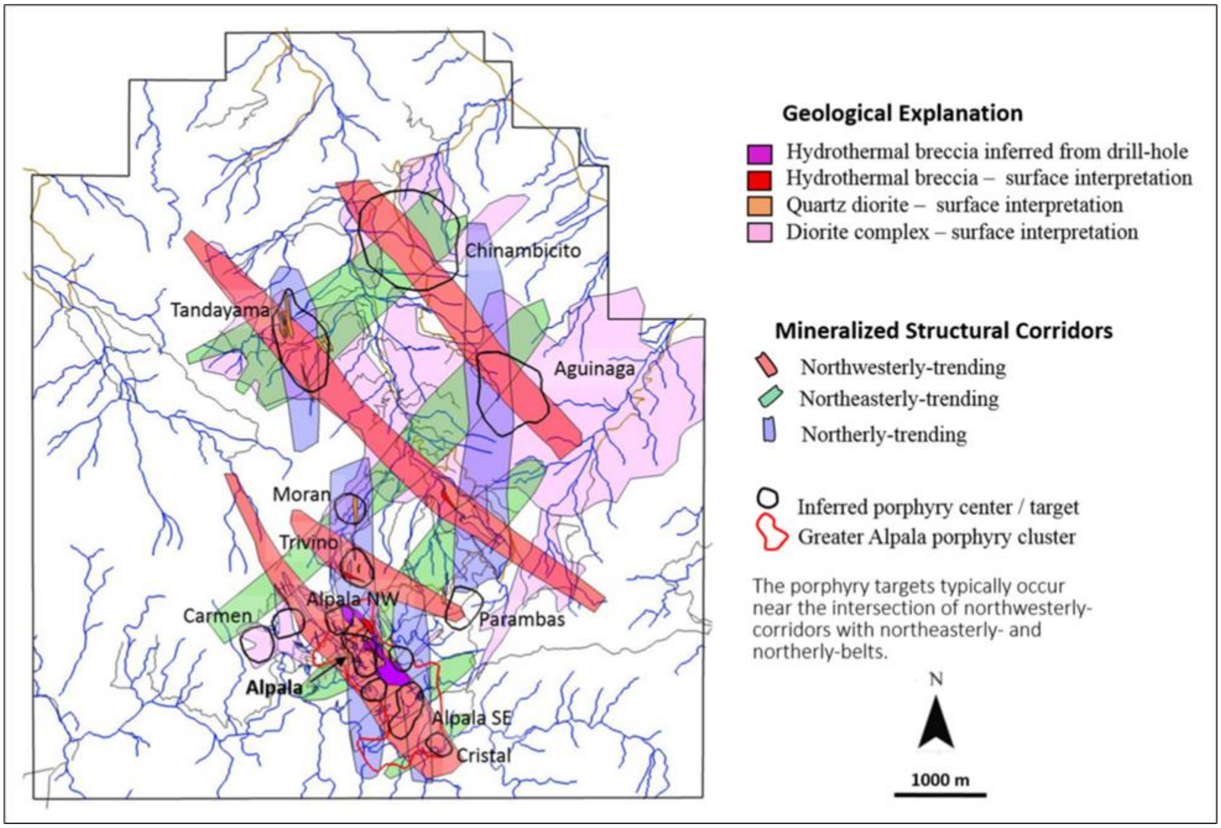
Figure 7-3: Structural Model of the Evolving Accretionary Arc

The Alpala Porphyry Copper-Gold-Silver Deposit lies adjacent to the TFZ, which is considered a significant fault of regional extent that accommodated collision and strike-slip deformation.

The superposition of Cretaceous rocks over and adjacent to the early Tertiary Macuchi rocks along the TFZ suggests the potential for significant vertical displacement. However, the displacement magnitude remains unknown to date. Strain partitioning along the oblique convergent plate margins results in competing northeast-southwest and northwest-southeast oriented maximum stresses.

7.4 Local Structural Framework – Cascabel Licence

An initial structural interpretation for the tenement area recognised several corridors of Cu-Au mineralisation determined by combining multiple datasets, including regional magnetics, electrical surveys, soil geochemistry, alteration mapping, topographic expression and the 1:500 to 1:2,000 scale mapping of Cu-bearing quartz veins, sulphide veinlets and fractures, as summarised by Garwin et al. (2017) (Figure 7-4).



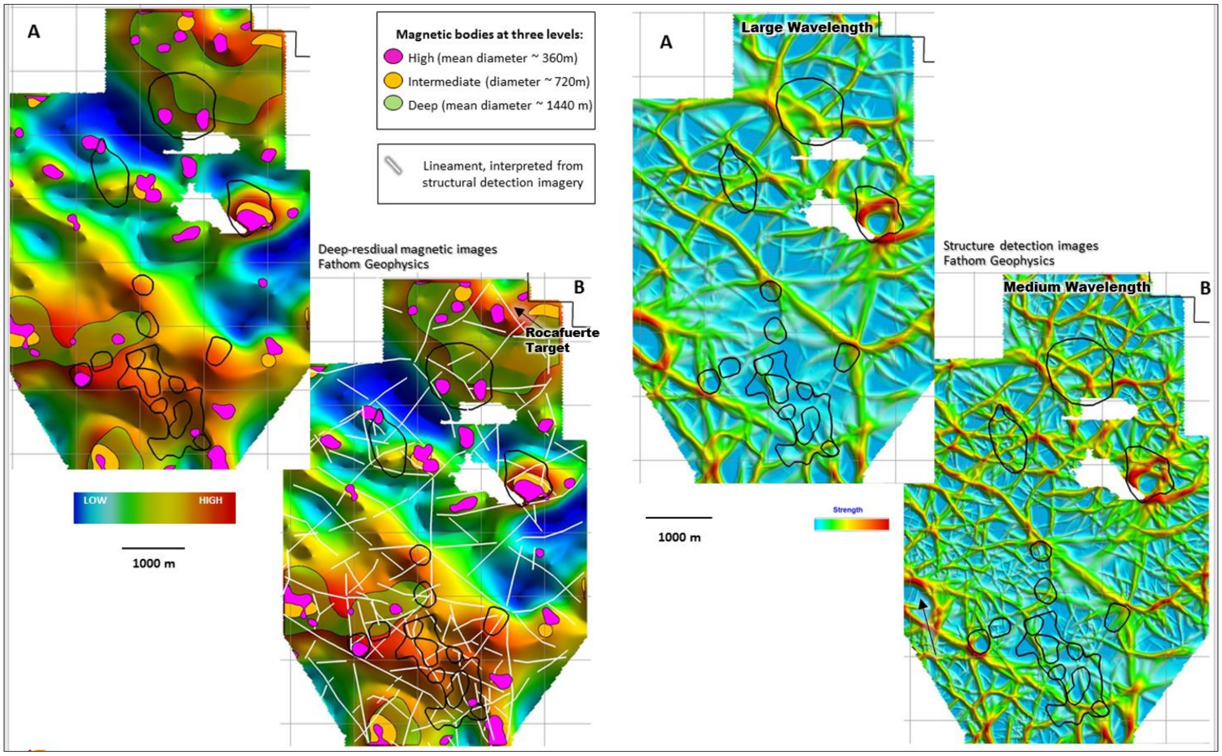
Source: Artica et al., 2022

Figure 7-4: Mineralised structural corridors, Cascabel

Mapping was completed using the Anaconda Method as described by Brimhall et al. (2006), Garwin et al. (2017) and Garwin (2018). Three major veinlet and fracture orientations exist, northwest, north, and northeast, similar to the orientations expressed by the intrusions and faults.

A total of 15 Cu-Au targets have been delineated from the results of geological mapping, soil and rock chip geochemical anomalies and magnetic expression. Many of the targets lie near the intersection of the mineralised corridors.

This mapping work was further enhanced by specialist reprocessing and filtering of ground magnetic data by Fathom Geophysics to provide Deep Residual Magnetics and Structure Detection Imagery (Figure 7-5).



Source: Artica et al., 2022

Figure 7-5: Deep residual magnetics and structure detection imagery by Fathom Geophysics

Interpretation of key structures within the concession areas revealed first order arc parallel northeast trending faults, such as the TFZ, second order arc normal northwest trending faults, such as the Alpala Structural Zone (APZ), and third order north and north-northeast trending faults.

The majority of the prospective porphyries within the Cascabel tenement occur at the intersection of first-order northeast to east-northeast trending faults, with second-order northwest-trending faults and third-order north and north-northeast trending faults. This supports the hypothesis that the northeast and northwest structural trends have deep-seated roots and may have provided a locus or crustal weakness for the ascension of mineralised porphyry stocks and their associated hydrothermal fluids in the Cascabel area.

Northwest trending structures exhibit important controls on intrusion emplacement and the localisation of mineralisation at intersecting northeast trending structures at Alpala.

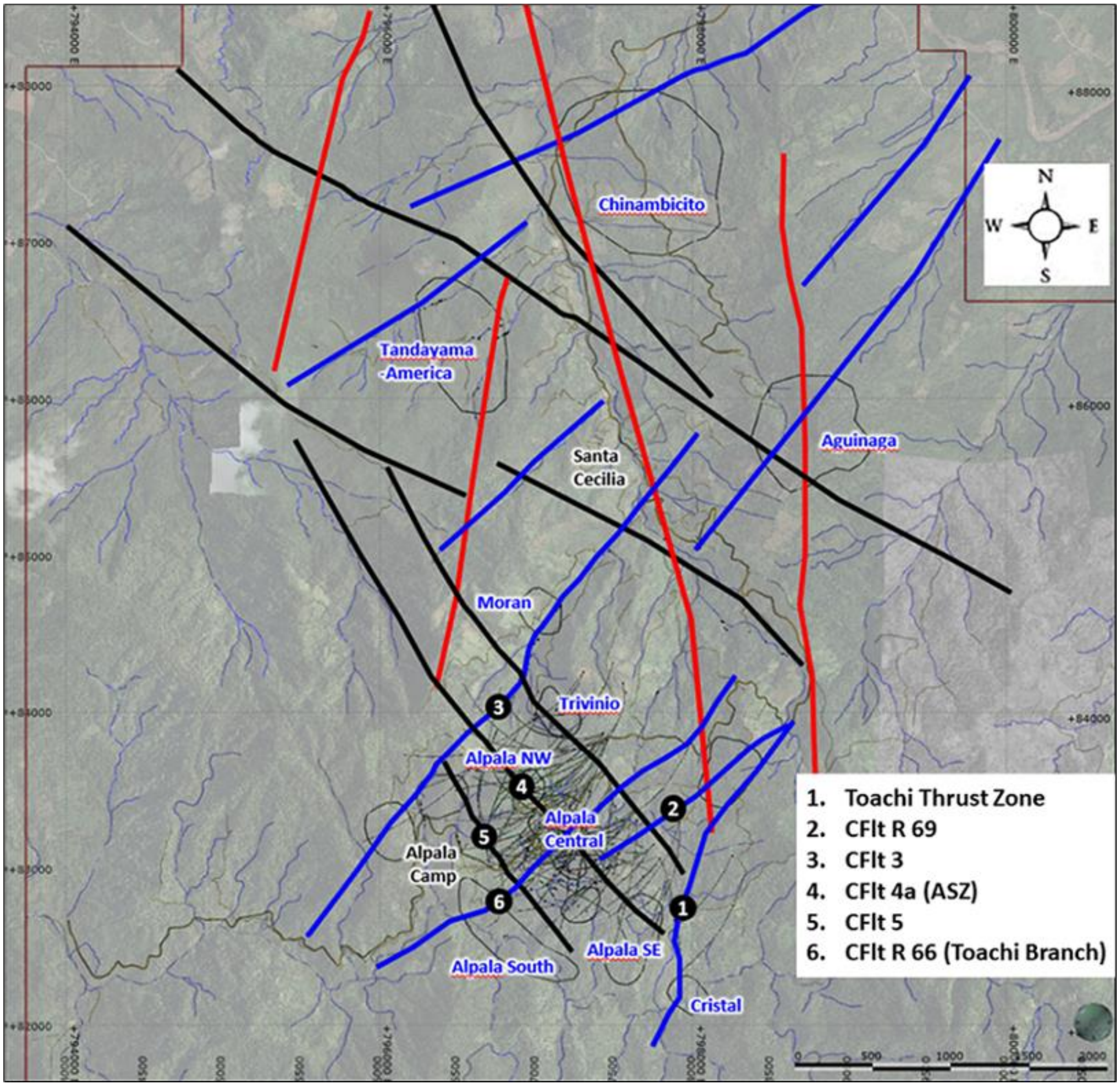
The dominant northwest trend is conspicuous in numerous datasets, including:

- Surface mapping of B-vein abundance
- Surface mapping of sulphide mineral abundances and chalcopyrite to pyrite ratio
- Zonation of clay mica alteration evident from mapping and spectral alteration mapping (which marks the root of the Alpala Lithocap)
- The measurement of downhole B-vein orientations shows a dominant cluster of average planes to maxima in a northwest-trending steeply northeast-dipping orientation

- Regional stream sediment Au anomalism also appears to be crudely controlled by northwest-trending structures
- Independent cross-section and level plan interpretations

Third-order north northwest to north northeast trending structures, evident in several of the above datasets, are also considered to play an important role in emplacement of intrusions and localising mineralisation.

Combining datasets from geological mapping, geochemistry, geophysics, and remote sensing identified a primary northwest-trending mineralisation trend likely related to deep-seated arc-parallel feeder structures with secondary north and northeast-trending structures. The southeastern corner of the tenement is dominated by a sequence of basement rocks uplifted along the TFZ. (Figure 7-6).



Source: Artica et al., 2022
Note: Showing Key Structures at Alpala that have been Interpreted Kinematically. The interpreted structures are colour-coded by strike orientation: blue is northeast to east-northeast, black is northwest to west-northwest, and red is north-northwest to north-northeast.

Figure 7-6: Key structures identified within the Cascabel tenement

The Alpala Deposit is located within a relay in the northeast to east-northeast trending TFZ, and because of the structural setting, there may be complex kinematic and slip histories. Regional tectonic interpretation by SolGold in 2019 identified three likely key kinematic constraints:

1. Oblique contractile strain along northerly oriented structures (TFZ), transitional to more Oblique extensional strain in more east striking segments (e.g., CFlt R 69)
2. Overall dextral strike-slip shear along northwest trending structures (e.g., ASZ, CZFlt5, CFlt4a) such that blocks could possibly rotate or tilt between bounding northwest striking faults

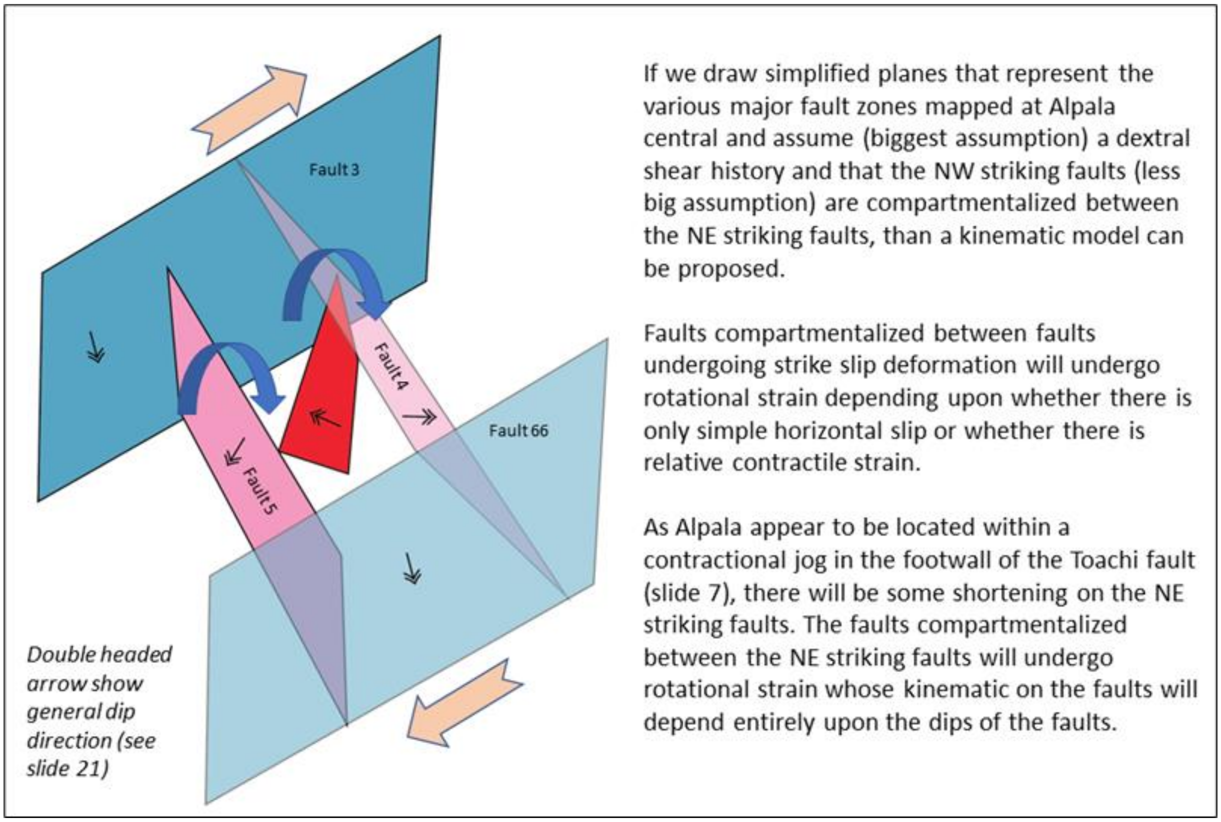
3. Strike-slip along northeast striking structures (e.g., CFlt R 66, CFLT3) such that dextral separation predicts the possibility that the Alpala Deposit and Moran porphyry target may be offset across the mapped fault CFlt 3. Such faults could possibly represent branch faults ahead of the main TFZ.

Constraints to a kinematic model generally include geometry, timing relations, and any kinematic indicators for slip at Alpala, including:

- The northeast striking fault should be considered part of the dextral TFZ. Evidence for shortening, as well as oblique dextral kinematics, has been viewed along various strands of the fault system.
- Kinematics of the northwest striking faults are elusive, and there is very limited preservation of fault rocks that suggest the extensional type of shear.
- Currently, in the fault model, Faults 4 and 5 appear confined between northeast striking faults. However, there is topographic evidence that this fault orientation may have a regional context, and thus, these faults may extend beyond the northeast striking fault. Whether this is true or not cannot be answered due to the lack of data.
- Furthermore, there are no constraints upon the timing of fault movement, but it has presumably been complex.

Hence, what remains is a limited data set that has a few kinematic constraints and largely depends upon known and inferred geometric constraints as well as the larger-scale regional tectonic setting of northern Ecuador and adjacent Colombia. Relying solely on geometry to construct a kinematic model is fraught with the potential for poor interpretation. Hence, the model sketched on the following slide should be viewed with a certain degree of skepticism until adequate data becomes available to confirm, disprove, or modify the current model.

In the context of the regional tectonic interpretation, it is likely the area has undergone an overall post-mineralisation dextral shear, displaying the following kinematics (Tosdal, 2019) (Figure 7-7).



Source: Tosdal, 2019

Figure 7-7: First pass kinematic model for possible post-mineral deformation of the Alpala deposit

Throughout the region and in the immediate area of the Alpala Deposit, numerous northwest striking faults are mapped, which are interpreted as pre-existing anisotropies that may extend largely into the basement and conceivably represent early Mesozoic rift-related faults that are common throughout the Central Andes.

This structural fabric predates the formation of the TFZ and likely reflects a basement fabric that may have controlled the formation of the porphyry magmatic complex and has formed passively to dextral shear. However, reactivation of northwest striking faults seems certain due to the conspicuous reappearance of dominant northwest trending features in the majority of geological datasets at Alpala.

Stratigraphic correlation and structural reconstruction work indicate the deposit has been tilted approximately 10 degrees toward the southwest between bounding northwest faults.

7.5 Local Stratigraphy – Cascabel License

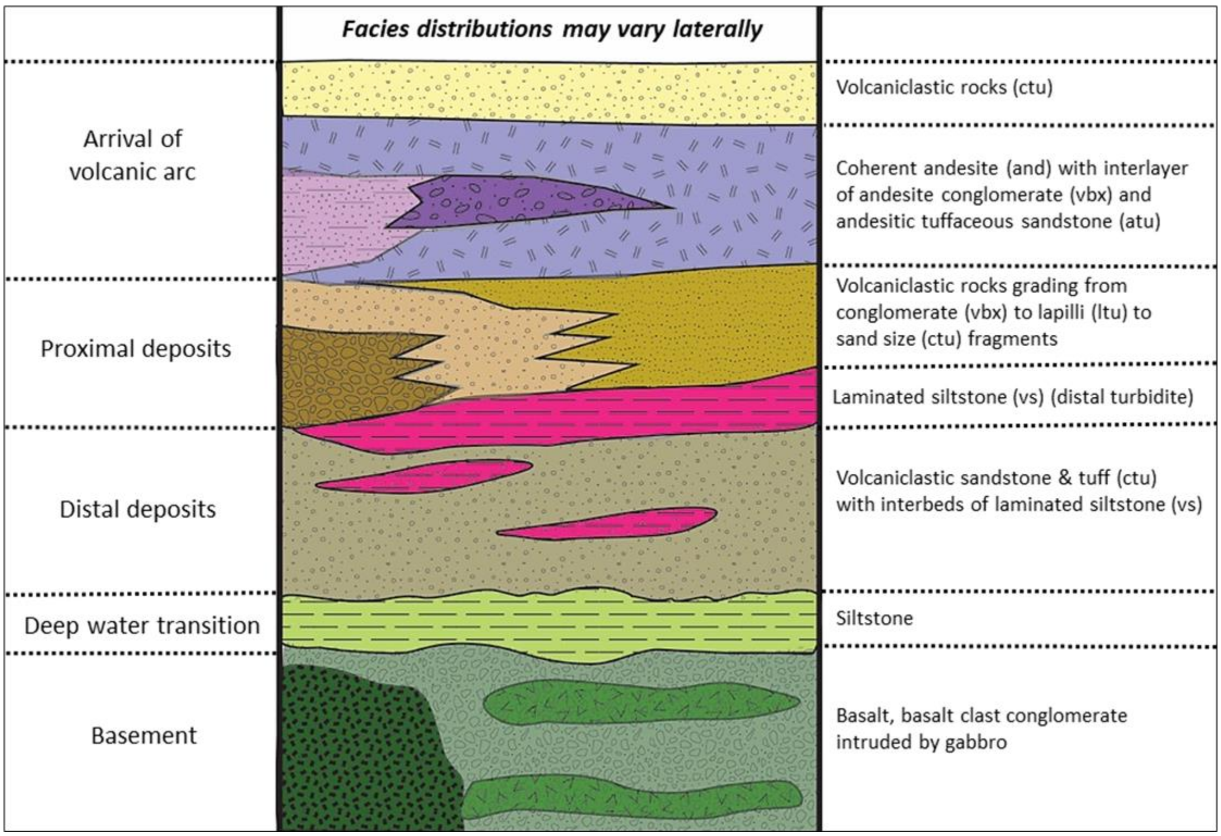
Three critical constraints form the foundation for a robust understanding of the geometry of a porphyry Cu-(Au) deposit:

- The geometry of porphyry intrusions
- The orientation of veins within and around each intrusive phase

- The geometry of the host rock stratigraphic units

To understand the geometry of the host stratigraphy at Alpala, general bedding orientations measured in the outcrop and the drill core were reviewed, as well as stratigraphic continuity of units across the deposit, the presence of marker horizons either in outcrop or in the subsurface data logged in drill core. This work was interrogated in a three-dimensional (3D) viewer and plotted on a stereonet.

No obvious marker horizon(s) occur(s) within the host stratigraphy at Alpala as it comprises a very complex volcanic, volcano-sedimentary, and sedimentary sequence where rapid and irregular lateral facies changes are common. Horizons of andesite (and), andesite tuff (atu), and well bedded volcano-sedimentary rocks (vs) provided the basis for establishing a simplified stratigraphic column (Figure 7-8).

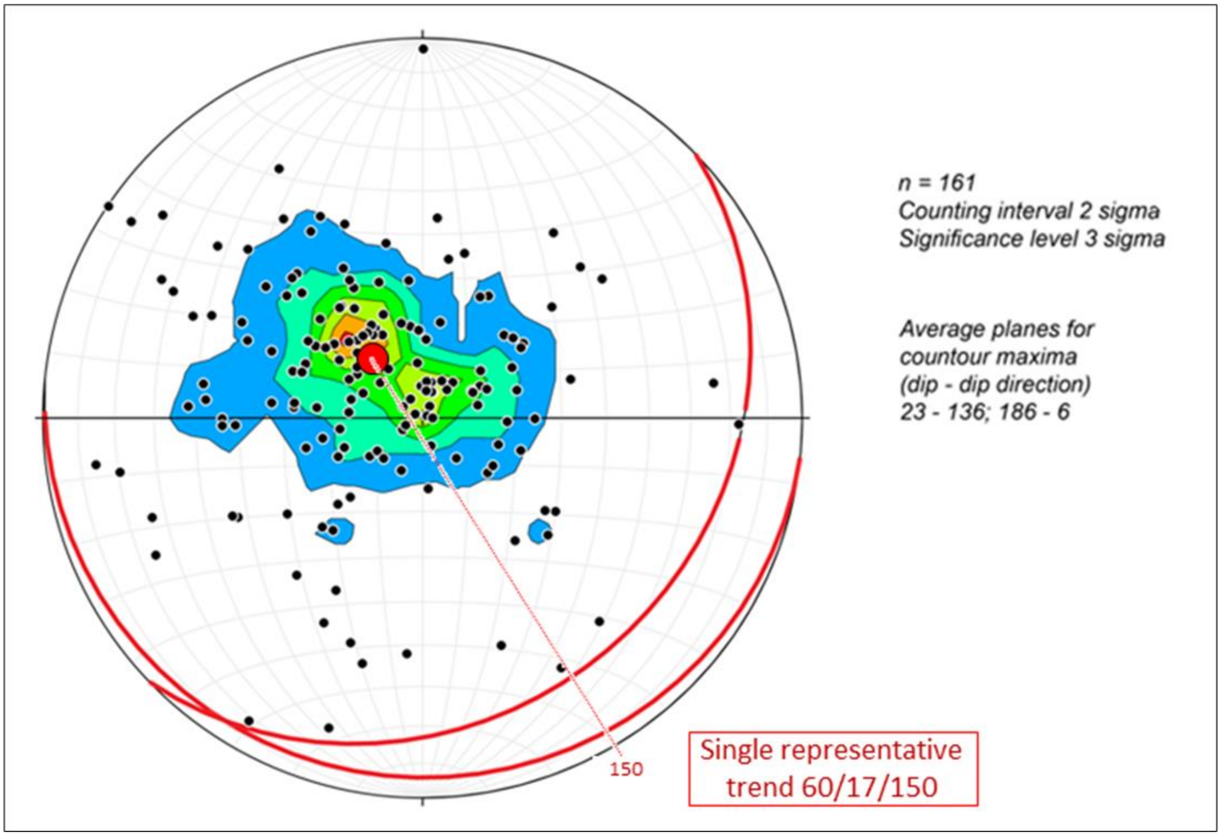


Source: Tosdal, 2019

Figure 7-8: Simplified stratigraphy of the Alpala deposit

Through the careful determination of different stratigraphic units and cross-referencing against bedding orientation data, an approximate average bedding trend was established, with a shallow dip towards the southwest (17° degrees towards 150°). The stratigraphy at Alpala appears to strike approximately parallel with the TFZ.

An overall geometry for the stratigraphic package was established using bedding orientation data from core logging. Bedding orientations measured in volcano-sedimentary rocks in both drill core and surface outcrop were filtered to composite multiple core measurements over intervals of less than 20 m, which revealed two very close maxima. The pole to a single representative maxima (060/17/150), favouring the primary trend group, was selected for 3D modelling (Figure 7-9).



Source: Tosdal, 2019

Figure 7-9: General orientation of stratigraphy hosting the Alpala deposit

This shallow southeast dip may represent an approximate paleo-horizontal surface. Deviations from this geometry occur within the dataset and can be explained where stratigraphy is locally more steeply dipping proximal to faults. The paleo-horizontal is further supported by the lack of deflection in bedding against intrusive bodies, and it is interpreted that the intrusions did not deform country rocks to any great extent upon emplacement.

At this stage, it appears that the modelled stratigraphy from the completed drilling demonstrates 3D continuity within fault blocks bounded by the northwest-trending, steeply southeast-dipping CFlt 5 and CFlt 4a faults. However, the pole to the postulated paleo-horizontal is not normal to the pole of the general trend of the Alpala Intrusive Complex, which does not support the idea that the entire area may have been tilted approximately slightly towards the southwest. The geometry of the stratigraphy at Alpala is, as yet, in a draft stage. It requires further detailed work to define the stratigraphic column fully and its potential offset by post-mineral faults.

Stratigraphic reconstruction work to date does, however, support the interpretation of a regional structural framework that indicates a possible overall dextral strike-slip shear along northwest trending structures (e.g., ASZ, CZFlt5, CFlt4a) such that blocks could possibly rotate or tilt between bounding northeast striking faults.

The geometry of the stratigraphy and resultant reconstruction work at Alpala is in the early stages. Further, detailed work is underway to fully define the stratigraphic column and its potential offset by post-mineralisation faults.

7.6 Local Geology – Cascabel Licence

The major rock types of the Cascabel tenement consist of Cretaceous siltstones and minor sandstones, which are unconformably overlain by a Tertiary sequence of andesitic lavas and volcano-sedimentary rocks. A series of hornblende-bearing diorites, quartz diorites and tonalities intruded the volcano-sedimentary sequence as plutons, stocks, and dykes (Figure 7-10).

Three of these intrusions have been dated by SolGold, utilising the SHRIMP U/Pb zircon method, which indicates results of approximately 39 Ma for the Alpala and Moran porphyry centres and approximately 37 Ma for the Aguinaga porphyry centre (Armstrong, 2015 and 2016).

Dykes, faults, and fracture zones mapped in the area typically strike northwest, north and northeast. The volcano-sedimentary host rock package at Cascabel has been mapped regionally by previous workers (British Geological Survey and Corporación de Desarrollo e Investigación Geologico, Minero y Metalúrgico, 1997) to belong to the Oligocene to Miocene San Juan de Lachas Formation. However, this unit is interpreted by SolGold and ENSA geologists to form part of the submarine to transitional emergent, Palaeocene to late Eocene Macuchi Formation (Vallejo, 2007), based on the volcano-sedimentary facies recognised in drill core and the late Eocene ages of intrusions at Alpala and Aguinaga. Recent work by M.Sc. student Carlos Diaz (as yet unpublished) revealed an age of approximately 41 Ma for the volcano-sedimentary host rocks at Alpala.

7.7 Alpala Deposit Geology

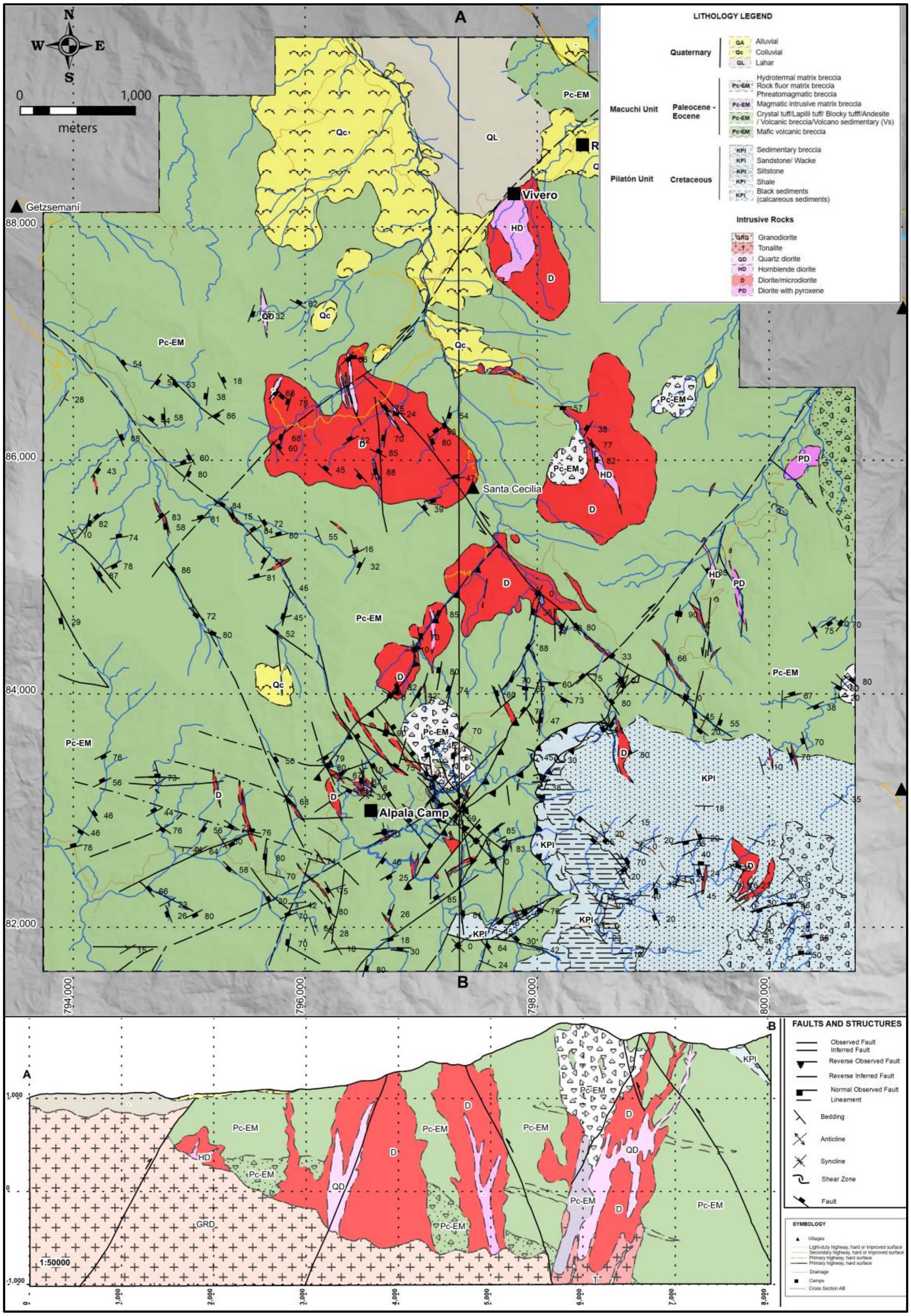
Major host rock types of the deposit consist of gabbroic and basaltic basement rocks, overlain by Cretaceous siltstones and minor sandstones that are unconformably overlain by a sequence of Tertiary volcano-sedimentary and andesitic lavas. The volcanic and volcano-sedimentary sequences that host the deposit were previously mapped as the Macuchi Unit (Vallejo 2007), a volcanic arc of tholeiitic and calc-alkaline composition which was formed on oceanic crust during the Eocene.

This sequence has been intruded by a series of Middle to Late-Eocene (Bartonian) hornblende-bearing diorites, quartz diorites and tonalities that form plutons, stocks, and dykes.

Drilling has defined a northwest-trending, steeply northeast-dipping corridor known as the Greater Alpala Trend. This trend is centred upon a syn-mineralisation causal quartz-diorite intrusion (QD10) cut by a series of intra-mineralisation, late-mineralisation, and post-mineralisation stocks, dykes and breccias of diorite, hornblende diorite, quartz diorite, tonalite and granodiorite. Intrusions have been emplaced episodically such that each subsequent intrusion has introduced mineralising fluids (notably as porphyry-type quartz and quartz-sulphide veins) into the Alpala porphyry system, and/or remobilised existing mineralisation or contributed to localised overprinting and destruction of the pre-existing mineralisation.

Thin-section petrography reveals the presence of very fine-grained quartz in the groundmass of the intrusions, which suggests compositions that range from quartz diorite to tonalite. However, the intrusive rock types have been classified on observations made by the field geologist using a 20 times magnification hand lens (Garwin et al., 2017).

Two of the Alpala intrusions have been dated by the sensitive high-resolution Ion Micro Probe (SHRIMP) U/Pb zircon radiometric dating method, which indicates an approximate age of 39 Ma for the intrusions (Garwin et al., 2017).



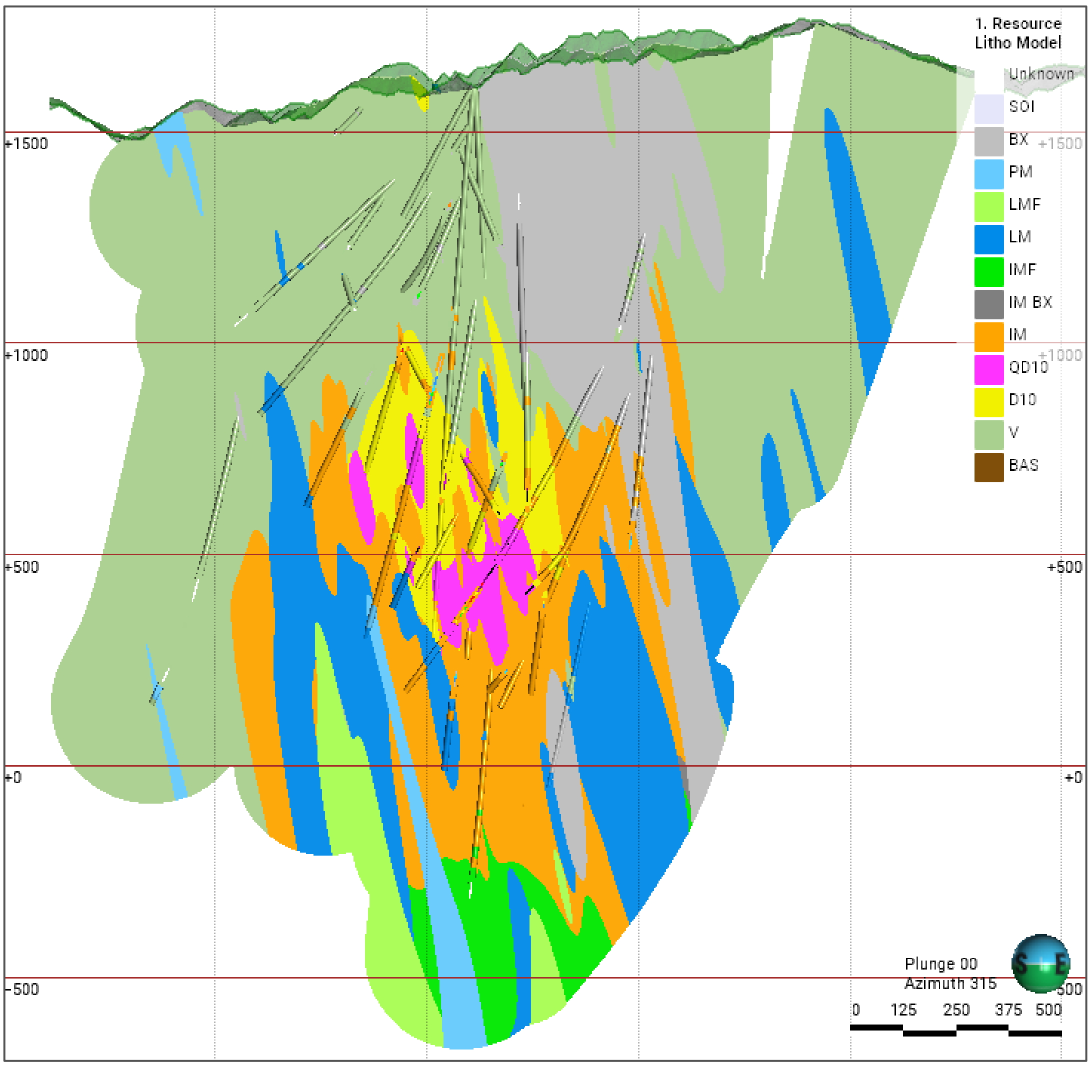
Source: Artica et al., 2022

Figure 7-10: Geology and structure of the Cascabel project area

Intrusions are typically emplaced with a stock-like geometry that is moderately elongate in a northwest direction. Intrusions often hold typically vertically and laterally extensive northwest trending, steeply dipping dyke extensions beyond their stock margins.

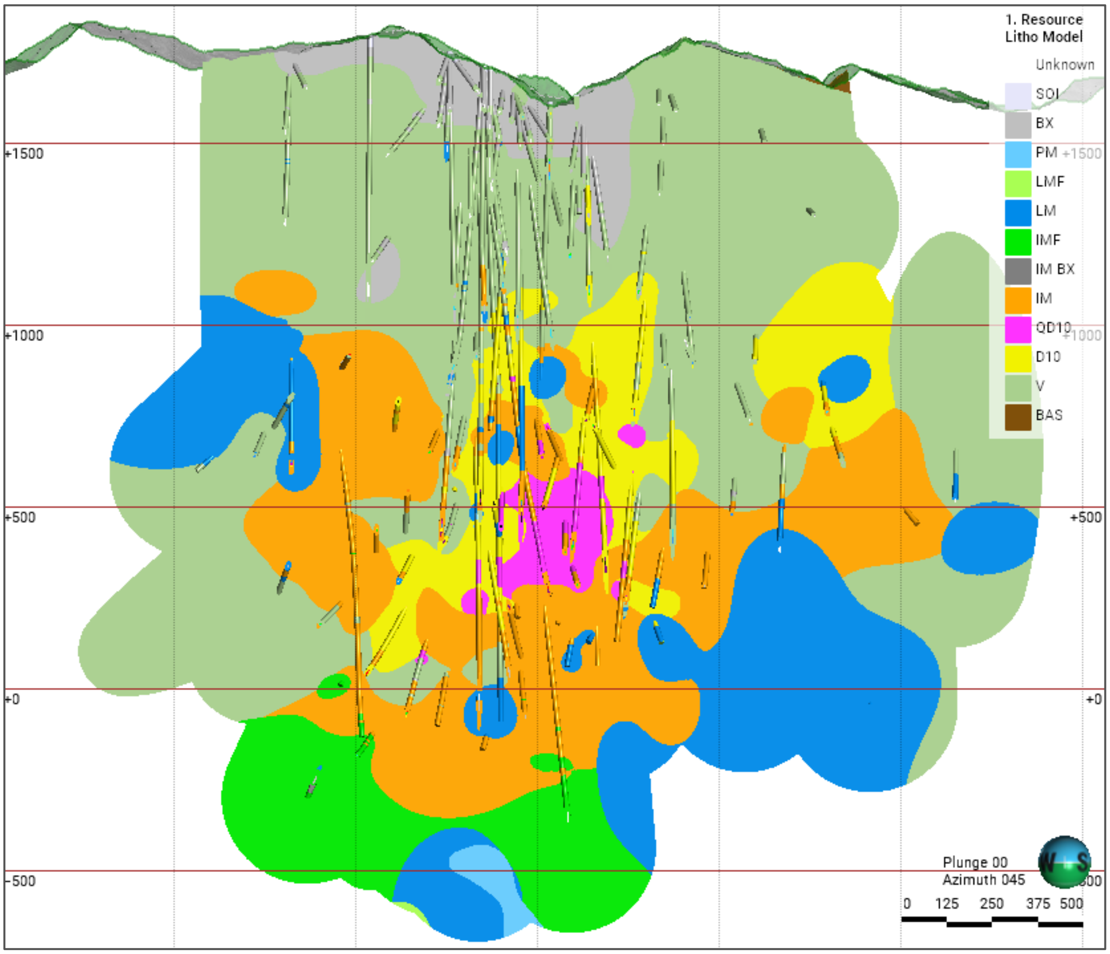
Due to the multi-episodic nature of the complex, several dykes appear rootless or 'hanging' in section view. However, these are seen to connect to the main stock or stocks in three dimensions or have been intruded and truncated by younger stocks.

The geometry of the various lithologies and intrusive bodies at Alpala is now well understood. It has been modelled from the completed drilling, demonstrating extensive sub-vertical continuity and highly complex intrusive relationships (Figure 7-11 and Figure 7-12).



Source: Artica et al., 2022
Note: Looking Northwest, along the trend of the Alpala intrusive complex, centred on a 100m wide viewing window, and highlighting the extensive vertical geometry and episodic nature of the intrusions

Figure 7-11: Vertical cross-section through the centre of the Alpala deposit



Source; Artica et al., 2022
Note: Looking West-Southwest, Centred on a 100m Wide Viewing Window, and Highlighting the Extensive Vertical Geometry and Episodic Nature of the Intrusions

Figure 7-12: Vertical long-section through the centre of the Alpala deposit

The application of the Anaconda method to geological mapping and drill core logging has identified a total of 31 likely intrusion phases. These intrusions have been grouped into 18 major rock groups which have been delineated based on their mineralisation phase and resource group. All the major rock groups have been incorporated into the three-dimensional geological models created of the Alpala deposit.

A total of 18 major phases of mineralisation have been delineated on composition, relative timing relationships and porphyry-related vein stages (Figure 7-13).

Mineralisation Phase	Resouce Group	Rock Group	Rock Phase	Rock Description	Rock Group Relative Timing	Rock Phase Relative Timing
Cover	NOT IN RES	SOI	SOI	Soil & Saprolite	350	350
		QT	QT	Quarternary	340	340
Post Mineral		FLT	FLT	Post Mineral Faults	330	330
		PM2	HD40	Post Mineral Hornblende Diorite	300	320
			D40	Post Mineral Diorite		310
			QD40	Post Mineral Qtz Diorite		300
		BX	RFBX	Rock Flour Matrix Bx	205	208
			UBX	Undifferentiated Late Bx		207
			HBX	Hydrothermal Matrix Bx		206
			IBX	Magmatic Intrusive Matrix Bx		205
		PM	GRD30	Post Mineral Tonalite	170	200
			T30	Post Mineral Tonalite		190
			D30	Post Mineral Diorite		180
			QD30	Post Mineral Qtz Diorite		170
Late-Mineral	LOW GRADE RES	LMF	G20	Granite	160	160
			T20	Late Tonalite	150	150
		LM QD	QD20	Late Mineral Qtz Diorite	140	140
		LM	HD20	Horblende Diorite	120	130
			D20	Late Mineral Diorite		120
Intra-Mineral	MED GRADE RES	IMF G	GRD15	Intra Mineral Granodiorite	115	115
	MED GRADE RES / HIGH GRADE RES	IMF T	T15	Intra Mineral Tonalite	110	110
		IM BX	IBXM	Intrusive Matrix Bx (Mineralised)	90	100
			HBXM	Hydrothermal Matrix Bx (Mineralised)		90
		IM	D15	Intra Mineral Diorite	80	80
		QD15	QD15	Intra Mineral Qtz Diorite	70	70
Syn-Mineral		QD10	QD10	Syn? Mineral Qtz Diorite	60	60
Pre-Mineral	HOST ROCKS	D10	D10	Pre Mineral Diorite	50	50
		V	V	Pre Mineral Volcanics	40	40
	BASEMENT	BAS	MFR	Gabbro, Basalt	30	30
			SED	Sst, Sltst, Mst, Lst		20
			MIBX	Mafic Magmatic Intrusive Matrix Bx		10

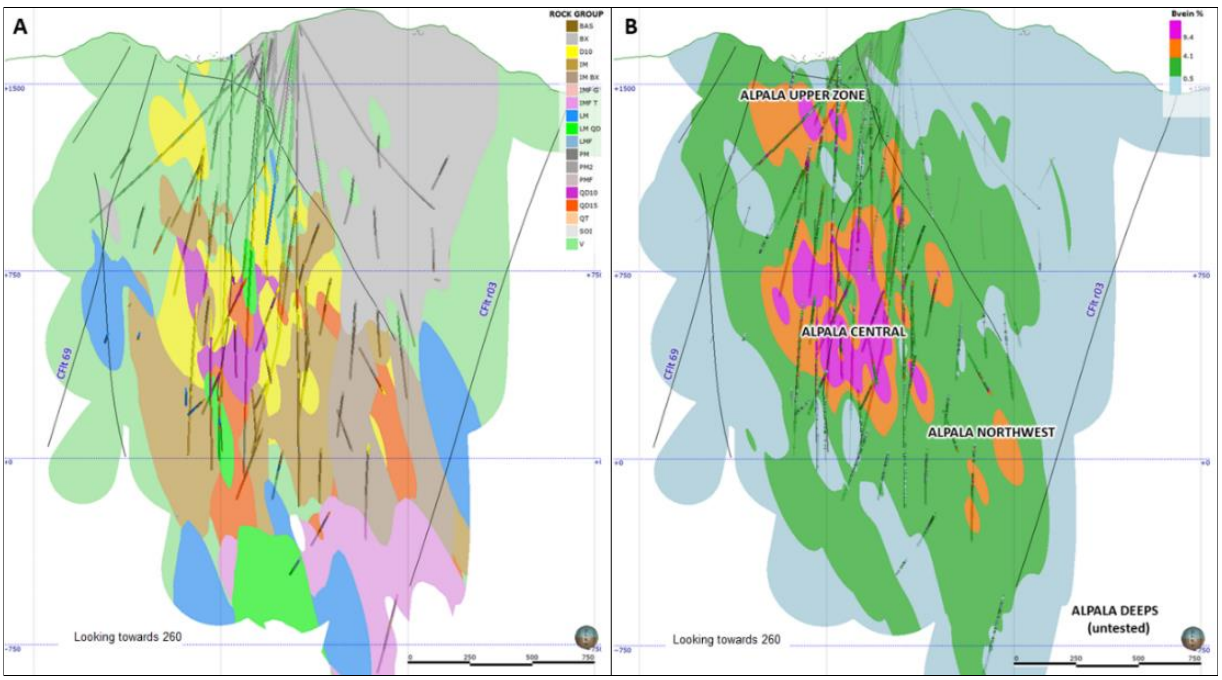
Source: Artica et al., 2022
Note: Codes form the Foundation of Geological Interpretation and 3D Modelling of the Alpala deposit, showing mineralised deposit forming porphyry phases inside red outline (QD10 to G20)

Figure 7-13: Geological modelling codes utilised for the Alpala deposit

These intrusive rock phases are simplified into 11 major rock groups for three-dimensional modelling of the deposit and form the major low-grade, medium-grade and high-grade stages of mineralisation at Alpala.

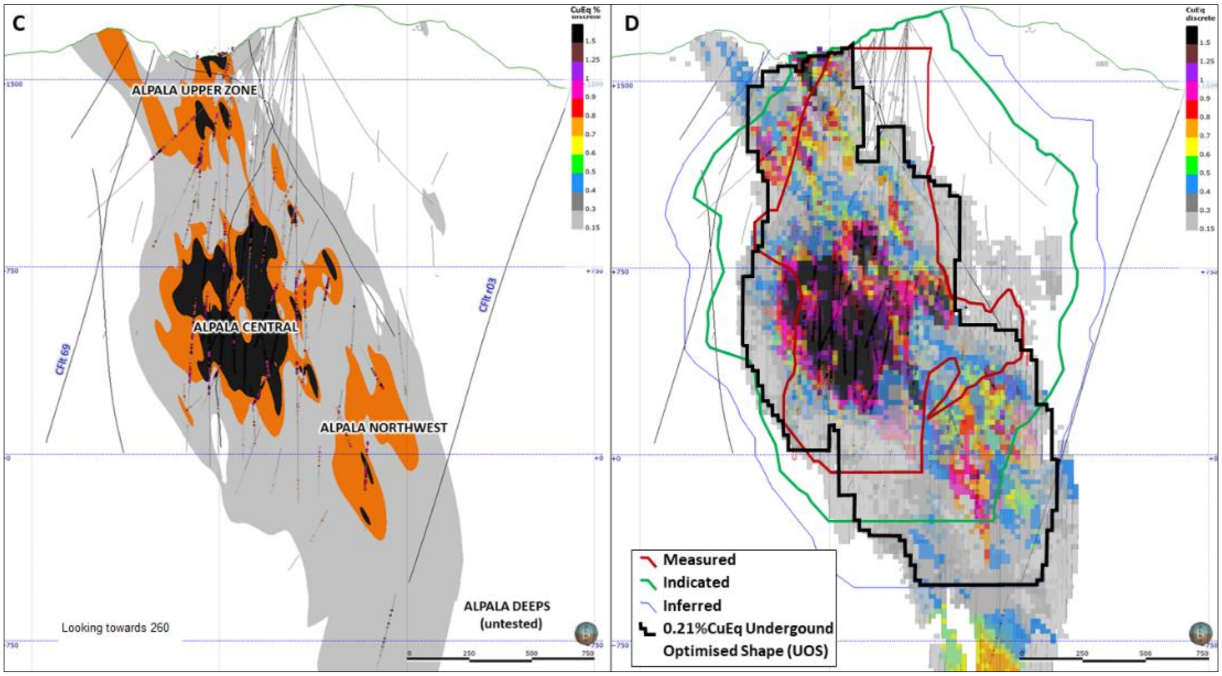
The majority of the Cu and Au mineralisation at Alpala was added to the system by the QD10 intrusion with supplementary additions through time with the injection of the intra-mineralisation QD15, IM, IM BX, IMF T intrusive phases, with only very minor metal addition from late-mineralisation and post-mineralisation intrusions as the system waned.

A key point in the understanding of the Alpala deposit relates to the manner in which the geometric framework of various lithologies and intrusive bodies subsequently controlled the successive geometries and zonation of the porphyry-style quartz vein abundance at Alpala, which in turn correlates with the distribution of Cu and Au (Figure 7-14 and Figure 7-15).



Source: Artica et al., 2022
Note: Looking West-Southwest, and Across the Trend of the Alpala Intrusive Complex, showing the geometric framework of intrusions (A), and high, medium and low abundance zones of porphyry-type quartz veins (B)

Figure 7-14: Example long-section through the centre of Alpala deposit



Source: Artica et al., 2022

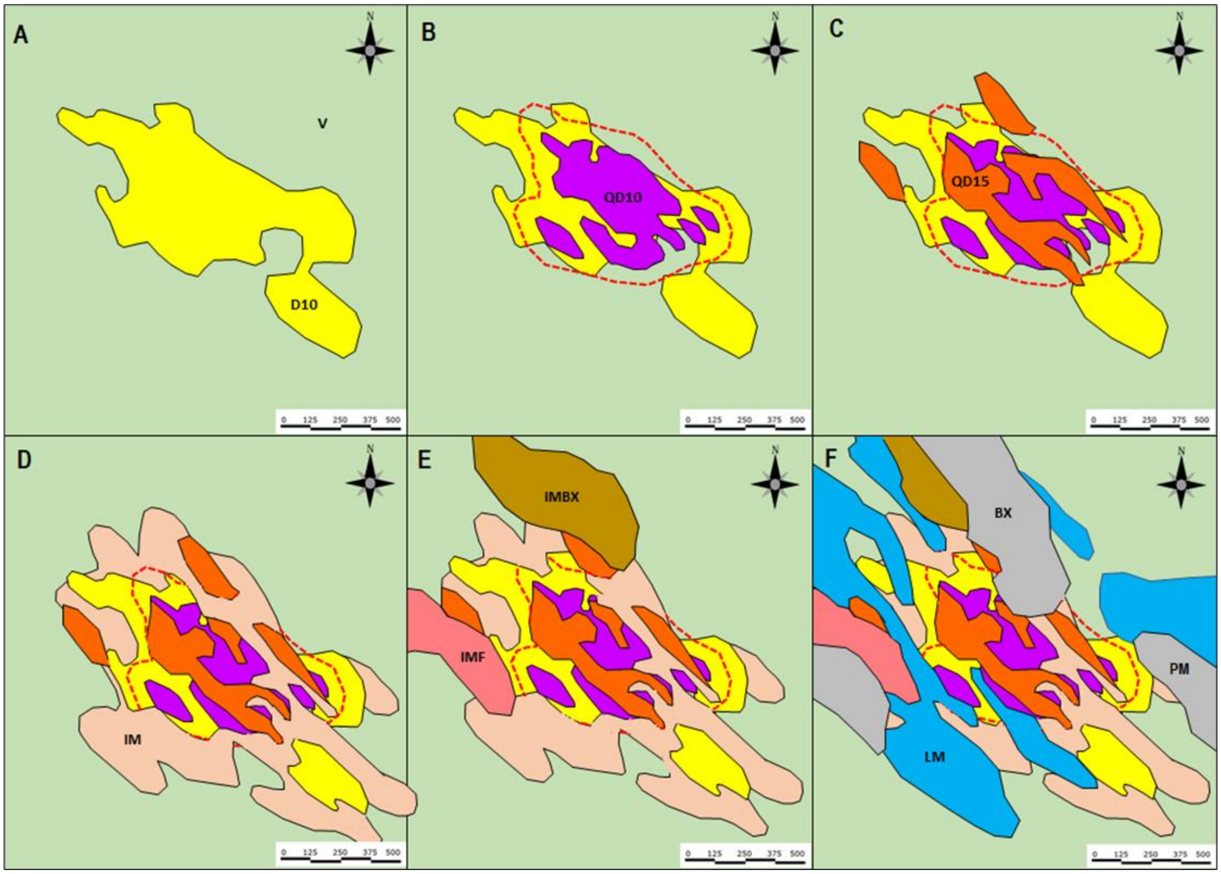
Note: Looking West-Southwest, and across the trend of the Alpala Intrusive Complex, showing excellent correlation with high, medium and low-grade zones (C), and block model grades above the 0.21% CuEq cut-off grade, classification shells, and the Mineral Resource Outline (MRO) limit (D)

Figure 7-15: Example long-section through the centre of Alpala deposit

Mineralisation at Alpala took place intermittently over approximately 800,000 years. Garwin et al. (2017) produced an accurate duration of the Alpala porphyry system of 800,000 years +/- 800,000 years at the 95th percentile confidence interval (2-sigma), which represents the difference in age from the early-mineralisation QD10 quartz diorite (39.4 ±0.6 Ma, SHRIMP U-Pb) to the emplacement of the late-mineralisation QD20 (38.7 +0.6 Ma, SHRIMP U-Pb) and formation of molybdenite in a D-type porphyry vein hosted within late-mineralisation D20 diorite (38.6 ±0.2 Ma, Re-Os).

Each of the 11 rock groups recognised (from QD10 to BX) sequentially added its own stage of porphyry-related veining and mineralisation to the Alpala Deposit. Interpretations of the deposit are built episodically by utilising Rock Group Relative Timing in the same manner that each subsequent intrusion has introduced mineralising fluids into the Alpala porphyry system and/or remobilised existing mineralisation or contributed to localised overprinting and destruction of the pre-existing mineralisation.

A simplified schematic intrusive and vein paragenesis model for the formation of the Alpala deposit has been developed (Figure 7-16).



Source: Artica et al., 2022

Figure 7-16: Simplified scaled schematic intrusive and vein paragenesis model for the Alpala deposit

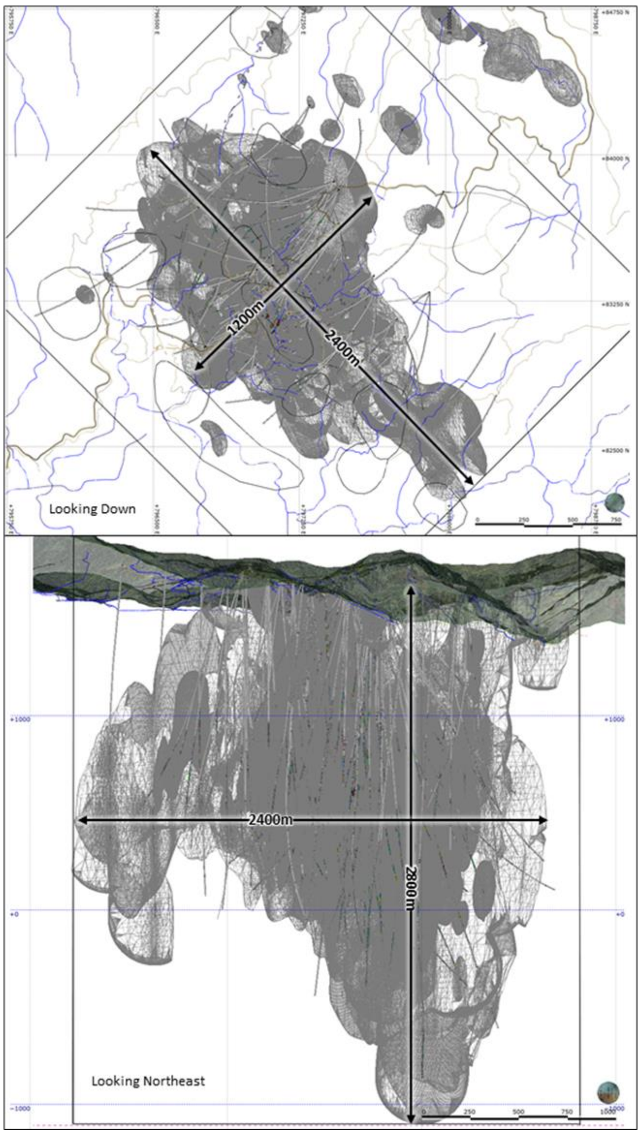
The earliest intrusion, the pre-mineral D10 diorite, was intruded into the host andesite volcanics (V) to form the host rock sequence for the deposit (Figure 7-16 A). The D10 intrusion probably intruded along a northwest trending structure (such as the ASZ), accounting for the elongated shape. The main phase of mineralisation was subsequently emplaced with the syn-mineralisation QD10 intrusion, resulting in a concentric zone of high-grade mineralisation marked by approximately greater than 10% B-type quartz-magnetite-chalcopyrite veining (Figure 7-16 B). The cupola of the QD10 dykes and stocks, when intersected by drilling, typically display Unidirectional Solidification Textures (UST). Mineralising fluids sourced from high-grade QD10 apophyses would further propagate upward into the D10 and volcanic host rocks. Due to the high density of these saline fluids, the mineralisation may have also intruded downwards along the apical margins.

A second weaker stage of mineralisation was introduced through later intra-mineralisation intrusions of QD15, IM, IM BX and IMF (IMF T and IMF G), which locally exploited the intrusive contacts of earlier intrusions and/or the re-activation of the previously significant structural framework (Figure 7-16 C, D, E). The intra-mineralisation group of intrusions crosscut the existing QD10 and its associated high-grade halo, leaving a second stage of weaker veining and intra-mineralisation. This intra-mineralisation intrusive complex was flanked by a Late-mineral (LM) group of intrusions followed by a final group of

Post-mineralisation intrusions, including a conspicuous late-stage hydrothermal breccia that continued through to the current topographical surface (Figure 7-16 F).

7.8 Alpala Deposit Mineralisation

The equigranular to sub-porphyritic, hornblende-bearing intrusions at Alpala are narrow, taper upwards, and are geometrically similar to grade models of Cu, Au and Ag mineralisation. Mineralisation occurs as a prolate body approximately 2,400 m northwest by 1,200 m northeast and 2,800 m in vertical extent, defined at a Cu equivalent (CuEq) cut-off criteria of greater than 0.15% CuEq and/or greater than 0.55% B-type quartz veins (Figure 7-17).



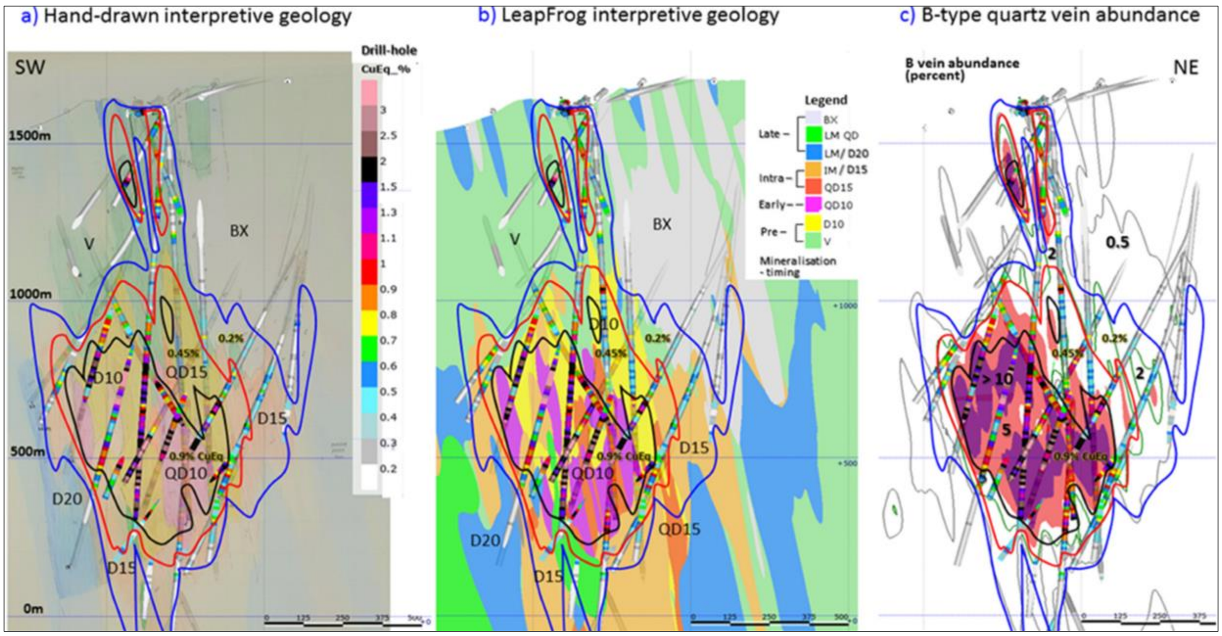
Source: Artica et al., 2022

Figure 7-17: Simplified scaled schematic intrusive and vein paragenesis model for the Alpala deposit

The porphyry-related vein types and mineralisation paragenesis at Alpala indicate a systematic progression in time and have been described by SolGold using the nomenclature originated by Gustafson and Hunt (1975).

Understanding the intrusive phase relationships within the Alpala Deposit was advanced following a detailed review of the drill core in 2014, which focused on identifying the different vein types and their paragenesis. This work resulted in advances in the recognition of inconspicuous intrusive contacts and clearly showed that each intrusive phase contains a specific set of veins and resultant Cu and Au grades.

The geometry of the intrusive units and porphyry style B-type quartz vein abundance zones correlates with Cu -Au grade distributions (Figure 7-18).



Source: Garwin et al., 2018
Note: Showing the relationships between rock units, porphyry style B-type quartz vein abundance, CuEq grade
Figure 7-18: Alpala deposit cross-section 82,950m N (window + 50m)

Planar and pervasive, B-type quartz veins crosscut the early vein types and consist of quartz-magnetite-chalcopyrite. At least two stages of B-type veins have been recognised, B1 and B2, with magnetite more abundant in early B1 veins and chalcopyrite more common in the later B2 veins. B-type veins contain the majority of the Cu and Au in the deposit.

Scanning Electron Microscopy (SEM) techniques indicate that the primary Cu minerals are chalcopyrite and bornite. Gold occurs as discrete grains of electrum (typically 65% to 85% Au) that range from one to 50 microns in diameter. The electrum grains occur within chalcopyrite, bornite, pyrite and rarely quartz and anhydrite. Grains of low-Ag electrum (greater than 90% Au) that are 1 to 3 µm in diameter are associated with sulphide grains and occur locally within silicate minerals.

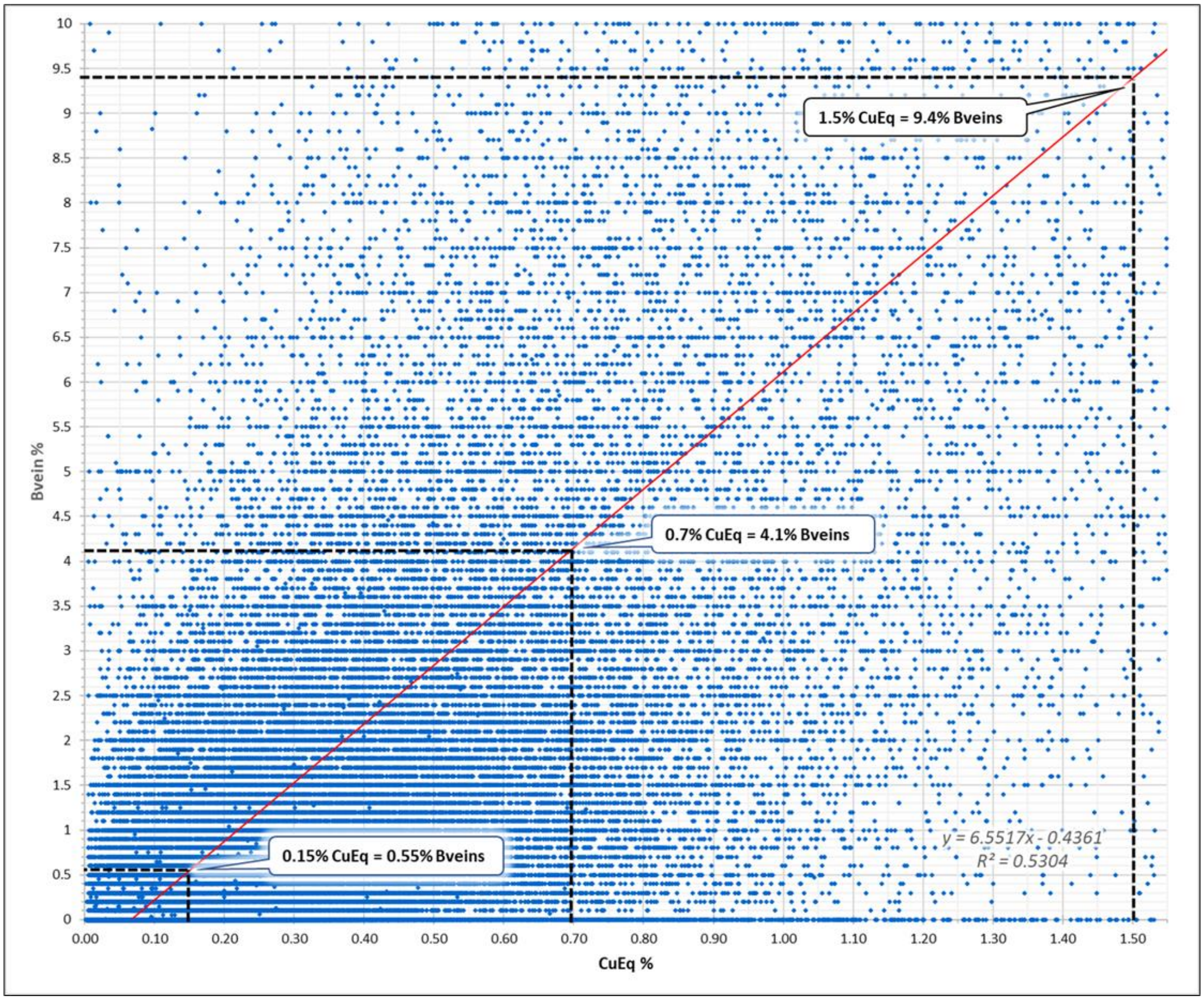
Chalcopyrite-rich, C-type veins contain rare to minor bornite and cross-cut earlier vein types. C-type veins contain significant amounts of metal but constitute a small volume of the drill core. B-type and

C-type veins are spatially associated with intrusions that show variable feldspar destruction and sericite-chlorite-clay overprinting of biotite-actinolite and chlorite-epidote alteration mineral assemblages.

Late-stage, pyritic D-type veins with quartz-sericite-pyrite selvedges contain chalcopyrite, minor bornite and, locally, molybdenite. Many of the later vein stages exploit and re-open earlier vein stages, as does anhydrite. Transitional to late stage, anhydrite-bearing veins are inferred to form a halo to the deposit core. Late-stage hydrothermal matrix breccia bodies and volumetrically small igneous matrix breccias, including pebble dykes, typically postdate sericite – chlorite ± clay alteration and are locally cut by pyritic D-type veins and anhydrite veins. The breccia bodies cut the volcanic host rocks and the pre-mineralisation, early-mineralisation, and intra-mineralisation intrusions.

A Re-Os age date determined by a commercial laboratory on molybdenite in a D-type pyrite-chalcopyrite-bearing anhydrite-quartz vein that cuts a late mineralisation D20 diorite dyke indicates an age of 38.6 ± 0.2 Ma (2 σ). The age dates of the QD10, QD20, and late-stage molybdenite are no different statistically.

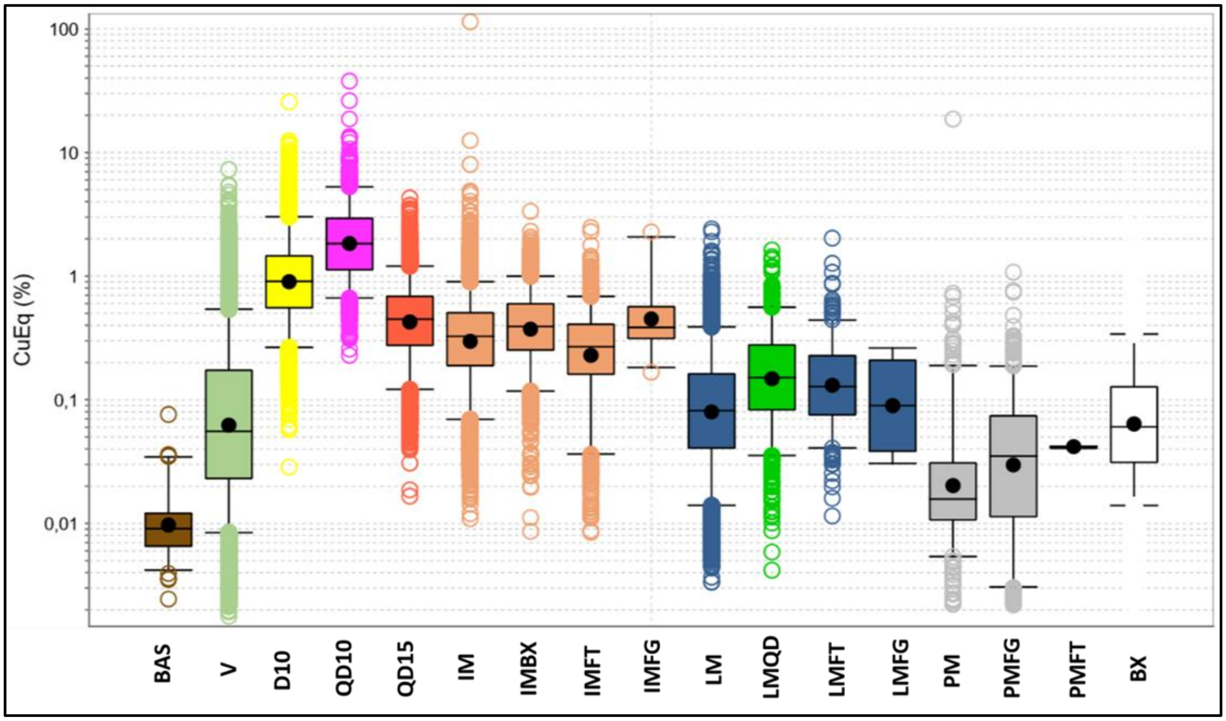
The relationship between B-type quartz vein (B-vein) abundance and Cu, Au and CuEq grades throughout the deposit show considerable scatter. However, a linear relationship has been defined with selected examples of 0.15% CuEq, 0.70% CuEq and 1.50% CuEq, equating to 0.55% B-veins, 4.1% B-veins and 9.4% B-veins respectively (Figure 7-19).



Source: Artica et al., 2022

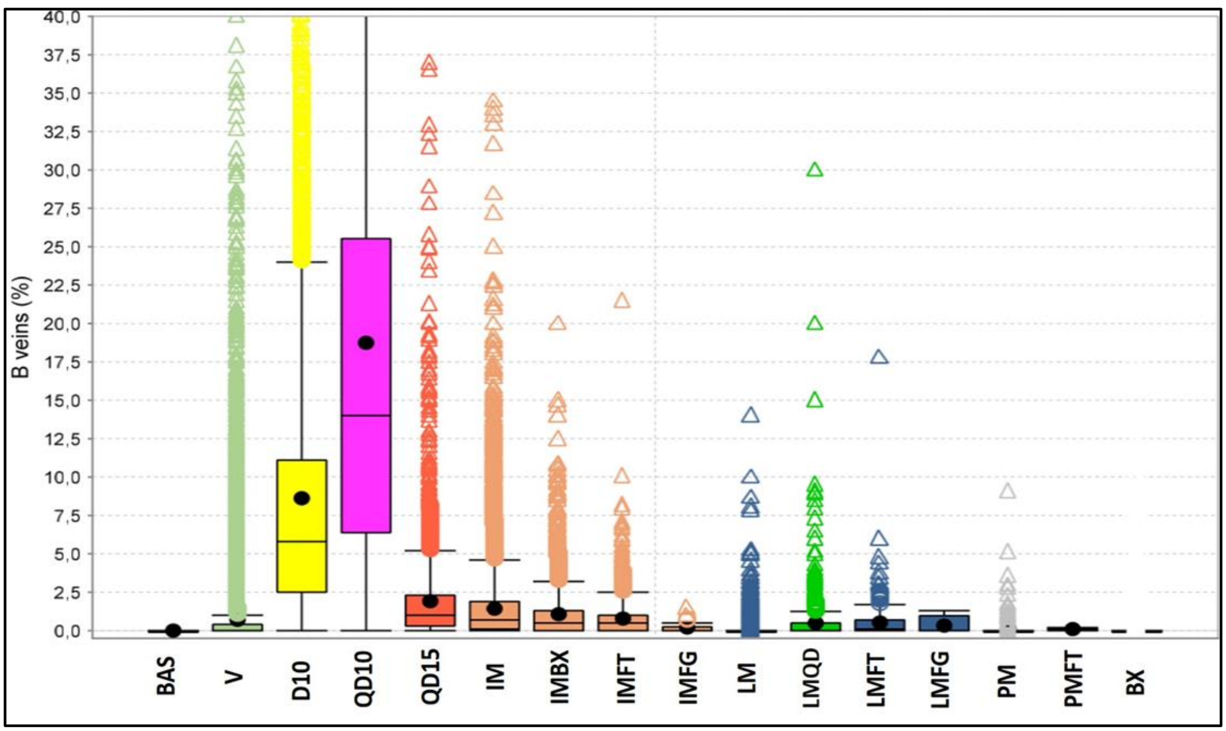
Figure 7-19: Linear relationship between B-type quartz vein (B-vein) abundance and Cu, Au, and CuEq grades at the Alpala deposit

The most important indicators of high-grade mineralisation include the presence of an early-stage Quartz Diorite intrusion (QD10) containing all early-stage porphyry style vein types with elevated vein abundance with an increased ratio of chalcopyrite to pyrite. These relationships are clearly evident when representing B-type quartz vein abundance and CuEq grades against each rock group in box and whisker plots (Figure 7-20 and Figure 7-21).



Source: Artica et al., 2022

Figure 7-20: Box and whisker plot showing average CuEq grade within each major rock group at the Alpala deposit

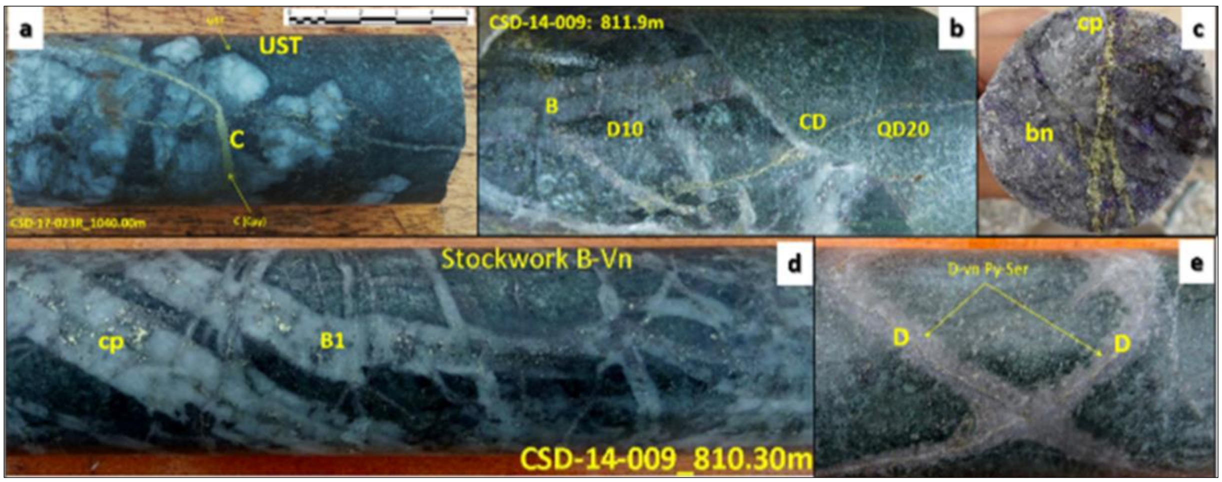


Source: Artica et al., 2022

Figure 7-21: Box and whisker plot showing B-type quartz vein abundance within each major rock group at the Alpala deposit

Figure 7-22 displays some examples of the most important vein types recognised in the Alpala deposit, which are:

- a) Prismatic quartz, showing unidirectional solidification texture (UST), cut by a chalcopyrite-rich C-vein
- b) Intrusive contact between late QD20 and early D10, showing truncation of early B-veins and a late CD-vein that cross-cuts the contact
- c) Chalcopyrite vein and late-stage bornite along fracture surface
- d) Magnetite bearing B1 quartz vein stockwork with clots of chalcopyrite (Cp)
- e) Late-stage pyritic D-vein with selvedges of quartz-sericite



Source: Artica et al., 2022
Note: Showing examples of the most important vein types recognised in Alpala
Figure 7-22: Photographs of drill core from the Alpala deposit

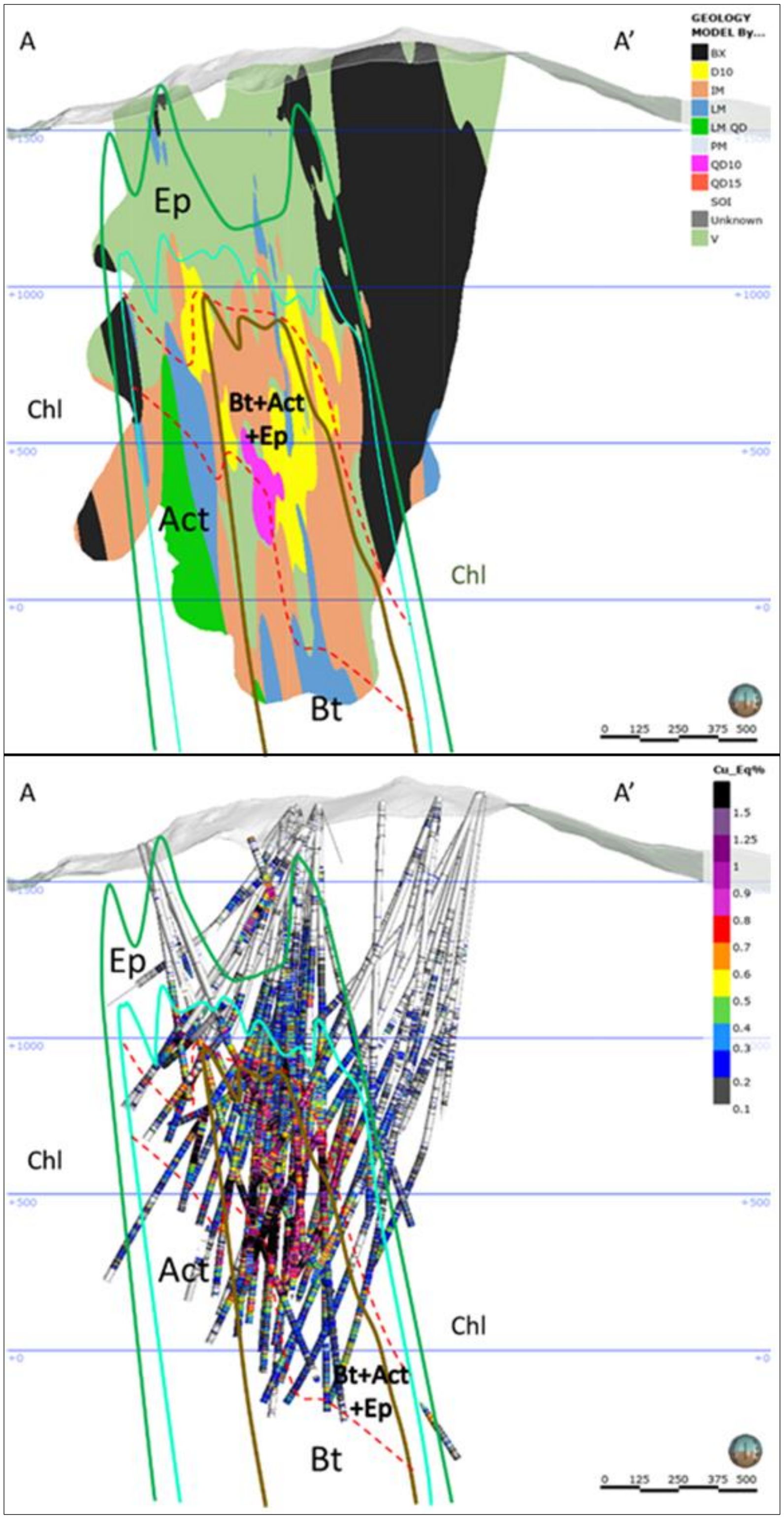
Early-formed, hydrothermal magnetite occurs within early AB- and B1-type veins, and as monomineralic veinlets, disseminated grains and replacements of hornblende. Magnetite is variably converted to metallic hematite and pyrite in the upper part of the deposit where chlorite-epidote altered intrusions and volcaniclastic rocks are moderately to strongly affected by feldspar-destructive alteration. The earliest formed Cu-sulphide minerals observed in the drill core consist of abundant chalcopyrite and rare bornite in B-type veins. Chalcopyrite most commonly forms after and surrounds cubic and massive pyrite in C-veins and D-type veins. It also occurs in anhydrite-rich veins and B-type veins that have been re-opened by later vein types. Late-stage bornite is in textural equilibrium with pyrite and chalcopyrite in C-type and D-type veins, which suggest that these later-stage veins formed at a lower temperature and a higher sulfidation state than chalcopyrite in early-stage B-type veins (Einaudi et al., 2003).

Scanning Electron Microscopy (SEM) techniques, including Backscattered Electron (BSE) imaging and Energy Dispersive X-ray Spectroscopy (EDS), indicate that gold occurs as discrete grains of electrum (typically 65% to 85% Au) that range from 1 to 50 microns in diameter (Muhling, 2014, 2015 and 2018). The electrum grains occur within chalcopyrite, bornite, pyrite and rarely quartz and anhydrite. Grains of low-Ag gold (>90% Au) that are one to three microns in diameter are associated with sulphide grains and occur locally within silicate minerals (Muhling, 2017).

7.9 Alpala Deposit Hydrothermal Alteration

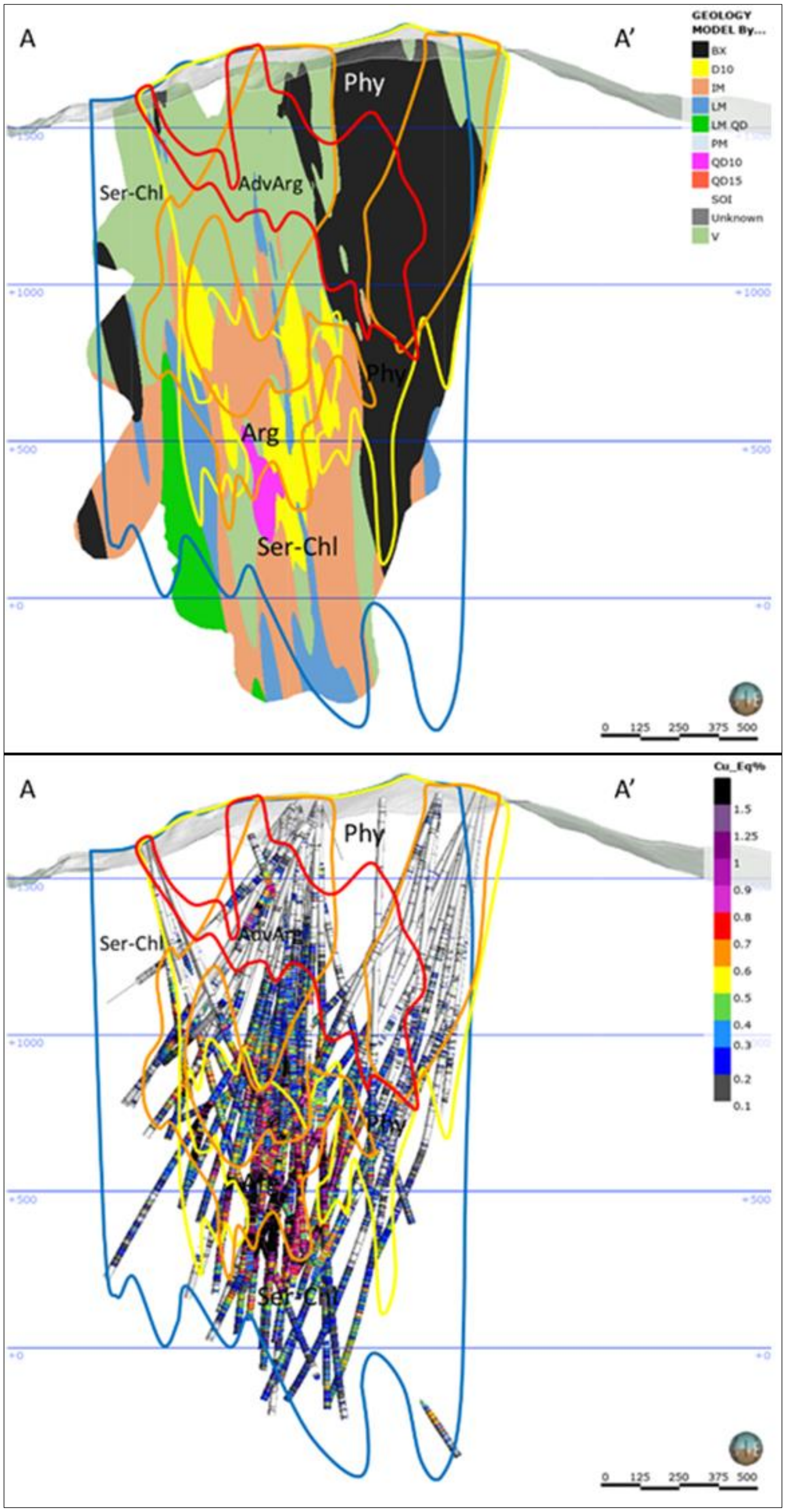
Distinct spatial and temporal relationships are evident between hydrothermal alteration and copper-gold mineralisation, most notably emphasised by the following major hydrothermal alteration features at the Alpala Deposit (Figure 7-23 and Figure 7-24):

- Intermediate argillic (chlorite-sericite-clay) alteration coincides with the majority of medium-grade to high-grade mineralisation (>0.7% CuEq) in the deposit, which is characterised by abundant chalcopyrite and lesser amounts of pyrite. The core of the deposit typically contains more than two percent hydrothermal magnetite over vertical intervals that exceed 300 m. Local zones of magnetite greater than 4% occur over vertical intervals of 50 m to 100 m.
- The high-grade core of the deposit, characterised by abundant chalcopyrite, coincides with the transitional-stage chlorite-sericite alteration that overprints the contact zone between early-stage potassic alteration and inner propylitic alteration. The combined silicate-mineral assemblage of this early-alteration zone consists of chlorite-biotite-actinolite + epidote.
- Late-stage quartz-sericite (phyllic), pyrophyllite-dickite (advanced argillic), and kaolinitic (argillic) alteration types are characteristic of the upper portions of the deposit, which contains abundant pyrite and lesser amounts of chalcopyrite and bornite.



Source: Artica et al., 2022

Figure 7-23: Early alteration phases at the Alpala deposit shown in section-view looking Northwest (window + 50 m)



Source: Artica et al., 2022

Figure 7-24: Late alteration phases at the Alpala deposit shown in section-view looking Northwest (window + 50 m)

7.9.1 Early Alteration

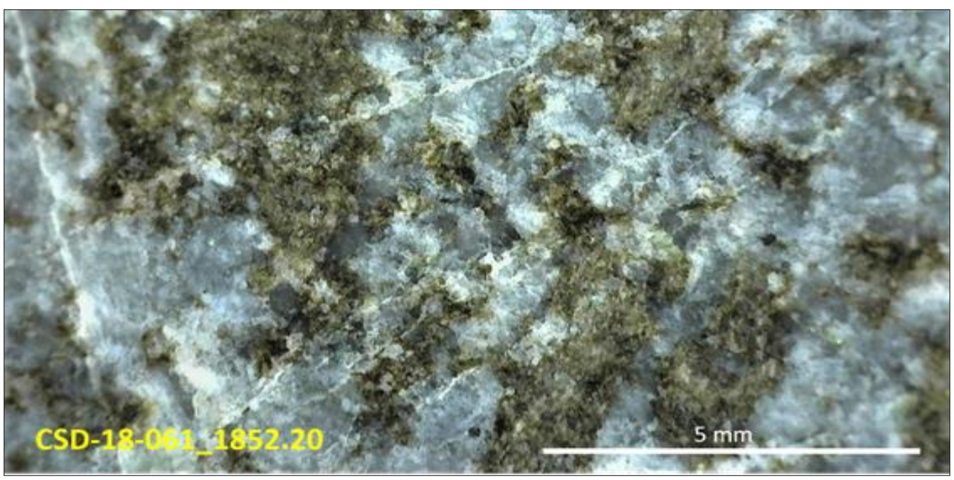
Early hydrothermal alteration occurs due to the dispersion of magmatic-hydrothermal fluids within and around porphyry intrusions. A common zonation from higher to lower-temperature minerals is spatially associated with the causal intrusion(s) margins.

In the Alpala deposit, early hydrothermal alteration assemblages develop in all intrusive phases and affect the volcano-sedimentary host rocks. Early alteration stages are characterised by an inner biotite ± magnetite-bearing, potassic alteration and surrounding actinolite-epidote-chlorite propylitic alteration assemblages.

Potassic Alteration

Potassic alteration (biotite ± magnetite) is typically related to the hottest zone of the porphyry system, where magmatic-hydrothermal fluids have circulated with a neutral to alkaline pH at a temperature greater than 350° (Meyer and Hemley, 1967; Corbett and Leach, 1998). At Alpala, this alteration occurs where secondary (hydrothermal) 'shreddy' biotite replaces hornblende and pyroxene. Plagioclase is replaced by secondary albite-oligoclase and, less commonly, orthoclase. Potassic alteration at Alpala mostly occurs below 1000 mRL, approximately 700 m below surface. The potassic alteration zones typically have high magnetic susceptibility due to the presence of hydrothermal magnetite.

In Alpala, the potassic alteration is better developed in the early-mineralisation QD10 and D10 intrusive phases, as well as in intra-mineralisation D15, T15 and the IBXM phases. This alteration was initially volumetrically much larger than what is currently evident, as much of the early alteration mineral assemblages have been altered by late fluids, which have transformed most of the biotite into chlorite (Figure 7-25) and locally converted magnetite to pyrite and less abundant hematite.



Source: Artica et al., 2022
Note: Showing secondary biotite-magnetite alteration and replacement of secondary biotite by Chlorite
Figure 7-25: Example of potassic alteration from drill hole CSD-18-061 at 1852.2 m depth

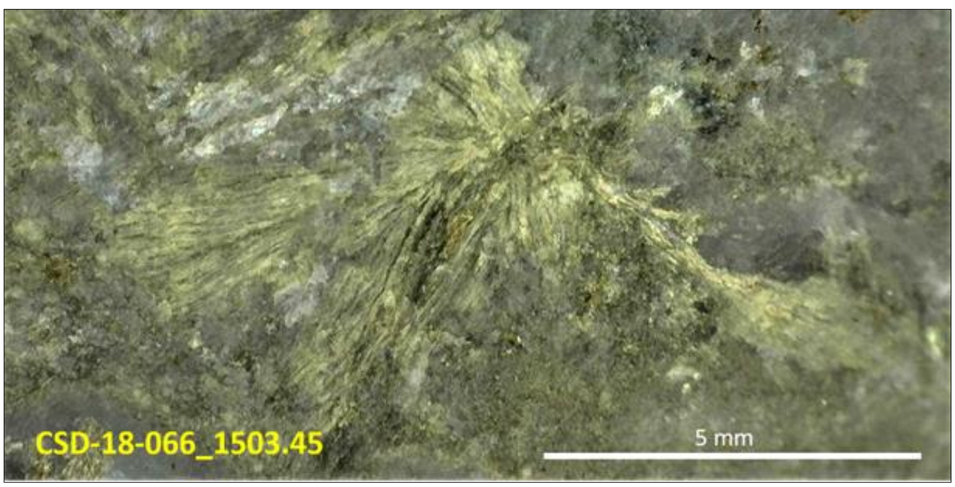
Potassic-Epidote Alteration

Potassic-epidote alteration (biotite + epidote ± actinolite ± magnetite) is evident in the transition from potassic alteration to inner-propylitic alteration but typically forms closer to the inner-propylitic zone. Potassic-epidote alteration is formed by the alteration of mafic minerals to biotite-epidote and of plagioclase to epidote at temperatures ranging from 260°C to 300°C (Corbett and Leach, 1998). This style of alteration, although conspicuous in part, is not widely developed at the Alpala deposit.

Propylitic alteration is formed at the same time as a potassic alteration in porphyry systems, as the magmatic-hydrothermal fluids disperse from the casual intrusion (Sillitoe, 2010). Propylitic alteration can be subdivided into subzones of innermost actinolite-propylitic, proximal epidote-propylitic and distal chlorite-propylitic alteration, which refer to the proximity to the porphyry centre. At Alpala, the propylitic alteration characteristically borders potassic and potassic-epidote alteration zones, and transitional silicate mineral assemblages are common.

Actinolite-Propylitic Alteration

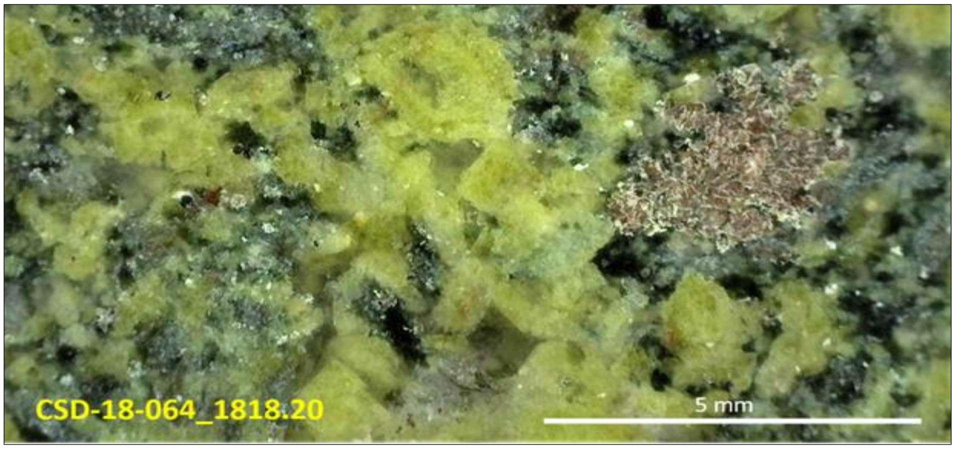
Actinolite-propylitic (inner-propylitic) alteration typically occurs adjacent to the potassic zone at Alpala, where hornblende and pyroxene mafic minerals are altered to actinolite and chlorite at temperatures between 280°C to 300°C by the presence of neutral to alkaline pH fluids (Corbett and Leach, 1998). This alteration is usually found below 1200 mRL, some 500 m below the surface. This zone is typically characterised by actinolite (Figure 7-26), chlorite and, less commonly, secondary biotite and epidote.



Source: SolGold
Figure 7-26: Example of actinolite mineralisation from drill hole CSD-18-066 at 1503.45 m depth

Epidote Propylitic Alteration

Epidote-propylitic alteration (epidote + chlorite) occurs outside of the inner-propylitic zone. This alteration assemblage likely formed at temperatures of 200°C to 280°C with neutral pH fluids (Corbett and Leach, 1998) and typically occurs within volcanic host rocks adjacent to late-mineralisation intrusive phases such as D30 and QD30. Figure 7-27 shows an example of epidote-chlorite alteration in the drill core from Alpala.



Source: Artica et al., 2022

Figure 7-27: Example of strong epidote alteration from drill hole CSD-18-064 at 1818.20 m depth

Chlorite-Propylitic Alteration

Chlorite-propylitic alteration (chlorite ± illite/smectite ± zeolite) forms extensively in the distal portions of the porphyry system at Alpala and can occur several kilometres away from the deposit.

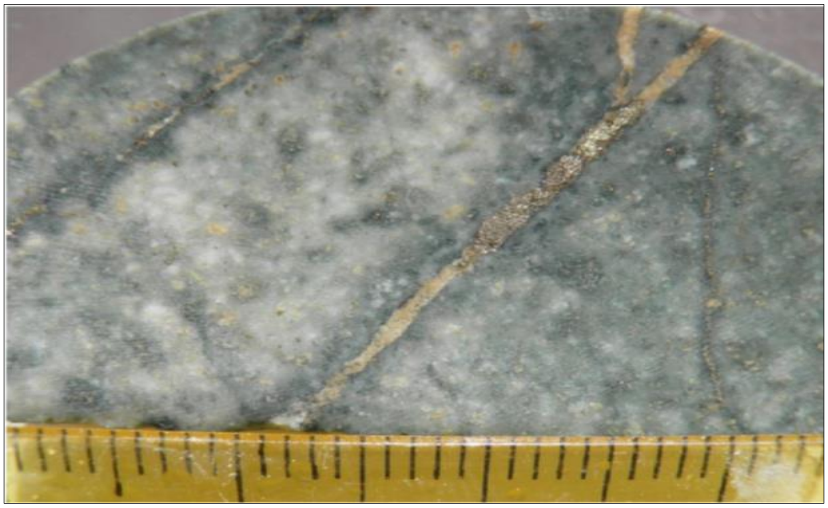
7.9.2 Transitional and Late-Stage Alteration

The transitional and late alteration stages in porphyry systems occur due to the acidification of hydrothermal and meteoric fluids and the progressive temperature collapse of the hydrothermal system. Resulting in overprinting or telescoping of these later alterations often forms above and within earlier alteration zones.

The late-stage alteration zones at Alpala include transitional-stage intermediate argillic alteration and late-stage phyllic, argillic and advanced argillic alteration mineral assemblages.

Intermediate Argillic Alteration

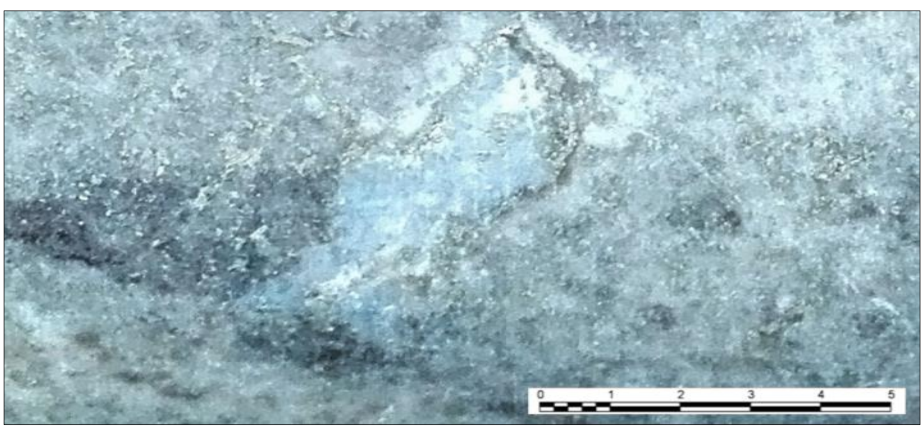
Transitional-stage, intermediate argillic (chlorite + sericite + illite-smectite) alteration mineral assemblages at Alpala may locally contain montmorillonite and likely formed at temperatures above 300°C with a pH of approximately 6 (Figure 7-28). Intermediate argillic alteration at Alpala is typically whitish green in colour, and magnetic pre-cursor rocks can maintain some remnant magnetism. Hematite may occur in fractures and as an alteration of earlier magnetite. Rutile and leucoxene typically replace mafic precursor minerals like hornblende and magnetite. Intermediate argillic alteration is extensive across the footprint of the deposit. It displays greater intensity in the deposit core, overprinting the transition between inner-propylitic alteration and the potassic alteration, characterised by a sulphide-oxide mineral assemblage of chalcopyrite-pyrite-magnetite ± hematite.



Source: Artica et al., 2022
Figure 7-28: Example of intermediate argillic (chlorite + sericite + clay) alteration from Alpala

Phyllic Alteration

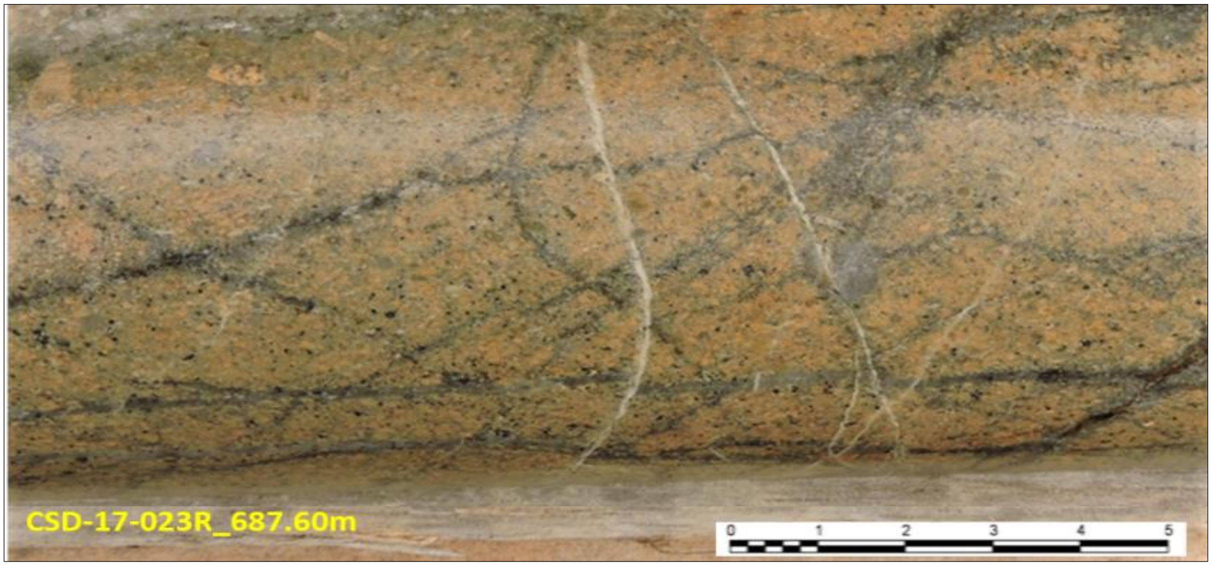
Late-stage phyllic alteration (sericite + quartz ± smectite) is usually located in the upper part of the deposit above 700 mRL but is also conspicuous within hydrothermal breccias that flank the eastern edge of the deposit and along intrusive contacts (Figure 7-29). At depth, phyllic alteration is restricted to faults, whereas in the upper portion of the deposit, it is overprinted by argillic and advanced argillic alteration. Phyllic alteration is typically light grey, surrounds the intra-mineralisation intrusions and forms an extensive zone that dips towards the south and becomes thicker in the southern part of the deposit. The rocks have a weak to null magnetism due to the conversion of magnetite to pyrite.



Source: Artica et al., 2022
Figure 7-29: Example of phyllic alteration (quartz + sericite) from Alpala

Argillic Alteration

Late-stage argillic alteration (kaolinite + smectite) overprints the previously mentioned alteration types in the upper part of the deposit, above approximately 700 mRL (Figure 7-30). This type of hydrothermal alteration is often yellowish to orange and inferred to form at temperatures below 250°C in acidic conditions with a pH of 3-4. The presence of this alteration below approximately 500 mRL may result from telescoping within the alteration, whereby mixing meteoric waters with hydrothermal fluids allows the leaching of argillic alteration minerals, permeating to greater depths. It may also be related to the alteration product of an intra-mineralisation intrusion (e.g., D15) at depth.

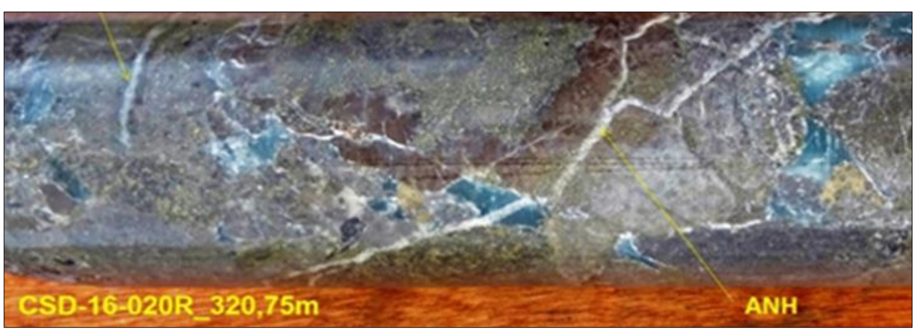


Source: Artica et al., 2022

Figure 7-30: Example of argillic alteration from drill hole CSD-17-023R at 687.6 m depth

Advanced Argillic Alteration

Late-stage advanced argillic alteration (dickite + pyrophyllite ± alunite) is usually formed by the action of very acidic fluids (pH less than 2) (Figure 7-31). Pyrophyllite precipitates with dickite at temperatures of about 250°C to 350°C. Advanced argillic alteration at Alpala is located near surface in the northern half of the deposit, where it appears to coincide with hydrothermal breccia units. The thickness of this alteration zone increases towards the southeast, where it lies above an intra-mineralisation D15 intrusion.



Source: Artica et al., 2022

Note: Showing the presence of blue-green dickite from drill hole CSD-16-020R at 320.75 m depth

Figure 7-31: Example of advanced argillic alteration

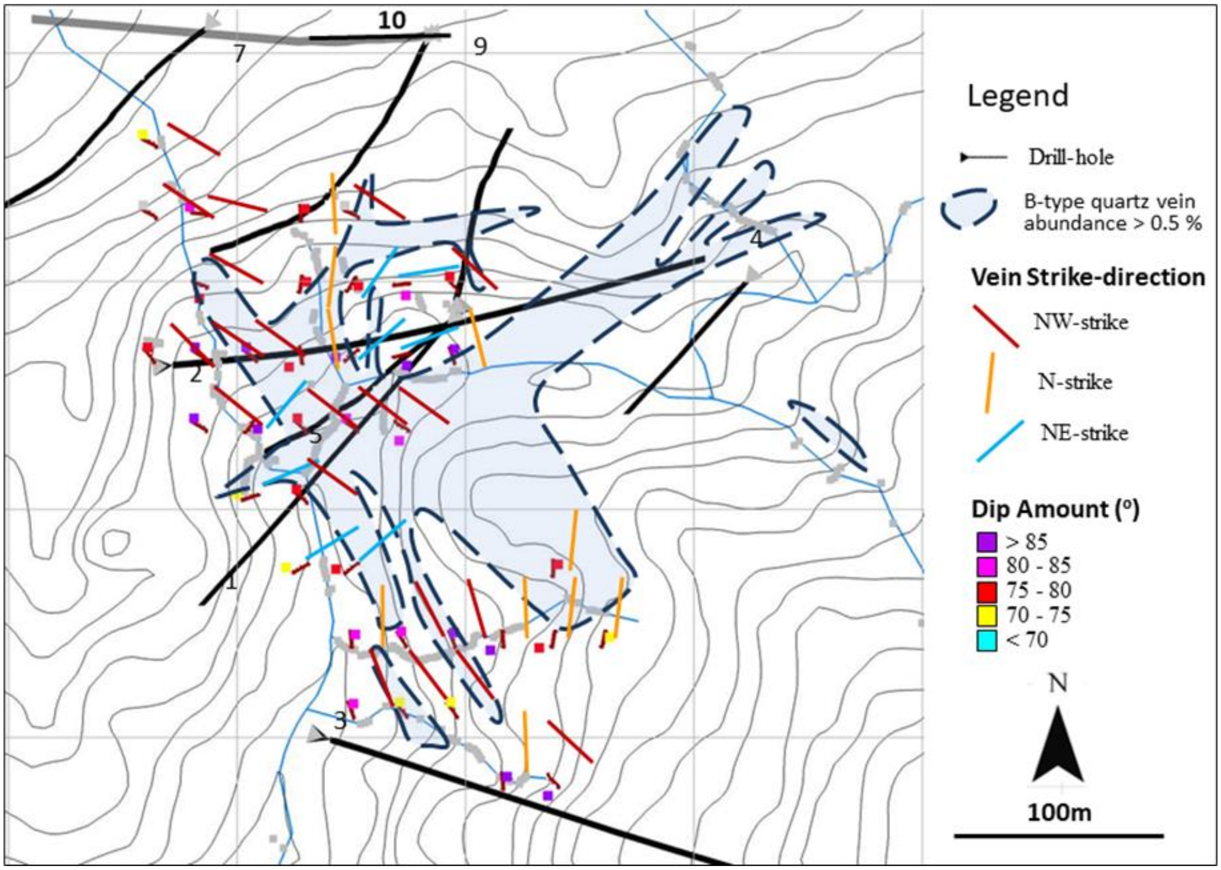
7.10 Alpala Deposit Mineralisation Trends

The Alpala deposit geological, structural and mineralisation all display a clear anisotropy which is prolate in a sub-vertical sense, dipping approximately 78° towards the northeast and oblate in plan (striking northwest). Recognition of this geometry in the field and various datasets has played a critical role in the path to the discovery of the Alpala deposit.

Representative mineralisation trends have been employed for each of the geology, vein and grade modelling processes utilising a combined trend model of the major trends identified from:

- Surface mapping of quartz vein orientations
- The Alpala Structural Zone (and CFlt4a interpreted from drill core)
- Intrusive contacts analysis
- Drill core vein abundance and orientation analysis (Global B-vein maximas)
- Localised B-vein analysis

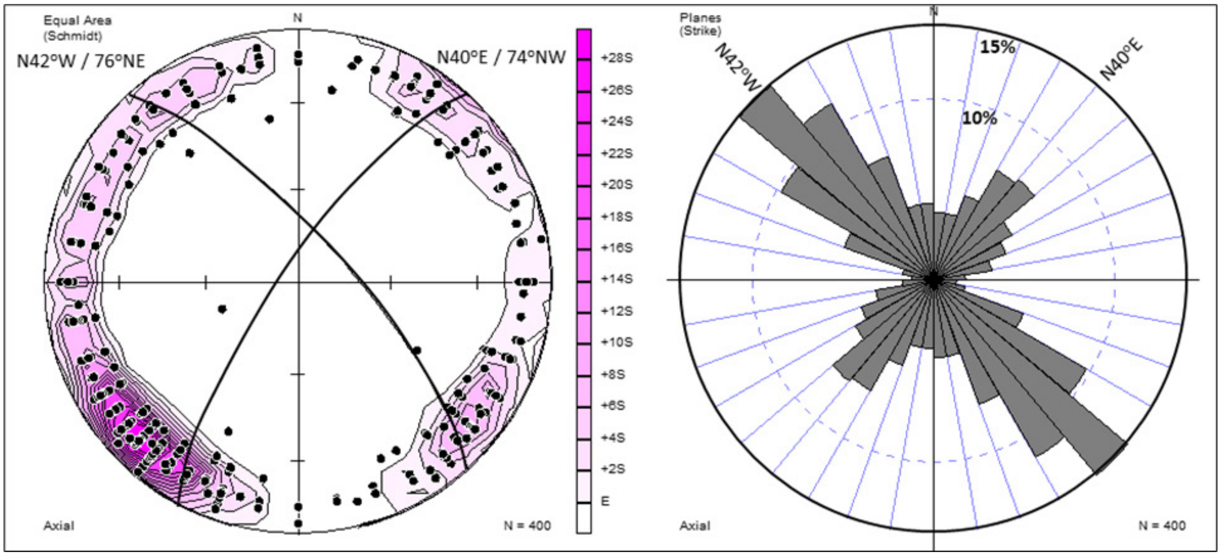
During initial exploration, quartz vein orientation analysis from the Alpala Central area assessed over 400 quartz vein measurements, the majority of which were from B-type quartz veins. This dataset, extracted from the Alpala deposit structural database, was assessed using a spatial averaging program (Spheristat™) to produce a map of the average strike and dip of the quartz veins (Figure 7-32).



Source: Artica et al., 2022
Note: Northwest and dips approximately 75°NE; a secondary vein set strikes Northeast and dips approximately 75°NW; a third vein set strikes Northerly and dips approximately 75°E

Figure 7-32: The most abundant v-set strikes

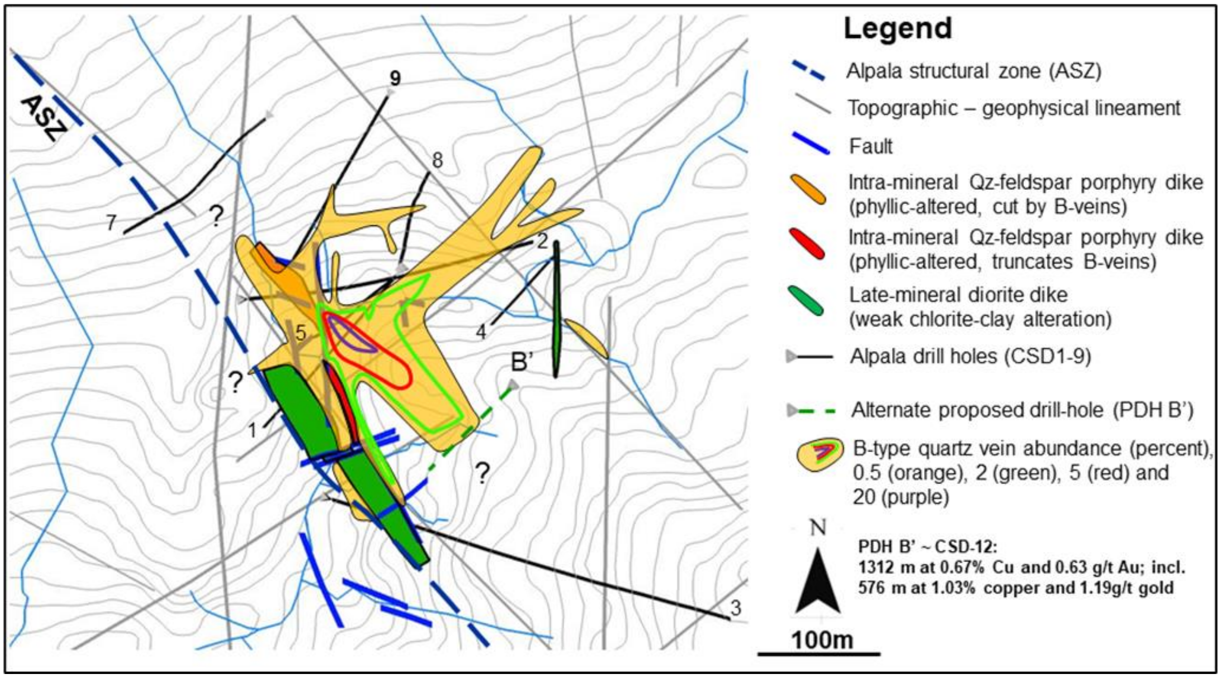
Lower hemisphere, equal area stereonet projection and rose-diagram of the dataset shows the most abundant vein set strikes northwest and dips approximately 75°NE; a secondary vein set strikes northeast and dips about 75°NW; a third vein set strikes north and dips greater than 80°E. The stereonet data has been contoured using the method of Kamb (1959), where the contour levels are expressed in multiples of the standard deviation, S, of the distribution of the data around the mean, or expected value, E. The rose diagram shows the strike directions grouped at 10° intervals and circles for frequencies of 10% and 15% (Figure 7-33).



Source: Artica et al., 2022
Note: ENSA in trenches and surface exposure outcrop from the Alpala central area

Figure 7-33: Stereonet projection and rose-diagram for 400 quartz vein measurements collected by SolGold

An interpretive geological map for the Alpala Central area, based on the 1:500 scale geological mapping, was completed by ENSA geologists in late 2014 (Figure 7-34). This work showed three different types of intrusive dykes, which include intra-mineralisation quartz-feldspar porphyries and late-mineralisation hornblende diorite. The dykes strike north-northwest to northwest and north and dip steeply towards the east. Faults strike northwest, north and northeast and commonly dip steeply towards the north and east. The approximate and inferred trace of the Alpala structural zone (ASZ) is also reflected in this mapping.



Source: Artica et al., 2022

Figure 7-34: Interpretive geology, structure and mineralisation for the Alpala central area

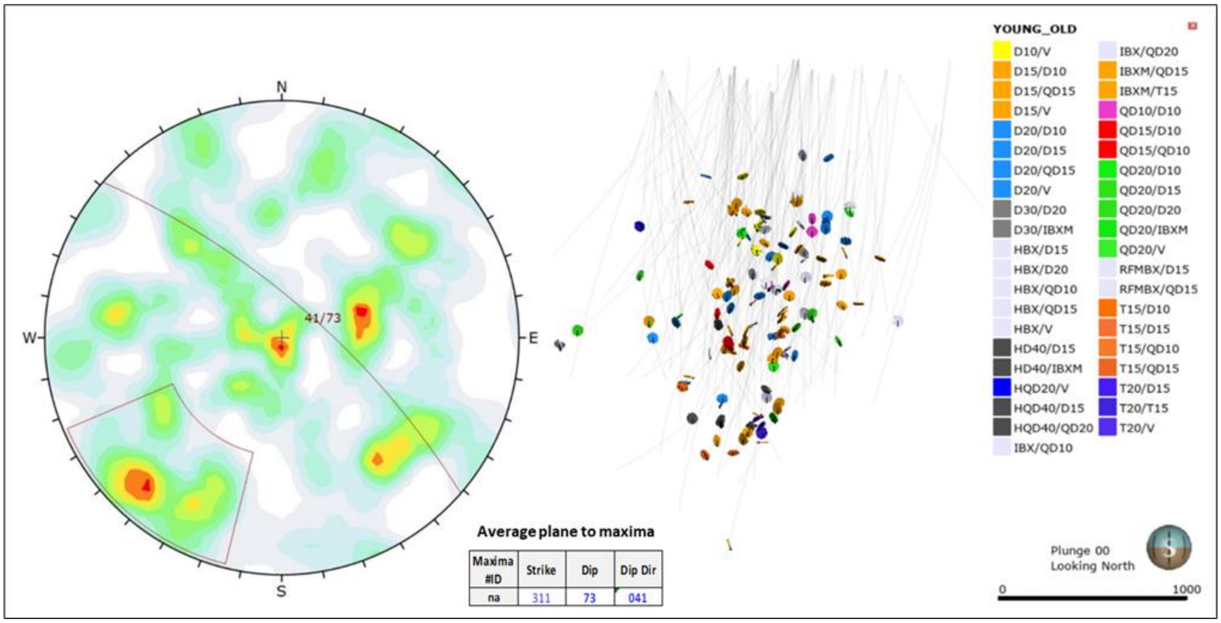
Interpreted distribution of B-type quartz veins (B-veins) using contours of 0.5%, 2%, 5%, and 20% forms a geometry of vein zones with northwest, northeast and a subordinate northerly trend. This is consistent with the strike directions of the B-type veins measured in the field and vein and fracture trends in the outcrop along Alpala Creek (Figure 7-35). These three vein sets typically dip steeply towards the northeast, northwest and east, respectively.

Source: Artica et al., 2022

Figure 7-35: Vein and fracture trend examples from Alpala creek, central Alpala area

Detailed structural analysis of intrusive contacts logged and oriented from the drill core throughout the Alpala deposit was completed from a dataset of 164 structural readings from the oriented drill core.

Intrusive contact orientations of the Alpala Deposit are dominantly northwest trending, steeply northeast dipping, with an average plane to maxima dipping 73° towards 041° (Figure 7-36).

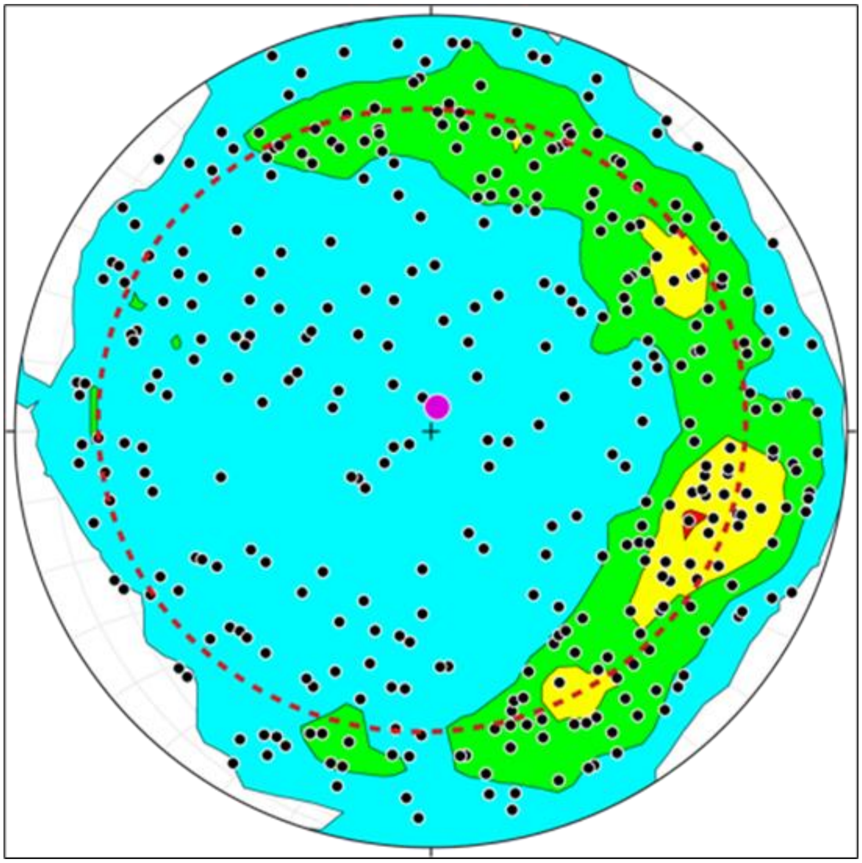


Source: Artica et al., 2022
Note: Intrusive contact orientations of the Alpala deposit are dominantly Northwest trending, steeply Northeast dipping, with an average plane to maxima dipping 73°towards 041

Figure 7-36: Intrusive contact orientations of the Alpala deposit

The global structural analysis of B-type quartz veins utilised a filtered dataset of 6,630 structural measurements. The original dataset of 12,946 measurements was reduced to remove poor survey data from magnetic compass survey equipment in the first 18 drill holes. B-type veins, characterised by straight walls, form a characteristic pattern typical of porphyry Cu deposits. Sheeted veins generally form with steep dips, near 90° as in the upper crust, because the earth's surface cannot support a shear stress (Tosdal, 2019). Emplacement of successive intrusions will deform the wall rocks such that in an upright porphyry deposit, veins will be distributed around the margins of a lower hemisphere stereonet.

Plotting all 18,543 B-type veins measured in oriented core demonstrated the issue with the data, in that drill holes with inclinations between 85° and 90° from vertical have poor control on the drill core orientation. In these cases, measured vein orientation data define a circle (red dashed circle) centred on the azimuth and inclination of the drill hole (Figure 7-37). Many of these such issues with the orientation data were related to using a non-gyroscopic, magnetic-based, Reflex Ezi-Shot camera for downhole surveying of the first 18 drill holes, which produced survey data displaying interference from local magnetic fields as well as inaccuracy from the many steeply dipping drill holes with inclinations in the order of 85°. Although veins generally distribute as expected, near the margins of the stereonet, their distribution and orientation may not accurately reflect their geometry in situ.

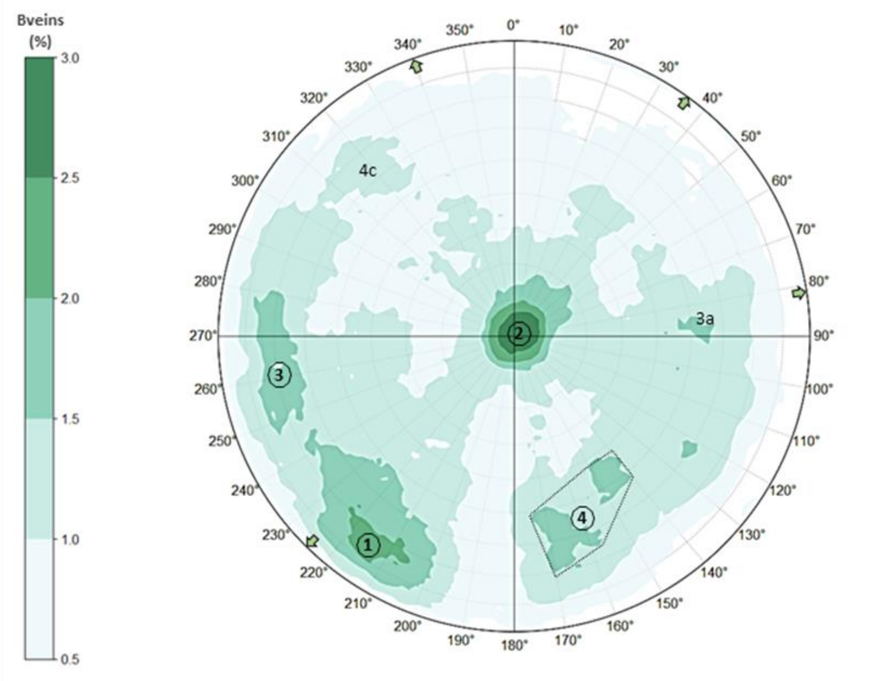


Source: Artica et al., 2022
Note: Example from drill hole CSD-14-008 drilled at -85° inclination illustrating poor control of drill core orientation. Measured vein orientation data define an inner circle (red dashed circle) centred on the azimuth and inclination of the drill hole (magenta dot)

Figure 7-37: Example from drill hole CSD-14-008 drilled at -85° inclination

A global structural analysis of B-type quartz veins throughout the Alpala deposit was completed from a filtered dataset of 6,630 structural readings from oriented drill core. B-vein orientations of the Alpala Deposit form four major average planes to maxima (Figure 7-38).

A northwest-trending steeply dipping maxima (1) and flat maxima (2) are clear major trends, with a secondary north-northwest steeply dipping trend (3) and a tertiary east-northeast moderately dipping trend (4). Lesser clusters form as conjugate vein sets to maximas 3 and 4 (at points 3a and 4c, respectively). This clean dataset of vein orientation data is important for modelling the Alpala deposit intrusions, veins, and grade.



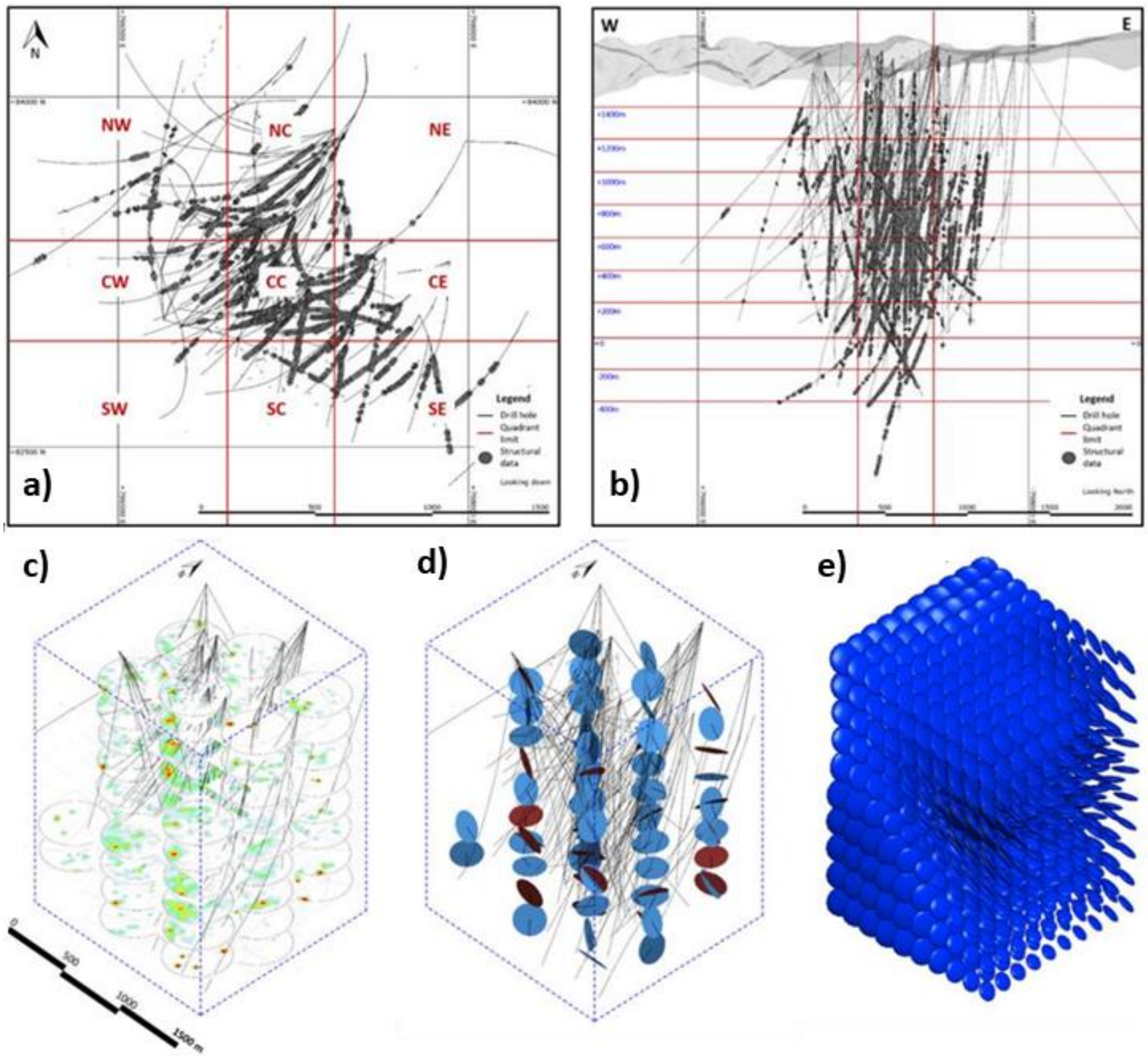
Average planes to maxima

Maxima #ID	Strike	Dip	Dip Dir
1	306	76	036
2	135	1	225
3	352	70	082
4	250	55	340

Source: Artica et al., 2022

Figure 7-38: Kamb contoured lower hemisphere polar stereonet plot of quartz vein (B-type) orientations of the Alpala deposit

A localised structural analysis of B-type quartz veins was also completed. This study was completed from an unfiltered dataset of 12,946 structural measurements of B-type quartz veins in drill core and analysed the data in nine quadrants and eleven elevation intervals to produce 96 spatially representative stereonets (Figure 7-39). The methodology allowed the identification of localised B-vein and mineralisation trends, emphasising the dominant northwest trending, steeply dipping orientation, and the tendency for sub-horizontal (or flat) vein orientations in the centre of the deposit within and above coincident D10 and QD10 intrusion cupolas. The localised vein trend method does not represent the overall geometry of the mineralisation. Still, it is important to consider the veining trends at intrusion cupolas and margins and assess the influence of any stratigraphic control on veining.

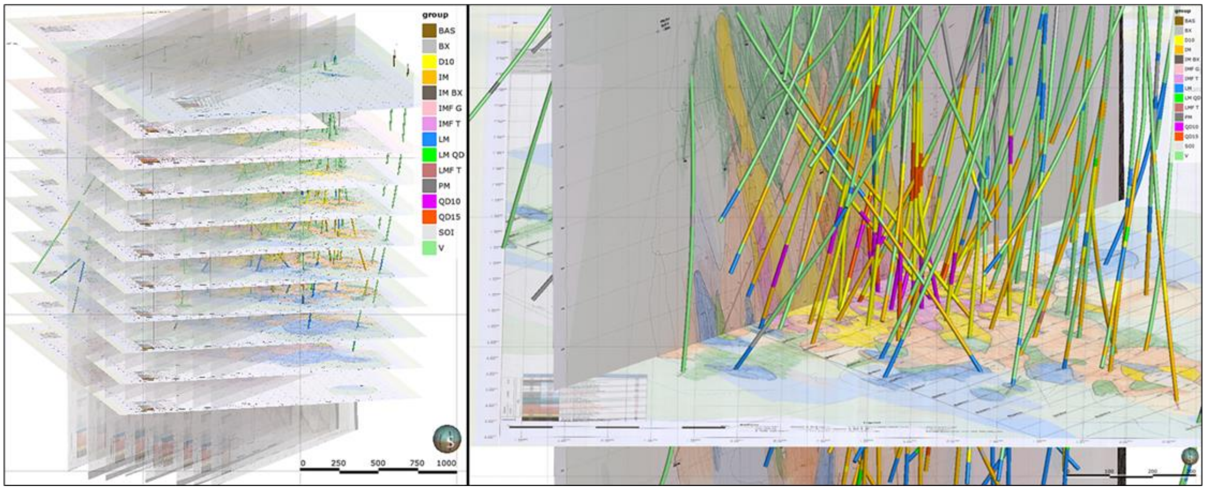


Source: Artica et al., 2022
Note: Segregated into nine quadrants in plan (a) and section view (b), showing resultant stereonets (c), 96 major trends (d), and blended structural trend (e)

Figure 7-39: Localised B-vein orientation analysis

The geometrical understanding of the deposit and its internal timing relationships between rock phases, veins, and Cu-Au grades was advanced by the completion of hand-drawn cross-sections and level plans for each dataset throughout the deposit (Figure 7-40). Hand-drawn cross-section interpretations of geology and B-vein abundance were performed at approximately 70 m (70.711 m) spacings across the deposit, with a window of approximately 35 m (± 35.356 m) from section 82,650N to section 83,450N and are orientated northeast and centred upon a northwest trending baseline.

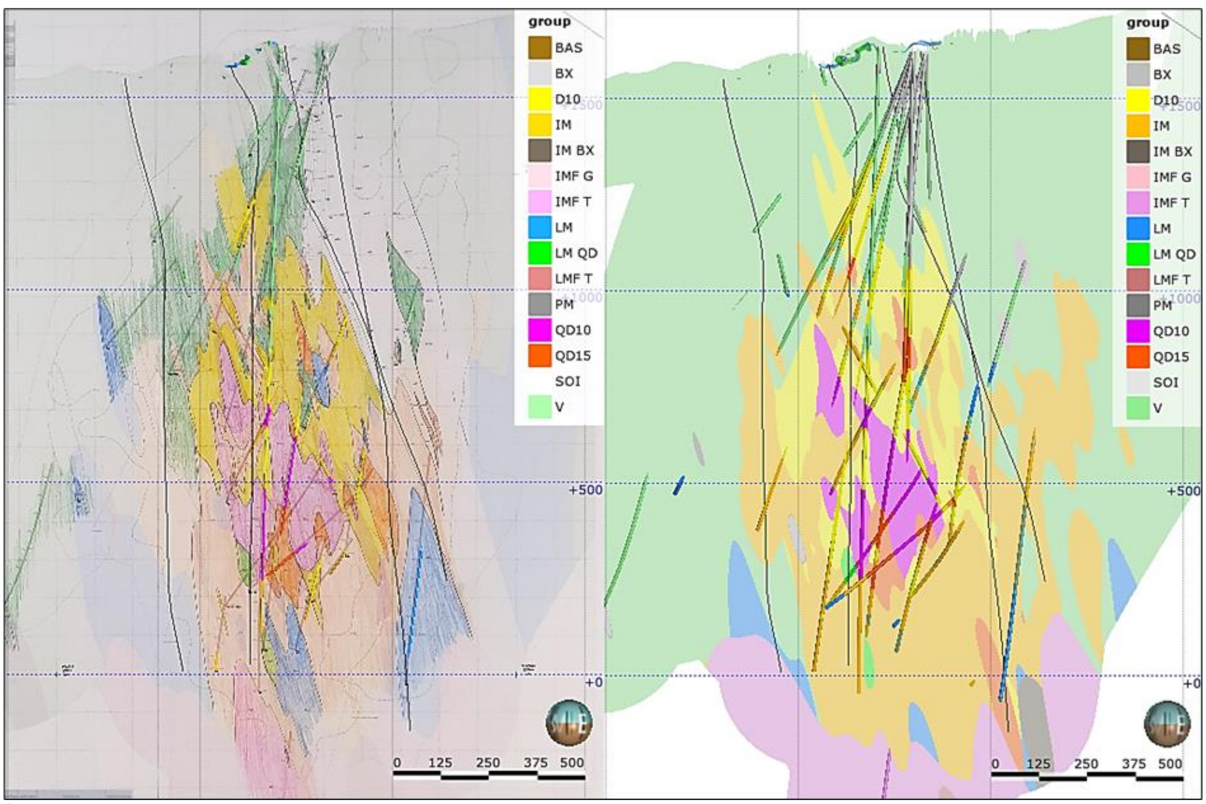
Hand-drawn level-plan interpretations of geology and B-vein abundance were also performed at 200 m levels throughout the deposit, with a window of ±100 m from the surface to -300 mRL.



Source: Artica et al., 2022

Figure 7-40: Examples of hand-drawn level plan and cross-section interpretations of the Alpala deposit

The geological modelling of rock groups was completed within the 3D software program Leapfrog Geo, which forms the foundation of the vein and grade modelling in the Mineral Resource estimation and compares well with hand-drawn level plans and sections (Figure 7-41).

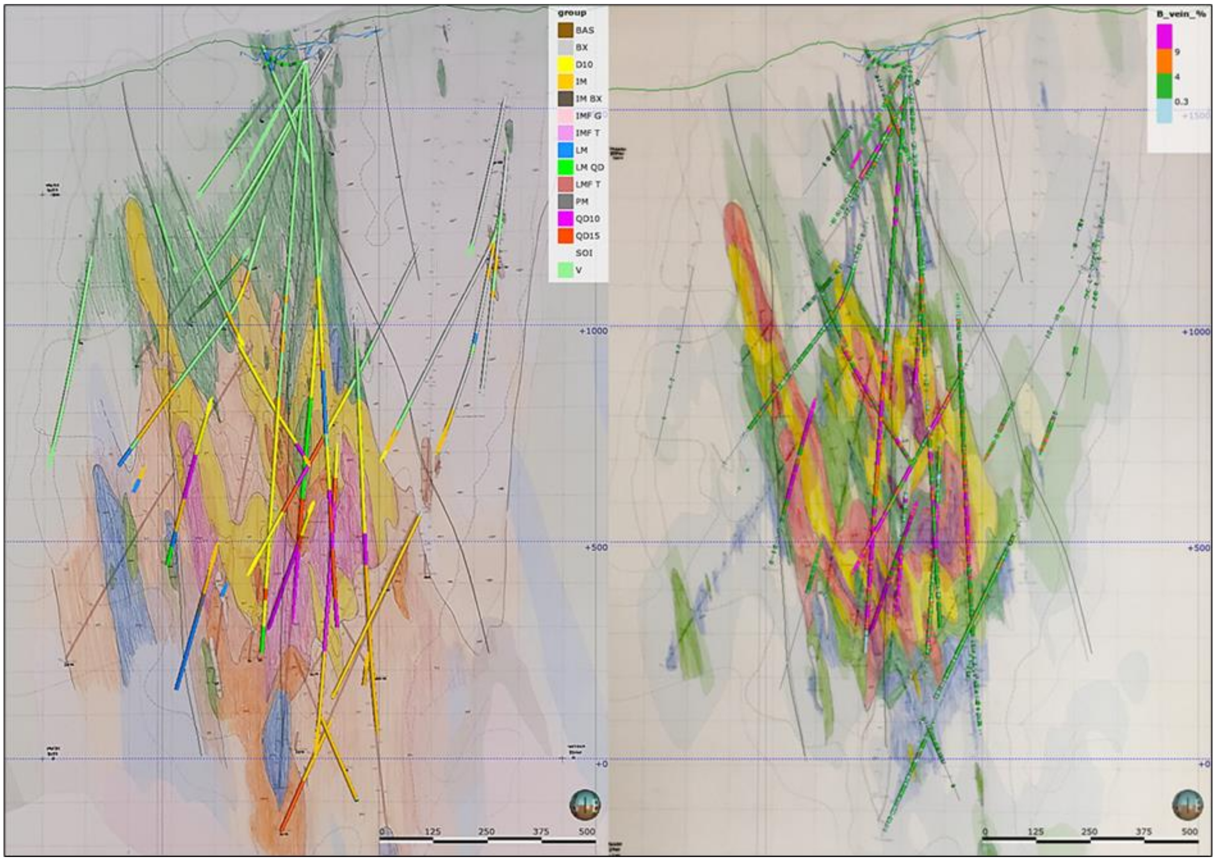


Source: Artica et al., 2022

Note: Hand-drawn geological interpretation on 82,900N cross-section (left), looking northeast, reproduced as a 3D Geological model in Leapfrog 3DTM (right) showing downhole geology

Figure 7-41: Hand-drawn geological interpretation on 82,900N cross-section

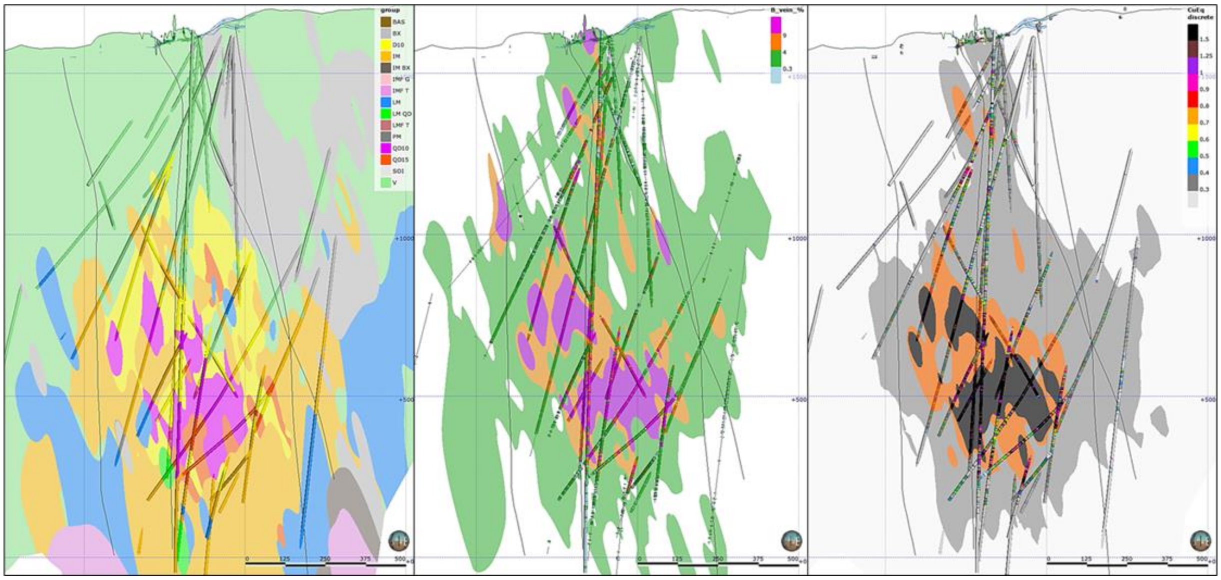
Geological interpretation was completed as a first pass, with subsequent B-vein interpretation utilising an underlay of geology such that interpretation can account for the sequence and relative timing between host intrusions (Figure 7-42).



Source: Artica et al., 2022
Note: Hand-drawn geological interpretation and vein abundance interpretation along 83,000N cross-section line looking Northwest, showing drill holes within a window of ±70 m

Figure 7-42: Hand-drawn geological interpretation and vein abundance interpretation along 83,000N cross-section line

The geometry of the intrusive units and accompanying zones of porphyry-style quartz vein abundance correlate exceedingly well with associated Cu and Au grade distribution (Figure 7-43).



Source: Artica et al., 2022
Note: Showing 3D geological models (left), B-type quartz vein abundance models (centre), and grade models (right) with drill holes displaying downhole geology

Figure 7-43: Example cross-section, looking Northwest, centred upon a 100 m wide window

7.11 Tandayama-America Deposit

7.11.1 Geology

Major host rock types of the Tandayama-America (TAM) deposit consist of a sequence of Tertiary volcano-sedimentary and andesitic lavas of the same age as those at the Alpala deposit. This sequence has also been intruded by a series of Middle to Late-Eocene (Bartonian) quartz diorites and diorites that form plutons, stocks, and dykes.

Drilling at Tandayama-America has defined a northwest-trending, steeply northeast-dipping corridor regionally associated with the Greater Alpala Trend. This trend is also centred upon a syn-mineralisation causal quartz-diorite intrusion (QD10).

Geological mapping and drill core logging have identified a total of 15 likely intrusion- and intrusion-related breccia phases at Tandayama-America. These intrusions form a similar geometry to that of the Alpala deposit and have been grouped into 8 major rock groups which have been delineated based on their mineralisation phase and resource group. All the major rock groups have been incorporated into the three-dimensional geological models created of the Alpala deposit (Figure 7-44).

Mineralisation Phase	Resouce Group	Rock Group	Rock Phase	Rock Description	Rock Group Relative Timing	Rock Phase Relative Timing
Cover	N.I.R	SOI	SOI	Soil & Saprolite	na	na
		FLT	FLT	Post Mineral Faults	na	na
Post-Mineral	BARREN	PM	QD30	Post Mineral Qtz Diorite	1190	1190
			D30	Post Mineral Diorite		1180
	BARREN - LOW GRADE		RFBX	Rock Flour Matrix Bx		1130
			HBX	Hydrothermal Matrix Bx		1120
Late-Mineral	LOW GRADE	LM	D20	Late Mineral Diorite	1110	1110
			QD20	Late Mineral Qtz Diorite		1100
Intra-Mineral	MED GRADE	IM	QD15	Intra Mineral Qtz Diorite	1080	1080
			D15	Intra Mineral Diorite		1070
Early-Mineral	Hi GRADE	EM	D11	Syn? Early Mineral Diorite	1060	1060
			QD10	Syn? Early Mineral Qtz Diorite		1055
Pre-Mineral	HOST ROCKS	IBX	IBX	Pre Mineral Magmatic Intrusive Matrix Bx	1050	1050
		D10	D10	Pre Mineral Diorite	1045	1045
		V	V	Pre Mineral Volcanics	1040	1040
			VBAS	Pre Mineral Mafic Volcanics		1030

Note: Codes form the foundation of geological interpretation and 3D modelling of the Tandayama-America deposit
Showing mineralised deposit forming porphyry phases inside red outline (QD10 to QD30)

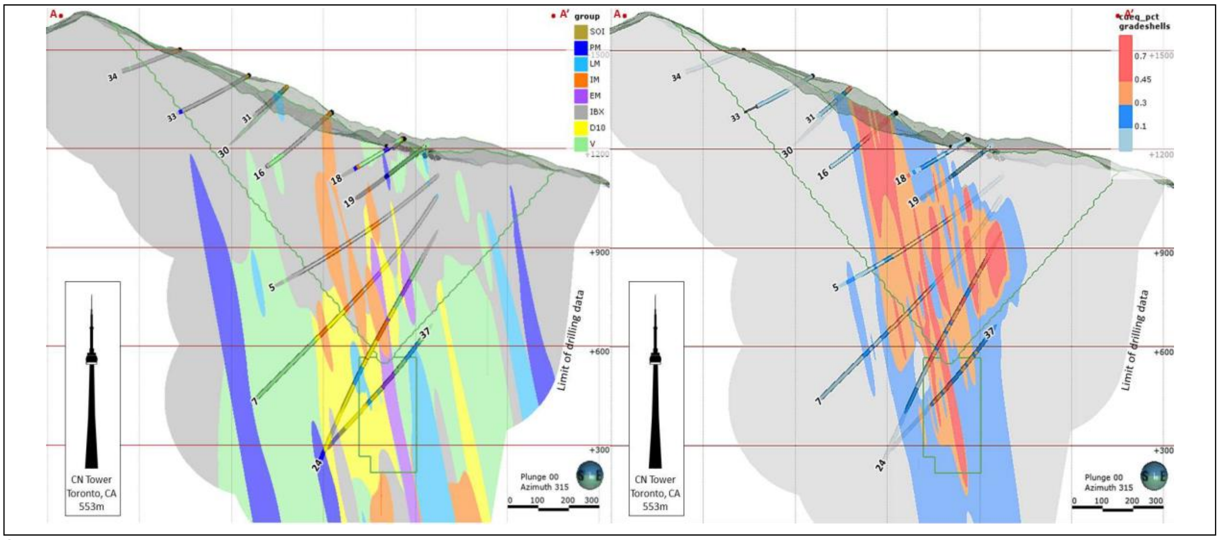
Figure 7-44: Geological modelling codes utilised for the Tandayama-America deposit

The majority of the Cu and Au mineralisation at Tandayama-America was added to the system by the QD10 intrusion with supplementary additions through time with the injection of the intra-mineralisation intrusive phases, with only very minor metal addition from late-mineralisation and post-mineralisation intrusions as the system waned.

The geometric framework of various lithologies and intrusive bodies subsequently controlled the successive geometries and zonation of the porphyry-style quartz vein abundance at Tandayama-America, which in turn correlates with the distribution of Cu and Au.

7.11.2 Mineralisation

The mineralisation style and trend at Tandayama-America is essentially the same style as that seen in Alpala (Figure 7-45).



Source: Artica et al., 2022
Note: Showing the geometric framework of intrusions (left) and copper equivalent grade shells (right)
Figure 7-45: Example cross-section through the centre of Tandayama-America deposit, looking Northwest

7.11.3 Hydrothermal Alteration

The following major hydrothermal alteration features at the Tandayama-America deposit are essentially the same style as that seen in Alpala, where intermediate argillic (chlorite-sericite-clay) alteration coincides with the majority of medium-grade to high-grade mineralisation (>0.3% CuEq) in the deposit, however, at Tandayama-America it is characterised by conspicuously abundant muscovite at approximately greater than 12%.

7.12 Comments on Section 7

In the opinion of the QPs, the knowledge of the deposit settings, lithologies, mineralisation style and setting, ore controls, and structural and alteration controls on mineralisation is sufficient to support Mineral Resource and Mineral Reserve estimation.

8 Deposit Types

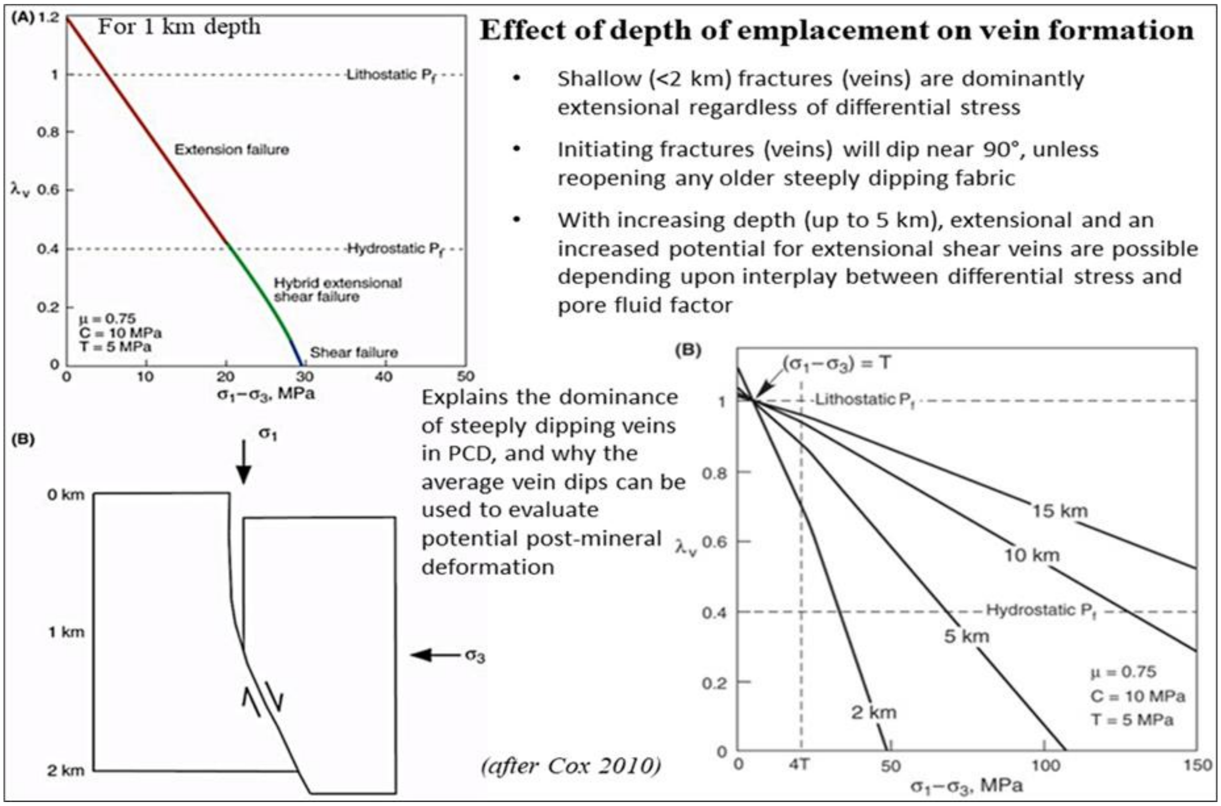
8.1 Deposit Types

The mineralisation observed at surface and in the drill core along the trend of the Alpala deposit is considered a classic porphyry Cu-Au system. Exploration has been designed with this in mind as the primary target. These mineralised systems are hosted within a linear belt (Andean Porphyry Belt) that extends from southern Chile through Ecuador and Colombia to Panama. The Andean Porphyry Belt hosts the largest concentrations of Cu in the world, including numerous deposits in active mining operations.

This geological setting is associated with the following mineral deposit types:

- Porphyry Cu-Au-Ag-Mo - related to the early stages of magmatism
- Epithermal Au, low and high-sulfidation - associated with volcanic regions above porphyry systems
- Polymetallic skarn - related to hydrothermal fluid flow from granite stocks through permeable and reactive calcareous host rocks, such as limestone and calcareous siltstone

Porphyry deposits typically form from magmatic-hydrothermal fluids that have evolved from a voluminous magma chamber several kilometres below the deposit. Predating or associated with those fluids are vertical dykes and stocks of porphyritic intrusive rocks from which this deposit type derives its name. Porphyry deposits typically host mineralisation within quartz veins that form dominantly in a vertical sense (Figure 8-1).



Source: Cox, 2010

Figure 8-1: Effect of depth emplacement on vein formation in porphyry deposits

As with many of the composite terranes across South America, the Western Tectonic Realm (Cedial et al., 2003) of Ecuador and Colombia hosts multiple intrusion-related systems. These include the Llumuriagua, Mirador, Fruta Del Norte, and Alpala deposits in Ecuador and the Buritica and La Colosa deposits in Colombia.

Porphyry Copper Systems

Porphyry deposits are characterised by disseminated, veinlet and fracture-controlled Cu-Fe sulphide minerals distributed throughout a large volume of rock in association with potassium silicate, sericitic, propylitic and less commonly, advanced argillic alteration in porphyritic plutons and in the immediate wall rocks (Myer and Hemley, 1967; Lowell and Guilbert, 1970; Gustafson and Hunt, 1975; Titley and Beane, 1981; Einaudi, 1982a). In porphyry systems, a close spatial and temporal link exists between volumetrically small causative intrusions and broadly dispersed magmatic hydrothermal alteration and mineralisation. Porphyry-copper deposits are large, commonly hundreds to thousands of millions of tonnes, and low to medium in grade (0.3% to 1.5% Cu). The majority of gold-rich porphyry deposits occur in the circum-Pacific area and commonly contain 0.3 to 1.6 g/t Au (Sillitoe, 1990 and 1993).

Porphyry systems are causal to different deposits, including:

- Porphyry deposits centred on the parent intrusion and its surrounding host rocks
- Epithermal and skarn deposits peripheral to porphyry deposits

- Carbonate replacement and sediment hosted deposits with increasing distance from the parent intrusion

Globally the most significant systems tend to be Mesozoic or Cenozoic in age. In South America, the most important metallogenic epochs are Eocene to Miocene. Porphyry deposits occur in linear belts related to composite plutons and convergent plate boundaries either within continental magmatic arcs or island arcs in association with subduction zones or post-collision volcanism. Porphyry deposits are often located at the intersection of the porphyry belt and intra-arc fault zones, forming at relatively shallow depths of one to four kilometres below the surface and related to magma chambers forming vertical elongate stocks or dyke swarms (Sillitoe, 2010). Several discrete stocks are often emplaced in one area, resulting in spatially and temporally clustered deposits or structurally controlled alignments, which consist of several generations of intermediate to felsic porphyritic intrusions.

In porphyry systems, there is a common upward and outward zonation of hydrothermal alteration, characterised by a core of biotite ± K-feldspar, with proximal actinolite through to epidote-propylitic and distal chlorite-calcite-propylitic alteration. In many systems, these early-stage alteration zones are overprinted by transitional-stage chlorite-sericite-clay (intermediate argillic); and late-stage quartz-sericite (sericitic-phyllic) and quartz-alunite-pyrophyllite-dickite (advanced argillic) alteration.

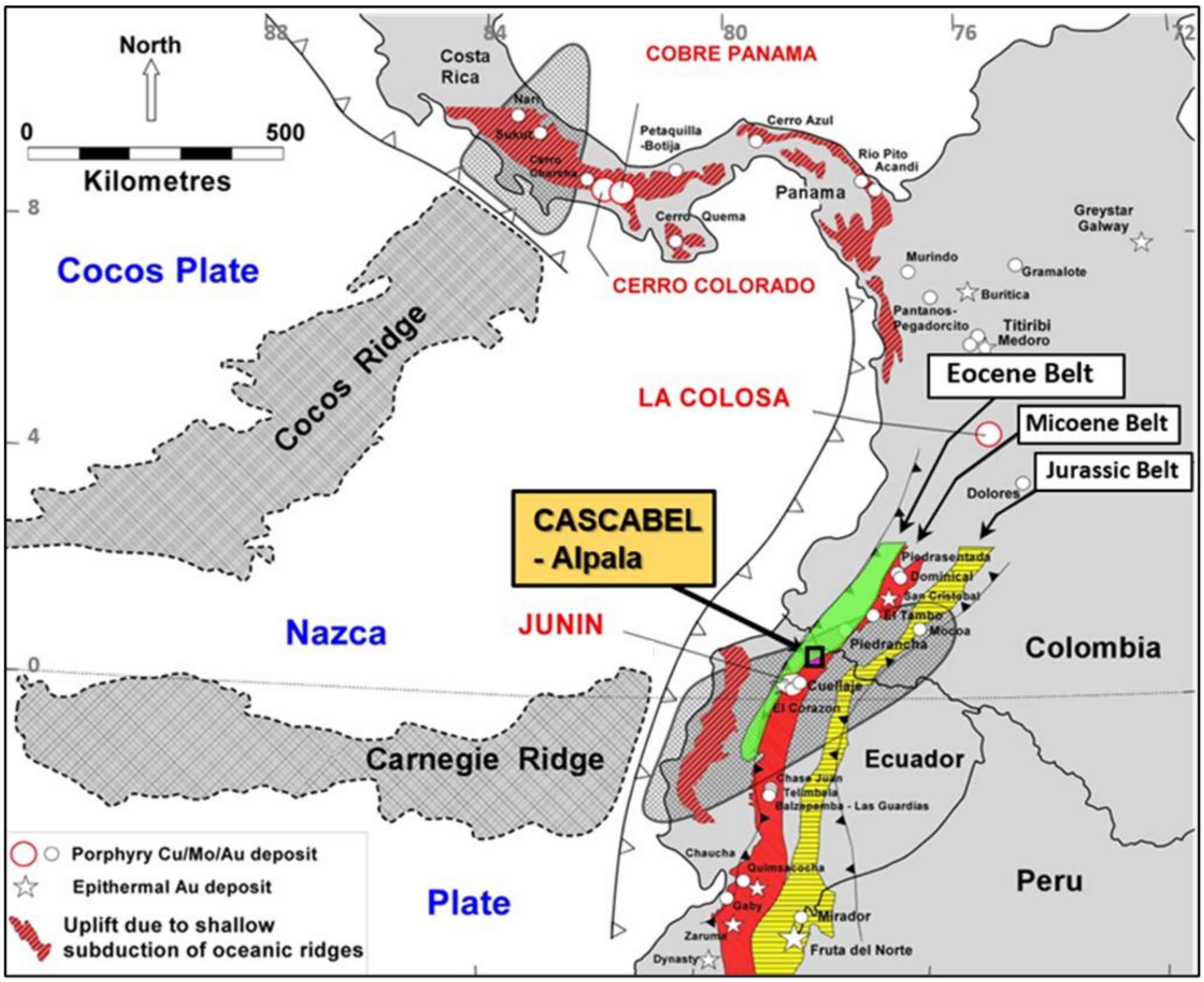
A similar zoning in sulphide minerals occurs, characterised by central, higher-temperature bornite-chalcopyrite, proximal chalcopyrite-pyrite and distal, lower-temperature pyrite-chalcopyrite-sphalerite-galena. This last assemblage is common in the late-stage, epithermal veins that locally flank porphyry centres.

The general characteristics of porphyry systems are:

- Small diameter, less than two kilometres, causative intrusions of intermediate to felsic composition
- Shallow levels of emplacement, typically one to four kilometres
- Porphyritic texture of causative intrusions, where feldspar, quartz and mafic phenocrysts are contained in a fine-grained to aplitic groundmass
- Multiple phases of intrusion, pre, syn and post-ore; late-stage diatremes are common in western Pacific volcanic-arc settings
- Several stages of hydrothermal alteration are associated with each mineralising intrusion
- Extensive development of fracture-controlled alteration and mineralisation in both porphyritic intrusions and adjacent wall rock
- A progression from early, discontinuous and irregular veins and veinlets (A-veins) through transitional, planar veins (B-veins) to late, through-going veins (D-veins) and breccia bodies (Gustafson and Hunt, 1975)
- A progression in hydrothermal alteration from early, central potassium silicate and distal propylitic styles to late sericitic/phyllic, advanced and intermediate argillic alteration types
- Sulphide and oxide minerals, which vary from early bornite-magnetite through transitional chalcopyrite-pyrite to late pyrite-hematite, pyrite-enargite or pyrite-bornite

- Fluid inclusion studies indicate that early alteration and Cu mineralisation are generated by magmatic fluids with 30 to >60 wt. % NaCl equivalent over a temperature range of 400°C to greater than 700°C, whereas the fluids related to late alteration and mineralisation commonly include a meteoric component and are more dilute (less than 15 wt. % NaCl equivalent) and lower in temperature (200°C to 400°C).

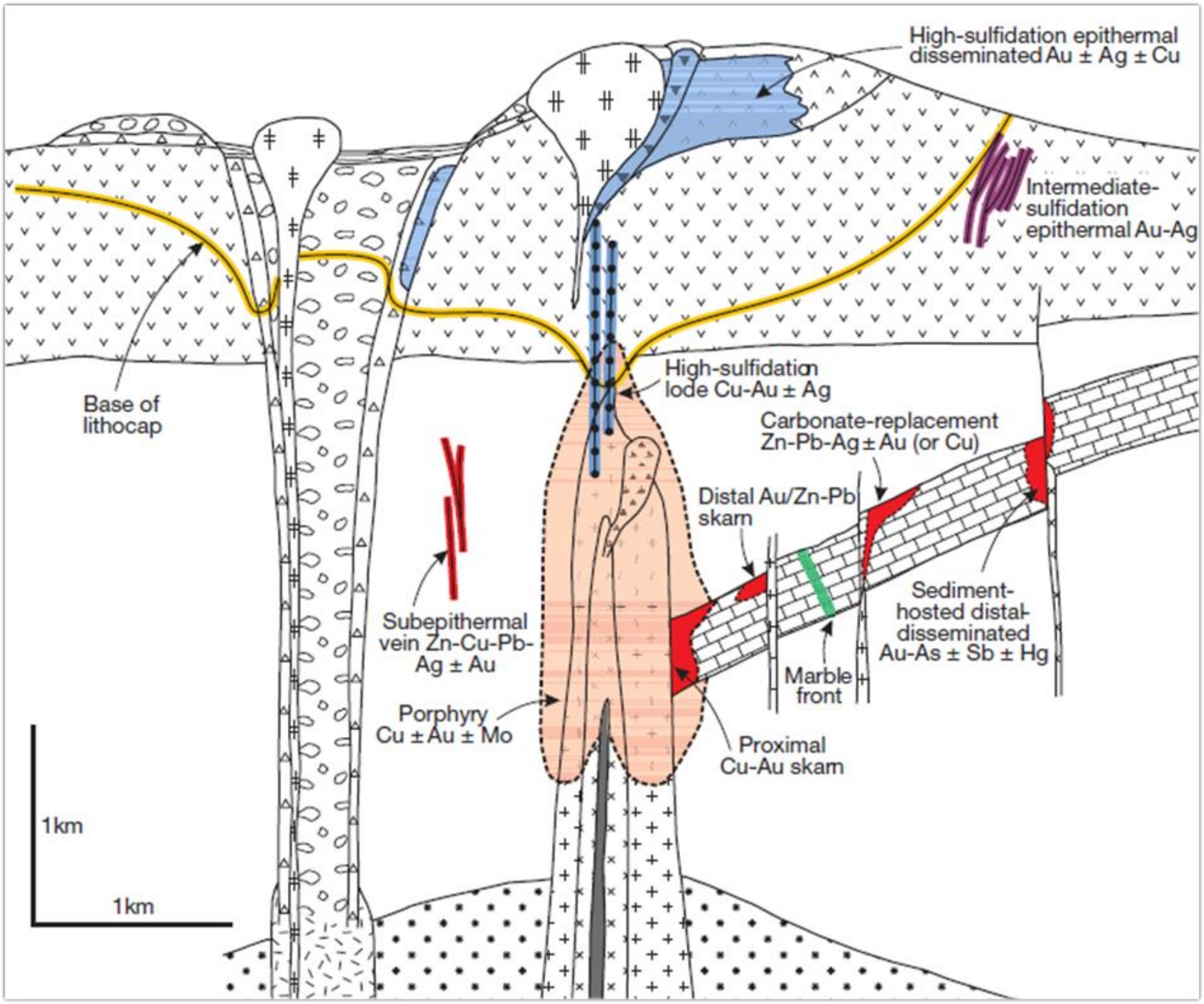
The Alpala porphyry copper-gold-silver deposit at the Cascabel project in northern Ecuador occurs near the overlap of Eocene and Miocene Andean porphyry belts that extend from Colombia through Ecuador and Peru into Chile and Argentina (Figure 8-2). The deposit, formed in the Eocene, is similar in age to the giant La Escondida and El Abra deposits in Chile (Cunningham et al., 2008).



Source: Artica et al., 2022

Figure 8-2: Location of the Alpala porphyry copper-gold-silver deposit, Cascabel project in the Eocene porphyry belt of northern Ecuador

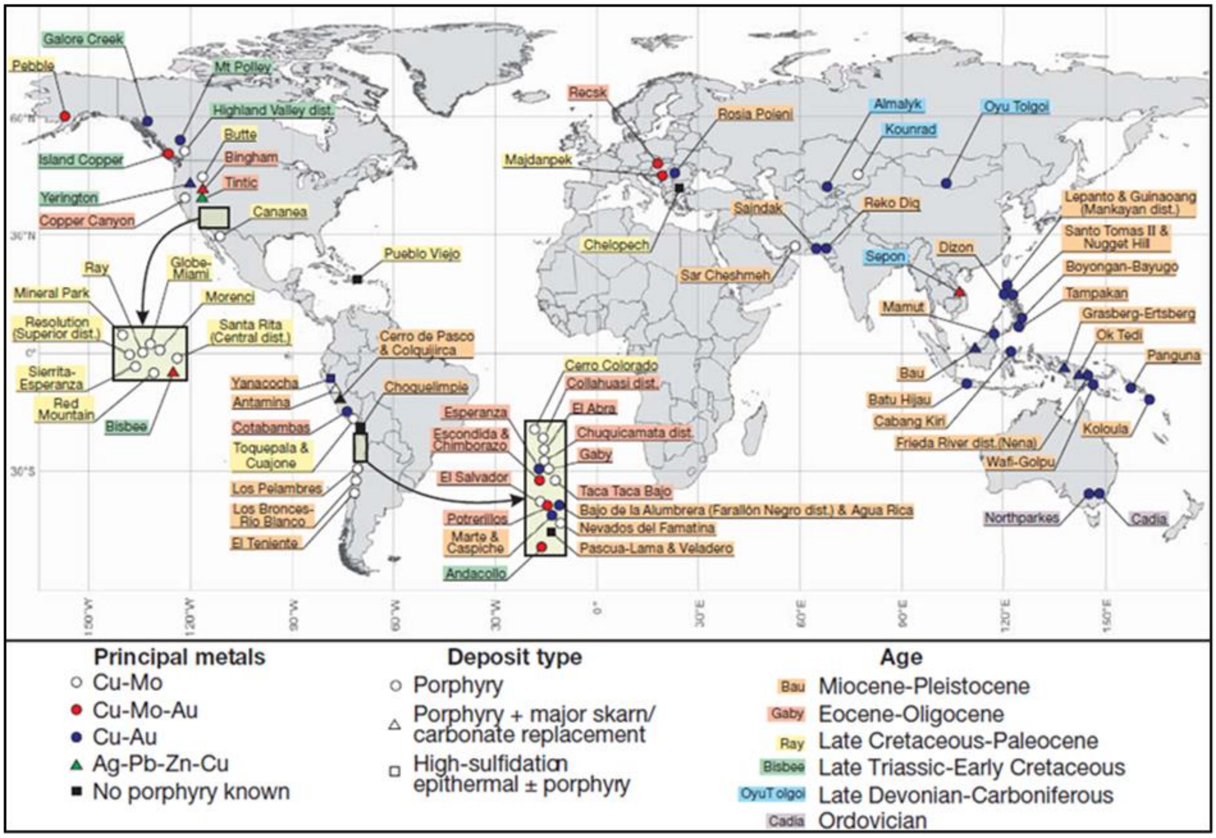
An idealised schematic of porphyry deposits illustrating the classic generic model and possible related deposit types is shown in Figure 8-3.



Source: Sillitoe, 2010

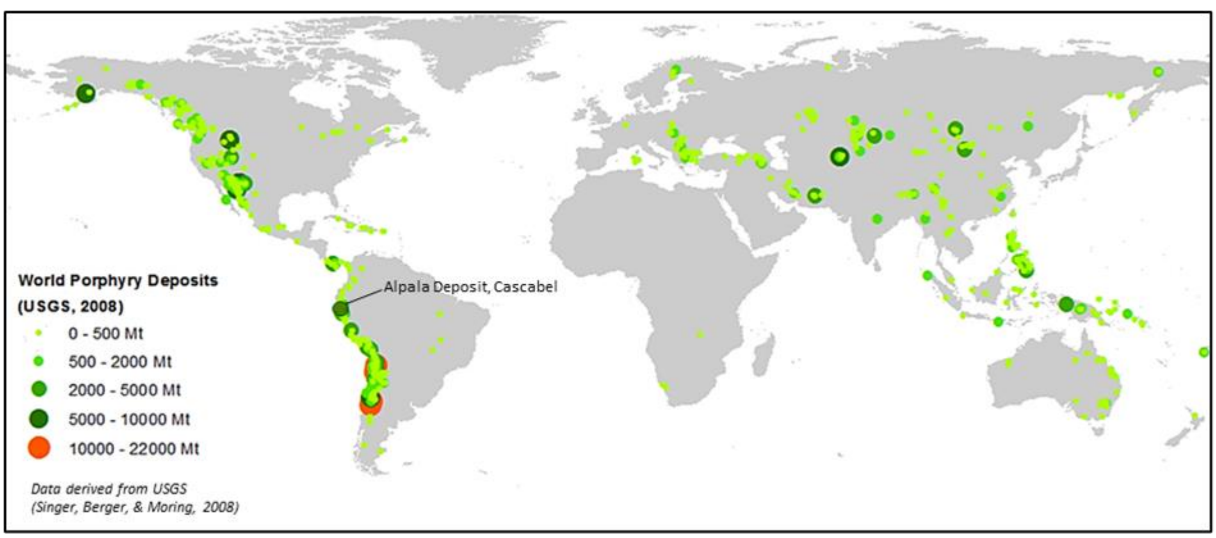
Figure 8-3: Schematic of an idealised copper porphyry deposit illustrating the classic generic model and possible related deposit types

The global distribution of porphyry deposits and their documented ages and size are shown below in Figure 8-4 and Figure 8-5, respectively.



Source: Sillitoe, 2010

Figure 8-4: Distribution of copper porphyry deposits and their documented ages



Source: Wood, 2019

Figure 8-5: Distribution of world porphyry deposits and their documented size by tonnage (Mt)

9 Exploration

9.1 Summary

Previous exploration of the Project area, extending from 1980 to 2011, focused on the source of Au, Cu, Pb and Zn in stream sediments, which led to the location of Au-bearing, polymetallic epithermal quartz veins in streams that flank the Alpala deposit.

Previous explorers focused on polymetallic base-metal Au veins and pan concentrate Au showings occurring some 800 m to 2,000 m from what would become the Alpala discovery outcrop at Alpala Creek. Intermediate epithermal polymetallic base metal Au veining and associated widespread Cu, Au, Mo, and Zn anomalism typically form peripheral to porphyry centres.

More recent exploration of the Cascabel project began with the acquisition of the Project by Cornerstone from Santa Barbara Copper and Gold S.A. in 2011. Cornerstone expanded on preliminary exploration conducted in the 1980s by completing reconnaissance mapping alongside stream sediment, panned concentrate and rock chip sampling totalling 93 samples from June to July 2011.

In May 2012, SolGold commenced the first systematic exploration of the concession with reconnaissance field mapping. SolGold's exploration has initially targeted the license as a whole and specifically targeted a number of the priority prospects within the license. Amongst the first exploration techniques employed by SolGold (ENSA) on the Cascabel project included reconnaissance mapping and the commencement of geochemical sampling of stream sediment, soil, and rock, which built on historical programs.

This early work noted the transition from chlorite-propylitic alteration in lower Moran Creek to epidote-(chlorite)-propylitic alteration in the Moran Creek headwaters, postulating a vector toward the south for a potential porphyry source. Soon after, a reconnaissance mapping team began exploring the southern watersheds of the concession and located an approximately 80 m wide zone of Cu and Au bearing, sheeted, porphyry-style quartz veins in Alpala Creek.

9.2 Recent Exploration

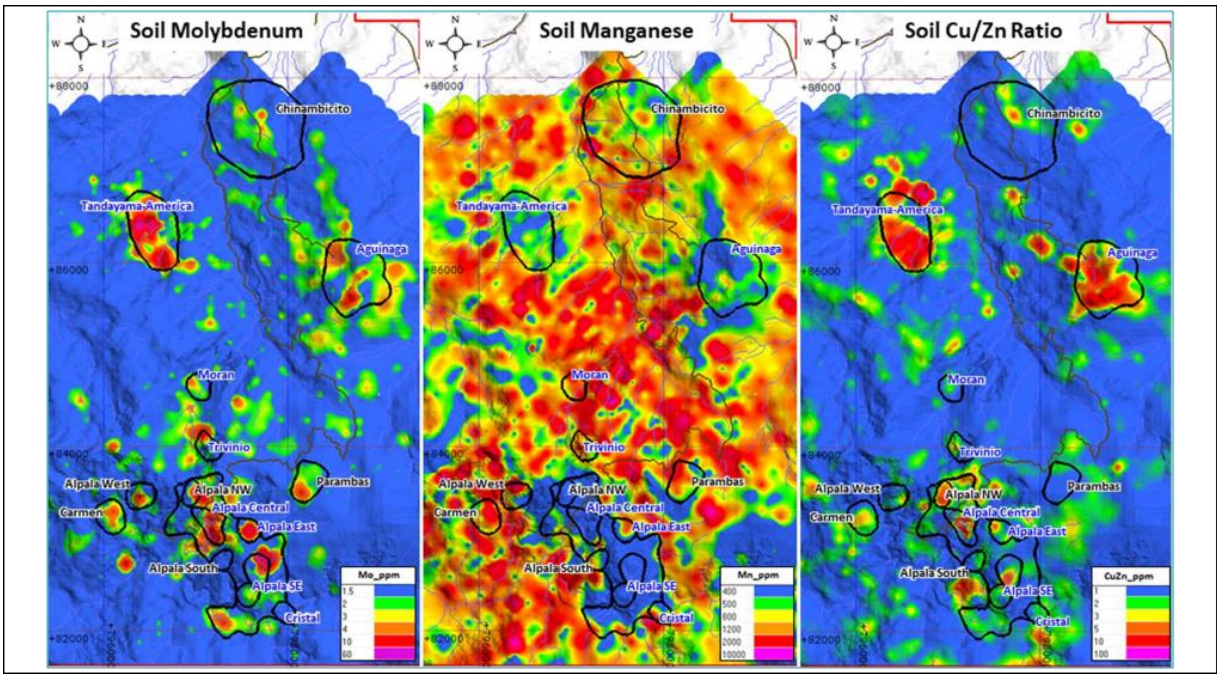
A helicopter-borne magnetics and radiometric survey was conducted over the entire Cascabel tenement in November 2012. A reduced-to-the-pole magnetic high/low complex was identified to be broadly coincident with a 1.5 km by 2.2 km Mo (greater than 1.4 ppm) soil anomaly that encompasses the Alpala discovery zone.

The grid that is used for all co-ordinates on the Project to date is WGS84-UTM-17N (EPSG:32617), and all historical data collected under the Peru South America Datum (PSAD56-17N) has been transformed to WGS84- UTM-17N. No local grids have been used for the Project as yet. However, the establishment of a local mine grid is planned. Six permanent first-order survey ground control points have been installed at the tenement using a differential DGPS and marked by solid concrete plinths. The control points were installed and calibrated following the procedure of the Manual of Technical Specifications - Geodetic Surveys - Horizontal Control of the Geographic Military Institute of Ecuador.

A 3D airborne laser scanning Light Detection and Ranging (Lidar) topographic survey was completed in November 2018. Following 3D airborne LiDAR and photogrammetry topographic surveys in February 2020, the DTM utilised for all studies was finalised.

Multi-element grid-based soil geochemistry studies took approximately 3,287 soil grid samples and 550 soil auger samples across an area of approximately 35 km^2. They indicated several zones of coincident Au, Cu, Mo, and Cu-Zn ratio anomalies across several interpreted porphyry centres. The recognition of geochemical zoning has assisted drill targeting within the deposit and tenement-wide exploration. This zoning is characterised by central Cu-Au, proximal Mo, proximal to distal Bi, Se and Te, and distal As, Mn and Zn. The central portions of the three major porphyry systems discovered to date show high Cu/Zn and Mo/Mn ratios in soil and rock-chip samples. Within the Alpala deposit, variations of Au/Cu ratios in drill holes assist in the delineation of different intrusion stages.

The soil survey identified widespread geochemical anomalies, including at least four major porphyry centres characterised by coincident Au, Cu and Mo highs which consist of the Alpala cluster, Moran, Aguinaga and Tandayama-America (Figure 9-1).



Source: Artica et al., 2022
Note: Showing Mo, Mn, and Cu/Zn ratio

Figure 9-1: Summary of soil geochemical results for the Cascabel concession

The application of the Anaconda method to geological mapping and drill core logging has facilitated the identification of more than six major intrusion stages and a vein paragenesis that allows for the prediction of Cu-Au grades in the Alpala deposit. The most important indicators of high-grade mineralisation include the presence of early-stage causal intrusion(s), elevated porphyry-style vein abundance and an increased ratio of chalcopyrite to pyrite.

The spectral analysis of the grid soil samples led to the identification of zoned clay-mica alteration assemblages over an area of approximately 2.5 km by 1 km, centred approximately over the discovery outcrop and hydrothermal alteration (at surface). These results were deduced to represent the structurally controlled roots of a lithocap above the Alpala porphyry system.

Building on past 1:10,000 scale mapping of the Project area, SolGold field teams commenced 1:2,000 and 1:500 scale Anaconda-style geological mapping over the tenement area, and updates to the local geology map remain ongoing.

A total of 524 rock chip and grab samples have been taken across the tenement during reconnaissance mapping to identify and define the surface mineralisation and geology. This formed the basis of an extensive channel sampling program, resulting in 1,434 rock-saw channel samples cut from 262 surface rock exposures.

Rock-saw channel results confirmed the significance of sheeted, porphyry-style quartz veins in Alpala Creek and inferred an upper margin of a mineralised porphyry system.

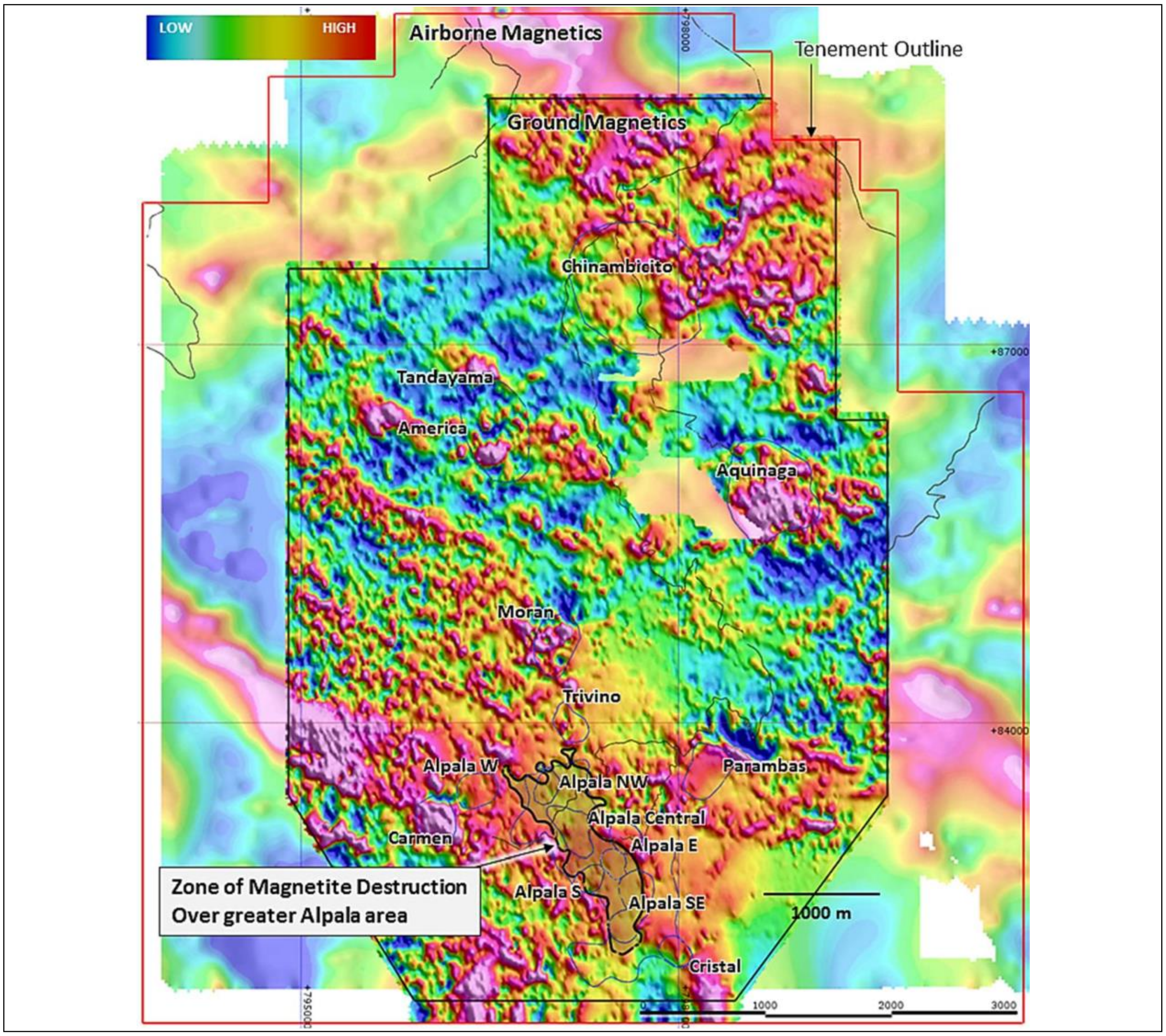
The channel sampling and structural measurements of quartz veins over an area of approximately 430 m by 200 m at Alpala provided the geological context for a diamond drilling program using a man-portable drill rig that commenced in September 2013. Channel sampling across the Alpala discovery outcrops within Alpala Creek returned 4 m at 0.99% Cu and 3.30 g/t Au; 33.3 m at 0.65% Cu and 1.02 g/t Au; and 56.9 m at 0.34% Cu and 1.16 g/t Au. These channel results allowed SolGold to plan the first targeted diamond core drill holes into the Alpala system.

The first four holes of the drill program confirmed the surface mineralisation to depths of approximately 200 m. However, the course of the program was changed by the length and high grades of chalcopyrite-bearing quartz vein stockworks encountered in Hole 5, which was started less than 18 months after the location of surface mineralisation. This fifth drill hole marks the discovery of the high-grade, world-class Alpala Porphyry Copper-Gold-Silver Deposit.

A ground magnetic survey was completed over an area of approximately 30 km^2 of the Cascabel tenement in February 2017. This represented 650 km of total-field magnetic data that was acquired from east-west oriented lines spaced every 50 m. The reduced-to-the-pole image for the ground magnetics data identified a major zone of magnetite destruction that occurs over much of the Alpala porphyry area.

The 3D magnetic inversion (MVI) models, based on the ground magnetic data in the Alpala region, mostly coincide with subsurface mineralised envelopes and reveal a northwest trending line of significant magnetic bodies at Moran, Trivinio, Alpala Northwest, and Alpala Central.

The reduced-to-the-pole image for the ground magnetics data shows that a major zone of magnetite destruction occurs over much of the Alpala porphyry area (Figure 9-2).



Source: Artica et al., 2022
Note: Showing high-resolution RTP magnetic imagery with the 2017 ground magnetic survey area (black outline) over previously flown heliborne RTP magnetic imagery

Figure 9-2: Cascabel concession area (red outline)

The zone of magnetite destruction at Alpala is related to intense hydrothermal, phyllic and advanced argillic alteration that has converted magnetite to pyrite (+hematite) and chalcopyrite from surface to depths of more than 750 m, as determined from drilling. Below this depth, high-grade Cu and Au mineralisation occurs with magnetite-rich, hydrothermally altered intrusions.

Petrographic studies have identified more than six major intrusion phases delineated on composition and relative timing relationships with porphyry-related vein-stages.

Scanning Electron Microscopy (SEM) techniques indicate that the primary Cu minerals are chalcopyrite and bornite. Also, Au occurs as discrete electrum grains (typically 65% to 85% Au) ranging from 1 to 50 microns in diameter. The electrum grains occur within chalcopyrite, bornite, pyrite and rarely quartz and anhydrite. Grains of low-Ag electrum (greater than 90% Au) that are 1 to 3 µm in diameter are associated with sulphide grains and occur locally within silicate minerals.

9.3 Comments on Section 9

Geochemical sampling, geological mapping and geophysical surveys have identified several anomalies, a portion of which have been drill-tested.

Exploration programs conducted are appropriate to the work phase conducted at the time.

The methods used were adequate for the models used of porphyry-style deposits, and the results were instrumental in properly outlining the extent of the mineralisation and defining the Cascabel concession Alpala and Tandayama-America deposits and other prospects.

There is considerable remaining exploration potential within the Project area, notably extensions to the underground mineralisation at:

- TAM deposit where the limits of mineralisation remain open towards both the northwest and the southeast
- The Aguinaga deposit, where mineralisation remains open at depth to the northwest

10 Drilling

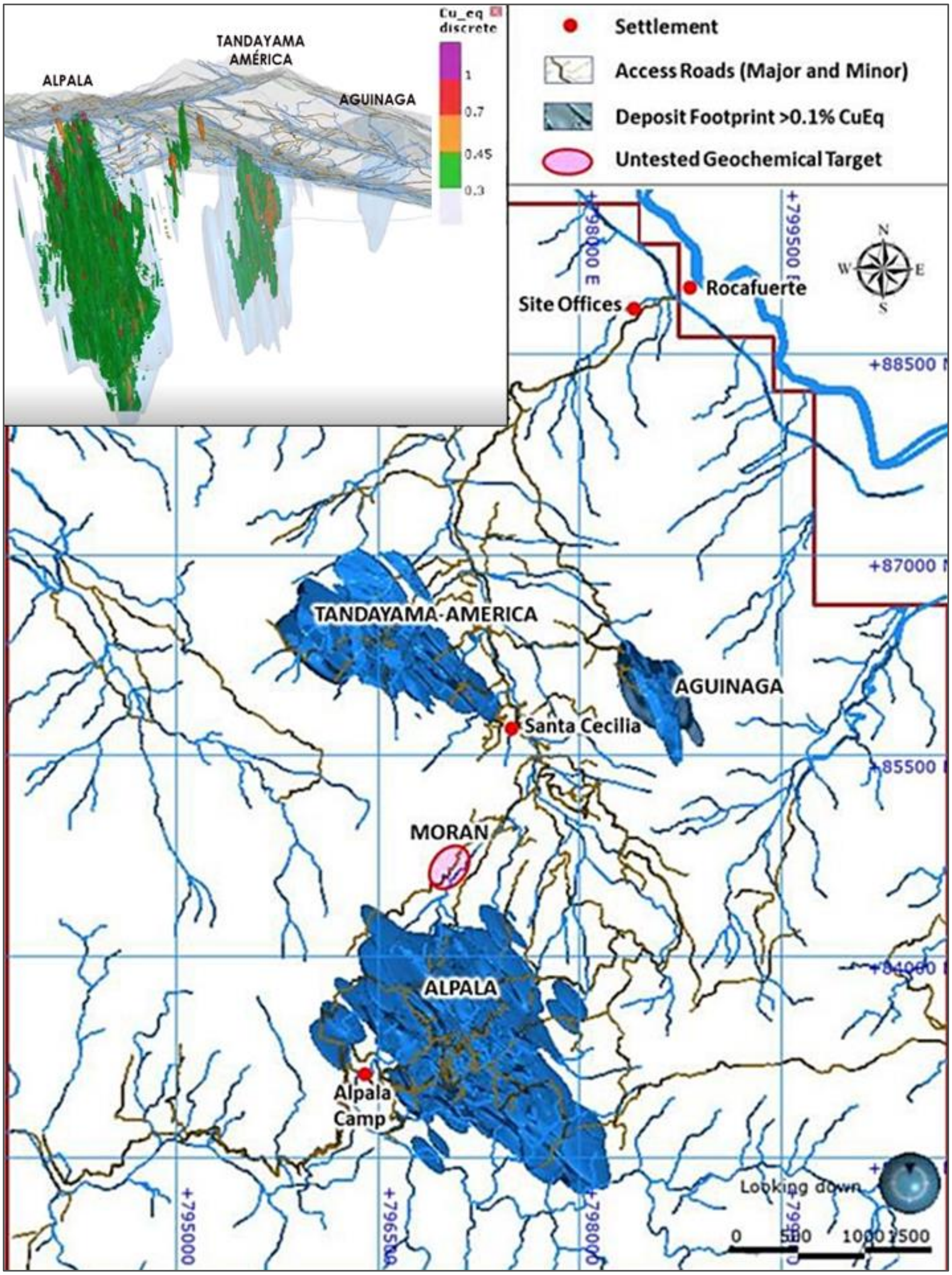
10.1 Summary

Drilling at Cascabel was undertaken in a professional manner in line with industry best practices:

- Drilling is undertaken by reputable drilling contractors with modern drilling equipment.
- Accurate location of drill hole collars is determined by differential GPS surveying methods.
- Recording of downhole surveys at 30 m intervals, or 1 m in the case of daughter holes.
- Achieving core recoveries of 97.5% and greater.
- Transporting core in lidded boxes and securing an undercover facility.
- Pioneering the world's first development of centrifugal and hydraulic Man-Portable Sediment Removal Units (MPSRU).
- Achievement of several record deep drill holes, including 11 drill holes over 2,000 m in depth and recognition of a world record for the deepest drill hole completed by a man-portable drill rig in hole CSD-19-110 (2,623.75 m).

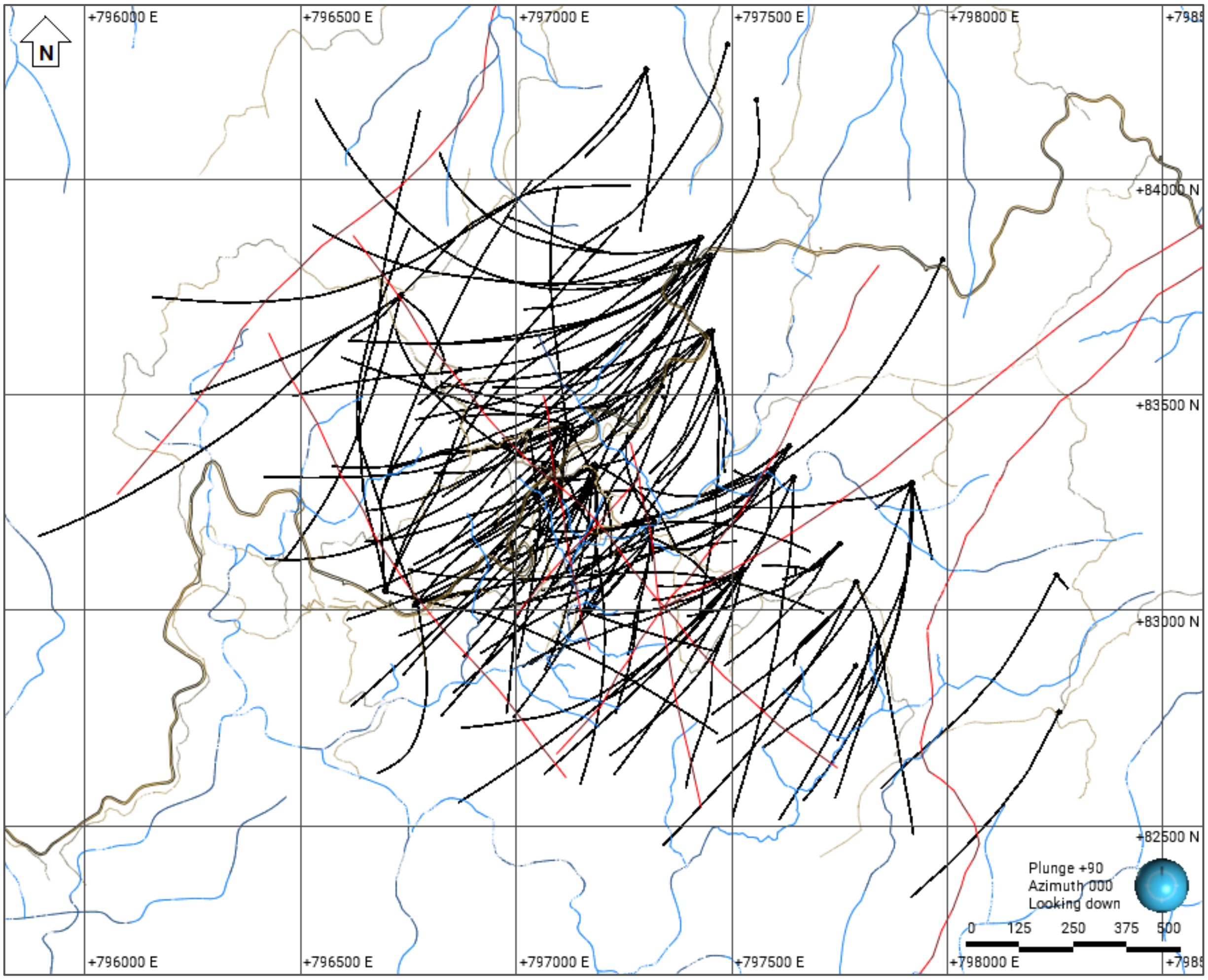
Existing drilling at the Cascabel project has focussed on delineating copper and gold mineralisation at a cluster of Eocene-aged porphyry deposits and prospects. Three significant deposits have been identified thus far at Cascabel, namely the giant Alpala porphyry copper-gold-silver deposit, the Tandayama-America porphyry copper-gold deposit, and the Aguinaga porphyry copper-gold deposit (Figure 10-1).

Since drilling commenced on 1 September 2013, a total of 310,335 m of diamond drilling has been completed at the Cascabel project to date. Drilling includes 265,224 m at the Alpala deposit, 36,111.09 m at the Tandayama-America deposit, 8,970.6 m at the Aguinaga deposit, and a further 6,774.2 m of drilling completed on infrastructure, sterilisation and water monitoring (Figure 10-2, Figure 10-3 and Figure 10-4).



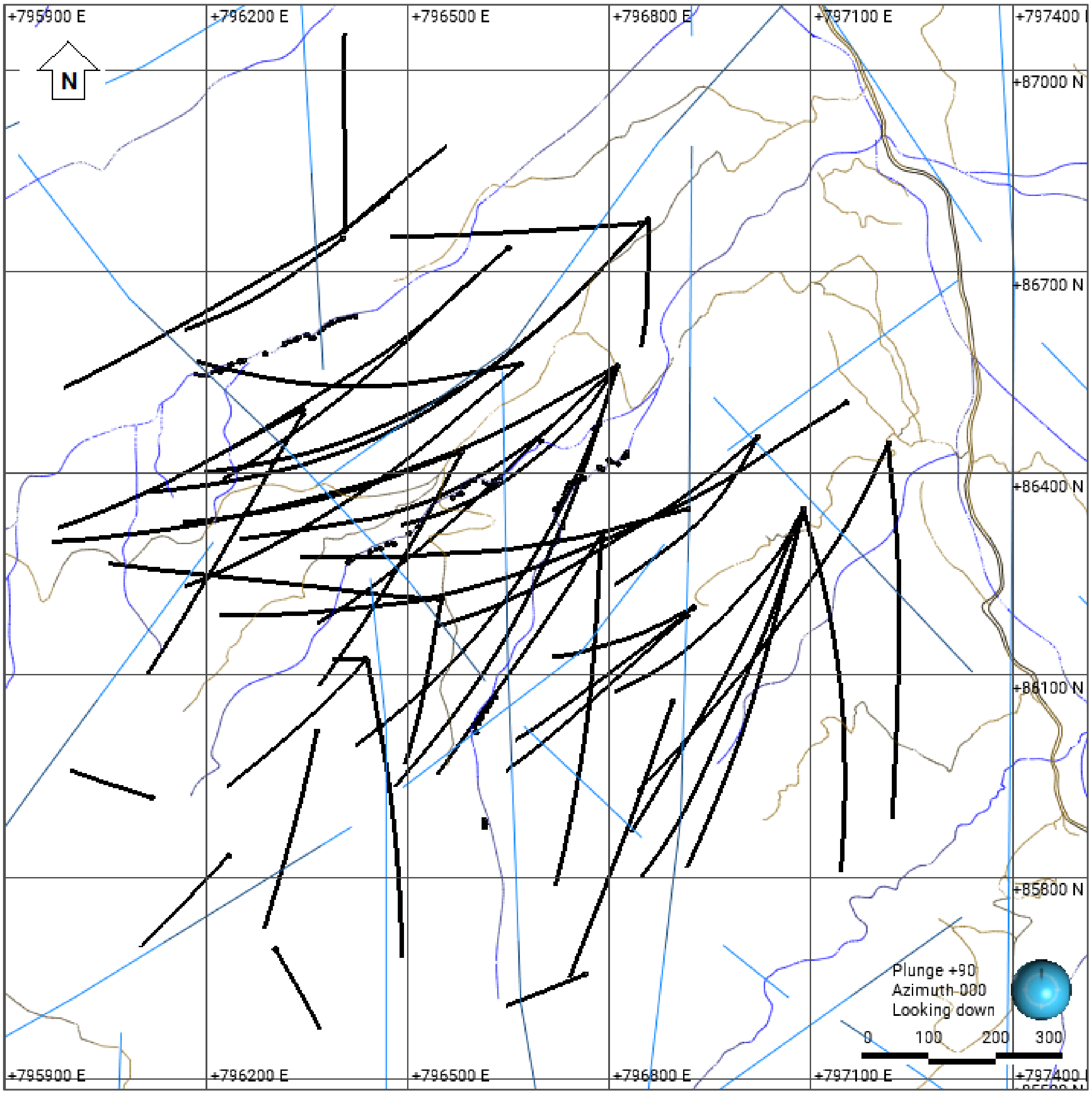
Source: Artica et al., 2022

Figure 10-1: Porphyry deposits at the Cascabel concession in northern Ecuador



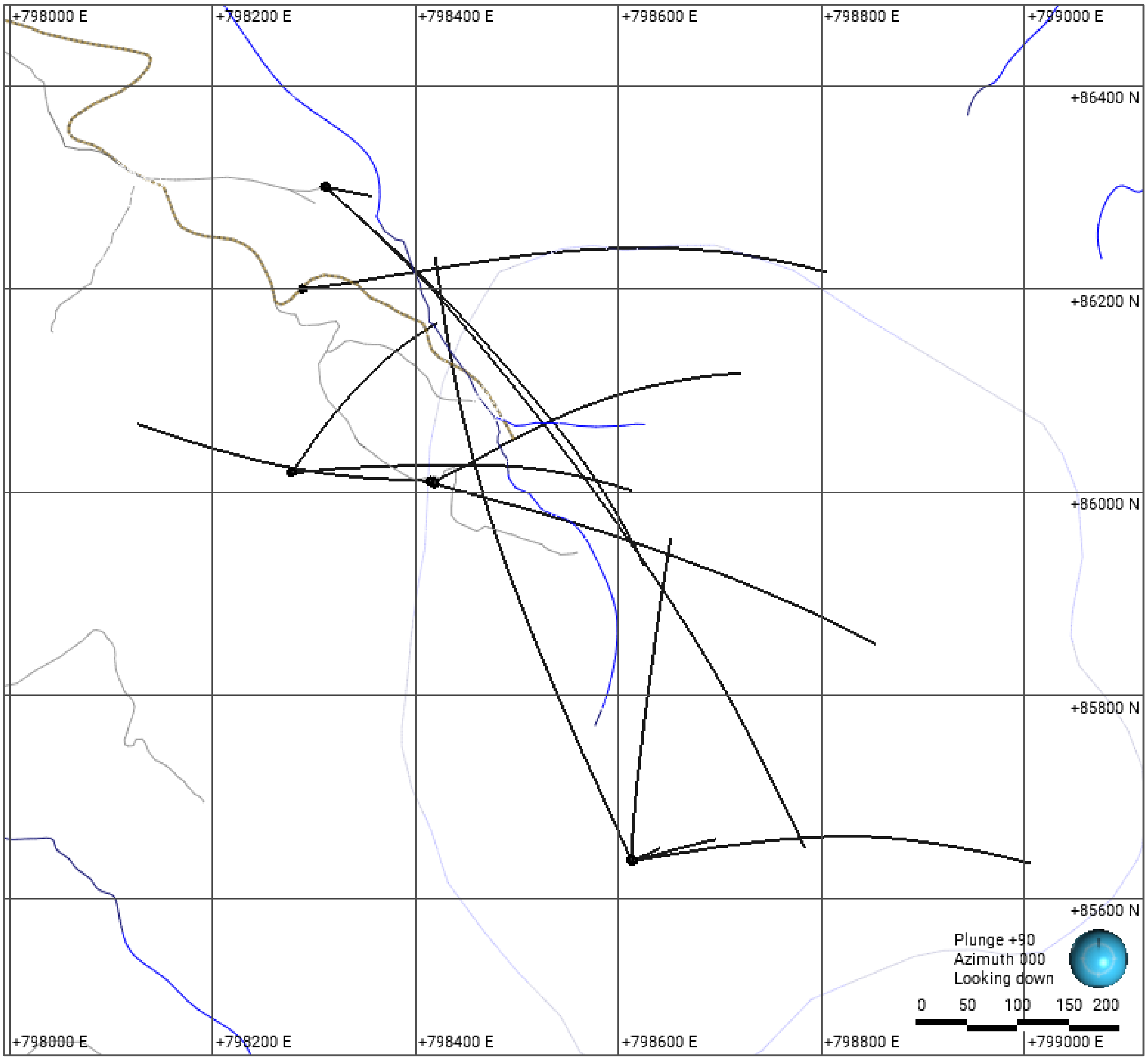
Source: Artica et al., 2022

Figure 10-2: Drill plan of the Alpala deposit



Source: Artica et al., 2022

Figure 10-3: Drill plan for the Tandayama-America deposit



Source: Artica et al., 2022

Figure 10-4: Drill plan for the Aguinaga deposit

The latest Alpala Mineral Resource Update (MRE#4) was estimated from drill holes and rock-saw channel samples that lie within the Alpala Block Model limits, which included 108,368 assays representing 240,663 m of diamond drilling in 185 drill holes and 696 assays representing 1441 m of rock-saw channel samples cut from surface rock exposures.

The Tandayama-America (TAM Mineral Resource (MRE#3) was estimated from drill holes and rock-saw channel samples that lie within the TAM Block Model limits, which included 17,558 assays, representing 36,144 m of diamond drilling in 51 drill holes and 220 assays representing 458 m of rock-saw channel samples cut from surface rock exposures.

Re-drills and down-hole over-runs were omitted from the MRE datasets, as were any drill holes that lie outside the extent of mineralisation.

10.2 Collar Surveys

The azimuth and dip of the drill rig setup are measured and checked ahead of the start of all drill holes. Collar locations are initially surveyed using hand-held GPS units by SolGold personnel; however, the final collar location is surveyed by professional surveyors after drilling has been completed using a Differential GPS (DGPS) theodolite.

All drill hole collars are clearly labelled and capped on completion (Figure 10-5).

Source: This study, 2024

Figure 10-5: Example of collar location for Alpala deposit

10.3 Downhole Surveys

A Single Shot Reflex Ezi-Gyro system was used to provide downhole survey information upon completion of each drill hole. Readings were initially taken at 50 m intervals for holes 1 to 33. Thereafter, readings were taken at 30 m intervals and provided to the SolGold geology team. Any deflection of more than 10 degrees in dip or azimuth within a 30 m interval is resurveyed.

Where daughter holes are drilled, a magnetic survey tool integrated into the steerable Devico DeviDrill navigational tool is used at 1 m intervals to steer the hole onto the new azimuth/dip.

10.4 Diamond Drilling Procedures

Drilling utilised modified custom-built man-portable machines, capable of drilling NQ to 2,500 m. Track-mounted Sandvik DE880 machines, capable of drilling NQ to 3,000 m, and track-mounted Tech 5000 machines, capable of drilling NQ to 3,400 m, were also used.

A steerable wireline core barrel was utilised to drill daughter holes from a parent hole and perform corrections to drill hole trajectories.

All drilling is undertaken using a Diamond Core triple tube at either PQ3 (83.0 mm core), HQ3 (61.1 mm core) or NQ3 (45.0 mm core) core size. Core was produced in 3 m core runs, and each core run underwent orientation. Drillholes were surveyed for dip and azimuth at 30 m intervals.

10.5 Core Recovery

Drill core recovery results achieved average recoveries of 97.5%.

10.6 Core Storage

The core is labelled and photographed before being transported from the drill sites by four-wheel drive (4WD) vehicles to the Project office at Rocafuerte, where it is logged, re-photographed, and sampled. All core from the Alpala project is stored under cover at the camp, adjacent to the logging area.

10.7 True Thickness

The term “true thickness” is not generally applicable to porphyry-style deposits as the entire rock mass is potentially mineralised and there is often no preferred orientation to the mineralisation. In areas that display porphyry-style mineralisation, in general, most drill holes intersect mineralised zones at an angle, and the drill hole intercept widths reported for those drill holes are typically greater than the true widths of the mineralisation at the drill intercept point.

Significant down-hole drill intercepts were reported using a data aggregation method based on copper equivalent (CuEq) shut-off grades with up to 10 m internal dilution, excluding bridging to a single sample and with a minimum intersection length of 50 m.

The true width of down-hole intersections reported is expected to be approximately 25-70% of the down-hole lengths, depending on the attitude of the drill hole. Drill hole inclinations range from -45 to -90 degrees.

10.8 Comments on Section 10

The initial spread and design of the drill holes at Alpala were impacted by site access and topography, hence the use of man-portable rigs. This has resulted in some low intersection angles with the mineralised deposit, which is not uncommon when drilling a steeply dipping porphyry deposit. With the introduction of further drill rigs and the use of the Devico orientation device, SolGold was able to better target and drill the mineralisation at Alpala and Tandayama-America.

SolGold has achieved a high degree of control over the complex multi-contractor drilling programs through the use of its own independent foreman.

From Dr. Arseneau's review during the technical site visit, the drilling at Alpala has been conducted in a professional manner using industry-accepted practices and produced core of sufficient quality and recovery to be used in Mineral Resource Estimations.

Dr. Arseneau is unaware of any material factors that would impact the accuracy and reliability of the sample results.

11 Sample Preparation, Analyses and Security

11.1 Sampling Methods

11.1.1 Core Sample Selection and Mark-up

Prior to cutting and the collection of drill core samples, all core is marked up for sampling. A standard sampling interval of 2 m is employed by SolGold, although smaller samples may be taken in significant zones or narrow veins of geological significance.

All drilling phases have applied a minimum sample length of 25 cm. In the situation where a small sample interval is required, the zone is typically sampled at its geological contacts or limits. All drill core is half-cut, excepting special intervals during wedging, over-runs, or deviations. Half of the sample is retained within wooden or plastic core boxes, and the sampled half is submitted for assay. On occasion, detailed lithotype, petrographic, or petrophysical samples may be taken as a quarter core, leaving a quarter sample retained within the core box.

11.1.2 Core Cutting

Field technicians mark all competent drill core longitudinally at the drill site during the core orientation process of each drill run. Where broken ground is encountered, a longitudinal mark-up is often not practical.

Before cutting, all core is again re-marked longitudinally, indicating the drill hole's vertical axis, hence ensuring that the volume of core sampled is representative of half the drill core generated during drilling. Core is cut along the marked line by field technicians or a geologist.

Intervals of highly broken core that may be washed away by the water supply during cutting are wrapped in plastic and/or masking tape to increase retention of fines. Intervals of extremely broken or fragmented or clay-rich core are left in the core tray without cutting using the saw and are split by cleaver and spatula.

Following cutting or splitting each core segment, the entire volume is returned to the appropriate location in the core box prior to being selected for sampling.

11.1.3 Core Sample Collection

Half core is sampled including coarse and fine rock fragments. Where there is significant fine material, a trowel is used to ensure that at least 50% of the fines are included in the sample. All material is placed into high-strength plastic sample bags, which in turn are placed into calico sample bags, ensuring that fine sample particles are not lost due to the open weave of the calico bag.

Sample numbers are written on the exterior of the plastic bags with a waterproof marker, and a corresponding barcoded plastic sample ticket is placed into each plastic bag.

11.1.4 Core Magnetic Susceptibility Analysis

Following the sampling of the core, magnetic susceptibility measurements are taken of the bagged half-core samples over the entire length of each drill hole at 2 m intervals. Each interval is measured three times for magnetic susceptibility and an averaged susceptibility value is utilised. All measurements are taken using a KT-10 Magnetic Susceptibility Meter manufactured by Terraplus.

11.1.5 Channel Sampling

Rock outcrop channel samples were taken with the use of a handheld rock saw utilising 12” diamond saw blades. The channels were cut continuously across the rock face while being mindful to sample both visibly and non-visibly mineralised rock equally. All samples were cut at a nominal 50 mm width by 50 mm depth and sampled at one or two-metre intervals depending on the outcrop position and exposed length. The depth and width of the channels was approximately equivalent to the diameter of the NQ3 drill hole diameter (45 mm), which has been used in the Alpala and Tandayama-America MREs.

Samples were geologically logged, bagged and labelled using the same procedures as detailed for the drill core, and sample dispatches were subject to the same QAQC protocols as the diamond drill hole, which are detailed below.

For the Alpala MRE, the channel samples were used to guide the geological interpretation. However, the grades were not used as input to the grade estimate.

For the Tandayama-America MRE, channel samples were used to guide the geological interpretation and their grades were also used as input to the grade estimate.

11.2 Bulk Density Determinations

A total of 12,478 at Alpala and 1,735 at Tandayama-America bulk density determinations were assessed for the Resource Estimation. Bulk density has been conducted on 10 cm long pieces of drill core overall resource drilling programs at Cascabel.

Bulk density has been measured using a wax-coated water immersion method. Core is sawed orthogonally to provide a smooth-ended, 10 cm long core cylinder and placed into a small drying oven at 150°C. PQ-sized core is dried for a period of 3 hours, HQ-sized core for 2 hours, and NQ-sized core or smaller is dried for 1 hour.

The core is then weighed to provide the mass of the dried unwaxed sample in the air (Measurement A). The dried core is then coated in wax before a second measurement of the mass of the wax-coated core in air is completed (Measurement B). The wax-coated core is then submerged in water and weighed to provide the mass of the submerged waxed-coated sample (Measurement C).

The bulk density is then calculated using the equation below, utilising a density of 0.914 g/cm^3 for the paraffin wax.

$$BD = \frac{A}{[B - C] - (B - A)/0.914}$$

11.3 QAQC Sample Protocols

11.3.1 Sampling and Analytical Methods

SolGold operates according to its rigorous Quality Assurance and Quality Control (QAQC) protocol, which meets industry-accepted practices.

Primary sample collection involves secure transport from SolGold's concessions in Ecuador to independent certified sample preparation facilities in Cuenca or Quito, Ecuador. Samples are then typically air freighted from Quito to certified laboratories in either Lima, Peru, or Vancouver, Canada, where the assaying of drill core, channel samples, rock chips and soil samples is undertaken. SolGold typically utilises ALS-certified laboratories in Lima, Peru for geochemical analysis and ALS certified laboratories in Vancouver, Canada for the analysis of metallurgical samples.

Samples are typically prepared and analysed using 100 g 4-Acid digest ICP with MS finish for 48 elements on a 0.25 g aliquot (ME-MS61). Laboratory performance is routinely monitored using umpire assays, check batches and inter-laboratory comparisons.

To monitor the ongoing quality of its analytical database, SolGold's QAQC protocol encompasses standard sampling methodologies, including the insertion of certified powder blanks, coarse chip blanks, standards, pulp duplicates and field duplicates. The blanks and standards are Certified Reference Materials supplied by Ore Research and Exploration, Australia (OREAS).

SolGold's QAQC protocol also monitors the ongoing quality of its analytical database. The Company's protocol involves independent data validation of the digital analytical database, including searches for sample overlaps, duplicate or absent samples, as well as anomalous assay and survey results. These are routinely performed ahead of Mineral Resource Estimates and Mining Studies. No material QAQC issues have been identified with respect to sample collection, security and assaying.

11.3.2 Certified Reference Materials (Standards)

Since the commencement of the drilling at Alpala, SolGold has utilised nine different Certified Reference Materials (CRMs) in the analysis sample stream. CRMs were sourced from CDN Resource Laboratories Ltd, Canada (CDN), used between October 2013 and July 2016, and OREAS CRMs have been used from March 2015 to the present.

CRMs are inserted at specific sample numbers, equating to approximately one every 50 metres of drilling. They are typically 60 gm matrix-matched pulp CRMs, providing a range of low, medium and high grades. All standards are re-bagged and labelled with ENSA sample numbers before being inserted into the sample lot.

CRM utilisation represents a rate of 1:52 samples or 1.9%.

11.3.3 Field Duplicates

Field duplicate samples are collected from the drill core during sampling and are selected at specific sample numbers, equating to approximately one every 33 m of drilling.

The duplicate sample is cut from the existing half-core assay sample to provide an original quarter-core and a second-quarter-core Field Duplicate. The two samples are submitted to the assay laboratory for analysis and follow each other through the sample preparation and assaying process.

Field duplicate utilisation represents a rate of 1:35 samples or 2.8%.

11.3.4 Blanks

Certified blank material sourced from CDN and OREAS has been inserted into the sample stream at a typical rate of 1:50 samples (2%). All blanks used are 60 gm pulp CRMs, except for the use of coarse chip blank (OREAS C27c), introduced in September 2017. The coarse chip blank was introduced in response to risks identified during a laboratory due diligence visit, noting that minor smearing of soft sulphide minerals and/or Au during ring milling pulverisation process could be monitored using a coarse blank.

Field duplicate utilisation represents a rate of 1:50 samples or 2.0%.

11.3.5 Umpire Laboratory Checks

Routine inter-laboratory Umpire or Lab Check analyses are undertaken on pulp samples returned to the Quito storage facility. Batches of pulps from previously assayed samples (by ALS Global) representing a range of high, medium and low grades are selected and dispatched to a second umpire laboratory (Bureau Veritas/ACME) for analysis.

The database geologists determine the number of required cross-check samples for a given period as 5-7% of the returned samples. They generate a range of consecutive sample numbers from each dispatch to be located by a technician at the storage facility. The technician locates a consecutive range of samples from within the defined range and dispatches them to the umpire lab as instructed by the database geologist.

Umpire pulp duplicate utilisation represents a rate of 1:20 samples or 4.9%.

11.4 QAQC Sample Results

Detailed analysis of the results from the QAQC samples has been undertaken by SolGold and Independently reviewed by the QPs who have worked on the Alpala Project.

11.4.1 Alpala MRE#4 Dataset

Dr Arseneau (QP) made the following observations from the review of the QA/QC data collected.

Certified Reference Materials

- Cu CRM results showed good compliance for the period, with very few results outside the error limits, and no significant grade bias was noted.

- Au results for CRM 501c showed two errors, while those for CRM 502c showed one. These represent 0.5% and 0.3% error rates, respectively, which the QP considers immaterial for the MRE.
- No significant bias was observed in the Au CRM results for the reporting period.
- Ag CRM results showed good compliance for the period, with zero results outside the error limits.
- CRM 501c did show a negative bias, which was significant at -5.4%. The QP considers that Ag values for this CRM should be monitored closely in future. However, the result is acceptable for the current MRE.

Field Duplicates

- The Cu field duplicate results showed no significant bias, and 78% of duplicate pairs had less than 30% Absolute Relative Paired Difference (ARPD), which the QP considers an acceptable result for this duplicate type.
- The Au field duplicate results showed no significant bias, and 78% of duplicate pairs had less than 30% ARPD, which the QP considers an acceptable result for this duplicate type.
- The Ag field duplicate results showed no significant bias, and 82% of duplicate pairs had less than 30% ARPD, which the QP considers acceptable for this duplicate type.

Blanks

- Acceptable failure rates are observed in Au and Ag with the pulp blank 22e and in Au with the coarse blank C27c.
- Coarse blank C27c. is a certified blank for gold only. This blank was introduced to reduce gold smearing in laboratory ring mills. Blank C27c has been successful in this endeavour. Unacceptable failure rates are observed in Cu with this coarse blank. The QP considers that although some of the failures may reflect contamination, it is more likely that C27c is not suitable for use as a blank material for copper, as the certification for C27c only certifies it as a blank for Au. The QP recommends against using it as a blank for Cu and Ag.

Umpire Laboratory Checks

- The Cu umpire laboratory duplicate results showed no significant bias and that 83% of duplicate pairs had less than 10% ARPD, which is acceptable for this duplicate type.

11.4.2 Tandayama-America MRE#3 Dataset

Certified Reference Materials

- Cu results for CRM 501c showed four errors, while results for CRM 502c showed one error. These represent 2.6% and 0.8% error rates, respectively, which the QP considers immaterial for the MRE.
- One extreme outlier for CRM 501c is most likely a labelling error and should be investigated.
- Au results for CRM 501c showed three errors, while those for CRM 502c showed one. These represent 1.9% and 0.8% error rates, respectively, which the QP considers immaterial for the MRE.

- No significant bias was observed in the Cu or Au CRM results for the reporting period.

Field Duplicates

- The Cu field duplicate results showed no significant bias, and 76% of duplicate pairs had less than 30% absolute relative percent difference (ARPD), which the QP considers an acceptable result for this duplicate type.
- The Au field duplicate results showed no significant bias, and 79% of duplicate pairs had less than 30% ARPD, which the QP considers an acceptable result for this duplicate type.

Blanks

- Acceptable failure rates are observed in Cu and Au with the pulp blank 22e, and in Au with the coarse blank C27c showed similar issues as noted for Alpala. Therefore, the QP is satisfied that cross-contamination during sample preparation has not been significant.

Umpire Laboratory Checks

- No significant bias was observed for the reporting period in either Cu or Au.
- The Cu umpire laboratory duplicate results showed no significant bias and that 79% of duplicate pairs had less than 10% ARPD, which is an acceptable result for this duplicate type.
- The Au umpire duplicate results showed that 62% of duplicate pairs had less than 10% ARPD, which should be investigated.

11.5 Analytical and Test Laboratories

Analyses of SolGold exploration samples have been performed by different laboratories over the life of the project (Table 11-1). All laboratories are independent of SolGold and ENSA and are accredited laboratories for the analysis methods used.

Table 11-1: Analytical and test laboratories

Lab Code	Sample Type	Start	End	Preparation	Analyses
ACME	Core Rock Chip Sediment Soil Trench Umpire	Jul 2012	Aug 2016	Aucay and Associados (Cuenca)	Acme (Vancouver) Acquired by Bureau Veritas SA on February 23, 2012 (see Bureau Veritas below)
METSOLVE	Rock Chip	Apr 2016	Aug 2016	Aucay and Associados (Cuenca)	Met-Solve (Vancouver) Certified by Intertek for ISO 9001:2008. Valid 29 April 2014 to 28 April 2017
ALS	Core Rock Chip Trench	Oct 2014	Present	ALS (Quito)	ALS (Lima)
	Metallurgical				Certified by SCC: ISO/IEC 17025:2017. Valid 20 March 2022 to 1 March 2026
Bureau Veritas	Umpire	Jul 2017	Present	NA	Bureau Veritas (Lima) certified by Instituto Nacional de Calidad: NTP- ISO/IEC 17025:2017. Valid 3 June 2019 to 2 July 2023

11.6 Sample Preparation and Analysis

11.6.1 ACME Laboratory – Historical Drilling

All samples were sent to Luis Aucay and Associados, Cuenca Laboratory who undertook the preparation of samples on behalf of ACME (using the ACME procedures), all rock, channel and drill core samples were prepared using standard rock preparation procedures (ACME Code: R200-250/PRP70-250) including crushing (1 kg to ≥ 70% passing 10 mesh (2 mm)), splitting (split to 250 g) and pulverising (≥ 85% passing 200 mesh (75 µm)).

Prepared samples were then assayed by ACME Laboratories in Lima, Peru, using three methods:

- Au by lead collection fire assay (FA) with Atomic Adsorption Spectrometry (AAS) on a 30 g charge sample (FA430/G601)
- Multi-acid digest ICP with an Emission Spectrometry (ES) finish for 35 elements on a 0.25 g aliquot (MA300/1E)
- Multi-acid digest ICP with ES finish for 23 elements on a 0.25 g aliquot (MA370/7TD) (for over limits Ag, Cu, Pb, Zn samples)

Method MA300 detailed above is only a partial analysis for some elements, especially in some S, Cr and Ba-bearing minerals and some oxides of Al, Hf, Mn, Sn, Ta and Zr. Volatilisation during fuming may result in some loss of As, Sb and Au.

Soil samples submitted to ACME have undergone SS80 preparation, which involved drying at 60˚C, sieving to 100 g passing -80 mesh), followed by AQ201 Aqua Regia 1:1:1 digestion ICP with a Mass Spectrometry (MS) analysis for 36 elements.

11.6.2 Met-Solve Laboratories – Historical Drilling

As detailed above, samples submitted to Met-Solve laboratories underwent sample preparation at Luis Aucay and Asociados, Cuenca laboratory.

Samples have been dispatched to Met-Solve laboratories in Langley, British Columbia, Canada, for assay by two methods:

- Au by lead collection FA with AAS finish on a 30 g charge sample (FAS-111)
- 4 acid digest ICP with Atomic Emission Spectrometry (AES) or MS finish on a 0.2 g aliquot (IMS-230/ICF- 6Cu – Ore grade)

11.6.3 ALS Global – Arequipa Geochemistry, Peru (Current Laboratory)

ALS subcontracted sample preparation to Carlos Puig Associados, Quito, until 2019, when they commissioned their lab in Quito.

Samples sent to ALS Laboratories in Quito are prepared by crushing (CRU-31), logging (LOG-22), weighing (LOG-24), pulverisation of 1 kg to 85% passing 75 µm (PUL-32) and splitting (SPL-21), before being transferred to a new sample bag (TRA-21) and re-weighed.

Prepared samples are then dispatched to ALS Lima, Peru, for assaying.

A variety of methods have been used for the analysis of rock, channel and drill core samples by ALS:

- Acid digest ICP with MS finish for 48 elements on a 0.25 g aliquot (ME-MS61)
- Acid digest ICP with AES finish for 33 elements on a 0.25 g aliquot (ME-ICP61)
- Aqua-regia digest ICP with MS finish for 51 elements on a 0.5 g aliquot (ME-MS41)
- Au by lead collection FA with AAS finish on a 30 g charge sample (Au-AA23)
- Au by lead collection FA with a gravimetric finish on a 30 g sample (GRA-21)
- Ag by aqua-regia digestion and AAS finish on a 0.5 g sample (Ag-AA46)
- Cu by aqua-regia digestion with AAS finish on a 0.5 g sample (Cu-AA46) (Over limits Cu)
- Cu by four acid digestion and AAS finish on a 0.4 g sample (Cu-AA62) (Over limits Cu)

Additionally, all samples submitted to ALS Laboratories have been scanned for hyper-spectral mineralogy, combining TerraSpec®, 4HR scanning and aiSIRISTM interpretation (HYP-PKG).

11.6.4 Bureau Veritas (Peru) (Formerly Inspectorate) – External Check Laboratory

External umpire check assays have been undertaken by Bureau Veritas (Peru) via their Quito office.

The analyses of pulps used the following three methods:

- Au by lead collection FA with AAS finish on a 50 g charge sample (FA450)
- 4 acid digest ICP with MS finish analysis on a 0.25 g sample (4A200)
- 4 acid digestion with AAS Finish (MA402)

11.7 Geological Database

All geological and sample analysis data is managed in the AcQuire geological database. AcQuire uses an SQL Server to store its database, allowing security to be applied at both the database and server levels.

Access to the database is restricted and carefully managed. The data integrity is enforced through the relational database structure, such as the application of primary keys, parent-child relationships and field validations, which, combined with rules established to monitor data import, ensure that only valid data is stored in the database.

Three dedicated database geologists (DBGs) manage the database, including the QAQC of the data and logging import schemes. Two additional geologists are responsible for managing the Terraspec™ spectral logging information.

Technicians in the core shed undertake direct entry of geotechnical and bulk density information. Read-only access is provided to the database through a web application through pre-configured views.

The data collection systems have been configured to ensure minimum manual entry of information, thus reducing the potential for errors to be introduced. Invalid data is flagged and must be corrected before storing it in the database.

Data is provided by geologists or assaying laboratories in predefined data templates. Data is loaded using import procedures designed specifically for those templates. This ensures that the data is loaded correctly and consistently, independent of the operator performing each data load.

11.8 Sample Security

Sample security from the drill site to the laboratory is ensured by the samples always being attended to, stored in the guarded on-site preparation facility, or stored in a secure area prior to laboratory shipment.

Chain-of-custody procedures consist of sample despatch forms and transport certification sent to the laboratory with sample shipments to ensure that all samples are received by the laboratory. Chain-of-custody management was achieved through the bagging of samples with tamper-proof tags.

11.9 Sample Storage

The Rocafuerte core facility is located in a secured compound with dedicated undercover processing and storage areas.

After logging and cutting, samples are bagged, and magnetic susceptibility readings are taken in the sample area. Samples remain there until they are dispatched.

Core is stored in stacked core boxes available for review and future sample requirements. Returned assay pulp samples are held at the laboratory for 90 days and then transported to the SolGold facility in Quito.

11.10 Qualified Person Comment

Based on the review of the QA/QC procedures and results, the QP is of the opinion that the sample preparation, security, and analytical procedures for samples collected at Alpala and the Tandayama-America deposits are adequate for the inclusion in a Mineral Resource and Mineral Reserve estimation and in keeping with best industry practices.

12 Data Verification

12.1 Alpala QP Verifications by Dr. Gilles Arseneau

The Independent Qualified Person (QP) undertook a site visit to the Cascabel Property on 2-3 October 2023, in relation to the Alpala MRE#4 and TAM#3 mineral resource estimations. The QP undertook the following data verification checks during the site visit:

- Cross-checking the location of drill hole collars with handheld GPS
- Drill hole platform visits
- Check-logging of drill core significant intersections for both deposits
- Observing core sampling and bulk density measurement procedures
- Observing storage and security of drill core and coarse rejects
- Reviewing database compilation and integrity
- Evaluating QAQC procedures
- Comparison of SolGold's digital assay database against original assay certificates obtained directly for the assay laboratory, and no errors were noted
- Interviews with key SolGold personnel
- Collection of independent samples (Table 12-1)

The QP made the following conclusions based on the site visit in October 2023:

- Core boxes were stored in a tidy and organised way, aligned with best industry practices. Coarse and pulp reject sample storage in Quito city was not visited by the QP.
- The QP's independent check-logging showed that the geology and contacts observed in the drill core agreed well with contacts logged by SolGold personnel and that the sample intervals marked on core and core boxes also corresponded well with intervals marked in the logs
- Collar survey procedures were, in general, according to best practices
- Hand-held GPS collar survey check measurements determined by the QP show a satisfactory agreement with the SolGold collar survey measurements

Table 12-1: Results of independent samples collected during 2023 site visit

Description	Original Au (g/t)	Original Cu (%)	Check Au (g/t)	Check Cu (%)
Hole CSD15-012 Fr 870 to 872	0.213	0.364	0.224	0.35
Hole CSD15-012 Fr 932 to 934	1.38	1.195	1.50	1.33
Hole CSD15-012 Fr 1074 to 1076	1.885	1.435	2.62	1.78
Hole CSD13-005 Fr 964 to 966	1.892	2.231	1.67	2.41
Hole CSD14-009 Fr 1030 to 1032	0.347	0.33	0.17	0.22
Hole CSD16-018 Fr 958 To 960	2.146	1.038	1.78	1.16
Discovery Outcrop grab sample	NA	NA	9.15	3.61

12.2 Tandayama-America QP Verifications by Dr. Gilles Arseneau

Dr. Arseneau conducted a site visit to the TAM area on 2-3 October 2023, in relation to the Tandayama-America MRE#3. The purpose of the QP's site visit was to appraise the quality of core drilling and logging processes, the core and channel sampling methods and the database integrity of these samples in support of the inaugural Tandayama-America MRE. The site visit encompassed:

- Check-logging of drill core over economically mineralised and lower-grade intervals
- Evaluations of drill core logging, sampling, QAQC and bulk density measurement protocols
- Observations of the storage and security of drill core and coarse reject samples
- Cross-checks of the locations of drill hole collars with a handheld GPS
- Reviews of database compilations and their integrity
- Evaluating internal QAQC procedures
- Interviews with SolGold personnel involved in the MRE
- Review the results of previous independent samples collected from the deposit

The QP has drawn the following conclusions from the site visit:

- The geological descriptions and logging conventions used by SolGold were deemed appropriate for an MRE in this particular style of mineralisation.
- The core sampling procedures have produced unbiased samples.
- The storage procedures and facilities at Rocafuerte base camp and warehouse facilities in Quito were deemed orderly and adequate for the preservation of drill core as well as their pulps and coarse rejects.
- The QAQC protocols were adequate and suitable for the style of mineralisation.
- The drill hole collar surveying procedures are in accordance with industry-accepted practices.
- The bulk density measurement facility and protocols are reliable.

12.3 Qualified Person Comments

In summary, the QP is of the opinion that the quality of the geological and assay data collected in relation to the Alpala and TAM deposits, as well as their validation and storage, were aligned with industry-accepted practices.

The independent samples collected by the QP from the Alpala drill core and the samples collected from the TAM deposit in 2022 agree very well with the original assay data provided by SolGold.

The verification of the assay database against the original assay data sheets provided from the assay laboratory identified no material errors. As such, the QP considered the geological and assay data to be robust and suitable for inclusion in mineral resource and mineral reserve estimations.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

Metallurgical testwork for the Alpala underground deposit commenced in 2014. Testwork was conducted by the independent metallurgical facilities Inspectorate Metallurgical Division, Bureau Veritas Commodities Canada, ALS Metallurgy Kamloops BC and Perth Western Australia, Core Resources Albion Queensland, Nagrom Perth Western Australia, Brisbane Met Lab Brisbane Queensland, JKtech Indooroopilly Queensland, Freemantle Metallurgy Perth Western Australia, and Base Metallurgical Laboratories Kamloops BC during the period of 2019 – 2023. Tests included mineralogy, material flow, comminution, open and locked cycle flotation, Davis tube magnetic separation, cyanide leaching, and solid-liquid separation.

The proposed process flowsheet has been refined and modified over time, with the current preferred option representing a conventional copper-gold flotation flowsheet with no additional gold cyanidation circuit. The flotation flowsheet consists of a single rougher stage and a multi-stage cleaning circuit to produce a copper-gold-silver concentrate.

Samples selected for the testwork programs considered lithology and alteration descriptions, grade of copper and gold, spatial location, different timing in the mine plan and various geological measurements.

13.2 Mineralogy

A total of 53 variability samples were analysed at a nominal P_{80} 150 µm through bulk modal analyses. Key points from the mineralogy evaluation are shown in Table 13-1.

Table 13-1: Variability sample bulk mineralogical analysis summary

Mineral/Group	Range %	Average %	Observations
Chalcopyrite	0.4 – 6.2	2.3	Primary copper mineral
Other Copper Sulphides	0.1 – 0.1	0.1	Bornite with some chalcocite, covellite and tennantite/enargite
Pyrite	0.4 – 7.8	2.6	Dominant sulphide mineral
Iron Oxides	0.1 – 12.6	3.0	As hematite or magnetite
Feldspars	1.2 – 37.2	11.8	
Quartz	28.6 – 69.4	45.7	
Micas	4.7 – 25.8	14.1	
Chlorite	4.7 – 19.7	11.7	
Calcium Sulphate	0.1 – 12.0	4.8	
Talc	0.1 – 0.3	0.2	Specific locations not quantified yet
Other Non-Sulphide Gangue (NSG)	1.1 – 19.0	5.7	Where kaolinite is detected in quantity, recoveries may be impacted

Across all variability samples, chalcopyrite contained an average of 98.1% of the copper, with a range of 83.9% to 99.9%. At P_{80} 150 µm, 36% to 77% of copper sulphides (an average of 55%) were considered liberated, meaning that by cross-sectional area ≥ 95% of a host particle is copper sulphide.

Additional microanalysis investigations were undertaken by the University of Tasmania (UTas) for the gold in pyrite. The gold was observed to be hosted in refractory form – chemically bound in the mineral structure – with compositional zoning and gold-rich inclusions. Au-Ag-telluride phases may be important to the overall Au forms present.

13.3 Comminution

Primary and regrind testwork was undertaken for the Alpala deposit, which is reported here.

13.3.1 Primary Grind

Parameters investigated included the determination of SG, drop weight index, SMC comminution tests, Bond Ball Mill and Bond Rod Mill indices, and abrasion index. The testing was completed on the same 53 variability samples that were examined mineralogically in the previous section; results are presented in Table 13-2.

Table 13-2: Primary grind comminution parameters

Parameter	Units	Range	80th Percentile	Comments
SG		2.69 – 2.96	2.85	
Bond Ball Mill Work Index (BWi)	kWh/t	12.0 – 18.7	16.2	150 µm screen
Bond Rod Mill Work Index (RWi)	kWh/t	12.7 – 20.2	19.3	1180 µm screen
Breakage parameter (A x b)		21.0 – 64.7	27.1	
Drop Weight Index	kWh/m^3	4.3 – 13.1	10.3	
Abrasion Index (Ai)		0.022 – 0.222	0.165	

The results show that as the head grade of copper decreases, the samples will generally be more competent and abrasive. The volcanic and D15 lithologies are typically more competent, and the deposit competency generally increases with depth.

13.3.2 Rougher Concentrate Regrind

A series of six Eliason grinding tests were conducted to determine preliminary power consumption estimates for rougher concentrate regrinding by high-intensity stirred mills. The Eliason rougher concentrate regrind signature plots generated for the composites provided the power estimates required to achieve 15 µm, 20 µm and 25 µm as shown in Table 13-3 below from a primary grind of 150 µm.

Table 13-3: Rougher concentrate regrind power estimates

Target P_{80} (µm)	25	20	15
Composite		Estimated Power (kWh/t)	
1	11.59	18.23	32.80
2	11.45	19.88	40.62
3	12.14	18.81	33.20
4	10.38	17.08	32.58
5	10.36	18.59	39.55
6	22.02	33.00	55.78
80^{th} Percentile	12.14	19.88	40.62

13.4 Flotation

The flotation testwork was conducted over several programs, including open circuit rougher and cleaner testing to identify preferred flotation conditions, locked cycle testing to confirm results and effects of recycled streams, and variability testing.

13.4.1 Open Circuit Rougher and Cleaner Flotation

A significant volume of flotation testwork has been completed since 2019 under the program names KM5922, KM5754 and KM6024. The flotation testing was completed using seven master composites made up of the 53 variability samples used in the comminution and mineralogical analysis described in the previous sections of this section. The composites were classified as being low grade (< 0.4% Cu), medium grade (0.4% - 0.9% Cu), and high grade (> 0.9% Cu).

The primary grind was varied from F_{80} 90 µm to 320 µm. During the KM5754 program, a series of grind versus recovery tests were conducted, and it was determined that flotation recovery was consistent at a primary grind between F_{80} 150 µm and 200 µm but generated additional mass yield at coarser sizes, with decreased recovery as the primary grind was coarsened to 300 µm. Grinds finer than F_{80} 150 µm did not improve the recovery of copper. This was consistent with each of the composites tested, and it was decided to proceed with F_{80} 150 µm as the primary grind for subsequent testing.

A range of different collectors and dosages were conducted while using MIBC as the frother. Comparative tests considered the use of Potassium Amyl Xanthate (PAX) as the base case and compared alternative collectors and combinations of collectors, including other xanthates, dithiophosphate, monothiophosphate, mercaptobenzothiazole, and alkyl thiocarbamate. The results determined that PAX, as the solo collector, provided the best flotation performance. Rougher flotation testing was conducted at natural pH, and a residence time of 9 to 11.5 min was required for this flotation stage.

The rougher concentrate was reground prior to cleaner flotation and trialled at F_{80} 13 µm to 72 µm before reducing the range to F_{80} 16 µm to 25 µm for cleaner flotation testing. For intermediate and high-grade composites, grinds finer than F80 25 µm increased copper grade in the concentrate, but recoveries did not improve, and grinds coarser than F_{80} 50 µm struggled to achieve typical saleable copper grades in concentrate. Lower-grade composites required a grind of F_{80} 20 µm to achieve marketable copper grades at acceptable recovery.

Cleaner flotation was conducted at pH 11.0 to suppress some pyrite and improve copper grade in the circuit. This resulted in reduced gold recovery at finer grinds as gold associated with liberated pyrite struggled to float. Poor froth quality was noted during cleaner flotation, which was overcome by replacing MIBC with a stronger polyglycol frother during locked cycle testing.

13.4.2 Locked Cycle Flotation

Following open circuit flotation testing, flotation parameters were selected for the locked cycle testing (LCT). LCT rougher testing was completed at a nominal primary grind target F_{80} 150 µm, natural pH using PAX and MIBC as reagents. Rougher flotation was completed as a single stage with no scavenging initially for 11.5 minutes but then extended to 15.5 minutes.

The rougher concentrate was reground based on composite feed grade, with the low-grade composite nominally reground to F_{80} 15 µm, medium-grade composites to F_{80} 20 µm and high-grade composite to F_{80} 25 µm.

Cleaner flotation was conducted in three stages, with scavenging of the first cleaner tail. The scavenged concentrate was recycled to cleaner flotation feed. Tails from the second and third stages of cleaning were recycled to the cleaner flotation feed. Cleaner flotation utilised a polyglycol frother in place of MIBC and was conducted at pH 11.0.

Table 13-4: Locked cycle testing results

Composite	High Grade (> 0.9% Cu)	Medium Grade (0.4% - 0.9% Cu)	Low Grade (< 0.4% Cu)
Copper in Feed - Average	1.54%	0.57%	0.21%
Average Mass Yield to Concentrate	5.14%	1.94%	0.63%
Copper Recovery – Range	94.6% – 94.9%	83.3% – 92.6%	78.2% – 81.5%
Copper Recovery – Average	94.8%	86.2%	80.4%
Copper in Concentrate – Range	26.0% – 30.1%	19.8% – 29.5%	22.8% – 28.5%
Copper in Concentrate – Average	28.6%	26.3%	26.4%
Gold Recovery – Range	70.0% – 80.5%	63.6% – 82.1%	46.5% – 50.1%
Gold Recovery – Average	75.5%	68.8%	48.5%
Gold in Concentrate – Range	16.1 – 16.8 g/t	7.8 – 16.5 g/t	8.7 – 10.3 g/t
Gold in Concentrate – Average	16.5 g/t	11.9 g/t	9.8 g/t

13.4.3 Variability

Open circuit flotation tests were completed on each of the 53 variability samples, with the flotation conditions matching those of the LCT. Results of the variability testing are provided in Table 13-5.

Table 13-5: Variability flotation testing

Parameter	Range	Average
Feed Grade - Copper	0.13% – 2.50%	0.86%
Copper Recovery	63.4% – 94.4%	81.6%
Copper in Concentrate	10.7% – 33.4%	25.3%
Gold Recovery	30.8% – 87.4%	61.3%
Gold in Concentrate	3.1 – 58.5 g/t	19.5 g/t

Compared with the variability tests, the LCT composites provided a good summary of the overall flotation performance. LCT recovery was approximately 5% higher on average, as expected, due to the recycling of cleaner tailing streams.

Of the variability tests 32% failed to achieve a copper concentrate greater than 22%. The head grades of the samples that failed to achieve 22% Cu in concentrate averaged 0.38% Cu, while samples that achieved greater than 22% Cu in concentrate averaged a feed grade of 1.08% Cu.

Medium-grade samples with a feed grade between 0.4% and 0.9% Cu made up 25 of the 53 variability samples, achieved an average concentrate grade of 24.7% Cu at 79.2% recovery of copper, with 36% of samples failing to achieve greater than 22% Cu in concentrate.

13.5 Solid-Liquid Separation

Testwork was undertaken with rougher and cleaner scavenger tailings and final concentrate from the locked cycle tests. All samples underwent flocculent screening, static and dynamic thickening, and underflow viscosity testing.

Rougher tailings thickening was based on 15% w/w solids content in the slurry feeding the dynamic thickener, and it was determined that 20 to 30 g/t of anionic flocculent was required and achieved an acceptable underflow density of 52% to 56% w/w. The flux rate was 0.6 t/m^{2}h.

Cleaner scavenger tailings had 8.7% w/w solids content in the slurry feeding the dynamic thickener, which achieved an underflow density of 35% w/w based on 80 g/t of anionic flocculent and a flux rate of 0.6 t/m^{2}h.

Final concentrate thickening was based on 15% w/w solids content in the feed slurry and achieved an underflow density of 55% to 65% w/w with 20 g/t of non-ionic flocculant. The flux rate was 0.25 t/m^{2}h.

Rheology testing of each underflow generated during the thickening tests determined that no rheological issues are expected based on maintaining less than 65% w/w solids content in the rougher tailings thickener underflow, less than 50% w/w solids in the cleaner scavenger tailings underflow, and less than 75% w/w solids in the final concentrate thickener underflow.

Pressure filtration of the final concentrate determined that 15 bar of squeezing and 5 bar blowing at greater than 100 L/hr filter cakes of 11% moisture could be produced. The filter cakes were found to be very dense and friable. Filtration rates of 275.5 to 388.5 kg/h/m^2 were observed.

13.6 Flotation Tailings Gold Recovery

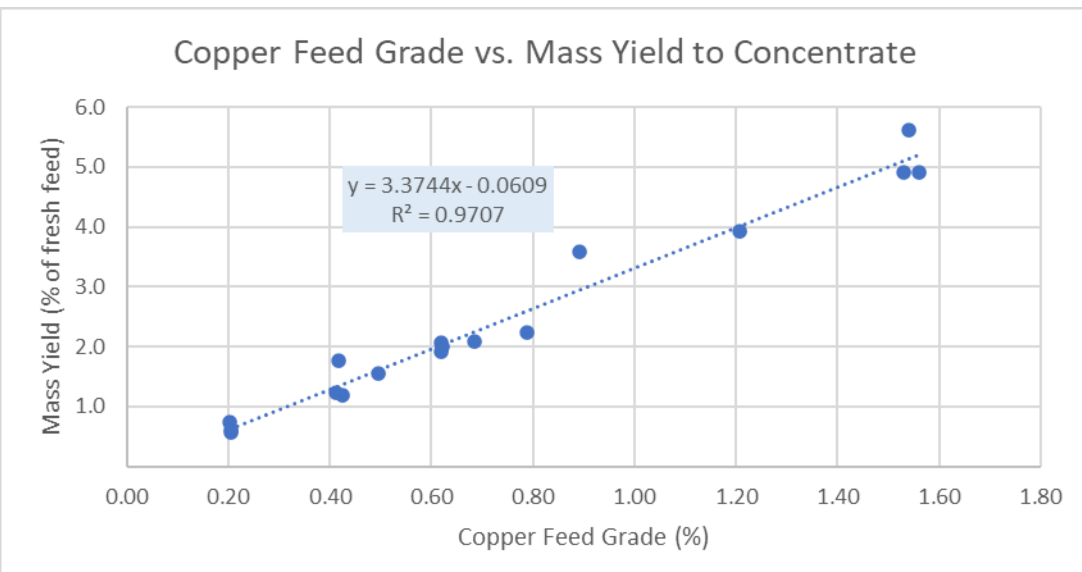
Using a combination of cleaner flotation tails and cleaner scavenger flotation tails, ten composites were prepared for cyanidation to investigate additional gold recovery through cyanidation. An additional composite was prepared when cleaner flotation tails were refloated to produce a gold-rich pyrite concentrate. The flotation tails composites had gold grades ranging from 0.27 g/t to 2.68 g/t, and the pyrite concentrate had a grade of 15.8 g/t.

Composites were leached for 48 to 72 hours, with cyanide concentration maintained at 0.05% - 0.1%. Pulp density in the leach tests was 25 and 40% solids. Leaching of the flotation tail composites recovered 46% to 88%, averaging 68.8% of the contained gold, with the pyrite concentrate leach recovering 76% of the contained gold. Cyanide consumption rates of the ten flotation tail composites averaged 3.2 kg/t, with the pyrite concentrate consuming 12.19 kg/t. The high cyanide consumption was expected due to the presence of copper which averaged 0.16% in the flotation tails composites and 2.2% in the pyrite concentrate.

Other leaching techniques, including thiosulphate, acid leach and cyanidation, bacterial oxidation, and oxidative series cyanidation, were also trialled but did not produce favourable results.

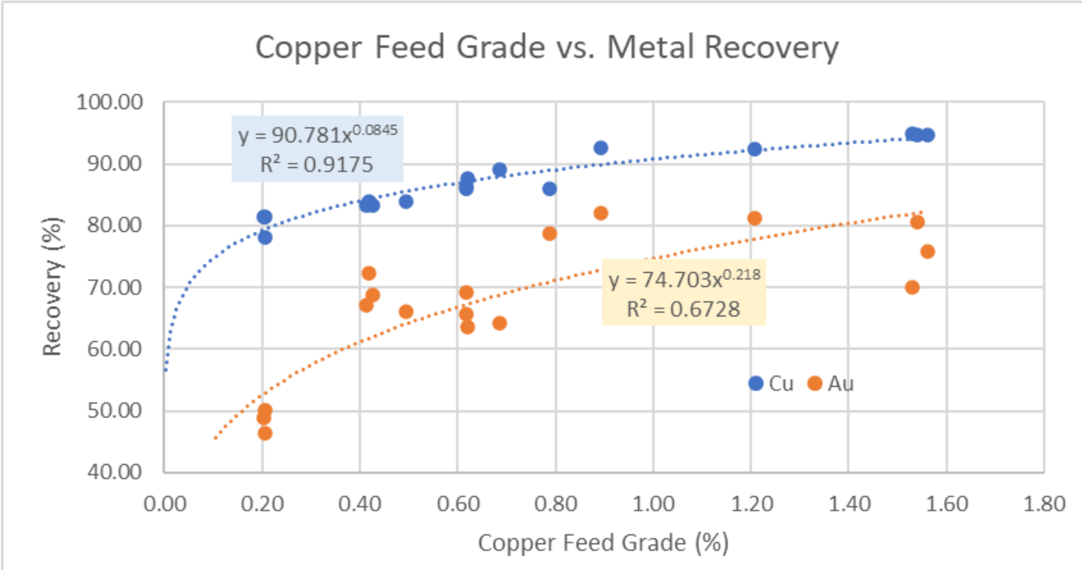
13.7 Recovery Estimates

The overall recovery relationships for the flotation plant are based on the available locked cycle testwork (LCT). No corrections have been applied at this time for operability-related losses. The LCT established strong correlations between copper feed grade and the mass yield to concentrate and to metal recovery as shown in Figure 13-1, Figure 13-2 below and Figure 13-3. These equations are based on a final average concentrate grade of approximately 27.9% Cu and 2.4% mass yield to concentrate (LCT average results).



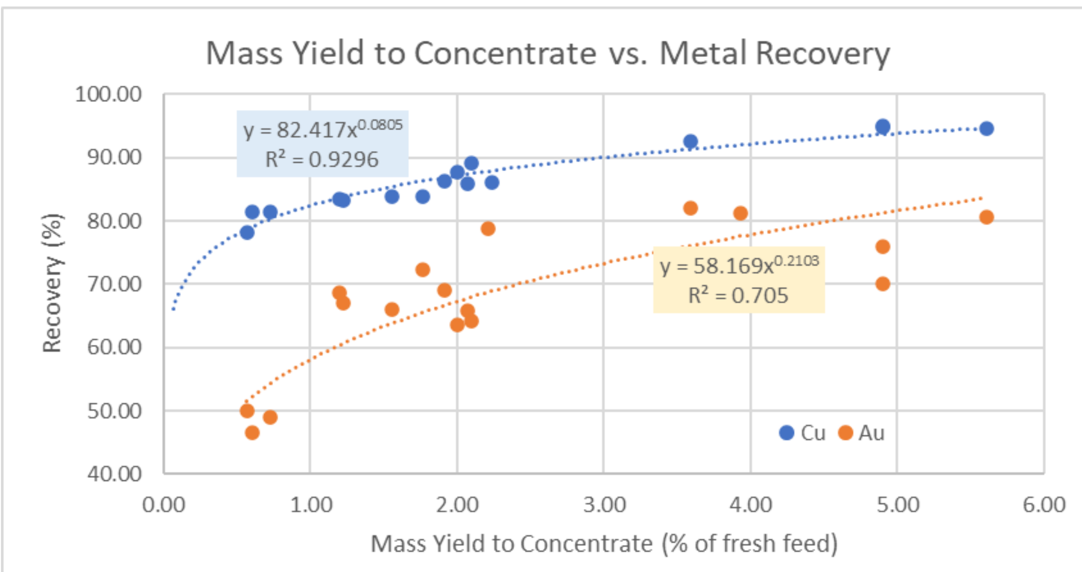
Source: Artica et al., 2022

Figure 13-1: Copper feed grade vs. concentrate mass yield



Source: Artica et al., 2022

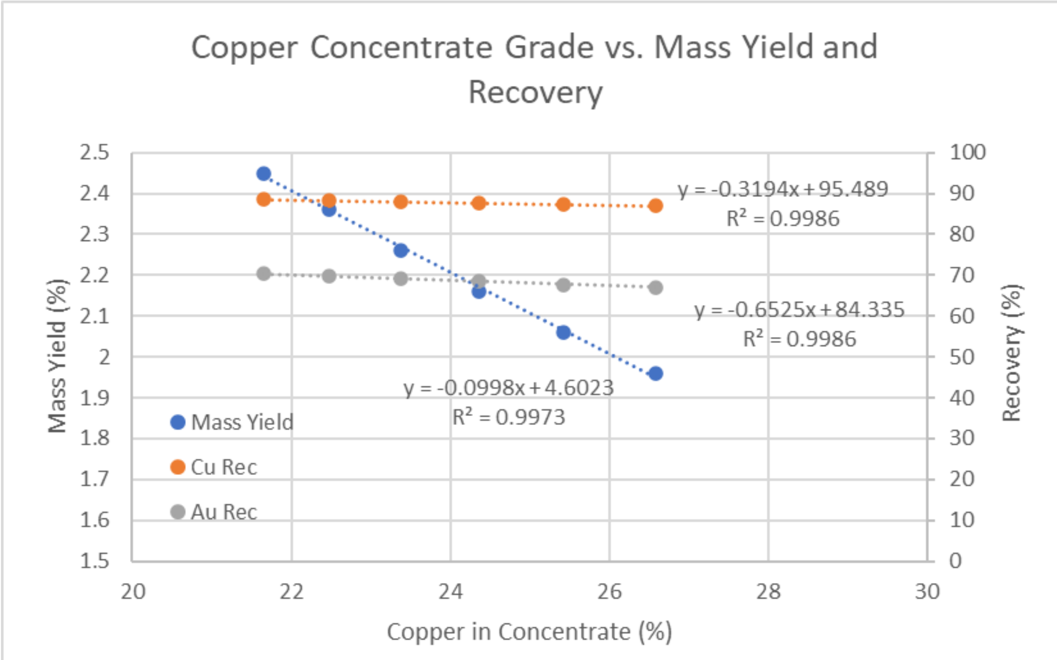
Figure 13-2: Copper feed grade vs. metal recovery



Source: Artica et al., 2022

Figure 13-3: Mass yield vs. metal recovery

Using the established relationships, the metal recovery and concentrate production can be adjusted based on a targeted final copper in concentrate grade, as shown in Figure 13-4. Based on a life of mine average copper feed grade of 0.6% Cu and a concentrate target of 22% Cu, the life of mine copper recovery is calculated to be 88.5%, and gold recovery is 70%. Gold in concentrate is calculated to be 15.7 g/t Au.



Source: Artica et al., 2022

Figure 13-4: Copper concentrate grade vs. mass yield and recovery

13.8 Concentrate Quality

The concentrate quality shown in the table below is based on the LCT concentrate quality reported in KM 6204.

Table 13-6: Alpala concentrate quality

Symbol / Element	Units	Minimum	Maximum	Average
Sb - Antimony	g/t	1	12	5
As - Arsenic	g/t	7	124	61
Bi - Bismuth	g/t	4	9	6
Cd - Cadmium	g/t	5	46	16
Cu - Copper	%	20	29	25
Au - Gold	g/t	8	17	13
Pb - Lead	g/t	98	576	235
Se - Selenium	g/t	111	170	143
Ag - Silver	g/t	38	122	66
Te - Tellurium	g/t	5	11	8
U - Uranium	g/t	< 1	< 1	< 1
Zn - Zinc	g/t	540	8,690	2,484

14 Mineral Resource Estimates

14.1 Introduction

The mineral resource estimates (MREs) for Alpala and Tandayama-America (TAM) were prepared following industry best practice as described in the "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" (CIM, 2019).

The QP who takes responsibility for the mineral resource estimates is Dr. Gilles Arseneau, Associate Consultant with SRK with over 20 years of experience in mineral resource estimation, including porphyry copper-gold deposits. Dr. Arseneau is independent of SolGold and of the Cascabel Property.

Geology was interpreted in two-dimensional vertical and plan hard-copy sections before being transferred to Leapfrog software and modelled as three-dimensional wireframes. The wireframes were then used to define estimation domains and domain boundary conditions through exploratory data analysis. Once domains were defined, variography was completed and search and estimation plans were developed. The estimations for the deposits were undertaken in Leapfrog Edge Software and then verified and validated using Geovia GEMs Version 6.8.4.

The estimation for both deposits was completed with ordinary kriging (OK), with check estimates completed using the nearest neighbour (NN) estimation method. Block model validation showed good agreement between the input composite grades and the estimated block grades in both Alpala and Tandayama-America.

The MREs have been classified in accordance with the CIM Definition Standards (CIM, 2014) and were both informed by a drill hole spacing study based on Alpala assumptions.

The reasonable prospects for eventual economic extraction (RPEEE) criteria were met by reporting inside optimised pit shells and/or underground optimisation shapes assuming block caving mining methods.

The cut-off or cut-off grades used for optimisation and reporting were based on third-party metal price research, forecasts of Cu and Au prices at the time of reporting, and operating cost structures for Alpala that were current at the time of reporting. Operating costs included mining, processing and general and administration (G&A). Net Smelter Return (NSR) included metallurgical recoveries and off-site realisation (TCRC), including royalties.

Open pit optimisation was completed using the conventional Lerchs-Grossman optimisation routine implemented in Whittle software, and the revenue factor one pit was selected for reporting the Mineral Resource.

Subsequently, a three-dimensional Underground Optimised Shape (UOS) was generated from the remaining mineralised material using Geovia GEMs Footprint Finder software. The UOS maximised the tonnes above the cut-off while ensuring all material was part of a minimum mining unit with geometry appropriate for a block cave. These minimum mining dimensions were consistent with the QP's experience, and the resulting shapes contained planned internal and edge dilution that the QP considered appropriate.

14.2 Data Supplied

The QP was supplied with drill hole databases in the form of CSV text files, which had been exported from SolGold's main database as maintained on-site. The data was checked for internal consistency when loading into Leapfrog and GEMs software. Minor inconsistencies were identified and resolved in consultation with SolGold.

14.3 Drill Hole Data Used and Excluded

14.3.1 Alpala Deposit

The Alpala Mineral Resource Update (MRE#4) was estimated from 112,131 assays (111,435 from drill holes and 696 from surface trenches). These represent an addition of twenty-eight drill holes since the last MRE was published on 22 May 2022. Drill core samples were obtained from a total of 265,225 m of drilling from 185 diamond drill holes. Drill holes comprise 120 parent drill holes, 37 wedged holes, 8 re-drills, 20 over-runs and 9 geotechnical diamond drill holes that were not assayed. Surface rock-saw channel samples were obtained from 1,441 m of rock-saw cuts from 118 surface rock exposure trenches.

Nine geotechnical drill holes GHD-19-003 to GHD-19-010 and GHD19-108-D1, totaling 7,764.76 m, were included in the geological database utilised for this resource estimate, providing important geological data improving geological models. However, these drill holes were not assayed because the whole drill core was retained for further geotechnical and geometallurgical testwork.

Three sterilisation drill holes, GHD-19-001, GHD-19-002 and GDH-19-002R, were assayed. However, these drill holes lay outside the Alpala deposit resource estimation area and were omitted from the estimation. Six water monitoring drill holes, totaling 277.8 m (MW-19-001 to MW-19-003B), were drilled specifically for hydrological studies, and these holes lie outside the Alpala deposit resource estimation area and are omitted from the estimation.

Of the 185 drill holes supplied in the collar file, 176 drill holes were used in the grade estimation of the MRE. Some holes contained logging information, not assay information, which was used to inform the lithological interpretation.

14.3.2 Tandayama-America Deposit

The Tandayama-America Mineral Resource was estimated from 17,767 assays, 17,574 assays from diamond drill core samples, and 220 assays from rock-saw channel samples cut from surface outcrops. Drill core samples were obtained from 36,111 m of drilling from 51 diamond drill holes. Surface rock-saw channel samples were obtained from 458 m of channel intervals from 72 surface channels.

14.4 3D Lithological and Mineralisation Modelling

Geological and grade domains were prepared by SolGold. Dr Arseneau reviewed the wireframes with SolGold, made suggestions for improvements where necessary and validated and accepted the geological and grade domains used in the MREs for the Cascabel project.

Mineralisation domain wireframes have been interpreted to honour lithological contacts and intrusion geometries and guided locally by structural measurements of B-vein orientations. Three grade wireframes were interpreted by modelling copper equivalent (CuEq) grade and B-vein abundance and using the following guidance criteria:

14.4.1 Alpala Deposit

- Low grade - where CuEq exceeds 0.1% and B vein intensity exceeds 0.55%
- Medium grade - where CuEq grade exceeds 0.7% CuEq and B vein intensity exceeds 4.1%
- High grade - where CuEq grade exceeds 1.5% CuEq and B vein intensity exceeds 9.4%

Copper equivalent grade for mineralisation modelling at Alpala was determined using metal prices of $3.60/lb copper and $1,700/oz gold and calculated using the formula CuEq = (copper grade (%)) + (gold grade (g/t) x 0.683).

14.4.2 Tandayama-America Deposit

- Low grade (LG) - where CuEq exceeds 0.1% and trace of B-veins
- Medium grade (MG) - where CuEq grade >0.3% and >2.4% B-veins

Copper equivalency used for mineralisation modelling was based on third-party metal price research, forecasting of Cu and Au prices, and a cost structure from mining studies data available at the time. Costs include mining, processing and general and administration (G&A). Net Smelter Return (NSR) includes metallurgical recoveries and off-site realisation (TCRC), including royalties and utilising metal prices of Cu at $3.60/lb and Au at $1,700/oz and a copper equivalency factor of 0.683 (whereby CuEq = Cu + Au x 0.683).

This CuEq formula for the cut-off or cut-off grade calculation was updated between the Alpala and Tandayama-America Resource Estimates to recognise metal price forecasts based on current information at the time of reporting.

14.5 Exploratory Data Analysis

The QP undertook Exploratory Data Analysis (EDA) by first investigating the statistics of each individual lithology and grade wireframe before iteratively generating various groups of lithologies within each grade wireframe and comparing their statistics. Lithology and grade wireframes were combined to form the final estimation domains used to prepare the mineral resource estimate.

14.6 Drill Hole Compositing

To capture most of the samples in a standard composite length, while also maintaining sufficient resolution, a 2 m composite length was selected with a minimum composite length of 1 m for both Alpala and Tandayama-America. Compositing honoured the lithology boundaries, and composite lengths less than 1 m were discarded. Compositing was conducted on capped assay data.

14.7 Estimation Domains

Estimation domains were defined by merging the lithology domains with the grade domains after considering the geological history, the geometry of the units and by iteratively combining various groups of lithologies within each grade wireframe and analysing the resulting statistics (Figure 14-1 and Figure 14-2).

<table>
<tr><th rowspan="2">CODE
(Lith Domain)</th><th rowspan="2">Group
(Lith Domain)</th><th colspan="4">Resource Model Domains (SG, Cu, Au, Ag)</th></tr>
<tr><th>Unmineralised MINCODE 0</th><th>Low grade MINCODE 1</th><th>Medium grade MINCODE 2</th><th>High grade MINCODE 3</th></tr>
<tr><td>100</td><td>BAS</td><td rowspan="3">DOM 10</td><td>na</td><td>na</td><td>na</td></tr>
<tr><td>200</td><td>V</td><td rowspan="2">DOM7</td><td rowspan="2">DOM5</td><td rowspan="2">DOM 2</td></tr>
<tr><td>300</td><td>D10</td></tr>
<tr><td>400</td><td>QD10</td><td>na</td><td colspan="2">DOM 4</td><td>DOM 1</td></tr>
<tr><td>500</td><td>IM</td><td rowspan="7">DOM 10</td><td rowspan="3">DOM8</td><td rowspan="3">DOM6</td><td rowspan="2">DOM 3</td></tr>
<tr><td>600</td><td>IM BX</td></tr>
<tr><td>700</td><td>IMF</td><td>na</td></tr>
<tr><td>800</td><td>LM</td><td colspan="2" rowspan="4">DOM 9</td><td></td></tr>
<tr><td>900</td><td>LMF</td><td rowspan="3">na</td></tr>
<tr><td>1000</td><td>PM</td></tr>
<tr><td>1100</td><td>BX</td></tr>
</table>

Source: This study, 2024

Figure 14-1: Alpala estimation domains

CODE (Lith Domain)	Group (Lith Domain)	**Resource Model Domains** (SG, Cu, Au, Ag)		
		Unmineralised MINCODE 0	**Low grade MINCODE 1**	**Medium grade MINCODE 2**
100	V	**DOM 6**	**DOM 5**	**DOM2**
200	D10	**DOM 6**	**DOM 4**	**DOM3**
300	EM	**DOM 6**	**DOM 4**	**DOM 1**
400	IBX	**DOM 6**	**DOM 5**	**DOM2**
500	IM	**DOM 6**	**DOM 4**	**DOM3**
520	LM	**DOM 6**	**DOM 5**	**DOM3**
540	PM	**DOM 6**	**DOM 5**	**DOM 5**

Source: This study, 2024

Figure 14-2: TAM estimation domain

For the Alpala deposit, Cu, Au and Ag grades were estimated. Silver used the same domain definitions as Cu as it showed better correlation with Cu than Au, and because Ag only adds a small incremental value to the deposit. Au had its own estimation domain definitions.

For the Tandayama-America deposit, only Cu and Au were estimated, and the estimation domains were developed based on copper equivalent data.

14.8 Outlier Management and Capping Strategy

For sample outlier population management, the samples were coded by estimation domain to interrogate the statistics for outlier capping or top-cutting. Histograms and log normal probability plots were used to identify outlier sample populations. These populations were subsequently confirmed not to form independent, volumetrically discrete, high-grade domains. Capping was applied to the un-composited assay data prior to compositing, as outlined in Table 14-1.

Table 14-1: Alpala and TAM capping levels

Deposit	Domain	Au Cap (g/t)	Cu Cap (%)
Alpala	1	9.0	5.0
	2	5.0	4.2
	3	2.5	3.0
	4	NC	2.0
	5	4.5	2.5
	6	1.7	1.7
	7	2.1	2.1
	8	2.5	1.7
	9	2.3	1.3
TAM	1	1.5	NC
	2	1.8	NC
	3	1.4	1.5
	4	NC	NC
	5	1.5	NC
	6	0.4	NC

NC= Not Capped

14.9 Contact Profiles and Boundary Analysis

Mineralisation contact profiles were used to establish if a mineralisation domain boundary represented "hard" (i.e., domains cannot share assays for grade interpolation) or "soft" boundaries (i.e., domains can share assays for grade interpolation).

For Alpala, boundary conditions were assessed and applied independently for each estimation domain boundary in Leapfrog and verified in GEMs.

For Tandayama-America, the contact plots within the medium and low-grade domains showed that soft boundaries were appropriate with the grade domains but that hard boundaries should be used between grade domains.

14.10 Variographic Analysis

Correlograms and normal score variograms were calculated to limit the influence of the proportional effect on the shape of the experimental variograms.

The experimental variograms were generated from capped composites in reference planes aligned with the overall strike and dip of the mineralisation. Other directions were investigated for some domains and elements where they showed a stronger continuity in a direction other than the overall strike and dip of the mineralisation.

Once the reference plane had been established, a variogram fan was produced at 10° increments in the reference plane. The direction that showed the best continuity (longest range) was modelled with the two perpendicular directions also modelled. Spherical models with a nugget and two or three structures were manually fit to the directional variograms.

Some domains had insufficient samples from which to model robust variography. In those cases, models from geologically similar domains were borrowed, or domains were grouped to generate meaningful variograms.

14.11 Block Model Construction and Estimation

14.11.1 Block Model Parameters

The Alpala and the Tandayama-America block models were constructed using sub-cells in Leapfrog Edge. The Alpala AND TAM models are aligned north-south, unrotated. Grades were estimated into parent cells with dimensions of 20 mE by 20 mN by 10 mRL. Sub-cells were used for the Alpala model to fit the geometry of the lithology and grade wireframes more precisely, with these sub-cells estimated at the parent cell scale. Minimum sub-cell dimensions of 5 mE by 5 mN by 5 mRL were used at Alpala, while Tandayama-America used full-sized cells. Table 14-2 summarises the model parameters for both deposits.

Table 14-2: Block model parameters

	Minimum			Parent Block size			No of Blocks			Sub Block Size		
Deposit	**X**	**Y**	**Z**	**X**	**Y**	**Z**	**X**	**Y**	**Z**	**X**	**Y**	**Z**
Alpala	795,750	82,000	-1,500	20	20	10	163	150	370	5	5	5
TAM	795,500	85,500	-600	20	20	10	120	85	230	20	20	10

14.11.2 Estimation Method

Grades were estimated by Ordinary Kriging (OK) estimation method for copper and gold in both deposits and inverse distance square (ID2) was used for the silver estimate in the Alpala deposit and the density for both deposits. Nearest neighbour (NN) estimations were also run for verification purposes.

Alpala Deposit

- Grade estimation was completed in four search passes:
  - The first search pass used ranges approximately equal to the range of the correlogram and required at least three drill holes to complete an estimate.

  - The second and third passes used ranges approximately double the range of the correlogram, with the third pass having more relaxed criteria so that a block could be estimated using two drill holes.
  - The fourth pass had more relaxed criteria again, with one drill hole required for the estimate to be completed. The same search dimension was used as the third pass.
  - Blocks that were not estimated in the fourth pass were assigned the 25th percentile grade for that domain.
- Parent cell estimation was used, and discretisation of 3 (X) by 3 (Y) by 5 (Z) points was applied. Five points were used to discretise the block vertically to approximately match the sample interval in the composite drill hole file.
- Hard boundaries were used for all domains.
- Nearest neighbour estimation used samples that were composited by the bench compositing technique so that the composite's vertical dimension matched the block's vertical dimension (10 m). This ensured that the drill hole data was fully represented in the NN estimate.

Tandayama-America Deposit

- Grade estimation was completed in three search passes:
  - The first search pass used ranges approximately half the range of the variogram and required at least three drill holes to complete an estimate.
  - The second used the full variogram ranges and three drill holes, and the third pass used twice the variogram range with a minimum of two drill holes.
- Parent cell estimation was used, and discretisation of 4 (X) by 4 (Y) by 5 (Z) points was applied. Five points were used to discretise the block vertically to approximately match the sample interval in the composite drill hole file.
- Soft boundary estimation was used within the grade domains. Hard boundaries were used between medium- and low-grade domains.
- Nearest neighbour estimation was completed using a block model with 20 mE by 20 mN by 10 mRL parent blocks so that the composite's vertical dimension approximated the block's vertical dimension. This ensured that the drill hole data was fully represented in the NN estimate.

14.12 Bulk Density

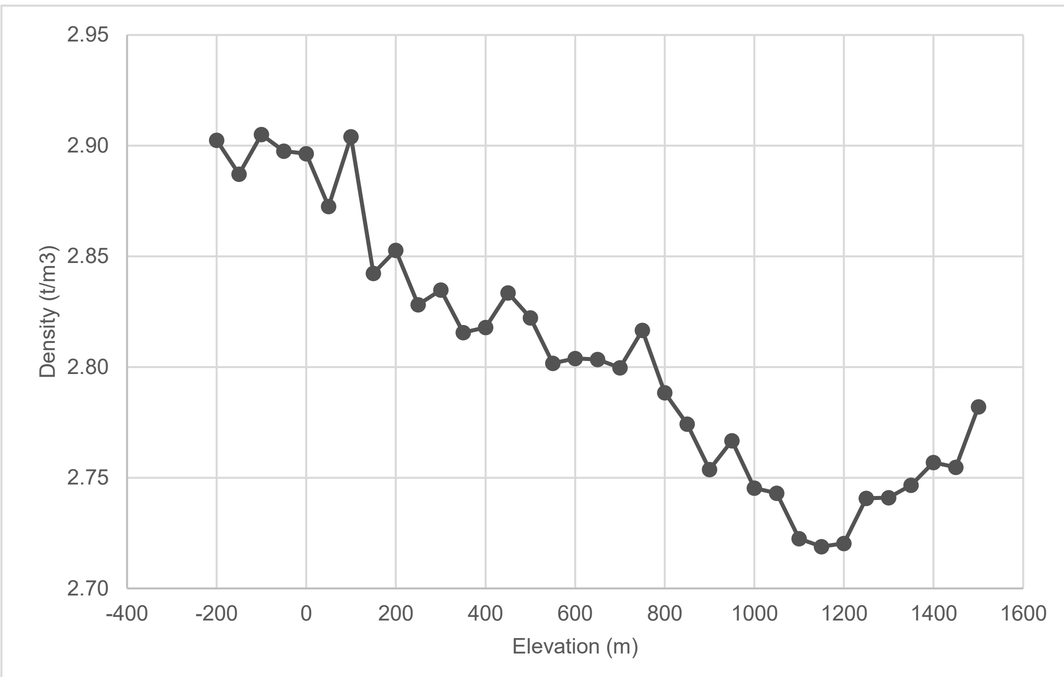
Density data were reviewed to determine if densities varied based on grade, domains or elevations.

14.12.1 Alpala Deposit

A total of 12,478 density measurements were collected from the core at Alpala. Bulk density values were observed to be very consistent throughout the deposit with no significant variation spatially or by lithology or grade wireframe. Therefore, it was concluded that there would be no material improvement gained by estimating the values into the block model. Instead, the bulk density values were assigned in the block model based on the mean values for each lithology.

14.12.2 Tandayama-America Deposit

A total of 1,735 density measurements were collected from the drill core at the TAM deposit. Bulk density values were observed to have a relationship with elevation, whereby lower bulk densities were generally found at higher elevations (Figure 14-3). Therefore, bulk density was estimated in a single pass using ID^2 using a flat search ellipse to keep similar data together. A minimum of 2 and a maximum of 16 samples were selected inside search ellipsoids with dimensions based on the range of the variogram for each grade domain. Variographic rotations and search ellipsoids were oriented to follow the geometry of the intrusions. Un-estimated bulk density was assigned a value of 2.77 t/m^3, which was the mean bulk density of all samples.



Source: Artica et al., 2022

Figure 14-3: Bulk density variation by elevation for the TAM deposit

14.13 Model Validation

Block model validation was undertaken to ensure the interpreted geological and grade characteristics were correctly modelled and estimated. Comparisons were made between the declustered composite drill hole grades and the estimated block model grades. The comparison was both visual and statistical.

The models were validated by:

- Comparing the estimated block values in plans and sections against the composited drill hole data

- Comparing the estimated blocks grades using swath plots of Nearest Neighbour estimate and the composite data used to estimate the model
- Comparing the estimated grades of blocks pierced by drill holes with the average grade of the data used to estimate these blocks (well-informed blocks)
- Comparing the results of the current estimate with the previous mineral resource estimate for the Project

14.14 Mineral Resource Classification

Mineral resources were classified based on the average distance of multiple drill holes informing the block model. Blocks were coded as measured if estimated by at least three drill holes within an average distance of 80 m from the estimated block. Blocks estimated with at least three drill holes within a 160 m distance were classified as Indicated mineral resources. Blocks estimated with at least two drill holes within a 240 m radius were classified as Inferred mineral resources. After estimation, the results were reviewed in plans to ensure uniformity within each class. A smoothing algorithm was run to ensure no isolated misclassified blocks existed within each class domain.

14.15 Reasonable Prospects for Eventual Economic Extraction

Open pit and underground optimisations were run for mineralised and classified material that was potentially mineable by open pit or underground methods, respectively. The underground optimised shapes (UOS) were then used to report the potion of the Mineral Resource that could potentially be mined from underground block caving methods. It is important to note that the resulting UOS should not be described as a “mineable shape.” Mining factors excluded in this analysis include but are not limited to capital costs (non-mining, access and footprint establishment), regional pillars, footprint geometries, unplanned dilution and the time value of money.

Nevertheless, the UOS did enclose a contiguous Mineral Resource that, by virtue of its grade and geometry, should be considered for inclusion within a mineable shape. As such, the QP considers that the reported underground portions of the Mineral Resources have reasonable prospects for eventual economic extraction by the block cave underground mining method at the specified cut-off grade.

An assessment of whether the Project as a whole was economically viable was not made under this analysis.

14.15.1 Alpala Deposit

The cut-off grade used for reporting was based on up-to-date third-party metal price research, forecasting of Cu and Au prices, and a cost structure from the May 2022 PFS at the Alpala porphyry copper-gold-silver deposit (Artica et al. 2022).

Costs included mining, processing and general and administration (G&A). Net Smelter Return (NSR) included metallurgical recoveries and off-site realisation (TCRC) including royalties. Metal prices used included Cu at \$3.60/lb and Au at \$1,700/oz. The input assumptions are provided in Table 14-3.

Table 14-3: Summary of inputs for underground optimisation

Description	Unit	Cu	Au
Metal Price	US$ per lb/oz	3.60	1,700
Government Royalty Gross Revenue		5%	5%
Refining Charge	US$ per lb/oz	0.07	5
Units Metal to Grade		2,204	0.0322
Units Factor		2,204	31.013
Smelter Payable	%	97.5	98.0
Smelting	US$ per dmt concentrate	70	
Water Content	%	5	
Shipping	US$ per wmt concentrate	90	
Shipping dry tonnes		95	
Concentrate Slurry Pipeline	US$ per dmt concentrate	0.13	
Metallurgical Recovery	%	93	83
NSR/grade unit		6,166	43
Factor	Gross Value to Mined Value	77%	77%
Equivalence Factor	Ratio to Cu grade unit	1	0.683
Mining Cost	US$ /mined t	3.73	
Processing Cost	US$ /mined t	6.33	
Site G&A Cost	US$ /mined t	1.58	
Total Increment Cost	US$ /mined t	13.24	
Shut-off (Cut-off grade)	CuEq grade	0.21%	

A UOS was generated at the 0.21% CuEq cut-off grade. This UOS maximised the tonnes above the cut-off while ensuring all material was part of a minimum mining unit with geometry appropriate for a block cave of 80 m length by 80 m width by 200 m height.

These minimum mining dimensions for a block cave were based on a mining study that was current at the time of reporting. As such, the resulting shape contained planned internal and edge dilution that the QP considered appropriate.

14.15.2 Tandayama-America Deposit

The cut-off or shut-off grades used for reporting were based on third-party metal price research, forecasting of Cu and Au prices that were current at the time of reporting, and a cost structure from mining studies that were being reviewed at the time of reporting. Costs included mining, processing and general and administration (G&A). Net Smelter Return (NSR) included metallurgical recoveries and off-site realisation (TCRC), including royalties and utilising metal prices of copper at $3.60/lb Cu and gold at $1,700/oz Au.

Cut-off and shut-off grades were developed independently for open pit mining methods and underground bulk mining methods. The cut-off grade for potentially open pit mineable material was calculated at 0.16% CuEq using a copper equivalency factor of 0.683, while the cut-off grade for material potentially mineable by a bulk underground mining method such as block caving was calculated at 0.19% CuEq using a copper equivalency factor of 0.683.

Optimisation was completed in two stages, with the open pit optimisation initially applied to the block model. The remaining material was then considered for underground optimisation.

The open pit optimisation was completed using the conventional Lerchs-Grossman optimisation routine implemented in Whittle software, and the revenue factor one pit was selected for reporting the Mineral Resource based on parameters outlined in Table 14-4. The QP considered that the open pit portion of the reported Mineral Resource had reasonable prospects for eventual economic extraction at the specified cut-off grade.

Subsequently, a UOS was generated using Datamine software at a shut-off grade of 0.19% CuEq. This cut-off grade was based on costs associated with the block cave mining method Table 14-5. The UOS maximises the tonnes above the cut-off while ensuring that all material was part of a minimum mining unit with geometry appropriate for a block cave of 120 m length by 120 m width by 200 m height. The resulting UOS contained planned internal and edge dilution.

Table 14-4: Summary of open pit optimisation parameters

Description	Unit	Cu	Au
Metal price	US$ per lb/oz	$3.60	$1,700
Government Royalty Gross Revenue		5%	5%
Refining Charge	US$ per lb/oz	0.079	5
Smelter Payable		97.5%	98.0%
Smelting	US$ per dmt concentrate	70	
Water Content		5%	
Shipping	US$ per wmt concentrate	90	
Shipping dry tonnes		95	
Concentrate Slurry Pipeline	US$ per dmt concentrate	0.0	
Metallurgical Recovery		0.87	0.75
Factor	Gross Value to Mined Value	73%	70%
Equivalence Factor	Ratio to Cu grade unit	1	0.683
Mining Cost	US$ /mined t	1.50	
Processing	US$ /mined t	6.33	
Site G&A	US$ /mined t	1.58	
Total Increment Costs	US$ /mined t	9.46	
Cut-off grade	CuEq grade	0.16%	
Overall slope angle	deg	45	

Table 14-5: Parameters for underground optimisation

Description	Unit	Cu	Au
Metal Price	US$ per lb/oz	$3.60	$1,700
Government Royalty Gross Revenue		5%	5%
Refining Charge	US$ per lb/oz	0.07	5
Smelter Payable		97.5%	98.0%
Smelting	US$ per dmt concentrate	70	
Water Content		5%	
Shipping	US$ per wmt concentrate	90	
Shipping dry tonnes		95	
Concentrate Slurry Pipeline	US$ per dmt concentrate	0.0	
Concentrate Grade		0.29	36
Metallurgical Recovery		0.93	0.83
Factor	Gross Value to Mined Value	78%	79%
Equivalence Factor	Ratio to Cu grade unit	1	0.701
Mining Cost	US$ /mined t	3.73	
Processing	US$ /mined t	6.33	
Site G&A	US$ /mined t	1.58	
Total Increment Costs	US$ /mined t	11.69	
Cut-off grade	CuEq grade	0.19%	

14.16 Mineral Resource Statement

The Cascabel Property is registered as an Advanced Exploration Licence for metallic minerals. Exploraciones Novomining S.A. (ENSA) is the registered holder of the mineral tenure of the Property, which is in northern Ecuador.

The Cascabel property has in place the necessary regulatory licenses and authorisations required for its current status as an Advanced Exploration Project. Furthermore, with the support of the community and government organisations, future license, and authorisation requirements to advance the Property are expected to be successfully attained.

To the best of the QP’s knowledge, there are no environmental, permitting, legal, title, tax, socio-economic, market, political or other relevant factors other than what is presented in this Technical Report that would affect the Mineral Resource Estimate.

14.16.1 Alpala Deposit

The MRE (MRE#4) for the Alpala deposit was reported in accordance with the Canadian National Instrument 43-101 – Standards of Disclosure for Mineral Projects. The estimation process followed the Canadian Institute of Mining, Metallurgy and Petroleum's "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" (CIM, 2019). Dr. Gilles Arseneau, P.Geo. is the Qualified Person (QP) responsible for the Alpala MRE#4.

Dr. Arseneau has estimated that the Alpala porphyry copper-gold-silver deposit contained 3,013 million tonnes grading 0.35% Cu and 0.28 g/t Au in the Measured plus Indicated categories, at a cut-off grade of 0.21% Cu equivalent (CuEq) (Table 14-6). The deposit contains an additional 607 million tonnes grading 0.26 % Cu and 0.19 g/t Au in the Inferred category.

Table 14-6: Alpala Mineral Resource Statement (effective date: 11 November 2023)

Cut-off Grade (CuEq%)	Resource Category	Tonnage (Mt)	Grade				Contained Metal			
			CuEq (%)	Cu (%)	Au (g/t)	Ag (g/t)	CuEq (Mt)	Cu (Mt)	Au (Moz)	Ag (Moz)
0.21	Measured	1,576	0.64	0.43	0.35	1.16	10.0	6.7	17.5	58.6
	Indicated	1,437	0.39	0.28	0.20	0.71	5.6	4.0	9.3	32.7
	Measured + Indicated	**3,013**	**0.52**	**0.35**	**0.28**	**0.94**	**15.6**	**10.7**	**26.8**	**91.3**
	Inferred	607	0.36	0.26	0.19	0.56	2.2	1.5	3.7	11.0

Notes:
1. Dr. Arseneau, P. Geo. Associate Consultant with SRK Consulting (Canada) is responsible for this Mineral Resource statement and is an "independent Qualified Person" as such term is defined in NI 43-101.
2. Reasonable prospects of eventual economic extraction were assessed by enclosing the mineralised material in the block model estimate in a 3D wireframe shape that was constructed with adherence to a minimum mining unit with geometry appropriate for a block cave.
3. Cut-off grade for the shape was defined as the cut-off grade under a breakeven, eventual economic extraction criterion. The cut-off grade of 0.21% CuEq was calculated using (copper grade (%)) + (gold grade (g/t) x 0.683).
4. All material within this shape was reported in the Mineral Resource statement as block caving is a non-selective method, and all material extracted is treated as mill feed.
5. The material inside the shape without a Mineral Resource category was reported as planned dilution.
6. The resulting shape contained planned internal and edge dilution that the QP considers appropriate.
7. Cut-off inputs included:
   a) Metal prices of $3.60/lb Cu and $1,700/oz Au
   b) Recoveries of 93% for copper and 83% for gold
   c) Costs including mining, processing and general and administration (G&A)
   d) Off-site realisation (TCRC), including royalties
8. The QP considers that the Mineral Resource has reasonable prospects for eventual economic extraction by an underground mass mining method such as block caving.
9. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
10. Mineral Resources are reported inclusive of those Mineral Resources that were converted to Mineral Reserves.
11. Figures may not add up due to rounding.

14.16.2 Tandayama-America Deposit

The Tandayama-America deposit lies approximately 3 km north of the Alpala deposit. Dr. Arseneau is the Qualified Person (QP) responsibility for the Tandayama-America MRE (MRE#3) (Table 14-7).

Table 14-7: Tandayama-America Mineral Resource Statement (effective date: 11 November 2023)

Potential Mining Method	Cut-off Grade (CuEq %)	Resource Category	Tonnage (Mt)	Grade			Contained Metal		
				Cu (%)	Au (g/t)	CuEq (%)	Cu (Mt)	Au (Moz)	CuEq (Mt)
Open Pit	0.16	Indicated	492	0.22	0.20	0.35	1.1	3.1	1.7
		Inferred	45	0.18	0.18	0.31	0.1	0.3	0.1
Underground	0.19	Indicated	230	0.26	0.18	0.39	0.6	1.3	0.9
		Inferred	201	0.21	0.21	0.36	0.4	1.4	0.7
Total Indicated			722	0.23	0.19	0.36	1.7	4.5	2.6
Total Inferred			247	0.21	0.21	0.35	0.5	1.6	0.9

Notes:

1. Dr. Gilles Arseneau, P. Geo., Associate Consultant with SRK Consulting (Canada), is responsible for this Mineral Resource statement and is an “independent Qualified Person” as such term is defined in NI 43-101.
2. Reasonable prospects of eventual economic extraction were assessed by:
   a) First presenting the mineralised material in the block model estimate to a conventional Lersch-Grossman open pit optimisation routine based on a cut-off grade of 0.16% CuEq, and the cost and revenue assumptions listed below. Mineralised material inside the revenue factor one pit and above the cut-off grade were then reported in the “Open pit” section of the Mineral Resource statement.
   b) Subsequently, the remaining material was enclosed in a 3D wireframe shape that was constructed with adherence to a minimum mining unit with geometry appropriate for a block cave.
3. Cut-off grade for the underground shape was defined as the cut-off grade under a breakeven, eventual economic extraction criterion. The cut-off grade of 0.19% CuEq was calculated using (copper grade (%)) + (gold grade (g/t) x 0.683).
4. All material within the underground shape was reported in the “Underground” section of the Mineral Resource statement, as block caving is a non-selective method, and all material extracted is treated as mill feed.
5. The resulting shape contained planned internal and edge dilution that the QP considers appropriate.
6. Cut-off/Cut-off inputs included:
   a) Metal prices of $3.60/lb Cu and $1,700/oz Au
   b) Recoveries of 93% for copper and 83% for gold
   c) Costs including mining, processing and general and administration (G&A)
   d) Off-site realisation (TCRC), including royalties
7. The QP considers that the Mineral Resource has reasonable prospects for eventual economic extraction by open pit or an underground mass mining method such as block caving, as presented in the Mineral Resource statement.
8. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
9. Mineral Resources are reported inclusive of those Mineral Resources that were converted to Mineral Reserves.

15 Mineral Reserve Estimates

15.1 Introduction

The Mineral Reserve for the Alpala underground resource of the Cascabel project was converted by applying modifying factors to the Mineral Resource Estimate. Only Measured and Indicated categories have been converted to Mineral Reserves, with Inferred categories considered as waste and grades set to zero.

The Mineral Reserve was estimated using the following economic assumptions for block caving (Table 15-1).

Table 15-1: Alpala economic criteria

Parameter	Units		Values	
	(Cu)	(Au)	(Cu)	(Au)
Metal price	US$/lb	US$/oz	3.60	1,700
Unit conversion	lb/t	g/oz	2,204	31.103
Government royalty	%	%	5	8
Refining charge	US$/lb	US$/oz	0.079	5
Smelting	US$/dmt		79	
Shipping water content	%		8	
Shipping (including port cost)	US$/wmt		56.30	
Average LOM concentrate grade	%	g/t	22	16
Metallurgical recovery	%	%	93	85
NSR	US$/t		77.73	

15.2 Underground Block Cave Mining

Access to the Alpala underground mine was assumed to be via twin declines commencing from a boxcut located near the surface and the undercut development level for the first block cave footprint is 360 mRL. Mining is planned to be a block caving mining method, whilst all horizontal development will be undertaken utilising conventional drill and blast practices. The vertical development for the main ventilation raises will be excavated using raisebore methods.

15.3 Mineral Reserve Statement

The Mineral Reserve Estimate for the Alpala Underground Porphyry Copper-Gold-Silver Deposit, Cascabel Property, with an effective date of 31 December 2023 has been prepared under the supervision of SRK Consulting (Canada) Inc. Corporate Consultant Jarek Jakubec, C.Eng., FIMMM, who is the Qualified Person responsible for the Mineral Reserve Estimate Table 15-2.

The Mineral Reserve estimation process followed the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) “Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines” (CIM, 2019). The Mineral Reserve Estimate is stated in accordance with the CIM Definition Standards (CIM, 2014) and Canadian National instrument 43-101 (NI 43-101).

The Mineral Reserve has been estimated for a block cave mining method and considers the effect of mixing of Indicated material with dilution from low grade or barren material originating from within the caved zone and the overlying cave backs.

Table 15-2: Alpala Mineral Reserve Estimate (effective date 31 December 2023)

Category	Tonnes (Mt)	Cu Grade (%)	Au Grade (g/t)	Ag Grade (g/t)	Total Cu (Mlbs)	Total Au (koz)	Total Ag (koz)
Proven	457.5	0.64	0.60	1.7	6,475	8,854.5	24,942
Probable	82.2	0.36	0.22	1.2	653	578.7	3,093
Total	**539.7**	**0.60**	**0.54**	**1.6**	**7,128**	**9,433.2**	**28,034**

Notes:

1. CIM Definition Standards were followed for Mineral Reserves.
2. Mineral Reserves for the Cascabel project have an effective date of 31 December 2023.
3. The Qualified Person responsible for the estimate of Mineral Reserves is Jarek Jakubec, C.Eng., FIMMM.
4. Mineral Reserves are reported using long-term metal prices of $1,700/oz Au, $3.60/lb Cu, $19.90/oz Ag.
5. Mineral Reserves are constrained within a block cave design, using the following input parameters: height of draw of 400 m; mixing horizon of 350 m; 15% dilution (at 350 m column height); metallurgical recoveries that range from 68-81% for copper and 85-92% for gold; a footprint development cost of $1,750/m2; cut-off value of $15.00/t.
6. Units are metric tonnes, metric grams, troy ounces, and imperial pounds. Gold ounces and copper pounds are estimates of in-situ material and do not account for processing losses.
7. Totals may not match due to rounding.

15.4 Factors that Could Affect the Mineral Reserve

The Mineral Reserve could be materially affected by changes to the following factors, but not limited to mining, metallurgy, infrastructure and permitting:

- Commodity price fluctuations
- Metallurgical recoveries
- Geotechnical analysis and assumptions
- Hydrological and hydrogeological analysis and assumptions
- Cut-off and shut-off grade criteria
- Permitting and licensing, including environmental, social and governance
- Input assumptions into the mine design such as cave layouts and outlines
- Cost assumptions used to determine the Mineral Reserve
- Potential impact of seismic events to project facilities

16 Mining Methods

16.1 Geotechnical Considerations

The QP – Mr. Jakubec has conducted a review of the geotechnical data and interpretation completed by Mining Plus as part of Cascabel project PFS 2022 (Artica et al., 2022). The geotechnical study that was completed by Mining Plus encompassed:

- Rock mass characterisation
- Numerical model inputs
- Numerical model analyses
- Ground support recommendations.

The primary inputs for the study comprised the following.

- The in-situ stress measurements
- Borehole surveys; digital wireframes
- Geotechnical database that includes rock mass assessments
- Discontinuity measurements and televiewer logs
- Rock strength laboratory testing

The QP notes that Mining Plus used empirical and analytical analyses of this data to determine the geotechnical characteristics of various lithological and alteration domains within the deposit. Where information was unavailable, Mining Plus used industry-accepted practices and assumptions to complete the study. These practices and assumptions were reviewed by Mr. Jakubec.

16.1.1 Geotechnical Domains

Geotechnical Domains (Volumes) were classified at the onset of the program by SRK in 2020, based on combinations of lithology, primary alteration, and secondary alteration after suggested methodology proposed by Flores and Karzulovic (2003) – the combined lithology and alteration profiles are referred to as Underground Geotechnical Units (UGT) within this report.

The damage zone study determined that the damage zones (RQD<50) match with the northwestern contact of the D10 intrusion, indicating a lithological control to the damage zone location and not necessarily a structural control. The strong damage zones (RQD<25) are bordering the D10 unit and contain high mineralisation (UGT units 4 and 5). The QP relied on the existing wireframes generated during the Mining Plus geotechnical study.

Faults were interpreted by SRK Consulting (Chile) and summarised in a memorandum titled “Cascabel Structural Model Update, Project Number 01-2783-01”, dated 15 May 2020. The faults were interpreted from geology and geotechnical boreholes and from structural interpretation of the region.

16.1.2 Stress Model

In situ stress state information was derived from an interim report documenting the results of overcore measurements completed at the Cascabel project (Mills, 2020) and a subsequent SRK memorandum reviewing the available Cascabel stress measurements (Russo, 2020).

The stress fields used in the geotechnical numerical models and associated work are provided in Table 16-1 (after Mills, 2020).

Table 16-1: Estimated major principal stresses for Alpala deposit

	σ1	σ2	σ3
Dip	23	25	55
Dip Direction	264	163	032
Ratio to σ3	3.1	1.7	1.0
MPa/m	0.043	.024	.014

16.1.3 Intact Rock Strength

Three separate batches of laboratory testing were completed in the past. The first round of testing (R1) was performed by Strata Testing Services (STS), located in Australia and was focused on the borehole breakout and overcore samples obtained from the Insitu Stress Estimation testing; the second round of testing was performed by STS on boreholes within the upper portion of the deposit and decline area with samples obtained from geotechnical boreholes (R2); and the third round of testing performed by SGS Santiago on Laboratory testing on select samples from over-drilled wedge holes and other miscellaneous whole core samples that provided better definition of the geotechnical units and the deposit at depth (R3).

Laboratory samples were classified by geotechnical units and assessments were made to determine the minimum, average, median, and maximum values for each test type for both valid tests (per ISRM Standards), and for all test performed.

16.1.4 Rock Mass Assessment

The QP notes that Mining Plus performed a statistical analysis of the rock mass ratings provided by SolGold. The database contained RQD, Fracture Frequency, RMR89, GSI, NGI Q' Index, RMR90, and IRMR. The provided database was coded per UGT such that each run interval was assigned to a UGT. Intervals logged as Damage Zones or Faults were separated from the UGT and analysed separately for rock mass properties. Further separation of the UGT were made for the Decline, Lower Ore Body and Upper Ore Body location. In general, the rockmass ranges from Poor to Good, on average Fair by most classification schemes. Faults and damage zones are generally Poor to Very Poor by the majority of classification schemes.

16.1.5 Geotechnical Material Properties

The geotechnical properties for the Alpala deposit were determined from the intact lab strengths, investigation of the core logging database, rock mass statistics, and empirical databases where no information was available. The upper (above 650 MASL) and lower databases (below 650 MASL) were separated to account for the differences noted in rock masses. For each database, the low, middle, and high cases were determined to reflect the variation within the rock strengths and rock masses. The rock mass inputs were used to determine the excavation stability and caveability through numerical analysis.

16.2 Mining Method Selection

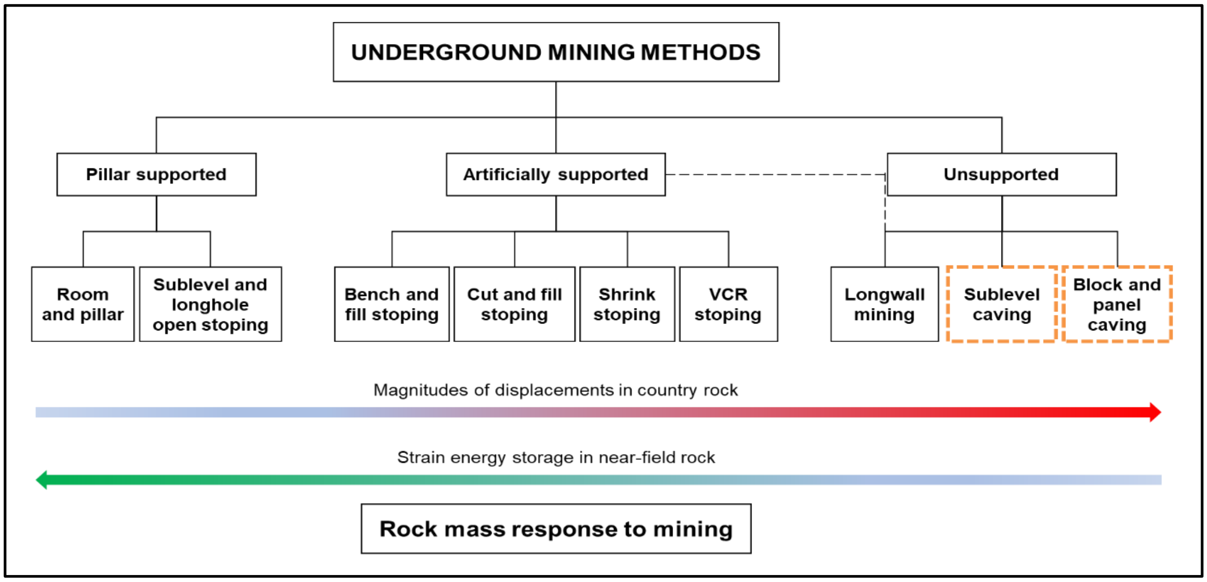
A mining method must be appropriate to the context of rock mass, orebody geometry and geology. Those are the conditions that cannot be changed and in which the mining method must be performed. If the mining method conflicts with the context, it will not perform as expected (and expectations usually come from other mines where the method is applied), as outlined by Jakubec et al. (2004).

In summary, the choice of a mining method depends on the following:

- Geology (pipe internal and external geology, phases and facies, etc.)
- Orebody size and geometry
- Grade (value) distribution
- Geotechnical conditions (rock mass competency, structures, rock mass strength, etc.)
- Disturbances (in-situ stress, groundwater, gases etc.)
- External constraints (such as desired production rates, etc.)

Other criteria that were taken into consideration are CAPEX, OPEX, and production rates. Based on SRK cave mining experiences, the direct mining cost for block caving is approximately 25-50% of SLC mining method and only 10% or less of the backfill method. Regarding production rates, block caving has the potential to produce the highest tonnage relative to SLC and backfill mining methods.

Based on value per tonne, desired production rates, in-situ geotechnical conditions and worldwide experience with underground mining, The QP has concluded that the most viable methods for effective extraction of the bulk of the Cascabel underground resource are “Unsupported Mass Mining Methods” (Figure 16-2), specifically block caving and its variations (‘caving’ methods).

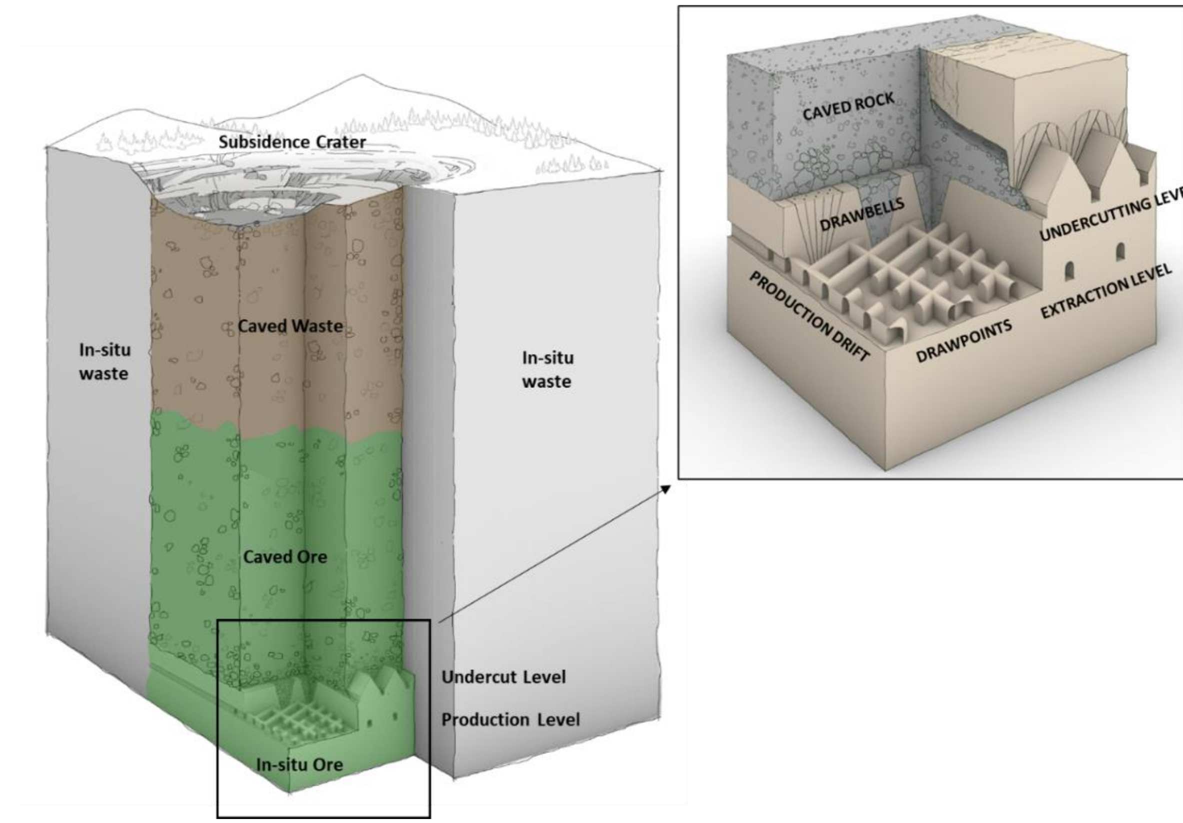


Source: Jakubec et al. 2004

Figure 16-1: Underground mining methods

16.2.1 Mining Method Descriptions – Block Cave

Block Caving (BC) is the mining method in which, panels, or blocks of ore are undercut to induce caving, permitting the broken ore to flow to the draw point via gravity to be gathered and then taken away for processing. The result of the caving process is a surface subsidence. Upfront development for block caving is typically extensive but generally less that for sublevel caving on a unit-cost basis. To offset the impact of the large capital expenditure, a consequent high rate of production is required. Today, several cave operations are mining at upward of 50,000 stpd (short tons per day) and super caves are being constructed with nameplate capacities of 100,000 stpd and more. The BC mining method is illustrated in Figure 16-2.



Source: Jakubec et al. 2023

Figure 16-2: Block cave mining method example

16.3 Strategic Option Assessment

16.3.1 Introduction

An Excel®-based Fast Evaluation Model (FEM) was developed to provide a high-level economic analysis of the mining options considered. The model allows for the rapid assembly of production cases by sequencing individual mining schedules and combining them with a selectable mill rate assumption. Automatic rescheduling allows the flow of ore to the mill to be re-scheduled to reflect the mill constraint and any relevant individual mining constraints.

The model is a discounted cash flow (DCF) model that estimates project cashflows. The model used constant (real) 2023 USD and modelled the project cashflows in annual periods.

The model considered only cashflows from the beginning of construction forward. It did not model expenditures for studies, exploration, optimisation, design, permitting, and other pre-construction activities.

The model does not place the Project within an estimated calendar timeline and is intended only as an indication of the Project's economic potential to assist in investment decisions.

16.3.2 Results

The work clearly indicates the economic potential of the Project. Positive NPV outcomes were identified for all cases modelled.

16.4 Production Shape Modelling

16.4.1 Cut-off Value

Cut-off grade (COG) or cut-off value (COV) is a standard, industry-accepted method used to determine which part of a mineral deposit to include in a Mineral Resource or a Mineral Reserve estimate or potentially in an operation's life-of-mine (LOM) plan. It is the minimum grade (or value) at which mineralised material can be economically mined or processed. The selected COV is essentially a trade-off between the revenue (inclusive of losses) that the potentially economic material contributes to the mine's cash flow vs. the cost to extract that same material. COV is an essential parameter for determining reserves, generating production and business plans, and ascertaining the potential profitability of a production shape.

The process of selecting a COV should begin with understanding the over-arching corporate and mine objectives for the deposit. Typically, it is valued in the form of NPV and IRR, but may also include a COV that produces high copper or gold content with the consequence of lowering the NPV.

The widely adopted method to calculate the COV is a break-even methodology. This approach accepts mining material, which will generate revenue from the sale of the finished product that is equal to the cost of certain modifying factors, such as mining, processing, G&A, sustaining capital costs, and treatment and is often inclusive of applicable royalties.

Net Smelter Return (NSR) Parameter

SRK used a calculated Net Smelter Return (NSR) value on the block model to determine the COG/COV. NSR values were then calculated for each block based on the copper and gold grades.

Mining Costs

Mining costs shown in Table 16-2 are based on a combined set of inputs provided by SolGold and SRK internal benchmarks and experience. These costs were used as inputs for the production analysis.

Table 16-2: Mining operating costs for COV

Description	Unit	BC
Mining Cost	$/t	5.50

Other Costs

Other operating costs, including processing, G&A, tailings storage facility (TSF), and port costs, were taken from the 2022 PFS and are shown in Table 16-3.

Table 16-3: Operating costs from 2022 PFS

Description	Unit	Cost
Processing Cost	$/t	6.38
G&A Cost	$/t	1.58
TSF	$/t	1.25
Port	$/t	0.29
Total Operating Cost	**$/t**	**9.50**

Selected Cut-off Values

Based on the estimated ore value and operating costs, SRK developed the preliminary mill feed break-even COV for the different mining methods presented in Table 16-4.

Table 16-4: Operating costs for COV

Description	Unit	BC
Mining Cost	$/t	5.50
Processing Cost	$/t	6.38
G&A Cost	$/t	1.58
TSF	$/t	1.25
Port	$/t	0.29
Total Operating Cost	**$/t**	**15.00**

16.4.2 Block Cave

Block caving was considered the most suitable method of extracting the most value from the Alpala deposit. GEMS™ Footprint Finder was used to determine the economic and practical footprint locations.

The process of determining the optimal mine plan is an iterative one and should continue to be assessed at future stages of study.

Input Parameters

Material Mixing - an initial Footprint Finder analysis was conducted for the entire block model with the following Laubscher mixing parameters:

- Mixing Horizon = 350 m
- First Dilution Entry = 60%

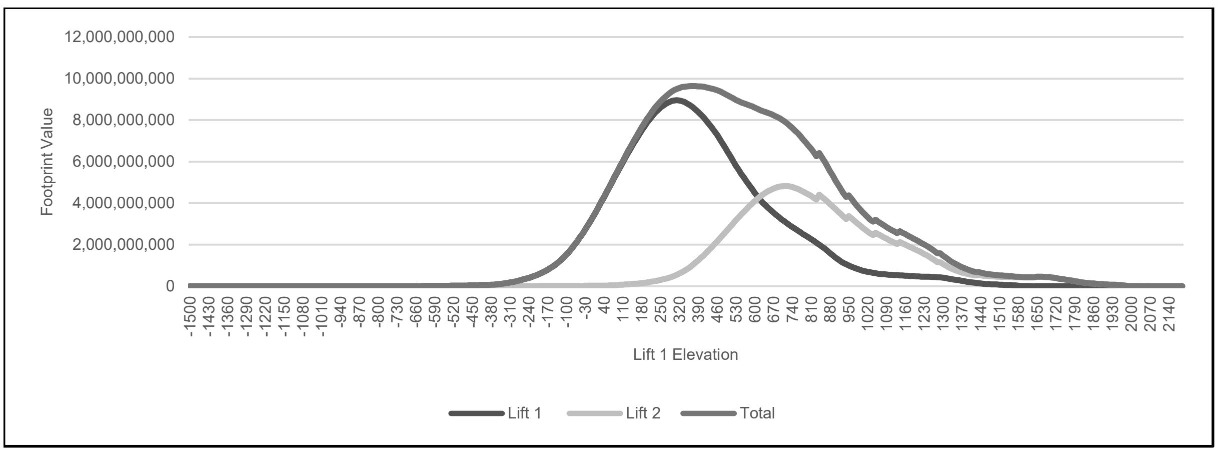
These settings correspond to 15% dilution for an ore column 350 m in height. However, the settings do not indicate a fixed dilution rate but rather the degree of mixing applied to all columns in the orebody. Ore columns with heights less than 350 m will have increased dilution, while ore columns higher than 350 m will have reduced dilution.

Height of Draw - each block column will be limited in terms of the maximum height of draw (HOD). A practical HOD for this deposit was estimated to be 400 m.

Economic Parameters - the COV mentioned above was used as the block value input for Footprint Finder. A footprint development cost of $1,750 /m^2 was used for the valuation based on SRK internal benchmarks. This cost includes footprint development (e.g., extraction drives, drawbell construction, and undercutting).

Initial Analysis

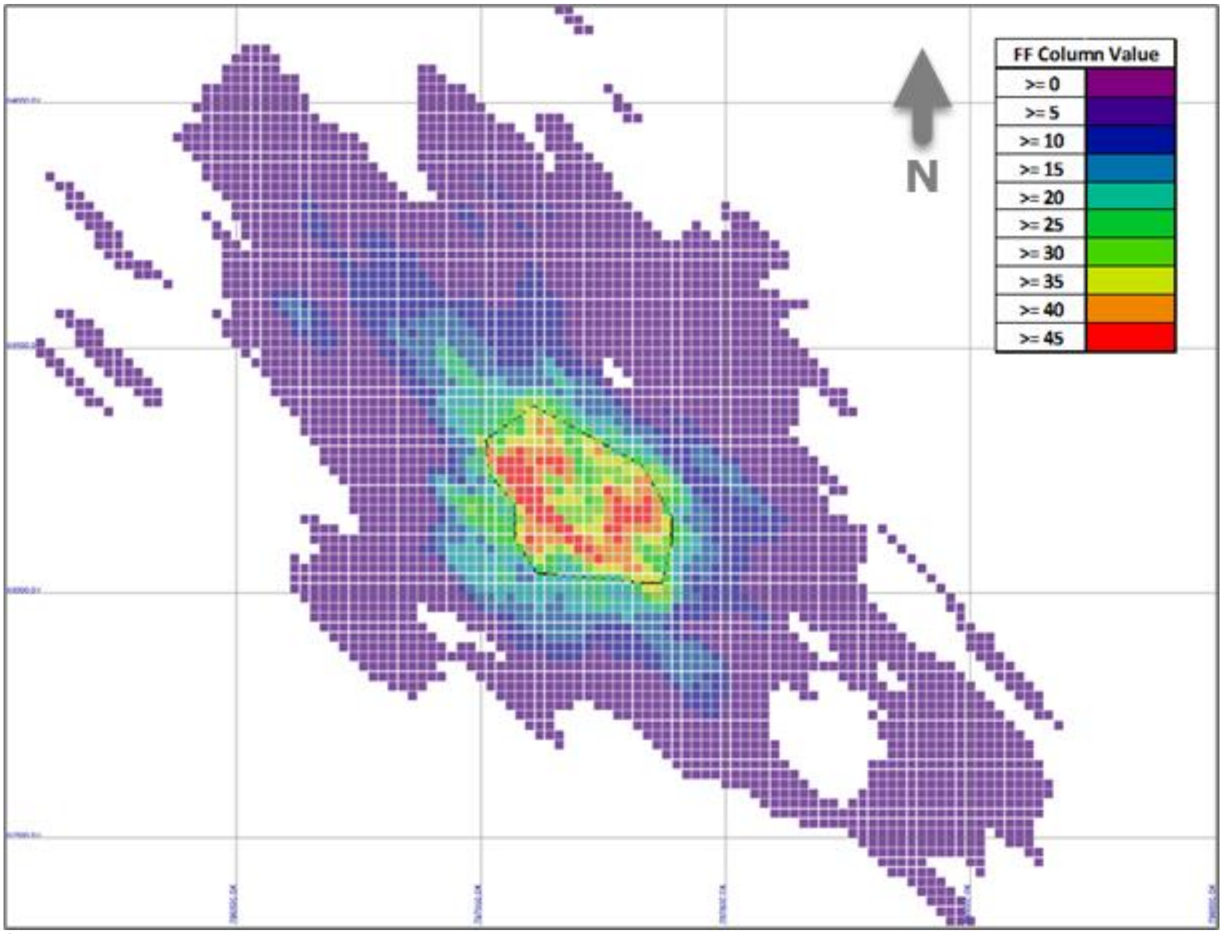
The initial analysis was run over the entire block model to determine the optimal location for the block cave footprints. Figure 16-3 shows the value of a block cave footprint at different elevations over the whole deposit.



Source: This study, 2024

Figure 16-3: Block cave value at different elevations (Block Cave 1)

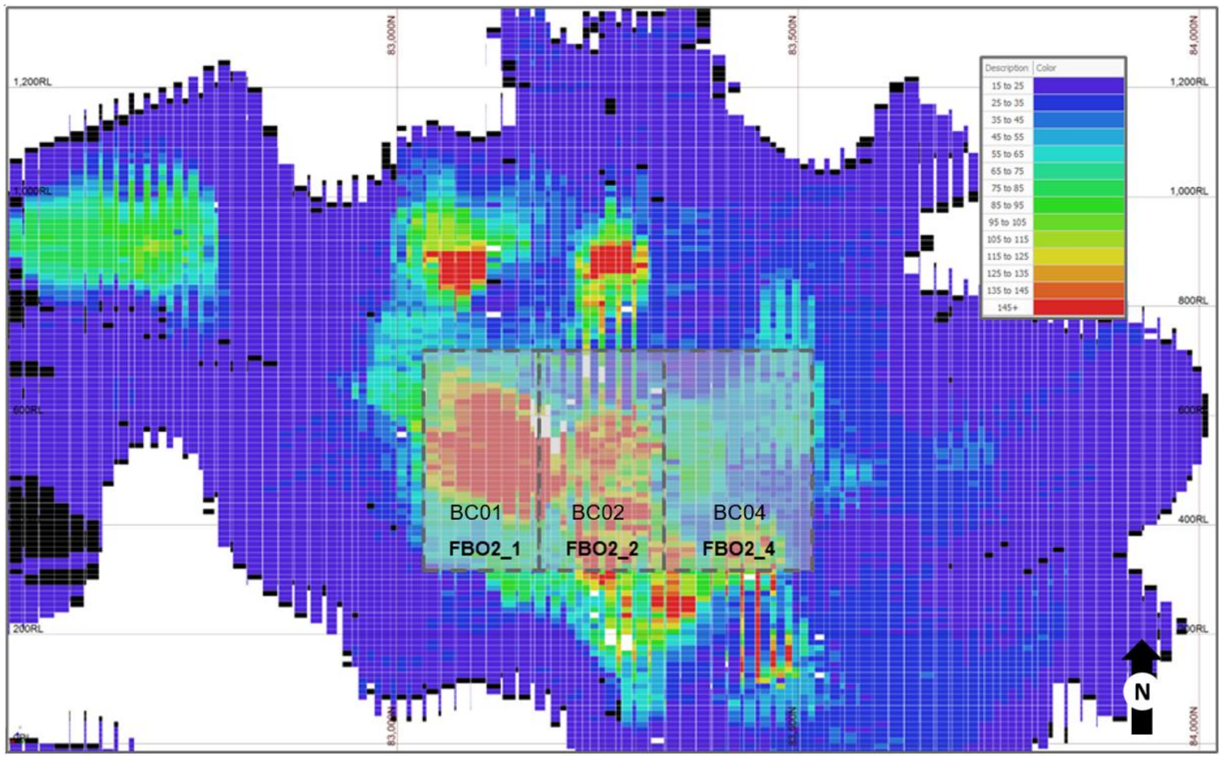
Figure 16-4 shows the result of this initial run at the best elevation, 380 mRL. The colours correspond to the value of a vertical column above the selected block.



Source: This study, 2024

Figure 16-4: Column values in millions of US$ for block cave footprint at 380 mRL

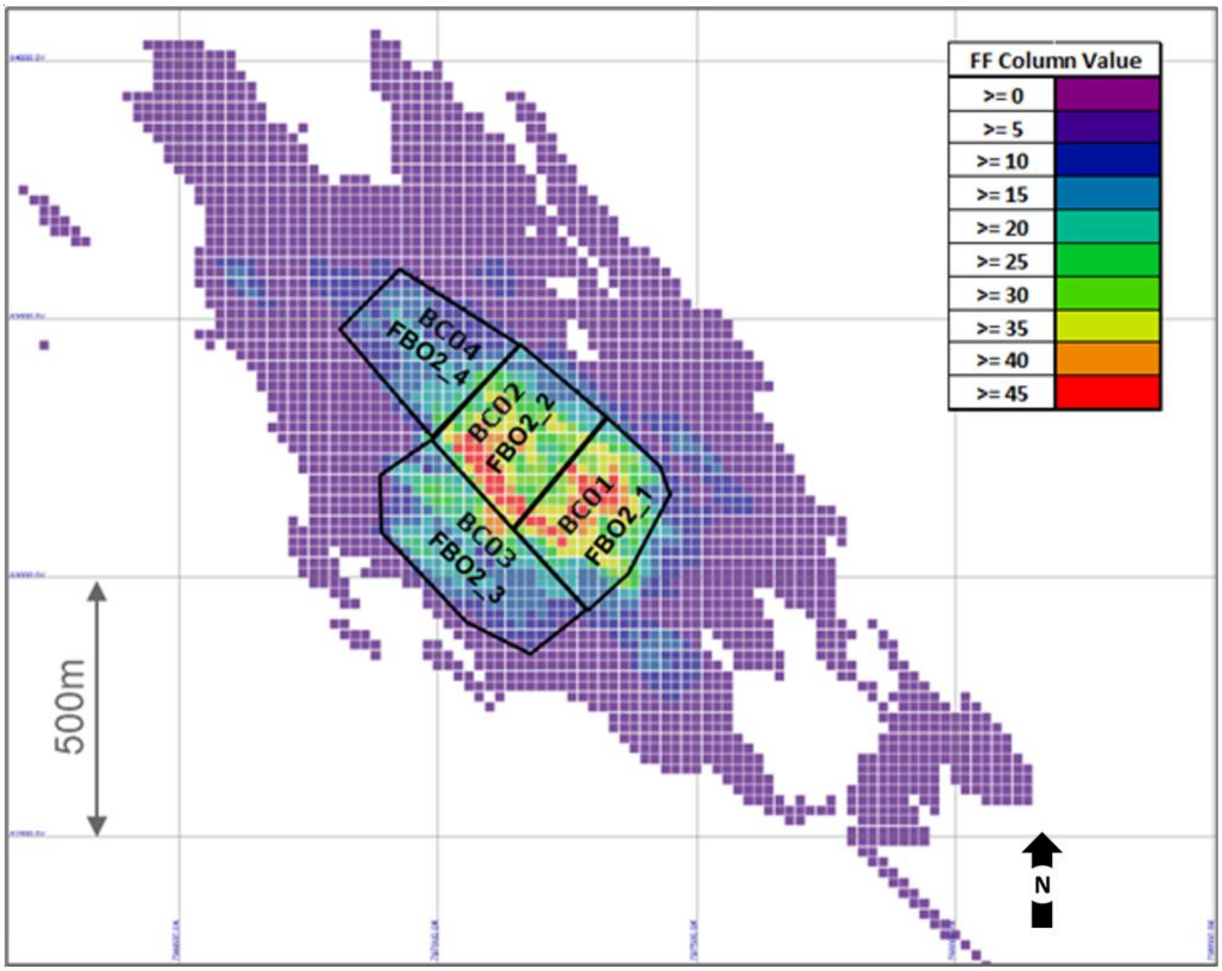
The block cave footprint was selected to target the highest-value columns, in this case, the central core of the deposit. The elevation of the block cave extraction level was selected to be 380 mRL as this is the highest value elevation below the high-grade core of the deposit, as shown in Figure 16-5.



Source: This study, 2024

Figure 16-5: Long section showing NSR value (US$) and selected block cave footprints

The selected footprint outlines are shown in Figure 16-6.



Source: This study, 2024
Note: For operational and practical reasons the highest value footprint was subdivided into four blocks

Figure 16-6: Selected block cave footprints

Selected Footprints

The production tonnes and grades from the four selected block caves are shown in Table 16-5.

Table 16-5: Block cave mineable inventories

Description	Unit	BC_01	BC_02	BC_03	BC_04
Extracted Tonnes	t	107,590,056	111,295,704	142,344,432	156,391,440
Grade (Cu %)	Cu %	0.74	0.76	0.56	0.44
Grade (Au gpt)	Au gpt	0.87	0.77	0.43	0.28
NSR Value	$/t	83.33	80.13	51.96	36.59
Grade (CuEq %)	CuEq %	1.33	1.29	0.86	0.63

16.5 Underground Development Design

16.5.1 Development Profile

Various development profiles were used throughout the design to take into consideration stability and the final use of the development. Table 16-6 outlines the development profiles used for the mine design.

Table 16-6: Development profile summary

Description	Development Profile	Total Metres
LATERAL - OPERATING		
Block Cave - Extraction Drive	A_4.5 mW X 4.5 mH	14,125
Block Cave - Drawpoint	F_4.5 mW X 4.5 mH	19,445
Block Cave Undercut Drive	A_4.5 mW X 4.5 mH	17,172
LATERAL - CAPITAL		
Decline	A_6.0 mW X 6.0 mH	8,770
Incline	A_5.5 mW X 5.5 mH	572
Conveyor	A_6.0 mW X 6.0 mH	9,711
Access Drive	A_6.0 mW X 6.0 mH	1,370
Main Footwall Drive	A_5.0 mW X 5.0 mH	2,731
Fresh Air Drive	A_5.0 mW X 5.0 mH	4,559
Return Air Drive	A_5.0 mW X 5.0 mH	4,387
Haulage Drive	A_5.0 mW X 5.0 mH	1,112

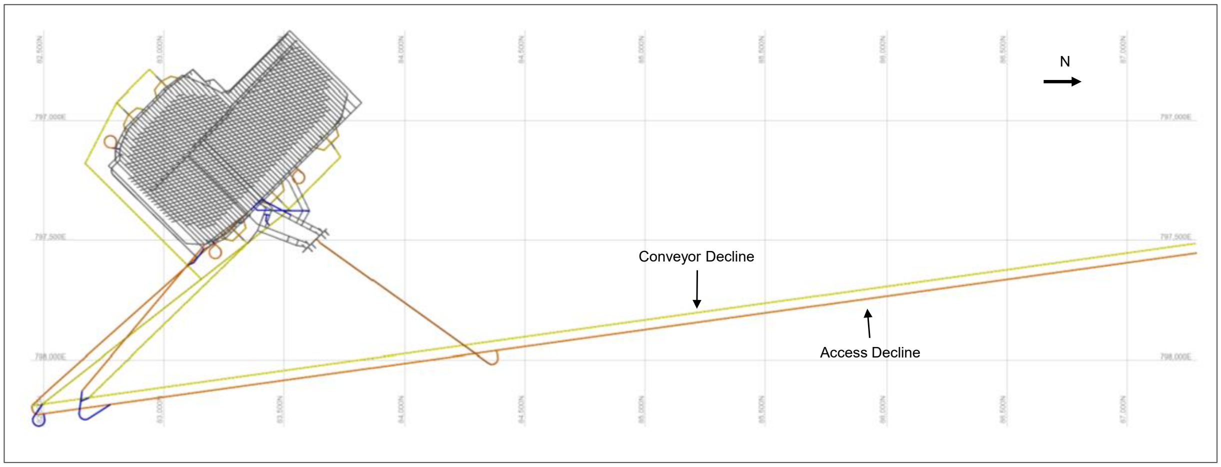
16.5.2 Advance Rates

Each drive type has a specific advance rate in metres per month and per day. Based on the number of headings open at any given time and the multiple headings constraint, these are the maximum number of metres that can be advanced per heading per month.

It should be noted that the assumed advance rates are individual face advance rates based on multiple headings being developed simultaneously, all using centralised blasting.

16.5.3 Mine Access

The mine will be accessed using dual decline from the surface. One decline will be used for the conveyor system, and the other decline will be used for equipment and logistics. The declines will daylight at a portal on surface (Figure 16-7).



Source: This study, 2024

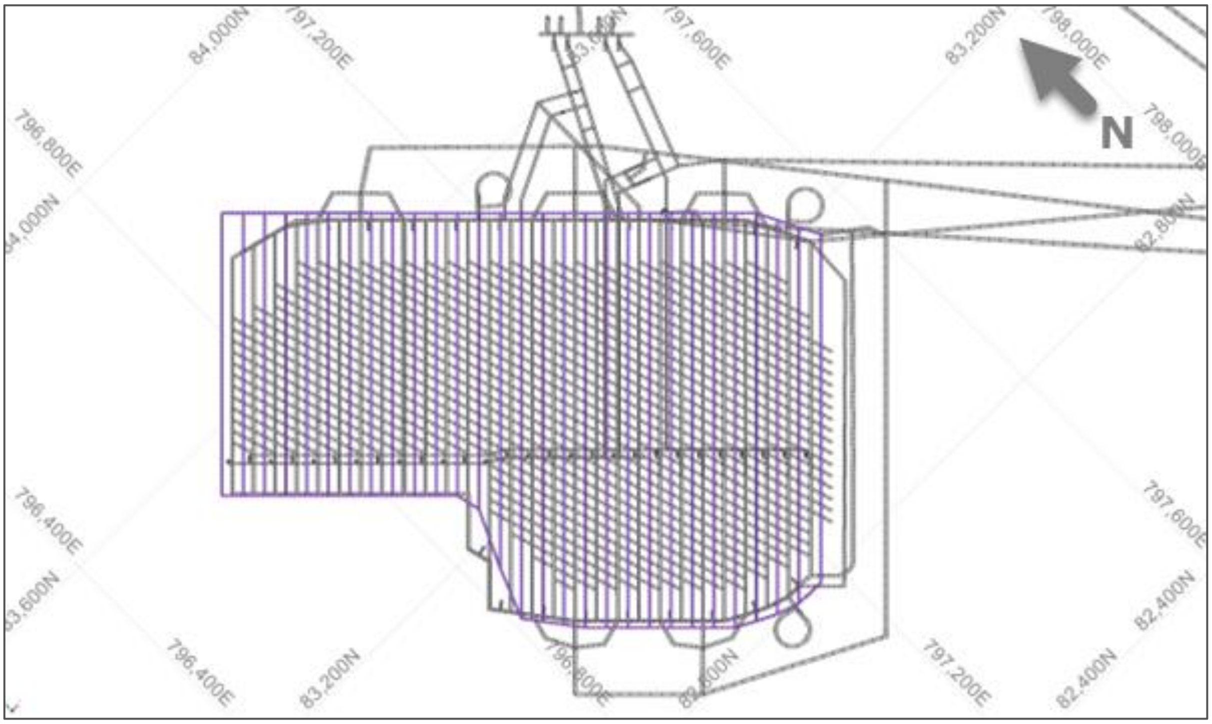
Figure 16-7: Mine access

16.5.4 Block Cave Layout

A block cave layout was designed based on the block footprints generated by Footprint Finder. The combined footprint of each lift included four development levels: undercut, extraction, ventilation, and truck haulage. These levels are connected by an internal ramp system.

Undercut Level

The undercut level is located 18 m above the extraction level and is accessed via ramps from the extraction level. The undercut drives are developed on 32 m centerline spacing and have the size of 4.5 mW x 4.5 mH. A 4.5 mW x 4.5 mH rim drive around the undercut level provides 360° access to the cave footprint. The layout of the undercut level is shown in Figure 16-8.



Source: This study, 2024

Figure 16-8: General layout of undercut level

Ventilation to this level is provided via intake air raises located along the northwest rim drive. Air is routed to the return air level via return air raises situated on the southeast side of the caving footprint. These raises also serve the extraction level.

The undercut will be taken to a height of 17 m above the floor of the undercut level. This is relatively high and is recommended to minimise potential problems arising from oversized rocks in the drawpoints.

Only 50% of in situ undercut tonnes will be mucked from the undercut level, with the remainder mucked with the underlying drawbells; this is to provide a buffer to protect against airblast risk.

The undercut ore will be transferred to the extraction level via an ore pass and will be re-handled to the primary crusher on that level.

An advance undercutting technique is proposed to destress the ground in the extraction level below prior to developing the drawpoints and drawbells. This technique will reduce the risk of potential damage to drawpoints during undercutting. Footprint Finder does not consider the specific undercut method, and such detail is generally beyond what would be justified for the level of this study. The post-undercut method offers some advantages from a production standpoint since broken material can be removed from the undercut level via the drawbells. However, as caves have become deeper and it is increasingly difficult to manage the abutment stress by ground support alone, other strategies such as advanced undercutting are available. The advanced undercut method addresses this issue by having drawbell development within the stress shadow of the undercut level once it has passed over the drawbell location. The advanced undercut strategy involves a high degree of scheduling and coordination between activities on the undercut and extraction levels. It may appear slower, but in the long run, it usually saves time by reducing time-consuming and costly repairs to the extraction level potentially associated with the post-undercut method.

Extraction Level

The extraction level of the block caves is located at 380 mRL for BC0 and 270 mRL for BC02, BC03, and BC 04. The footprint area, perimeter and hydraulic radius (HR) for each of the four-block cave areas are shown in Table 16-7.

Table 16-7: Block cave footprint dimensions

Description	Unit	BC_01	BC_02	BC_03	BC_04
Footprint Area	m^2	67,453	73,980	93,038	103,482
Perimeter	m	1,018	1,090	1,302	1,274
Hydraulic Radius		66	68	71	81

An “El Teniente” layout was selected for the extraction level. The extraction drive and drawpoint size is 4.5 mW x 4.5 mH.

Ore will cave via drawbells from above the undercut level. The number of drawpoints for each cave area is shown in Table 16-8.

Table 16-8: Drawpoint quantity by block cave area

Description	Unit	BC_01	BC_02	BC_03	BC_04
Drawpoints	#	125	136	168	193

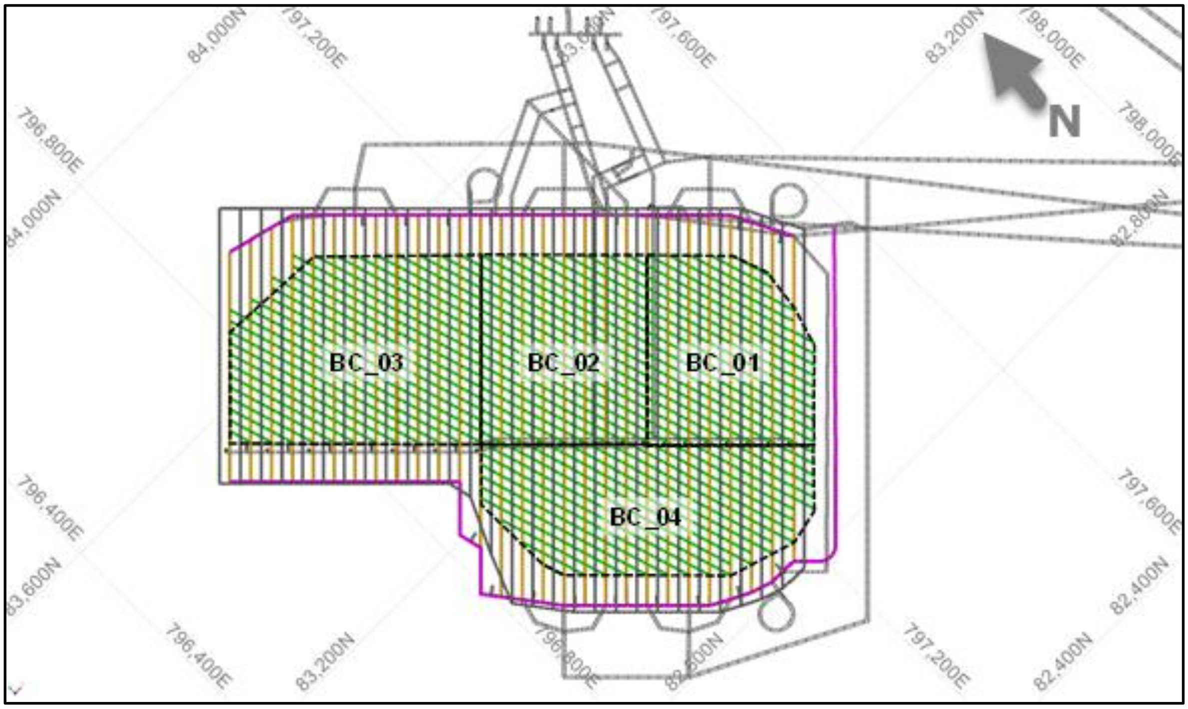
Drawbell dimensions are based on consideration of fragmentation and stability of workings. The width of the pillar across the major apex is 20 m, with the drawbell being 11 m in that dimension. Across the minor apex, the drawbells connect at a height of 12 m above the extraction level floor to form a continuous trough; this design was chosen as it is more favourable in terms of oversize being able to move towards the drawpoint, but does result in a reduced minor apex pillar compared to drawbells connecting at the undercut level elevation.

Material is mucked from the drawpoints and transported directly into the crusher tip point.

A single 5.0 mW x 5.0 mH rim drive will be developed around the extraction footprint. Rim drives are provided for access to the cave footprint, with all non-LHD traffic using these for access, including supervisory, secondary breakage, maintenance, technical services, rehabilitation, and construction traffic.

Ventilation to the extraction level is provided via intake air raises located on the perimeter footwall drive. Air is routed to the return air level via return air raises located in the center of the extraction footprint.

The general layout of the extraction level is shown in Figure 16-9.



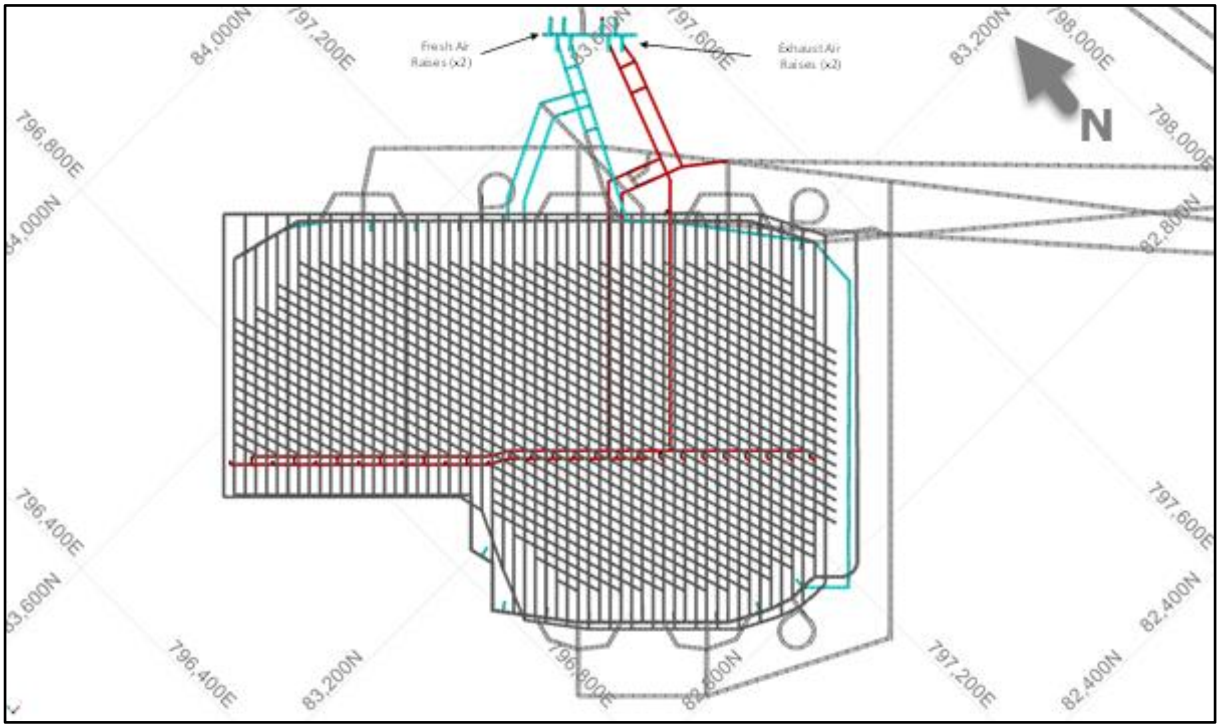
Source: This study, 2024

Figure 16-9: General layout of extraction level

Ventilation Level

Fresh air for the mine is circulated along one side of each footprint and flows to the exhaust air system across the extraction and undercut levels (Figure 16-10).

Exhaust air from the working levels is sent down to a return 4.5 mW x 4.5 mH airway in the center of the footprint using 4 m diameter raises. This drive is connected to the main return air raise with a 5 m x 5 m drift. These drives would also be used as a water collection level below the footprints.

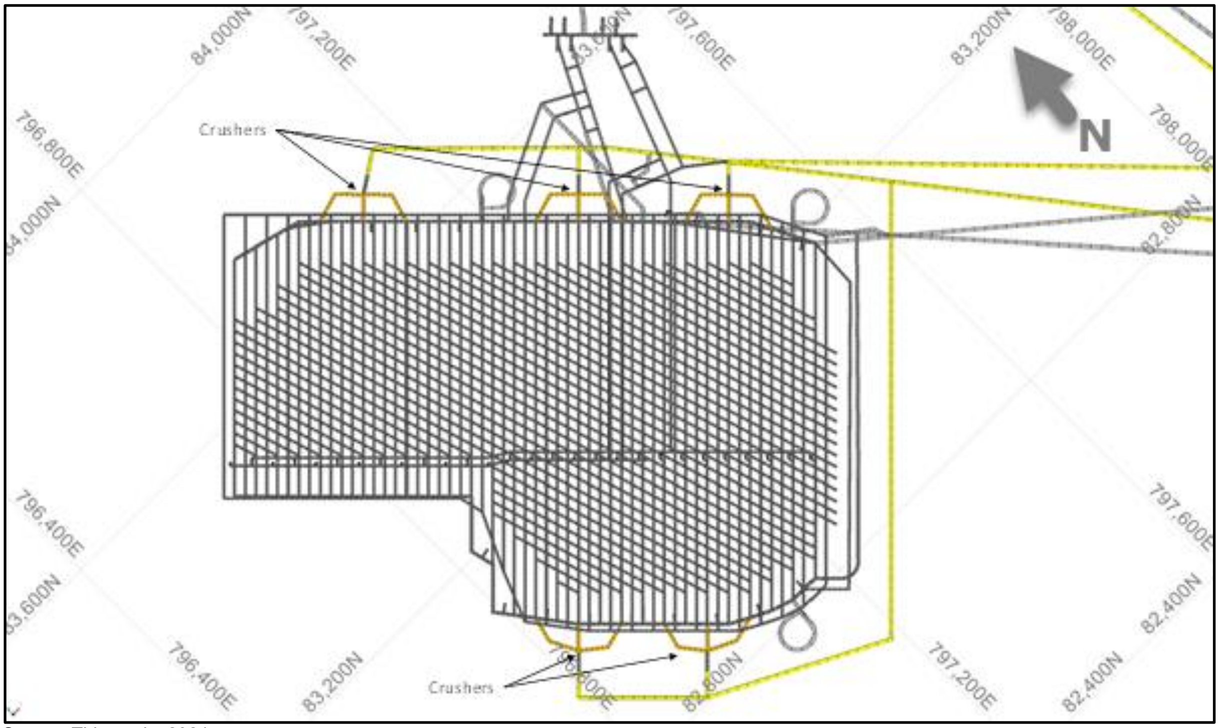


Source: This study, 2024

Figure 16-10: General layout of ventilation system

Materials Handling Level

Ore is mucked from the block cave drawpoints and dumped directly into the crusher for each block cave area. The crushed material is then transported to the surface on a conveyor system. Figure 16-11 shows the material handling level configuration for the block caves.



Source: This study, 2024

Figure 16-11: Material handling level configuration

16.6 Underground Infrastructure Design

16.6.1 Materials Handling System

Crushing

Five single-stage jaw crushers will be installed underground. Table 16-9 summarises the criteria for the crushing system required to maintain the 24 Mtpa production rate contemplated in the PFS.

Table 16-9: Crushing parameters

Parameter	Value	Unit
Annual Tonnage Throughput	24	Mt/y
Operating hours	5,508	h/y
Crushing Rate	3,400	t/y average
Crusher System Proposed Throughput	4,175	t/h
Conveying System Design Tonnage	4,500	t
Max Particle Feed Size	1,000	mm
Feed Particle Size Distribution	450	P80
Particle Size Distribution on Belt	150	mm
Clay content	2%	%
Moisture Content blasted ore	4%	%
Conveyor Belt Width	1,500	mm
Maximum Angle	15	degrees

Conveying

The LOM requirements for the conveying requirements were established based on the ramp and access layouts developed by SRK. The power requirements were established based on the tonnages and slopes provided.

The design criteria used for the estimation process are the following:

- All drives are support from foundations set on the ground of the conveyor gallery
- The conveyor structures will be supported from the roof and not the floor to permit cleaning under the belts
- Belt length limited to 850 m – 890 m in the inclined sections. Longer belts are possible when level or downhill. The length has been selected to permit a standardisation of the belt drives to less than 3 MVA. Longer belts are possible, but a detailed design is required to optimise the belt system.
- All belts require: drive stations, hydraulic tensioning structures, tail pulley structure and belt installation station
- Belt turners have been contemplated in this design as they will reduce the maintenance and operating costs of the conveying system
- Belts will be steel core due to the lengths
- All belts should be fed with a mechanical feeder to reduce belt wear
- All belts are assumed to be fixed speed, but the use of variable speed drives should be considered at the Feasibility level
- The conveyor galley must be separated from the main ramp, and fire doors are required where the two ramps communicate. The conveyor gallery must have upcast ventilation.
- The conveyor gallery cannot be considered as an emergency exit or travel way

All material that will be transported on the belt conveying system will have been previously crushed and the tramp iron removed. The particle sizes that must be supplied to the conveying system are found in Table 16-10.

Table 16-10: Particle size requirements

Passing	Dimension
P100	300 mm
P80	150 mm
P50	85 mm

It is assumed due to operational and maintenance considerations that the crushers will produce at a maximum throughput of 62% of their nameplate capacity values based on operating hours.

The design considerations for sizing the conveying system are found in Table 16-11. It defines the operating, maintenance hours based on a normal operating mine. The conveying system will have a nominal nameplate capacity of 4,600 t/h. The effective operating hours in the day have also been considered in the calculations.

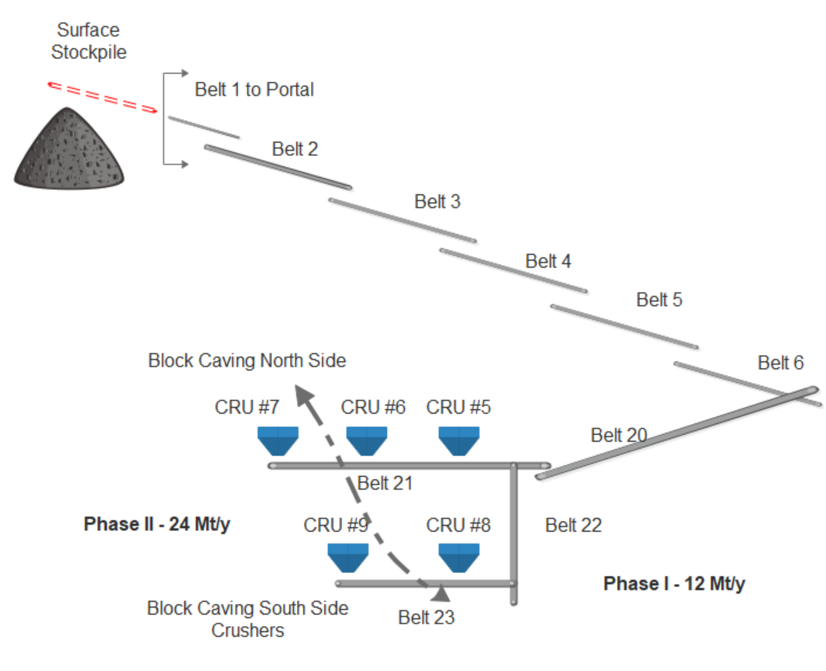
Table 16-11: Conveying design criteria

Description	12 Mtpa	24 Mtpa	Unit
Annual Production Mine	12,000,000	24,000,000	t/y (including waste)
Mine operating days	360		d
Belt Operating Days	27		d/m
Daily Operating Hours	17		h/d
Belt Speed	2.5		m/s
Ore bulk density	1.6		t/m^3
Belt Width	1,600		mm
Particle Size	165		mm (P80)
Troughing Angle	35		degrees
Surcharge Angle	20		degrees
Belt Edge Distance	120		mm
Belt Cross Section (Material)	0.162	0.296	m^2
Belt kg/m	58		kg/m
Ore kg/m	259.4	473.1	kg/m
Ore kg/m/s	648.5	1,182.7	kg/s
Max design slope angle	15%		% slope
Tension Safety Factor	2		
Efficiency	60%		
Maximum Belt Angle	18		degrees
Operating hours	5,508		h/y
Design throughput	2,335	4,258	t/h
Moisture Content	4%		
Throughput based on Hours	2,428	4,428	t/h

Figure 16-12 illustrates the conveying and crushing proposed layout. It comprises of 10 principal belts and five secondary feed belts one for each crushing station. The ore will be transported to surface using a belt conveying system with the characteristics found in Table 16-12.

Table 16-12: Ramp conveying power estimate

Parameter	Value	Unit	Notes
Belt Width	1,600	mm	
Throughput Nameplate	4,600	t/h	
Substation Power Capacity	5.0	MVA	Assume 0.93 kW/m of conveyor on 15 % slope
Motor Sizing	3,000	kW	Motor sizing to be limited to this value if single drive units are available. If not possible then dual drive units would be considered.
Motor Voltage	4,160	V	50 Hz



Belt	Length (m)	Width (mm)
M1	775	1600
M2	878	1600
M3	878	1600
M4	876	1600
M5	876	1600
M6	519	1600
Total	4802	
B20	573	1600
B21	249/124	1600
B22	711	1600
B23	300/199	1600
Total	2156	
Total M & B	6958	

Source: This study, 2024

Figure 16-12: Conceptual conveying system

The belts should be adjustable for practical reasons to approximately 800 m to 880 m in length prior to requiring a transfer. This distance is mainly defined for practical reasons in terms of belt changeout as well as repair time once a major mechanical issue occurs on the belt. Where possible, the belt drives should be as standard as possible so they can be interchanged in case of a mechanical issue with the drive units. If possible, the drives will be single motor drives. If standard gearboxes make it practical to operate with integral drive units, this should be considered.

The belts will have to be protected by a sprinkler system even though the belt material itself is fire retardant. All belts must be equipped with anti-rollback and an additional braking system. Belts must have “inching” capacity to permit the safe maintenance of the belt. All belt tensioners will be hydraulic. All belts should be steel mesh conveyor belts to reduce to a minimum the damage caused be a tear.

Over the course of the mine life, nearly 7.4 km of conveyor belt structures over 15 belts will need to be maintained (Table 16-13). The belt widths will vary based on their location and capacity requirement. The belts have been standardised to 1,600 mm for the main drives and 1,500 mm in the crushing and intermediate belt area. The block caving belts are located on the 240 mRL elevation.

Table 16-13: Belt summary

Belts	Length (m)	Number of Belts
Block Caving	2,156	4
Decline	4,802	6
Intermediate Belt	400	5
Grand Total	**7,358**	**15**

The main decline belts are the belts located in the ramp conveyor galleries going down to the block cave area. The intermediate belts are the belts that transport the crushed rock from the crushers to the main transport belts. The crusher belts will be the sacrifice belts prior to feeding the Intermediate belts. All transfers from the intermediate belts to the main belts will be done using mechanical feeders to avoid overloading the main belts.

Up to four crushing stations will be feeding the main belts simultaneously. To optimise throughput some form of automation to reduce delays in the crushing circuits. No surge bins or silos are considered for the system. Each crushing station should have a remuck bay and an oversize muck bay to optimise the use of the conveying system.

16.6.2 Electrical

Due to the length, tonnage, and the vertical rise of the conveyors; 4,160 V motors will be the preferred voltage choice.

The conveyor motors will be the largest motors being used underground and in the range of 3,000 kW divided into a multi-motor configuration. This permits the use of TEFC motors that are less susceptible to dust conditions. Beyond this size, dual motor drives will be used.

The water inflow for the mine is estimated at 700 m^3/h. Twelve pumping stations will be required to ensure that the mine remains dry. The conveying and pumping substations will normally be shared substations to minimise the number of substations required.

Table 16-14 summarises the power requirements for mining and development, excluding fans.

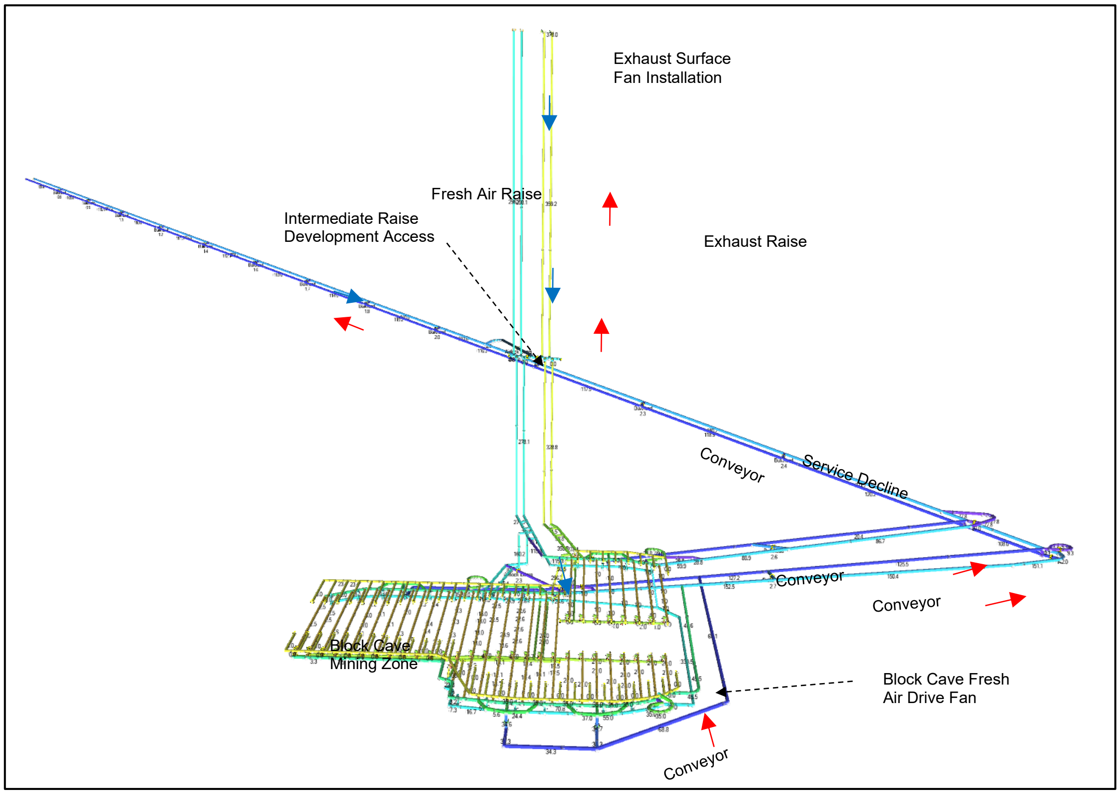
Table 16-14: Mine voltages and distribution

Parameter	Value	Unit	Notes
Mine Voltage	13,800	V	
Underground Voltage	380 V	V	Peru/Chile
Large Motor Voltage	4,160	V	All conveyor/crusher motors will be fed with this voltage
Main Ramp Inter-Substation Distance	600	m	Due to the ramp development configuration
Power Requirements for Development Substations	1	MVA	1- Bolter – 80kW 2- Jumbo – 205kW 3- Fans 4 x 140 kW 4- Pumps 2 x 40 kW 5- Pumping Permanent – 100 kW
Inter-Substation Distance	500	m	This is an average inter-substation distance in most mines and facilitates the distribution. Where belts are required, a substation will be required due to the motor size requirements.

The substations would be supplied from the surface substation by high voltage cables suspended from the roof of the main drives. The mine layout only permits the installation of the redundancy power line in the conveying gallery. The high voltage cables would be installed in cable trays whereas, the low voltage cables (380 V) would be suspended from anchors to roof by a guy wire with insulated supports. Armouring requirements would follow National Standards.

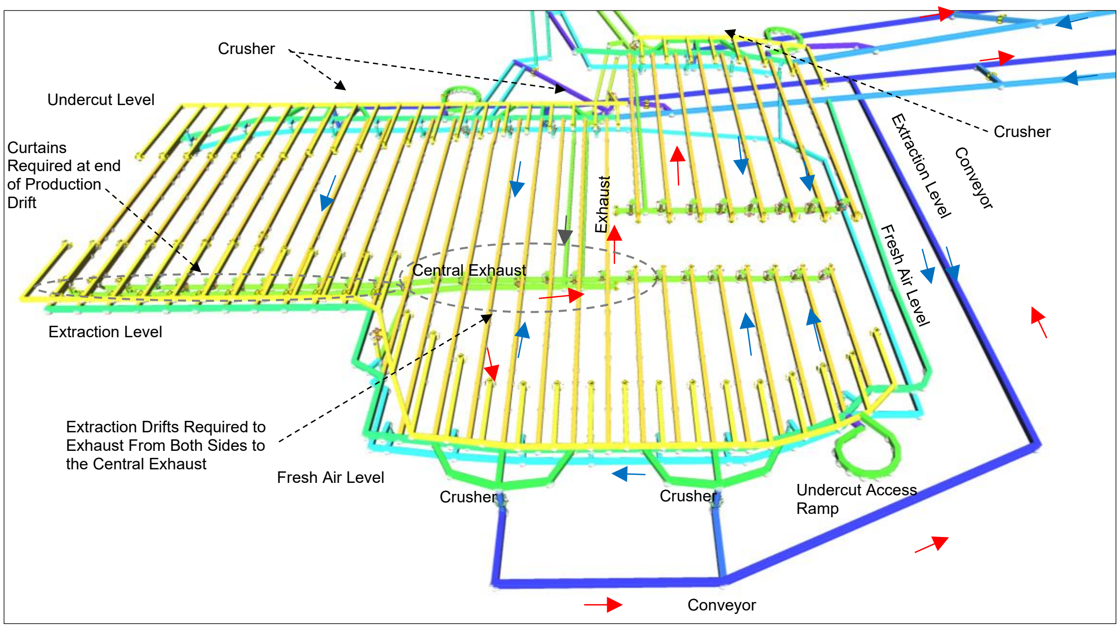
16.6.3 Ventilation System Layout

The mine design incorporates two exhaust raises to surface, two fresh air raises to surface, and twin declines (one exhausting conveyor and one providing access). The fresh air and exhaust air raises are split so that they can be developed in two lifts which will reduce the raise bore development length (drill string). An isometric view of the mine is provided in Figure 16-13, with a footprint view shown in Figure 16-14.



Source: This study, 2024

Figure 16-13: Layout projections (isometric view)



Source: This study, 2024

Figure 16-14: Footprint ventilation layout

Required Airflow for Mining Criteria Establishment

Several factors must be considered when determining the airflow requirements for the mine such as gas dilution, diesel particulates, heat, maintaining minimum air velocities, and meeting government regulations. These factors need to be applied to target areas to determine the actual total mine airflow requirement. Any fixed facilities underground (e.g., fuel and lubricant storage) will also demand dedicated airflow splits.

Gases

Harmful strata gases are not expected to be encountered at this site. The configuration of the system as an exhausting ventilation system minimises the blast clearance time/possibility of exposure to blast-generated gases by maintaining the ramp clear of blasting fumes. Each level will have access to an exhaust connection point and a fresh air connection point which will provide a limited compartmentalisation of the ventilation system. No specific airflow requirements were established based on this criterion, though these hazards factor into direction of airflow and general ventilation configuration.

Diesel Particulates

General best practices require a minimum factor of 0.06 m^3/s per kW of engine power (for modern diesel equipment supplied with 50 ppm diesel fuel) to ensure gaseous and aerosol contaminants from diesel equipment are sufficiently diluted which is a typical minimum design value for many ventilation systems. This is the recommended minimum airflow to ensure sufficient dilution of contaminants with new equipment. If used equipment is purchased, or diesel equipment is poorly-maintained this value may be insufficient.

Ventilation Raises

Two types of ventilation designs were used in the developing the underground ventilation system. Raise bored raises were used for all long raise segments when access exists to both top and bottom of the raise. Additional short raises were modeled within the cave footprint for both the central exhaust and perimeter fresh air connections.

Table 16-15: Friction factors

Raise Type	Method	Dimension (m)	Friction Factor (Ns^2/m^4)
Surface Ventilation Raise	Raise Bore	5.0	0.005
Internal Small Exhaust Raise	Raise Bore	3.0	0.005
Duct		1.4	0.006
Perimeter Ventilation Raise	Blast Hole	3 × 3	0.011

Horizontal Airways

Horizontal airways in the ventilation system were designed based off the Deswik output mine designs which were imported into VentSIM™ ventilation modeling software. Modeled airway dimensions (Table 16-16) were generally based off the dimensions of the airways as exported from Deswik. Levels and

Ramps are excavated at 6 meters wide by 6 meters high limited arch airways. Where needed fan cutouts will be used for auxiliary fan locations which will create enough room for housing fans and silencers in the footwall drift and decline.

Table 16-16: General airway dimensions

Airway Type	Dimension (m)	Friction Factor (Ns^2/m^4)
Primary Main Access Decline	6 x 6	0.012 (developed with long straight stretches)
Secondary Conveyor Decline	6 x 6	0.012 (developed with long straight stretches)
Ramp	5.5 × 5.5	0.012 (tight spiral)
Undercut	4.5 × 4.5	0.012 (assumed clutter)
Extraction	4.5 × 4.5	0.012 (assumed free of clutter)
Fresh Air Level	5 × 5.5	0.012 (assumed clutter)
Exhaust Level	5 × 5.5	0.012 (shock losses will be individually added, also control infrastructure will be added separately)

<u>Shock Losses</u>

Anytime airways converge, diverge, or change directions, the airflow will experience additional resistance from increased turbulence. All airway segments along fresh air and exhaust air raises have shock losses modeled (all intersections and transition).

<u>Air Velocities</u>

Air velocity limitations vary according to airway type. In areas such as return airways and shafts where personnel are not expected to work, higher velocities are acceptable. Airway velocities typically used by SRK-US for various airway types are shown in Table 16-17. Air velocity limits and recommended values for travelways are established to accommodate work and travel by people and equipment, optimising dust entrainment and temperature regulation.

Table 16-17: Recommended maximum air velocities for various airway types (design)

Airway Type	Maximum Air Velocity (m/s)
Travelways	7
Conveyor	6
Primary dedicated ventilation intake and exhaust accesses	8-10
Primary ventilation shaft	20
Ventilation shaft with conveyance or escape	10
Minimum air velocity	0.3

Low airflow volumes may insufficiently dilute/remove airborne dust, but high air velocities will entrain larger dust particles, resulting in a potentially hazardous environment for personnel. An air velocity between 1.5 m/s and 2.5 m/s should be maintained to minimise dust concentrations in areas effected by dust generation (Vutukuti and Lama). Air velocities in this range represent the provision of sufficient airflow to dilute the dust, without excessive air velocity to re-entrain dust.

In general, the minimum air velocity in a heading (without diesel equipment in operation) is based on the perceptible movement of airflow which, based on best practice, is between 0.3 m/s and 0.5 m/s. The higher value of 0.5 m/s is used in areas with possible diesel equipment operation to both ensure compliance and air mixing, the value of 0.3 m/s is used as a minimum air velocity for areas with only electrical equipment.

Air velocities in long upcasting shafts should be maintained outside of the range of 7 m/s to 12 m/s to avoid water blanketing. Variability of the number of equipment and mining locations throughout the mine life makes this hard to plan for in advance by manipulating the size of raises. A solution to the problem may be to slightly increase or decrease flow in problematic shafts. This may require some shifting of mining activities.

Heat

Detailed rock and water temperature data was not available for the proposed mining zone. However, when this data becomes available it will be incorporated into the ventilation design.

Heat produced by equipment (diesel or electric) may not dissipate quickly in areas of minimal velocity, and could result in high air temperatures which could pose a hazard to workers.

Specific Area Ventilation Requirements

The basic ventilation model was developed with the following general area ventilation requirements.

Main Decline Ventilation

The development of the twin declines (access and conveyor) will require both auxiliary ventilation systems installed at the development faces, and a fan installation in one of the portals. This will allow for length of the duct systems to be minimised, and the circulation through the declines to be maximised. The airflow quantity through the declines (200 m^3/s) is based on an air velocity criteria, and the airflow supplied by the face auxiliary systems (50 m^3/s) is based on the operation of a truck and LHD in each heading along with the general duct leakage.

Undercut Ventilation

The Undercut Level will be developed off of the North and South fringe drives requiring an auxiliary ventilation system to be installed in each cave heading. Approximately 35 m^3/s will be applied to each of the Undercut Level perimeter drift to allow for the operation of both one LHD and associated auxiliary equipment.

Extraction Level Ventilation

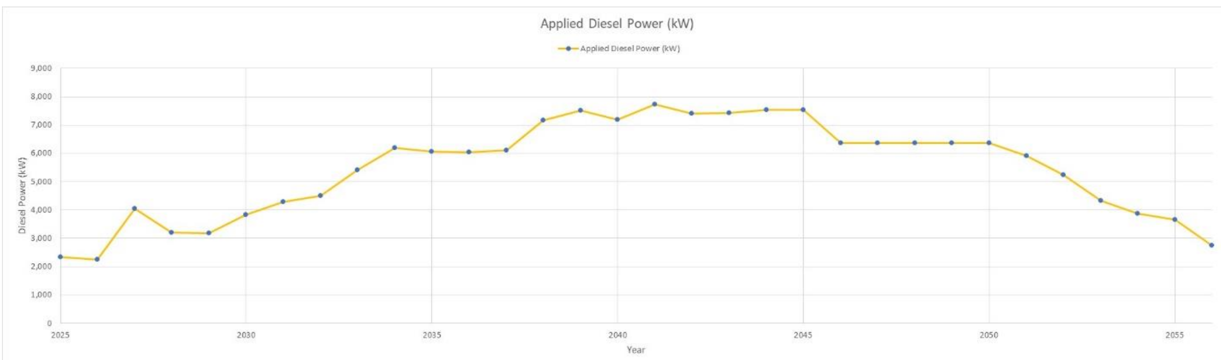
Each extraction drive segment will be required to have approximately 21 m^3/s exhausting it. The majority of the extraction drives will be ventilated in one direction, the drives at the center of the footprint will be split in half with both sides ventilated from the perimeter to the central exhaust. Each side will be separately ventilated, if airflow is not required on one side then a curtain can be used to limit flow. A booster fan will be required to be installed in the fresh air circuit to slightly pressurise the back side of the footprint so that the fresh air from the perimeter drives can be applied more evenly.

Crusher Ventilation

Each crusher will be required to be ventilated with approximately 35 m^3/s. In order to draw the airflow into the conveyor drive a small booster fan will be required to be installed at each crusher to both capture the dust from the LHD dumping and provide cooling for the crusher.

Airflow Calculations and Equipment

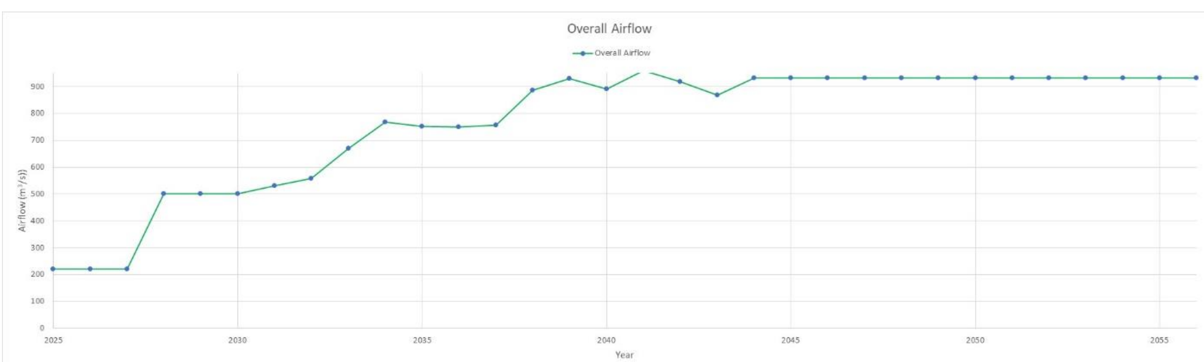
SRK engineers developed an equipment schedule from which the overall applied diesel power for the mine could be estimated. The applied diesel equipment load power used in the airflow calculation is shown in Figure 16-15.



Source: This study, 2024

Figure 16-15: Applied diesel power

The overall airflow requirement utilises the diesel dilution airflow as a base value then adds additional airflow for leakage, point of use applications, and development areas. The overall airflow requirement staged during the life of mine is shown in Figure 16-16.

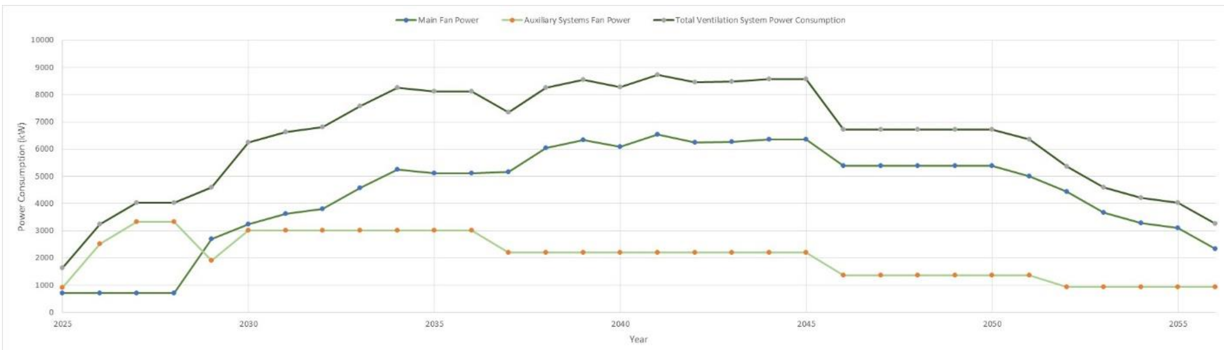


Source: This study, 2024

Figure 16-16: General airflow requirement based on diesel dilution, leakage, and factors

In addition to the actual headings with active draw, additional headings will be required to be ventilated to provide in-shift flexibility and simultaneous development of the block cave footprint. A ventilation-on-demand type system could be used to reduce the number of parallel "flexibility" headings resulting in a reduced airflow requirement. However, changes made to the ventilation of the extraction level will be difficult to re-balance.

The ventilation model was developed with the VentSim™ software package to estimate the overall fan power for the decline development, and for a life of time frame as shown in Figure 16-17. The fan power was then calculated for the intermediate time phases by modulating it against the applied diesel power. The auxiliary ventilation fan power was determined by the placement of assumed operating fan locations.



Source: This study, 2024

Figure 16-17: Ventilation power requirements (kW)

The required fan operating characteristics were developed based on the life of mine time frame during which both production and development activities are taking place. These fan operating points are identified in Table 16-18.

Table 16-18: Ventilation system fan requirements

Fan List	Airflow (m^3/s)	Pressure (kW)	Installation Losses (20%) (kW)	Fan Total Pressure (kW)	Fan Power (85% Efficiency) (kW)
Portal Development Fan System	220	2.3	0.5	2.7	710
Crusher Fan	35	2.2	0.4	2.6	110
Footprint Fresh Air Fan (2 Parallel)	332	1.0	0.2	1.2	470
Main Exhaust Fans (2 Parallel)	380	4.9	1.0	5.9	2630
Long Development Auxiliary System (Twin 1.4m Duct, Twin Fans)	50	6.8	n/a	6.8	400
Short Development Auxiliary System (Single 1.4m Duct)	50	3.9	n/a	3.9	229
Short Auxiliary Heading Fan System	n/a	n/a	n/a		50
Extraction/Undercut Development Auxiliary Fan System (1.2m Duct)	25	2.9	n/a	2.9	85

16.6.4 Piping

The mine layout only permits the installation of the piping in the main access ramp. All critical piping services, except the conveyor sprinkler line be passed in the main access gallery. The pipes would be hung from the back of the main access drive with anchors and chain.

The piping specifications have been determined for the development and permanent infrastructure.

Five types of piping services have been identified:

- Compressed air – mainly used for blast hole loading holes, air pumps, mechanical tools, shotcreting, emergency air for refuges. The major use will be during the development of the ramp and levels. It is assumed that the compressor station will be located at the portal. The sizing of the piping has been determined based on a mine of similar size with modern hydraulic equipment and shotcrete requirements. For design purposes the compressor station should be rated for 84 m^3/min. The operating line pressure would be 6 bars. The compressor requirement will vary significantly depending on the equipment models selected. The largest air consumer would for shotcrete use.
- Drill water – this water is used mainly in the development of the tunnels, dust control and washing. The source of the water is assumed to be a water supply at the portal entrance. The water is assumed to be non-potable. Clarified water from the main pumping system can be utilised for this purpose. The main water use will be for dust control and drilling. Due to the vertical height mine, pressure reducing stations will be required to ensure that the pressure in the water line is between 3-6 bars of pressure. The size of the main drill water line was selected to compensate for the line losses dues to friction. Dust control at the drawpoints will be a significant water user resulting in low pressures at the drills.
- Dewatering – one dewatering 14-inch diameter pipeline is required until year 12 of the mining operation. Thereafter, the flows are expected to increase rapidly as the caving impacts on surface and the second 14-inch line would be installed in the main decline.
- Sprinkler system conveyors – this installation is required in the locations where conveyors are installed to reduce the risk of fire. NFPA 122 recommendations must be considered for the underground conveying system.
- Cement line – this will be used to convey the cement underground for preparing the shotcrete.

The piping specifications for the mine are summarised in Table 16-19.

Table 16-19: Piping - main lines

Function	Material	Nominal Size
Compressed Air	Steel Sch 40	6-inch (for refuge and other utilities
Drill water	HDPE	6-inch SDR 9
Dewatering	Steel	2 x 14-inch Sch 80 Steel
Sprinkler System Conveyors	Steel	6-inch Sch 40 pipe
Cement line (dry conveying)	Stainless Steel	6-inch- SS

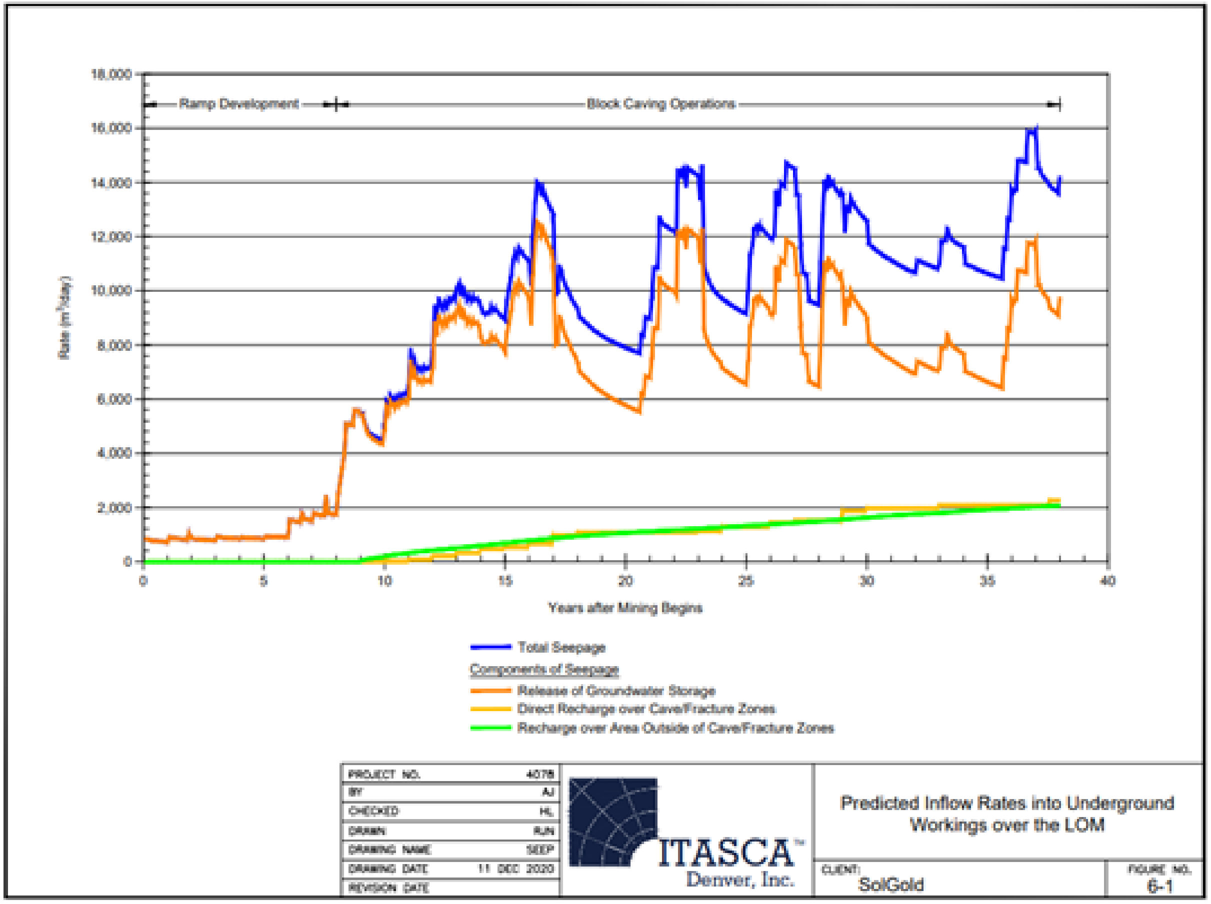
A fire sprinkler system will be required in the conveyor galleries. Additional sprinklers will be required at the head end, take-ups, and tail end. Heat detection sensors should be installed at the large pulleys where high friction could cause high heat due to bearing failure or belt stalling. Pressure reduction stations will be required for the sprinkler system as well as transfer tanks. This system will have to be developed in the feasibility stage of the Project.

The shotcrete requirements were not defined but it is proposed that a dry cement line be installed to send the cement to a silo underground. The aggregate would be sent to a stockpile underground by truck. The mix would be done dry underground and loaded to the supply trucks.

16.6.5 Dewatering

The main dewatering system will consist of sumps, pumping stations and pipelines. Due to the mine configuration, the water will be stage pumped up the main access ramp to surface. The pipes will be hung from the back of the galleries with chains. Steel pipe has been selected to handle the required pumping pressures. HDPE lines were not found to be adequate for this application due to the operating pressures and wall thicknesses.

The predicted inflow rates vs time is illustrated in Figure 16-18. The maximum inflow rate for the ramp development is around 2,000 m^3/day (23.15 l/s). The water inflow model was prepared by Itasca in 2020.



Source: ITASCA 2020

Figure 16-18: Predicted water inflows

During ramp development phase there only small tunnel voids exists, dewatering system will be designed to be able to handle ~30 L/s (108 m^3/h – 814 USGPM). However, in most pump stations the installation of additional pumps will be phased in as additional pumping is required.

When production starts and the cave begins propagating, pumping needs to be increased to 120 L/s (228 m^3/h – 3,613 USGPM. The last 25 years of production have the greatest groundwater inflow as it is assumed that the cave propagation has broken through to surface, as such and pumping needs to be able to handle ~185 L/s (666 m^3/h – 2,932 USGPM).

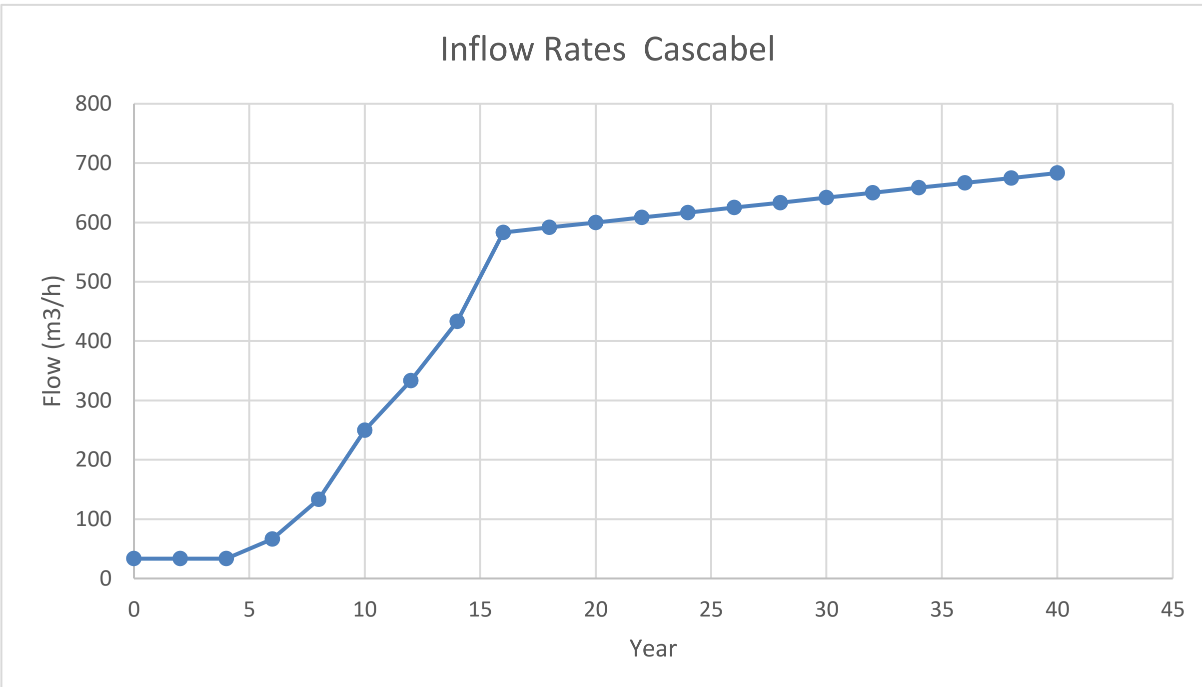
These pumping requirements assume that surface water (rainwater run off) is recharging groundwater aquifers and is not directly contributing to the mine's water inflow. Hydrological conditions are not considered to materially affect mining development rates or pumping costs relative to other block caving operations (reference Itasca report).

The Project will start with a 14-inch Sch 90 steel pipe and by Year 12, will require the second line. Steel could also be used but preference would be given to HDPE for the lifespan.

The dewatering system will be composed of following components:

- Sumps
- Clarification System
- Pumps
- Pipelines

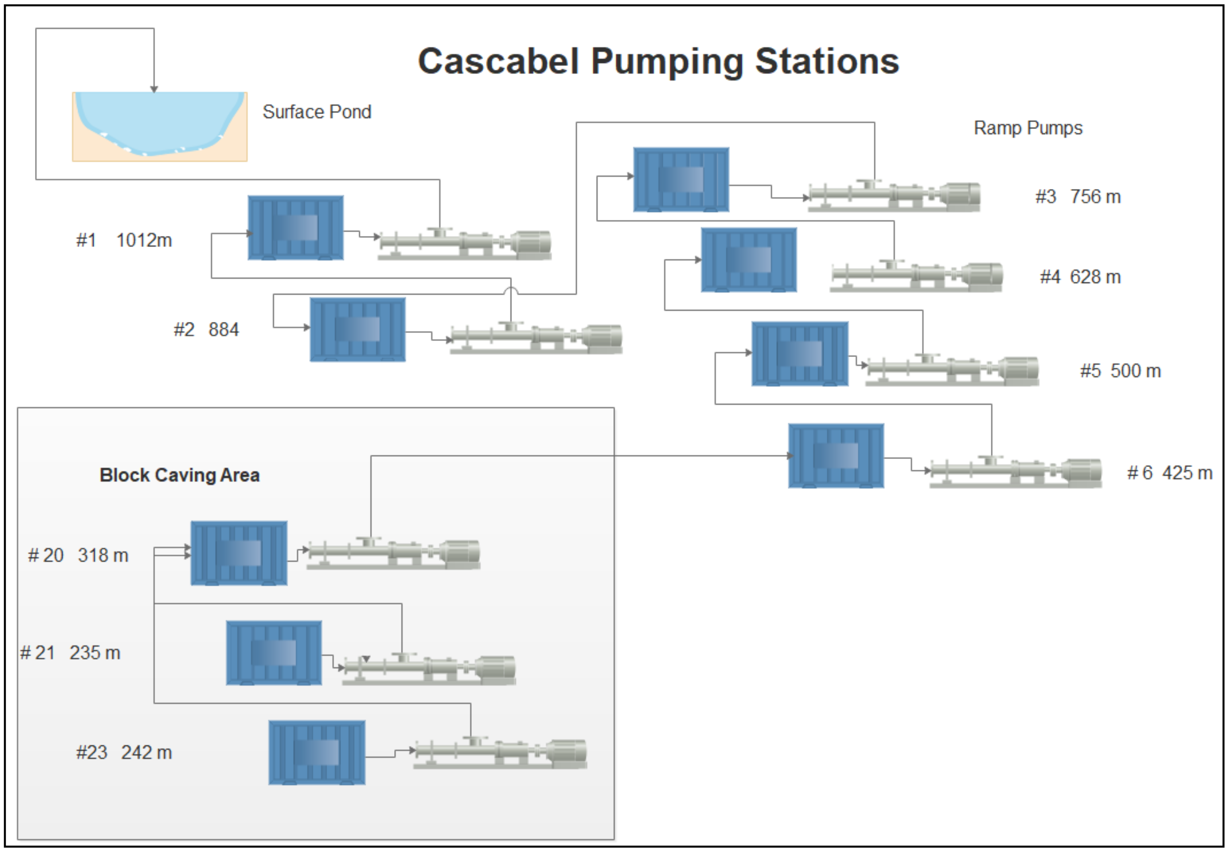
The water inflow model presented in Figure 16-18 was simplified and illustrated in Figure 16-19. This figure was used to determine the installed capacity requirements over the course of the mine life.



Source: Allnorth 2023

Figure 16-19: Underground pumping installed capacity requirement

The conceptual layout of the pumping system is illustrated in Figure 16-20.



Source: Allnorth 2024

Figure 16-20: Conceptual pumping layout

The layout of the mine requires the installation of a total of 12 main pumping stations. The average vertical distance between the pumping stages in the main access ramp is 128 m. The pumps and sump locations were matched with the conveyor drive stations to centralise the substations and reduce infrastructure.

The number of pumps will vary depending on pumping distance and Total Dynamic Head. A total of 27 pumps are required for the initial 8 years. As the surface impact of the cave reaches surface and expands, the second phase of pumping will be required. This will include the installation of the second pipeline. A total of 56 pumps are required at the end of the mine life to maintain the water level in the mine.

16.7 Underground Life-of-Mine Schedule

16.7.1 Mine Development Strategy

The production sequence is designed and optimised to balance the requirements of metal production, personnel resources, and equipment resources.

Consideration was given to the overall mining rate required, the geotechnical conditions likely to be encountered, the ventilation and logistics requirements, and general practice and equipment availability.

The mine development schedule and production plan were initially developed on an unconstrained basis using only expected unit advance and production rates. This was then resource-levelled to produce a practical plan that required a consistent availability of personnel and equipment.

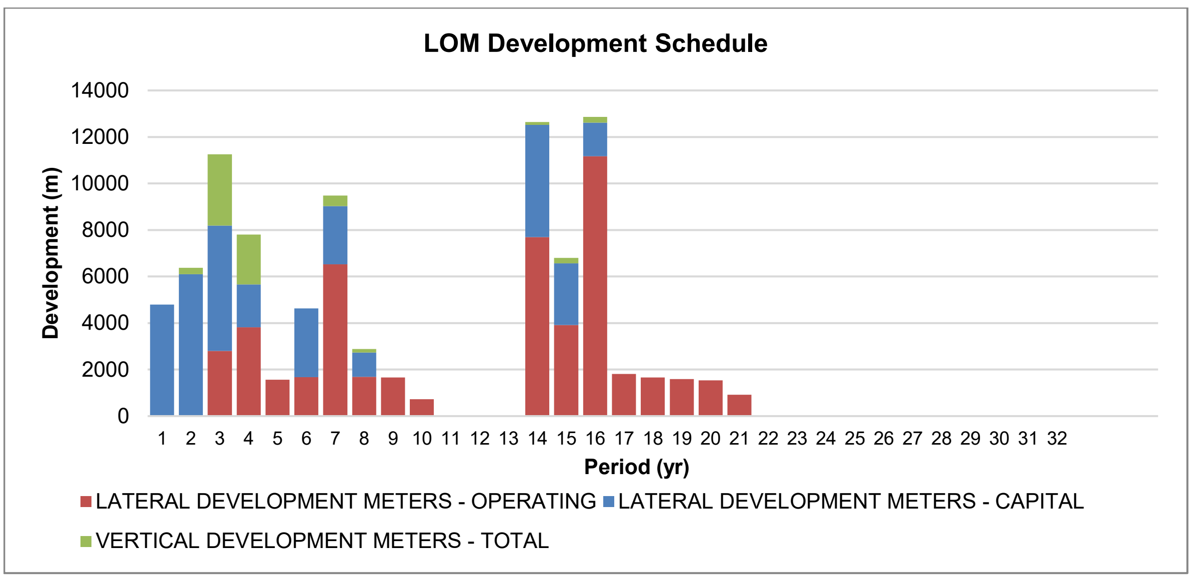
16.7.2 Development Constraints

The completion of the conveyor and access declines as far as the BC_01 footwall drive allows for the start of the development of the footprint and undercut levels. From the footwall drive, the priority development is completing the surface ventilation raises and the decline to complete the crusher chamber and material handling system.

The installation of the primary ventilation fans is a critical path in the initial development schedule as it is required to increase the number of active development headings. This is the priority of the development on the upper levels of the mine.

16.7.3 Development Schedule

Figure 16-21 shows the lateral, operating and vertical development schedule over the LOM. Operating development has been defined as the cross-cuts and slot drives development.



Source: This study, 2024

Figure 16-21: LOM development schedule

The total LOM development quantities for lateral development, operating development and vertical development by production area are shown in Table 16-20.

Table 16-20: LOM development summary (meters)

Description	Mine Access	BC_01	BC_02	BC_03	BC_04	Total
LATERAL - OPERATING						
Block Cave - Extraction Drive		2,962	3,025	3,428	4,710	14,125
Block Cave - Drawpoint		3,891	4,281	5,257	6,016	19,445
Block Cave Undercut Drive		3,284	3,378	4,276	6,234	17,172
LATERAL - CAPITAL						
Decline	8,770					8,770
Incline		189		188	195	572
Conveyor	7,857	22	87	1,367	379	9,711
Access Drive	511	361	416	41	41	1,370
Main Footwall Drive		266	261	1,383	821	2,731
Fresh Air Drive	349	971	867	1,808	564	4,559
Return Air Drive	952	1,491	688	364	892	4,387
Haulage Drive		228	225	435	224	1,112
VERTICAL DEVELOPMENT						
Fresh Air Raise	2,803	71	71	160	107	3,211
Return Air Raise	2,851	187	129	133	193	3,492

16.7.4 Production Schedule

Ramp-Up Strategy

The ramp-up of the block cave areas is dependent on completing the block cave crushers and starting the undercut and drawbell development. It was assumed that the block cave undercut advances at a rate of 5.5 m/day and a maximum of five drawbells are developed per month in each area.

Mining Area Sequence

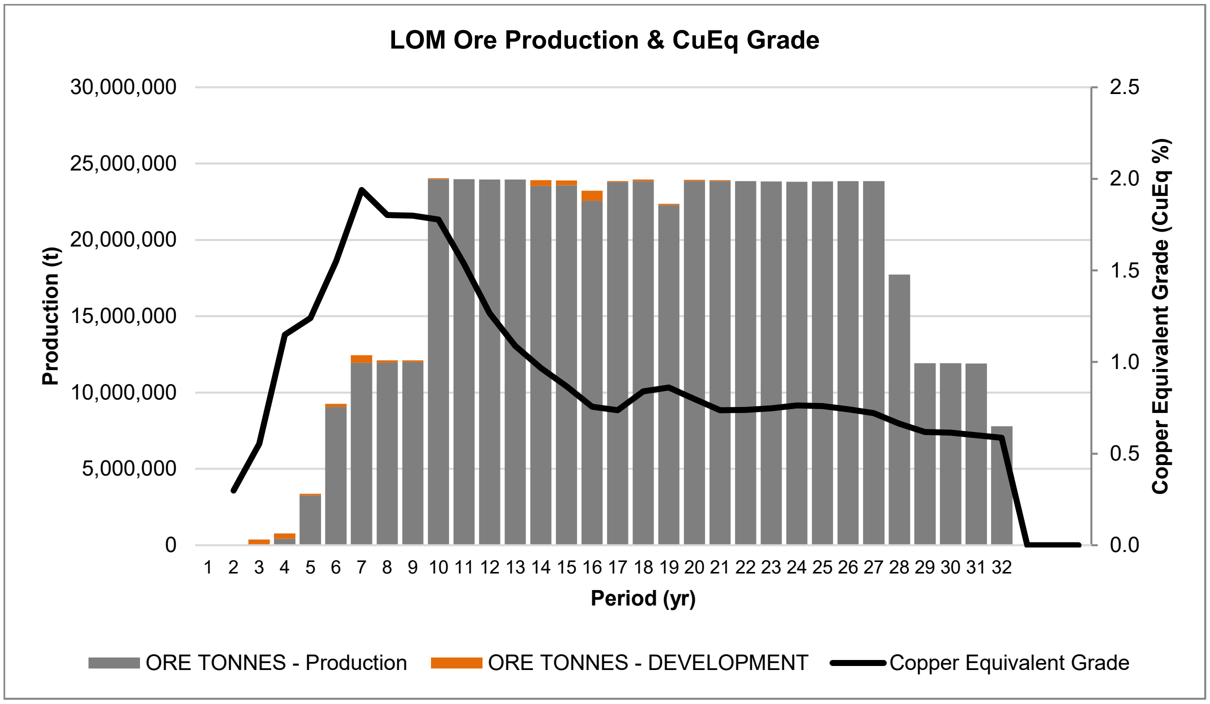
Mining areas were sequenced based on mining practicality and economic value. The sequence is listed below, and Figure 16-23 shows the production tonnes by mining area of the life of the mine.

1. BC_01 is the first production area (first ore starts in Year 4 from start of access development).
2. BC_02 is the next area to ramp up (Year 8) to fill the throughput requirement after the first mill expansion (Year 10).
3. BC_03 (Year 15) and BC_04 (Year 18) ramp up as BC_01 and BC_02 ramp down.

LOM Production Schedule

A total of 539.7 Mt of ore is mined from the production shapes and development with an average grade of 0.60% Cu and 0.54 g/t Au. Pre-production development mining is expected to last four years, followed by 28 years of ore production in the presented designed mine.

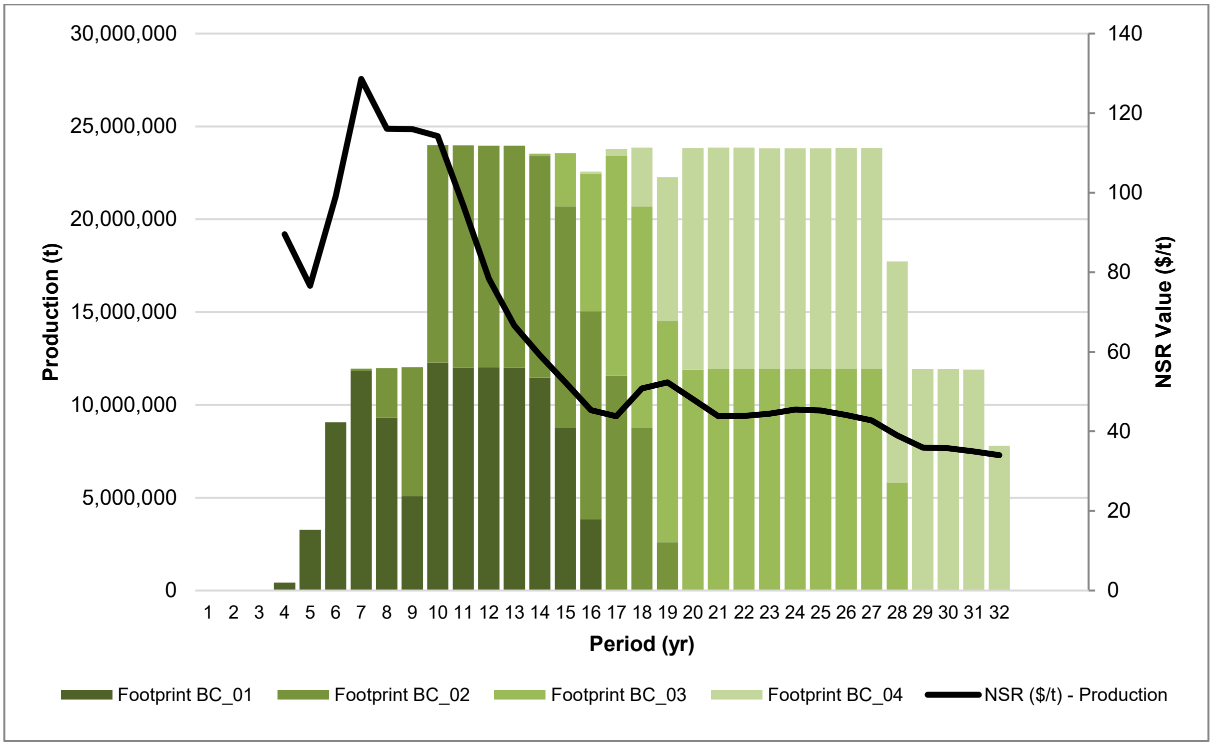
Figure 16-22 shows the LOM production and copper equivalent grade.



Source: This study, 2024

Figure 16-22: LOM ore production and copper equivalent grade

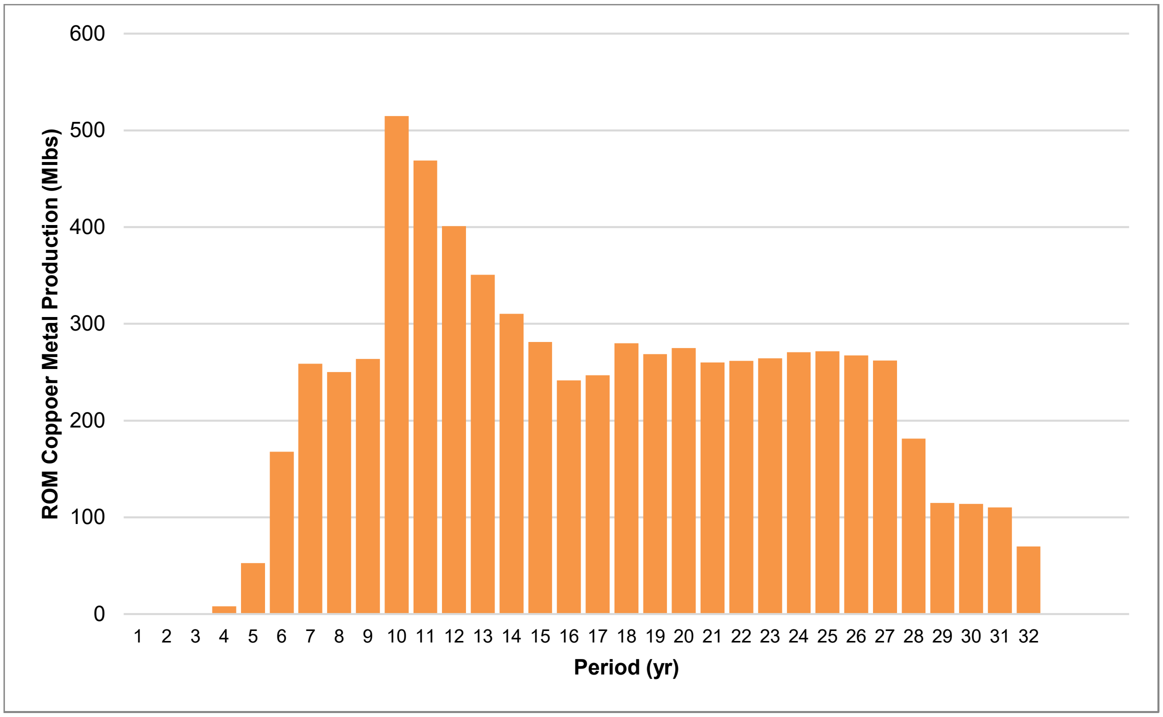
Figure 16-23 shows the LOM production schedule by mining area.



Source: This study, 2024

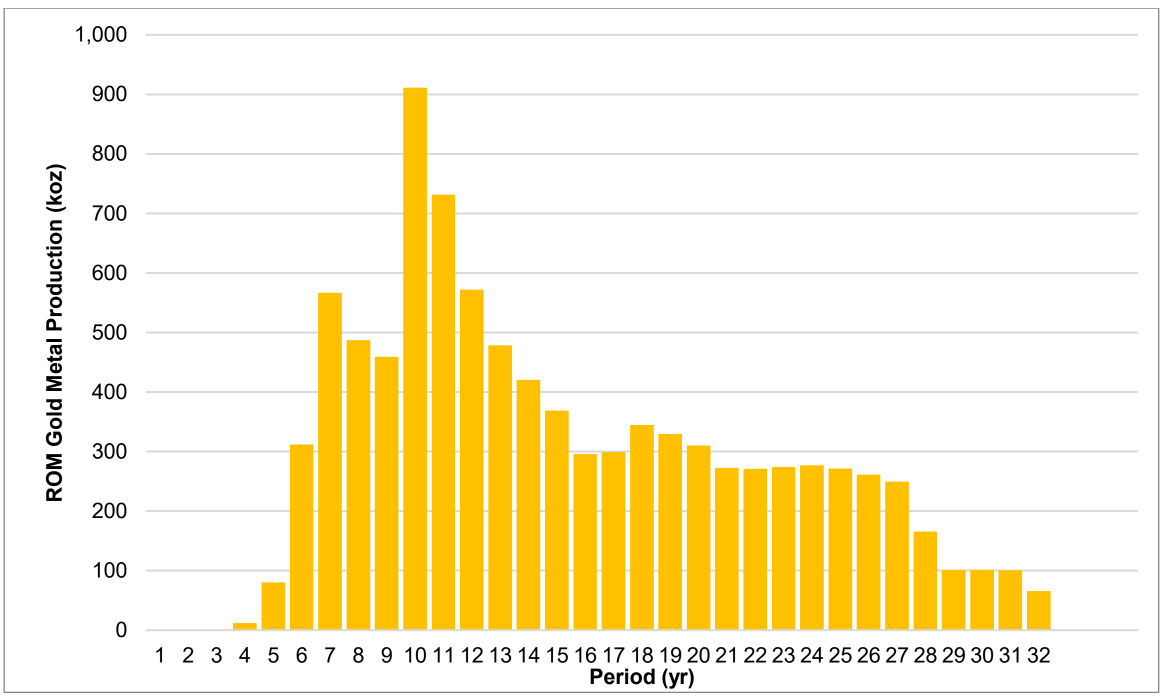
Figure 16-23: Underground mine production by mining area

Over the course of the mine life, 7,121 Mlbs of copper and 9,426 koz of gold are mined from the production shapes and sent to the processing plant. Figure 16-24 shows copper metal production for the LOM, while Figure 16-25 shows gold metal production.



Source: This study, 2024

Figure 16-24: Copper metal from underground production



Source: This study, 2024

Figure 16-25: Gold metal from underground production

16.8 Underground Equipment Selection

16.8.1 Fleet Sizing

The criteria used in the selection of underground mining equipment include:

- Mining method and preliminary development plan
- Mineral deposit geometry and dimensions
- Mine production rate
- Reliability and availability of after-sales support

The size of the equipment fleet was based on the scheduled quantities of work, estimated from first principles cycle times and productivities, benchmarking and practical experience. The selected equipment size satisfies the maximum size of excavations based on geotechnical recommendations.

The following input factors were considered to calculate the required number of operating units:

- Development and production schedule.
- Shift efficiency of 81% for 8-hour shifts to account for non-productive time due to shift change, equipment inspection and fueling, lunch and coffee breaks, equipment parking and reporting.
- An operational hour efficiency of 83%, accounting for 50 minutes of usable time in one operating hour for manual tasks.
- Equipment operation efficiency (e.g., 75% efficiency for the second boom on the drill jumbo; 85% and 80% fill factors for LHD bucket during pre-production and production, respectively; 90% fill factor for the truck box).
- Additional time for travel, setup, and relocation.

The estimated number of operating units was converted to a fleet size by accounting for equipment mechanical availability of 80% to 90%, depending on the type of equipment.

Development Equipment

During the initial stage of the mine development, 14t LHDs and 51t haulage trucks will be used to transport blasted development material out of the mine. Once the materials handling system is in place, material generated during production-level development will be re-handled through the ore pass system by the production LHDs.

The development crew also includes two boom jumbos, emulsion loaders, and ground support equipment (rockbolter, cablebolter, scaler).

16.8.2 Underground Mobile Equipment

The mobile equipment is typical of that used in underground mines: production, development, and services.

The criteria used in the selection of underground mining equipment include:

- Mining method and preliminary development plan
- Mineral deposit geometry and dimensions
- Mine production rate
- Reliability and availability of after-sales support

The size of the equipment fleet was based on the scheduled quantities of work, estimated from first principle cycle times and productivities, benchmarking and practical experience. The selected equipment size satisfies the maximum size of excavations based on geotechnical recommendations.

The following input factors were considered to calculate the required number of operating units:

- Development and production schedule.
- Shift efficiency of 83% on 12-hour shift length to account for non-productive time due to shift change, equipment inspection and fueling, lunch and coffee breaks, equipment parking and reporting.
- An operational hour efficiency of 83%, accounting for 50 minutes of usable time in one operating hour, for manual operation and 92% efficiency, accounting for 55 minutes of usable time in one operating hour, for autonomous operation.
- Equipment operation efficiency: 75% efficiency for the second boom on the drill jumbo, 90% fill factor for LHD bucket, and 90% fill factor for the truck box.
- Additional time for travel, setup, and teardown.

The estimated number of operating units was converted to a fleet size by accounting for equipment mechanical availability of 80% to 90%, depending on the type of equipment.

During the pre-production period, a mining contractor will use its mobile equipment fleet for mine development, haulage, and services. Equipment for conveying, crushing and ore/waste pass systems will be in place when the owner commences operations with its own crews and equipment.

The owner's mobile equipment fleet for the Cascabel project is completely rubber-tired and diesel-powered. The mobile equipment schedule was prepared on an annual basis. The required underground mobile equipment fleet will vary over the life of the Project. The underground mine mobile equipment list and maximum units in the equipment fleet are presented in Table 16-21..

Table 16-21: Underground mine mobile equipment

Description	Model	Make	Peak
Development and Production Equipment			
Jumbo, 2 boom	DD422i	Sandvik	3
Emulsion Loader, Development	EC-3	MacLean	2
Scaler	Scamec	Normet	2
Rockbolter	DS412i	Sandvik	3
Cablebolter	DS422i	Sandvik	2
Shotcrete Sprayer	SS5	MacLean	2
Production Drill	DL4222i	Sandvik	3
ITH Drill with V30 Reamer	DU 412i	Sandvik	1
Emulsion Loader, Production	MC5	MacLean	2
Water Cannon	WC3	MacLean	5
Blockholer	BH3	MacLean	6
Mobile Rockbreaker	RB3	MacLean	9
LHD, 8.6 m3, 17 t	LH517i	Sandvik	2
LHD, 10.7 m3, 21 t	LH621i	Sandvik	22
Haulage Truck, 51 t	TH551i	Sandvik	4
Auxiliary Equipment			
Grader	GR5	MacLean	2
Scissor Lift	SL3	MacLean	4
Scissor Lift Attachments	SL3A	MacLean	1
Boom Truck	BT3	MacLean	2
Flat Deck Truck	DT3	MacLean	2
Fuel/Lube Truck	FL3	MacLean	2
Explosives Truck			1
Transmixer	TM3	MacLean	6
Forklift/Telehandler	MHT	Manitou	2
Water Sprayer	WS3	MacLean	2
Roller	Roller		1
Backhoe	310 P-TIER	John Deere	1
Bobcat	Bobcat	Bobcat	1
Mechanics Truck	Toyota Hurth	Miller	3
Personnel Carrier MacLean	PC3	MacLean	2
Personnel Carrier Toyota	Toyota Van	Miller	3
Supervisor/Engineering Vehicle	Toyota	Miller	15
Cassettes System Prime Mover	CS-3	MacLean	2
Cassettes Attachments	CA-3	MacLean	1

17 Recovery Methods

17.1 Introduction

The process design basis ()Table 17-1 has been derived from the metallurgical test work results provided by SolGold. The Cascabel project will be developed in two phases, with each phase utilising a dedicated processing line suitable for a production rate of 12 Mtpa, recovering copper and gold using a conventional flotation process.

Primary crushing will take place underground, and crushed ore will be conveyed to the surface, where a radial stacker will create a separate crushed ore stockpile to feed each processing line. Reclaimed ore from the stockpiles is ground in a SAG Mill – Ball Mill – Crusher (SABC) configuration with 80th percentile data used to size the mills by Sedgman and verified by vendors.

The flotation circuit was sized based on metallurgical testwork, flotation kinetic curves and standard scale-up parameters. Following rougher flotation, rougher concentrate is reground prior to multi-stage cleaner flotation. Rougher and cleaner flotation tailings are thickened individually for separate deposition into the tailings storage facility. The final concentrate is thickened and filtered. The design of the flotation circuit production schedule has been undertaken utilising the recovery and grade equations developed from the testwork completed and target a 22% Cu concentrate grade.

Table 17-1: Process design basis

Parameter	Unit	Value	
		Phase 1	Phase 2
Mine Life	Years	28	
Feed Grade LOM Average – Copper, Cu	%	0.60	
Feed Grade LOM Average – Gold, Au	g/t	0.54	
Ore Throughput (each process line)	MTPA	12	12
Ore Throughput (total)	MTPA	12	24
Process Plant Throughput (each process line)	t/hr	1490	1490
Process Plant Throughput (total)	t/hr	1490	2980
Mass yield to final concentrate	% plant feed	1.55 – 4.16% (LOM Average: 2.41%)	
Concentrate production (each process line)	t/d	680	680
Concentrate production (total)	t/d	680	1360
Concentrate Grade - Cu	%	22.0% Target	
Overall Recovery – Cu	%	85.3% - 92.4% (LOM Average: 88.4%)	
Concentrate Grade – Au	g/t	9.9 g/t – 28.4 g/t (LOM Average 15.9 g/t)	
Overall Recovery - Au	%	70.5% - 80.9% (LOM Average: 70.8%)	
Total Production – Copper, Cu	kt	2,859	
Total Production – Gold, Au	koz	6,876	

17.2 Process Flowsheet

The Cascabel project process flowsheet (Figure 17-1) is a conventional copper-gold flotation flowsheet.

Primary crushing will take place underground in jaw crushers to achieve a coarse ore top size of 370 mm. Crushing will occur at a rate of approximately 1,930 t/hr during Phase 1 and 3,860 t/hr during Phase 2, with both phases considering a 70% crusher availability.

Crushed ore will be delivered from underground to surface by a conveyor, which will transfer ore to an 85 m radial stacker. The stacker will create kidney-shaped stockpiles, and each stockpile will have a single reclaim tunnel with three feeders and an escape route. Each stockpile will hold approximately 18,000 tonnes, equal to approximately 12 hours of live volume. A radial stacker was selected to accommodate the additional processing line in the planned expansion.

For each process line, reclaimed ore from the coarse ore stockpile is conveyed to a grinding circuit consisting of a SAG mill, pebble crusher and ball mill in a closed circuit with classifying cyclones (SABC). A front-end loader loads SAG mill grinding media from a bunker onto the SAG feed conveyor.

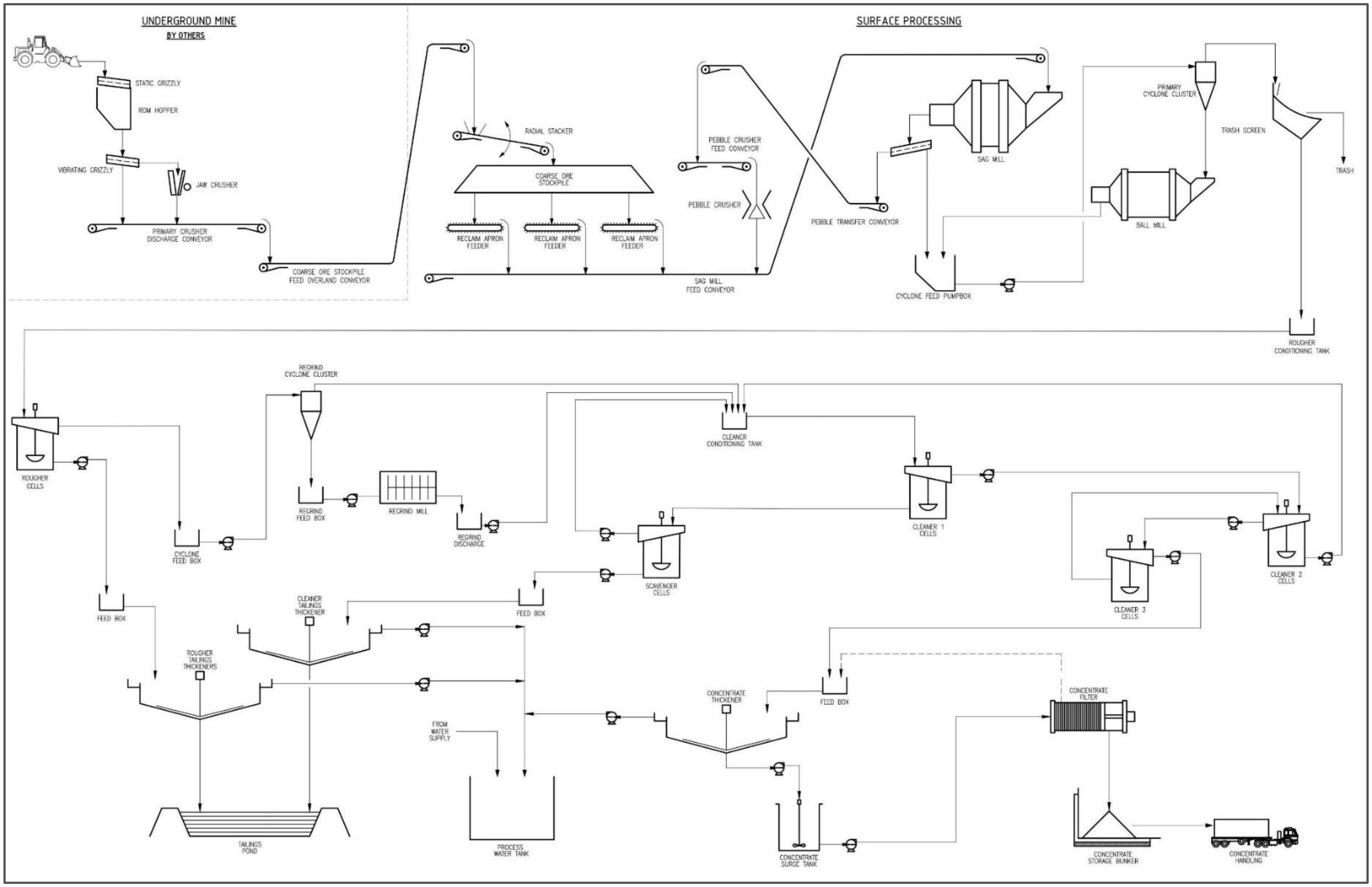
Process water will be added to the SAG mill to achieve a slurry density of approximately 72% solids by weight. The SAG mill discharge will pass over a vibrating screen with oversized material being transferred to the pebble crusher. The crushed pebbles are returned to the SAG feed conveyor. Screen undersize material is discharged into the cyclone feed pump box and pumped to the cyclone manifold. Cyclones will classify the slurry to achieve an overflow slurry stream containing 200-micron (P_{80}) particles at approximately 40% solids by weight. Cyclone underflow will be recirculated to the ball mill at approximately 74% solids by weight. The ball mill circulating load is nominally 250% of the new feed.

Ball mill discharge will flow through the discharge trunnion equipped with a magnet to remove any metal scats, which, when separated, will discharge to a concrete scats collection bunker. Afterward, the metal removal slurry will discharge into the cyclone feed pump box. The cyclone overflow will report to a trash screen, where the trash will be removed, and screen undersize will report to the flotation circuit.

The cyclone overflow will be directed to the flotation circuit, which consists of a rougher and cleaner circuit with scavenging of the cleaner tails. Conventional mechanical tank cells will be used in all parts of the flotation circuit. Rougher concentrate will be pumped to cyclones with underflow reporting to a regrind mill to regrind to P_{80} 25 µm. The cyclone overflow and regrind mill discharge will be upgraded through a multistage cleaner flotation circuit.

LOM average final cleaner concentrate is expected to have a copper grade of 22.0% Cu, with 88.4% of the contained copper recovered, and 70.8% of the gold recovered at a grade of 15.9 g/t Au. In the first ten years of the mine life, higher-grade ore will increase copper recoveries to 90.8% and gold recoveries to 76.9% with a grade of 21.4 g/t Au. Mass pull and recovery ranges have been calculated based on a fixed final copper concentrate grade of 22.0% Cu.

In the laboratory, copper concentrates assays revealed relatively low levels of deleterious elements such as arsenic, bismuth, cadmium, chlorine, selenium, and/or tellurium.



Source: Artica et al., 2022

Figure 17-1: Simplified process flowsheet

Process water will be used for slurry dilution and launder sprays. Flotation reagents used will include Potassium Amyl Xanthate (PAX) as a collector, W31 as the frother and lime for pH adjustment. The metallurgical testing considered Methyl Isobutyl Carbinol (MIBC) as the frother for the rougher stage but observed that W31, a stronger polyglycol frother, was required for the cleaner circuit likely due to high levels of xanthate affecting froth quality. For simplicity, W31 will also be utilised for the rougher circuit with no expected impact on performance.

The final concentrate will be pumped to concentrate thickeners ahead of filtration. Flotation tails will be pumped to tailings thickeners prior to discharge to the tailings storage facility.

The initial phase includes separate thickeners for rougher and cleaner flotation tailings. Each thickened tailings stream will be pumped to a collection tank to flow by gravity to the tailings storage facilities.

The rougher flotation tailings require an initial 55 m diameter high-rate thickener with a second duplicate thickener added for the Phase 2 expansion. The cleaner flotation tailings require a 40 m diameter high-rate thickener, which is sufficiently sized for both phases. Flocculant will be added at a rate of 22 g/t to rougher tailings thickener and 80 g/t to the cleaner tailings thickener. Reclaimed water from the thickener overflows will flow to the process water tank for reuse in the process facilities.

Final concentrate from the cleaner flotation cells will be pumped to a high-rate thickener. Flocculant will be added to the thickener feed at a rate of 21 g/t. The final thickened underflow is expected to be 55-60% solids by weight and will be pumped to a filter feed tank. Thickener overflow water will report to an overflow tank before being pumped to the process water tank.

In the first phase, the filter feed tank will provide 8 hours of capacity, which will be reduced to 4 hours following the expansion. This capacity is to allow filter maintenance to be conducted without affecting mill throughput, and the risk of production interruption decreases when additional filters are added as part of the expansion. The filter feed will be pumped to plate and frame pressure filters to produce a cake with approximately 11% moisture. The filter cake will drop into a bunker where a front-end loader will collect the concentrate and load it into a concentrate truck or into a covered storage area.

Both processing lines will share a 19 m high-rate concentrate thickener and filter feed tank. Each phase will have two 1.5 m x1.5 m plate and frame filter presses with a total concentrate production of 680 tpd during the first phase and 1,360 tpd following the expansion.

Process water is used for filter cloth washing and to flush the filter manifolds. Filtrate, cloth wash and manifold flushing water will report to the concentrate filtrate tank before being returned to the concentrate thickener. Dedicated filtrate separators remove excess air from the filtrate streams.

17.3 Process Design Criteria

The process design criteria was developed based on interpretations of the metallurgical test work results presented in Section 13. The process plant is expected to operate 24 hours per day and 365 days per year. The summary process design criteria in Table 17-2 provides the common values for each processing line. Concentrate and tailings thickening are shared, and design criteria information is presented in Table 17-3.

Table 17-2: Summary process design criteria

Description	Unit	Design Value
Ore throughput (nominal)	Mtpa	12
Overall availability		
Primary Crusher	%	70
Grinding / Flotation	%	92
Concentrate Filter	%	80
Ore specific gravity	-	2.8
Stockpile		
Live Volume (each)	t	18,000
Operating Time	hr	12.1
Primary Grinding		
Throughput	t / hr	1490
RWi – 80th Percentile	kWh/t	19.3
BWi – 80th Percentile	kWh/t	17.3
A x b – 20th Percentile		26.2
SAG Mill Feed Size F_{80}	mm	150
Product Size P_{80}	µm	200
SAG Mill		Φ11.5m x 6.7m EGL – 2 x 11.25 MW
Ball Mill		Φ7.9m x 12.4m EGL – 2 x 8.25 MW
Pebble Rate – Nominal / Design	%	15 / 30
Rougher Flotation		
Cell Type		Mechanical Tank Cell
Cell Number x Size		6 x 300 m^3
Residence Time	min	30
Stage Recovery – Mass	%	12.3% – 19.0% of fresh feed
Stage Recovery – Cu	%	85.2% – 97.2% (Average 92.4%)
Stage Recovery – Au	%	75.3% - 94.7% (Average 85.8%)
Regrind		
Feed Rate – Design	t / hr	223
Product Size P80	µm	25
Specific Grinding Energy	kWh/t	12.1
Cleaner Flotation		
Feed Rate – Design	t / hr	267
Cleaner Stages		3 Stages + Scavenger
Cell Type		Mechanical Tank Cells
Cell Number x Size		CL1 + 3 x 100 m^3 + Scav 2 x 30 m^3 CL2 3 x 30 m^3 CL3 3 x 20 m^3
Final Recovery – Mass	%	1.55 – 4.16

Description	Unit	Design Value
Final Recovery – Copper, Cu	%	85.3 – 92.4
Final Grade – Copper, Cu	%	22
Final Recovery – Gold, Au	%	63.6 – 80.9

Table 17-3: Dewatering process design criteria

Description	Unit	Design Value	
		Phase 1	Phase 2
Rougher Tailings Thickening			
Throughput - Nominal	t / hr	1266	2532
Number x Diameter		1 x 55 m	2x 55 m
Settling Rate	t / m^{2}h	0.6	
Underflow Density	%	50	
Cleaner Tailings Thickening			
Throughput - Nominal	t / hr	195	390
Number x Diameter		1 x 40 m shared	
Settling Rate	t / m^{2}h	0.4	
Concentrate Thickening			
Throughput - Nominal	t / hr	28.3	56.6
Number x Diameter		1 x 19 m shared	
Settling Rate	t / m^{2}h	0.25	
Underflow Density	%	60	
Concentrate Filtering			
Feed Tank Residence Time	hr	8	4
Filter Number		2	4
Filtration Rate	t / m^{2}h	0.275	
Filter Cycle Time	Min	20	
Filter Cake Moisture	%	11	

17.4 Reagents, Consumables and Utilities

17.4.1 Reagents and Consumables

The reagents will be prepared and distributed from the make-down systems allocated to each processing line. Reagents will be delivered to the required addition points by individual metering pumps. The design of these areas considers requirements such as section bunding, with dedicated sump pumps for individual reagents, segregated ventilation, and dust and fume control around reagents where potential for dust or fume release exists. The layout and general arrangement of the reagent area accounts for the separation between incompatible reagent types. Reagents include:

- Collector – Potassium Amyl Xanthate (PAX). PAX is a sulphide collector supplied in 900 kg bulk bags as a dry reagent. Water will be added to an agitated tank to produce a solution concentration of 20% w/v. The diluted mixture will be transferred to a distribution tank. Consumption will be approximately 275 tpa in Phase 1, increasing to 550 tpa in Phase 2.
- Frother – Polyfroth W31, glycol ether frother. W31 will be supplied in 1000 L intermediate bulk containers (IBCs). W31 will be delivered to the required flotation dosing points directly from the IBCs by dedicated metering pumps. Consumption will be approximately 1,190 tpa in Phase 1, increasing to 2,380 tpa in Phase 2.
- Flocculant – Anionic Polyacrylamide AN 923 SH or similar. Flocculant is added as a settling aid to each of the thickeners. Flocculent is diluted to 0.5% w/v and dosed at 0.05% w/v. Consumption of flocculent will be 392 tpa in Phase 1 and 784 tpa in Phase 2.
- Slaked lime – Calcium Hydroxide $Ca(OH)_2$. Lime is used for pH modification in the regrind mill prior to cleaner flotation. Lime will be delivered via bulk truck and stored in lime silos. Lime consumption will be 4,284 tpa during Phase 1 and 8,568 tpa during Phase 2.
- Grinding media and liners. The consumption rates for mill liners are expected to be 1.5 sets of liners per year for each SAG mill and regrind mill and one set per year for each ball mill. The grinding mills will require regular addition of balls to replace worn media and maintain grinding efficiency. The media consumption has been estimated based on abrasion test results and mill operating conditions. Media consumption rates are estimated as follows:
  - SAG Mill 6” Balls: Phase 1 media consumption is estimated to be 12,400 tpa, increasing to 24,800 tpa in Phase 2
  - Ball Mill 3.5” Balls: Phase 1 media usage is estimated to be 9,100 tpa, increasing to 18,200 tpa in Phase 2
  - Regrind Mill 4 mm Ceramic media: Phase 1 is estimated to be 157 tpa, and Phase 2 is estimated to be 314 tpa

17.4.2 Water

The facility will include facilities and equipment to store and distribute the following water services:

- Fresh water
- Reclaim water – thickener overflow
- Fire water
- Potable water (including safety shower/eye wash)
- Gland water
- Cooling water
- Process water
- Underground mine dewatering water (treated)

Fire water will be distributed throughout the process plant via a dedicated jockey pump and electric fire water pump. A diesel-powered fire water pump will also be installed as an emergency backup to the electric fire water system. Water is recycled inside the process where possible and treated as needed.

17.4.3 Air

Compressors and blowers will provide air to the systems within the process plant:

- Plant air
- Instrument air
- Low-pressure blower air for the flotation tank cells
- High-pressure blower air for the filter presses

17.4.4 Diesel and Lubricants

Diesel will be supplied to the mine site by truck and stored in the main fuel storage area. The storage area will be fully bunded with a containment capacity of 110% of the volume of one tank. The bunded will be fitted with a sump pump and an oily water separator. The fueling pad and truck offloading area will be concrete pads graded to ensure spillage reports to the storage area sump. The area will also have a fire detection system and foam-type cannons for firefighting.

A smaller diesel storage tank will be installed at the emergency generator plant in the process plant. This will supply the emergency generator system and will be refueled by truck as required. It will be laid out in a comparable manner to the main storage area but will not have fuel dispensing facilities.

Oils and lubricants will be stored in bunded areas within the workshop area in isotainers or drums, dependent on the supplier packaging.

18 Project Infrastructure

18.1 Introduction

The Project infrastructure is designed to support the operation of a 12 Mtpa underground mine operation and processing plant, expanding to 24 Mtpa in Phase 2, operating on a 24-hour per day, 7-day per week basis. The Project infrastructure is designed with local conditions and topography in consideration.

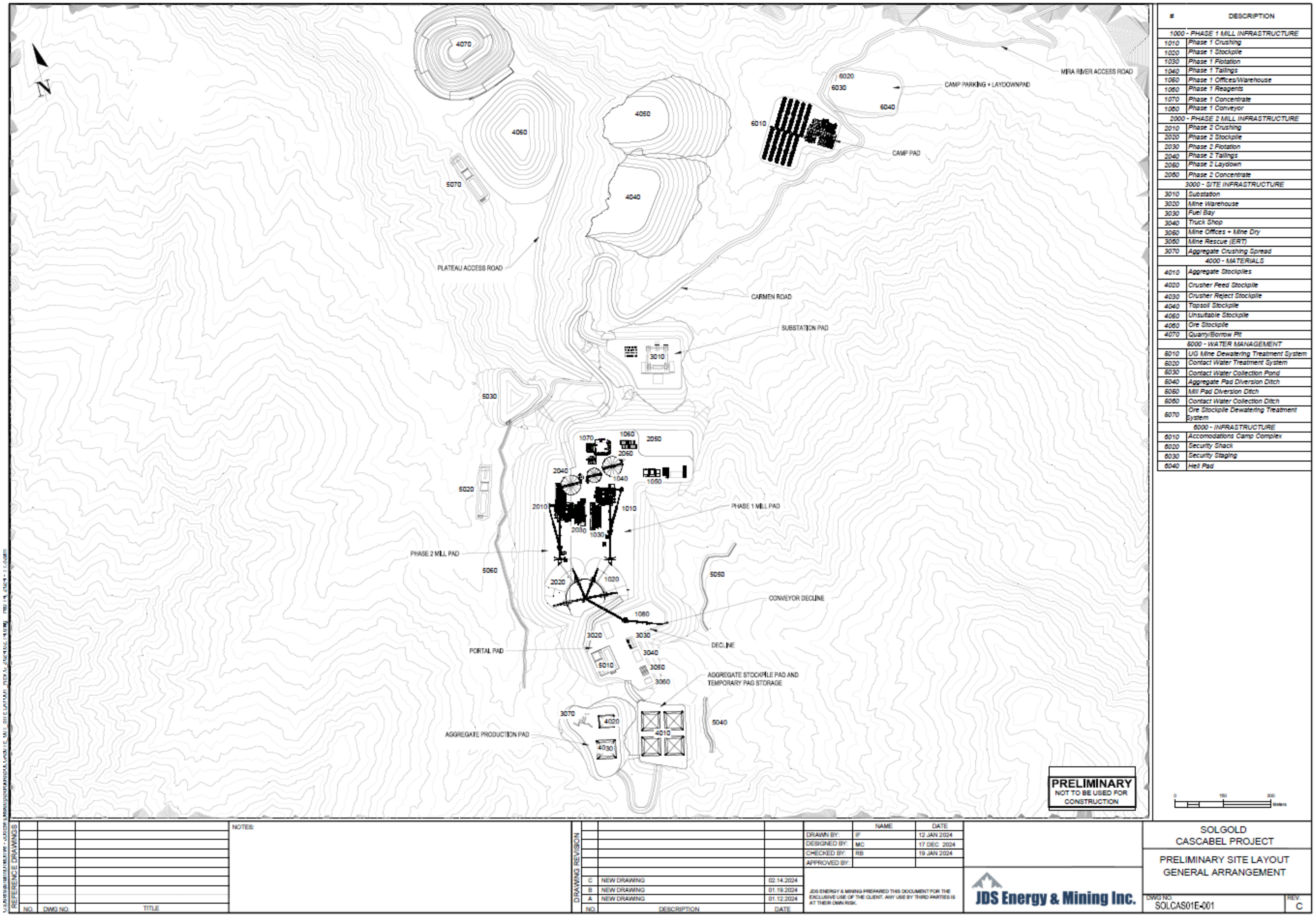
18.2 Site Layout

The overall Project site plan with the onsite infrastructure is presented in Figure 18-1.

18.2.1 Site Infrastructure

The required site infrastructure for the Cascabel project includes the following:

- Road works (access road, site roads, ramps and accesses, connections and signage, weighbridge)
- Site parking/delivery staging and security control facility for site access
- Waste management
- Surface water management (drainage networks, sediment control, bridges, culverts, etc.)
- Administration offices
- Maintenance facilities
- Warehousing and reagent storage
- Camp – construction and permanent
- Firefighting systems (detection and control)
- Communications – telephone networks, radio systems, data networks, CCTV
- Site substation and power distribution
- Emergency power generation
- Bulk fuel storage
- ROM temporary storage stockpile
- Waste rock storage facility
- Aggregate borrow pit
- Topsoil and unsuitable overburden stockpiles.



Source: This study, 2024

Figure 18-1: Site layout

18.2.2 Offsite Infrastructure

Critical infrastructure located away from the Project and not shown on the site layout include:

- Tailings storage facility and pipeline
- Power substation and transmission line to site
- Port facility

18.2.3 Cascabel Project Facility Design Considerations

Preliminary seismic hazard assessment for the Project area was conducted by Knight Piésold (2021a, 2021b) to provide a preliminary indication of the design earthquake parameters for the key facility locations (Table 18-1).

Table 18-1: Summary of estimated PGA for the Cascabel project

Seismic Level	Average period of recurrence (years)	Horizontal PGA (g)				
		Process Plant Site	Parambas Dam	Coastal Plains TSF	Port	Pipelines Route
Operating Basis Earthquake (OBE)	475	0.64	0.64	0.77	0.93	0.86
Safety Evaluation Earthquake (SEE)	MCE[1]	-	0.89	1.07	-	-

[1] MCE = Maximum Credible Event no return period assigned to this event

18.3 Roads

18.3.1 Access

Primary access to the Cascabel project process plant and mine will be via a new section of road starting from the E10 highway (two lane sealed road) that runs along the Rio Mira valley. The Project site is within mountainous terrain, ranging from 40% to 80% slope.

Culvert crossings with corrugated steel pipe are used on this alignment for major water crossings, and to maintain continuation of water flow.

18.3.2 Internal Roads

Internal roads have been designed as two-way roads to accommodate the planned traffic on each section, with an all-weather unsealed surface.

The planned roads at site are listed in Table 18-2.

Table 18-2: Site roads

Project Roads	Surface Width (m)
Carmen Access Road Section 1	12.2
Carmen Access Road Operating Section	21
Substation Powerhouse Pad	9.2
TSF Pipeline Road - 3.7 km	9.2
Mira River Access Road	9.2
Aggregate Pad Access Road	15.2
Plateau - Access Road	21

Surface water overland flow will be managed with roadside ditches, and use swales and culverts for road crossings, directed into the storm water pond.

18.4 Waste Management Facilities

18.4.1 Sewage

The Project will use modular sewage plants using a bio-oxidation process. Sewage plants will be located at the process plant area, the mine infrastructure area, the main administration building and at the camp. Each facility will be a standalone unit and process sewage and wastewater from the connected facilities located in that area. Pump stations will be used to feed each plant, and these will be located, so as to enable gravity flow of waste, to the pump station. Water from the facilities will be treated to the required standard and released to the environment. Solid treated waste will be periodically removed from the facility for disposal off site.

18.4.2 Solid Wastes

Solid waste materials will be stored on site before being transported off-site for disposal be a third-party contractor. Solid wastes will be classified as either:

- Recyclable – paper, glass, metal, plastics
- General waste – putriable or non-recyclable waste suitable for landfill
- Contaminated – soils and materials including used spares such as oil filters

All materials will be stored in suitable containers and where required stored in bunded or enclosed areas prior to removal from site. Only qualified and licensed contractors will be used for disposal of waste materials from site.

18.4.3 Liquid Wastes

These consist primarily of used or contaminated oils, fuels or lubricants derived from maintenance activities. These will be stored on site in suitable sealed containers and stored in bunded or enclosed areas prior to removal from site by an authorised contractor for disposal.

18.5 Tailings Management

SolGold commissioned Knight Piésold Consulting (KP) to undertake the PFS-level design for the tailings management component of the Project. The PFS designs considered the staged ramp up in production starting at 12 Mtpa for six years, followed by an expansion to 24 Mtpa thereafter. The total tailings requiring storage considered in the design was 529 Mt generated from 539 Mt of ore feed. The tailings will comprise 460 Mt of rougher tailings and 69 Mt of cleaner tailings.

This section of the Report provides details of the tailings management system designed to match the production profile. The key design objectives for the Tailings Management System are summarised as follows:

- Eliminate, manage or control environmental, health and safety risks with a zero-harm aspiration
- Design of the tailings management system to meet or exceed the requirements of Ecuadorian (Government of Ecuador, 2020) and international tailings design guidelines, standards and regulations including ICOLD (2022), ANCOLD (2019) and GISTM (2020)
- Permanent, safe and secure containment of all solid waste materials in facilities designed, constructed and operated engineered to international best practice
- Maximise tailings densities using effective tailings deposition strategies
- Minimise the risk of oxidation of potentially acid generating tailings by minimising the exposure to atmospheric oxygen
- Minimise the water retained in the tailings facilities, when possible, to only that required for operational requirements
- Design for and manage basin and embankment seepage to acceptable levels
- Design that can be constructed and operated in an efficient manner
- Design that allows for expansion should additional resources be processed, or processing rates increase above those considered as part of the PFS scenario
- Allow for effective rehabilitation at cessation of use of the facilities in line with the closure objective

18.5.1 Design Criteria

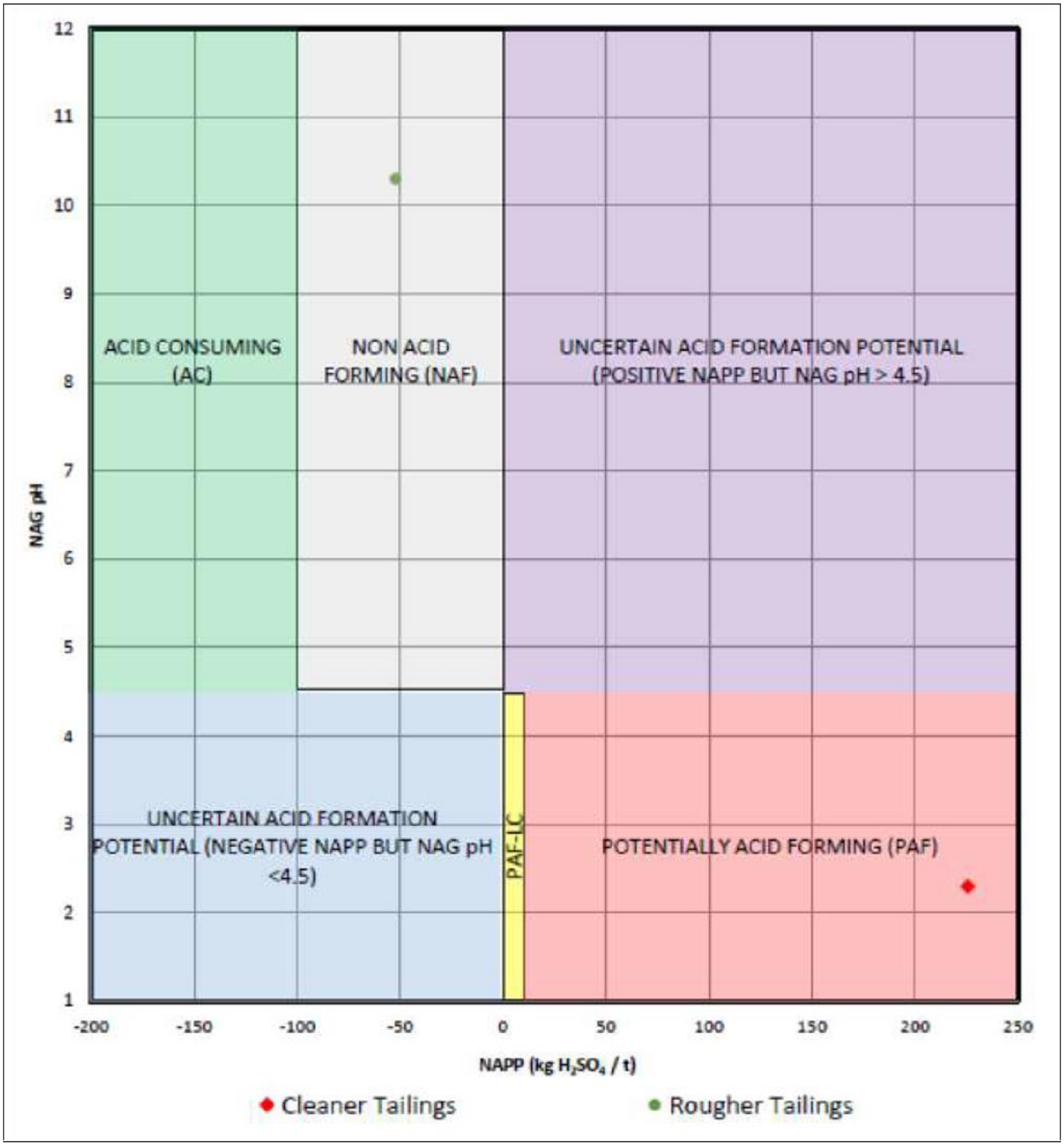
Based on the Global Industry Standard on Tailings Management (2020), all TSF options have been assigned an “extreme” dam failure consequence category at the PFS stage and the corresponding design criteria reflective of this consequence category have been adopted. This consequence category acknowledges the large environmental impact that could occur should a facility fail. The key design parameters are provided in Table 18-3.

Table 18-3: Key TSF design parameters

Parameter	Value	Source
Operating Basis Earthquake (OBE)	1 in 475 Year ARI (0.77 g)	ANCOLD (2019)
Safety Evaluation Earthquake (SEE)	Maximum Credible Earthquake (MCE) (1.07 g)	ANCOLD (2019)
Post Closure Earthquake	Maximum Credible Earthquake (MCE) (1.07 g)	ANCOLD (2019)
Static long-term drained FOS	> 1.5	ANCOLD (2019)
Static short-term undrained FOS	> 1.5	ANCOLD (2019)
Post Seismic FOS	> 1.2	ANCOLD (2019)
Residual Strength FOS	> 1.1	ICOLD (2022)
Post Seismic Deformation (OBE)	Maintain Serviceability	ANCOLD (2019)
Post Seismic Deformation (SEE)	No Loss of Containment	ANCOLD (2019)
Design Storage (fully operational spillway)	1 in 100-year ARI wet year + 1 in 100 year ARI 72 hour event + contingency freeboard of 1 m.	KP
Design Storage (without emergency spillway)	1 in 100-year ARI wet year + 1 in 100 year ARI 72 hour event + 24 Hr PMP + contingency freeboard of 1 m	KP
Operational spillways	PMP, with no catchment losses + diversion failure	ANCOLD (2019)
Closure spillways	PMP, with no catchment losses + diversion failure	ANCOLD (2019)

18.5.2 Tailings Geochemical Characteristics

A geochemical assessment has been conducted on both the cleaner and rougher tailings. The rougher tailings were classified as non-acid forming (NAF) with a low number of enriched elements in solids. The cleaner tailings classified as potentially acid forming (PAF) with a moderate number of enriched elements in solids. A combined tailings stream would be considered PAF. The acid formation plot for the rougher and cleaner tailings samples is provided in Figure 18-2.



Source: Artica et al, 2022

Figure 18-2: Tailings acid formation plots

18.5.3 Tailings Settling Characteristics

Samples of rougher and cleaner tailings were tested to determine their settling and consolidation parameters. The coarse-grained rougher tailings (P_{80} 150 µm) were found to settle moderately quickly achieving a high dry density of 1.4-1.5 t/m^3 prior to consolidation. The fine-grained cleaner tailings (P_{80} 33 µm) were found to settle moderately quickly but only achieved a low dry density of 0.9-1.0 t/m^3 prior to consolidation. Blending of the two streams is anticipated to produce a combined tailings stream with similar settling properties to rougher tailings based on the anticipated proportions of the two materials.

18.5.4 Site Characteristics

Knight Piésold have undertaken an assessment of the hydrometeorology at the Project. The regional climate around the Project is characterised by humid weather, with a bi-modal rainy season that typically peaks in December and March, when rainfall often exceeds 200 mm/month. The mean annual precipitation is 2,610 mm at Rocafuerte (location of the processing plant) increasing with elevation and local orographic effects. Estimates of extreme rainfall events have been produced by Knight Piésold, which have been employed in the design of the water management components of the Project.

Knight Piésold conducted a seismic hazard assessment for the Cascabel project to provide the design earthquake parameters for the proposed TSF location and the processing and mining area. The assessment included a probabilistic analysis and a deterministic analysis to evaluate a maximum credible earthquake (MCE). The seismic hazard assessment indicates:

- The Cascabel project is in a region of high seismicity associated with interface (intraplate) subduction earthquakes, intraslab (inslab) earthquakes in the subducted oceanic tectonic plate and shallow crustal earthquakes.
- Given the Cascabel project is in a high seismicity region with frequent earthquake occurrence it is appropriate to adopt the deterministically derived MCE events for design of the TSF rather than using probabilistic seismic hazard values associated with a low AEP. The results from the deterministic seismic hazard found that intraslab subduction and interface subduction earthquakes represent the major hazards to the sites, while the shallow crustal earthquakes were minor in comparison.
- It is recommended that a PGA of 0.77 g for Coastal Plain TSF and 0.65 g for the processing and mining area be adopted for the Operating Basis Earthquake (OBE) for a 1 in 475-year return period.
- For the Safety Evaluation Earthquake (SEE), a PGA of 1.07 g for Coastal Plain TSF and 0.89 g for Cascabel Mining Concession should be adopted.

18.5.5 Technology Study

A technology study was conducted to assess various tailings technologies available for the Project. Filtered and paste technology were ruled out as they have not been applied to throughputs commensurate with the design throughput at the Cascabel project and are considered an untested technology for these throughputs. Unthickened and deep-sea disposal were ruled out as they were not preferred based on the environmental and sustainability goals of SolGold. Thickened tailings was selected as the preferred disposal method with the use of cyclone separation to produce construction material to be assessed at the next phase of study when larger tailings samples are available for testing.

18.5.6 Site Selection

A comprehensive site selection study was carried out to identify potential tailings sites in an area up to 70 km around the site. A preliminary screening exercise reduced the potential sites down to four moderate size sites and two small sites located close to the mine and three sites approximately 40-50 km west of the mine. A multiple account analysis was conducted to select the preferred tailings configuration for each of the throughput scenarios. The multiple accounts analysis considered technical/engineering, financial, environment and social criteria with the criteria weighted at 40% technical/engineering, 25% social, 20% financial and 15% environmental. Technical and engineering criteria was considered the critical aspect as a technical failure could have major financial, social, and environmental impacts.

18.5.7 Tailings Storage Facility Options

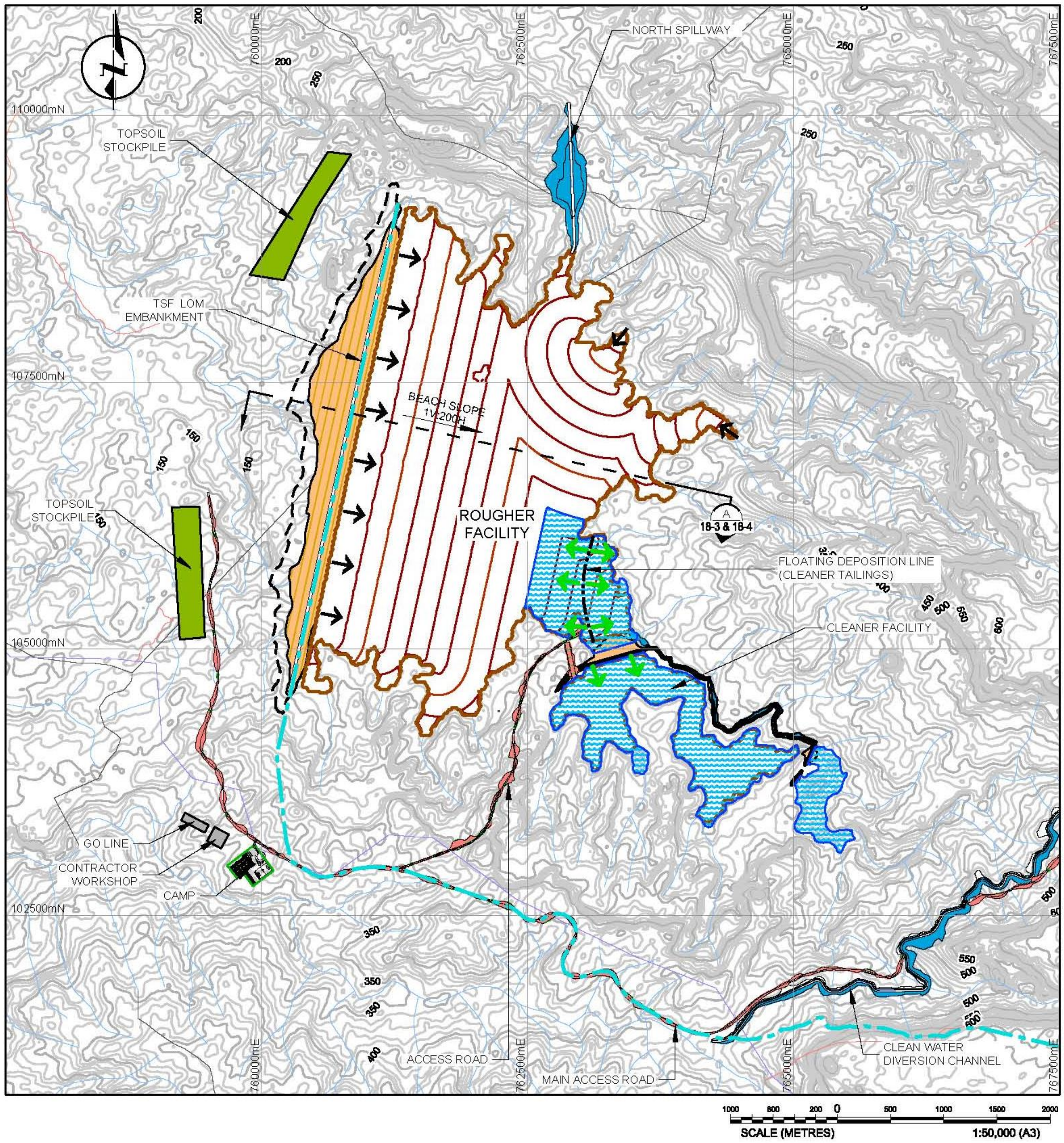
Four tailings storage facility (TSF) options were assessed as part of this Study as detailed below:

1. Coastal Plains Main TSF (Distal to Cascabel Concession)
2. Coastal Plains East TSF (Distal to Cascabel Concession)
3. Parambas TSF (On Cascabel Concession)
4. Cachaco TSF (On Cascabel Concession)

The option selected for the estimation of capital and operating costs provided as part of this study was the Coastal Plains East TSF but all four options were found to be viable options. The options located on the Cascabel Mining Concession (Parambas and Cachaco) should be reassessed if open pit mining at Tandayama Americana deposit is incorporated into the mine plan as these facilities become more economical should ex-pit mine waste be available and provide environmental control solutions for the valleys potentially impacted by open pit mining and waste disposal. The Coastal Plains Main TSF provides a much larger capacity (in excess of 1.8 Bt) than required under the current study and should be reassessed if the mine production plan increases. Details of the facilities are provided in the following sections.

Coastal Plains Main TSF

The Coastal Plains Main TSF is located distal to the proposed processing plant. The main embankment of the Coastal Plains Facility is approximately 40 km from the processing plant. The facility will comprise two storage areas, the Starter TSF located approximately 3.4 km upstream of the main facility and the main facility. The Starter facility could either be operated to store cleaner tailings during initial production or used as a starter facility for combined tailings (depending on final ramp up schedule) to defer the cost of construction of main embankment. The general arrangement of the Coastal Plains Main TSF showing both Main and Starter facilities is shown in Figure 18-3.



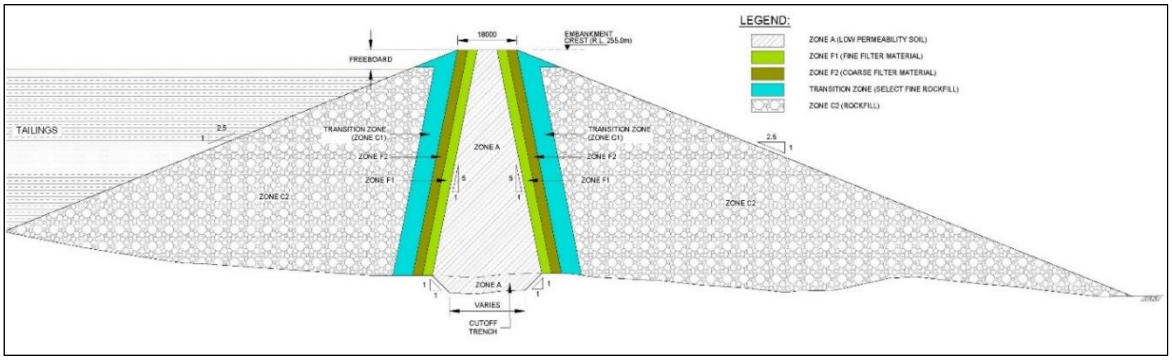
Source: This study, 2024

Figure 18-3: Coastal Plains Main TSF general arrangement

The starter facility will be constructed approximately 3.4 km upstream of the Coastal Plains Main Embankment at a natural constriction of the valley. The starter facility embankment will be 1,323 m long at crest and will be 87 m high from crest to downstream toe. The Starter Facility embankment is designed as a central core dam with a low permeability core protected on both upstream and downstream by two stage filter zones. The upstream and downstream structural zones have been designed to be constructed of competent rockfill with a slope of 1V:2.5H both upstream and downstream with embankment cross section provided on Figure 18-4.

A small embankment of 25 m height will be constructed at the eastern extent of the starter facility to act as a saddle dam and to direct water into the starter facility surface water diversion channel. The facility will be equipped with an ungated open channel spillway designed to safely pass the Probable Maximum Flood (PMF).

The embankment will be fully instrumented with real-time monitoring of pore pressures and deformations. Additional real-time monitoring of surface water flows, pond volumes, seepage outflows and pond volume will be proposed at the next phase of the design.

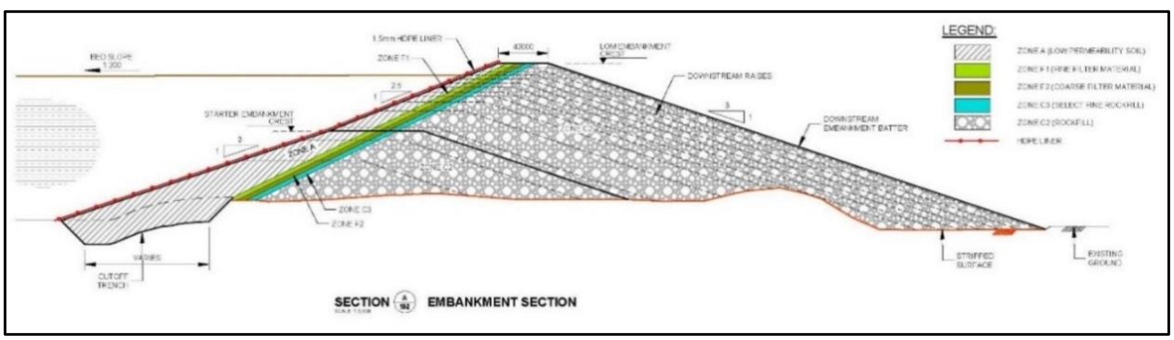


Source: This study, 2024

Figure 18-4: Coastal Plains Main TSF - starter facility embankment section

The Coastal Plains Main TSF Embankment will be a cross-valley embankment with a crest length at final height of 4.6 km with a height from crest to downstream toe of 132 m. The Coastal Plains Main TSF Embankment will be constructed as a zoned earth and rockfill embankment with a sloping upstream low permeability zone, two stage filter and downstream structural zone. Stripping of low strength material from the footprint of the embankment will be undertaken prior to construction of a filter compatible basal drainage blanket below the embankment. The upstream batter will be HDPE lined to reduce seepage and protect the low permeability zone from erosion when exposed to the high intensity rainfall at the site.

The facility has been designed with the downstream slope of 1V:3H and the upstream slope commencing at 1V:3H and then steepening to 1V:2.5H. The cross section of the Coastal Plains Main embankment is provided in Figure 18-5. Construction materials will be sourced from the spillway excavations, foundation stripping and a large borrow pits located along the ridgelines either side of the facility.



Source: This study, 2024

Figure 18-5: Coastal Plains Main TSF – main facility embankment section

To reduce the inflow of surface water, the facility will be equipped with a surface water diversion channel which will divert approximately 50% of the catchment around the facility. However, the facility will still have a positive water balance and will require active discharge of water from the facility.

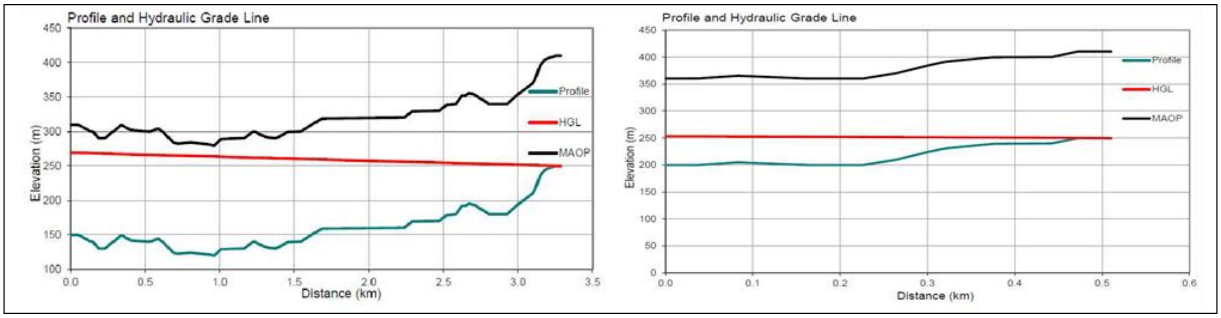
Design of the water management pumps required to control the volume of water within the facility has been undertaken by Fortin Pipeline. The system will comprise of pontoon mounted electrically driven pumps discharging via a continuously welded HDPE pipelines to the upstream surface water diversion channel of the TSF. The required pump capacity for the TSF have been calculated in the water balance model with the following pump capacities included in the model:

- 1.5 Mm^3 / month (~2,050 m^3/hour) from the Starter TSF when operational
- 8.0 Mm^3 / month (~11,000 m^3/hour) from the Main TSF when receiving rougher tailings only
- 9.5 Mm^3 / month (~13,050 m^3/hour) from the Main TSF when receiving rougher & cleaner tailings

The pump system from the Main and Starter TSF facilities designed by Fortin Pipelines includes:

- Starter – one HDPE pipeline with pontoon mounted pump station
- Main (rougher only) – six HDPE pipelines operating parallel each with pontoon mounted pump station
- Main (rougher and cleaner) – seven HDPE pipelines operating parallel each with pontoon-mounted pump station

The profile and hydraulic grade lines for the two systems is shown in the Figure 18-6 below.



Source: This study, 2024

Figure 18-6: Coastal Plains decant pipelines (Main and Starter) long section and hydraulics

The volume of water discharged will be capped based on the flow in the upstream surface water diversion channel, into which the discharge will be released, such that sufficient mixing and dilution occurs within the diversion channel to ensure Ecuadorian water discharge criteria are met before the final release of the water (mixed surface runoff and discharge) to the receiving environment.

No water will be returned to the processing plant from the Coastal Plains facility with water for processing to be sourced from mine dewatering and the Mirra River adjacent to the processing plant.

The Coastal Plains Main TSF will have a spillway to the north in later stages of operation that will be cut through the ridgeline. This spillway with then be raised as a masonry concrete dam with a reinforced concrete cascade structure. The spillway has been designed to safely pass the Probable Maximum Flood

(PMF). Prior to the spillway being constructed the facility has been designed to store PMP on top of the 1 in 100-year ARI 72 hours event.

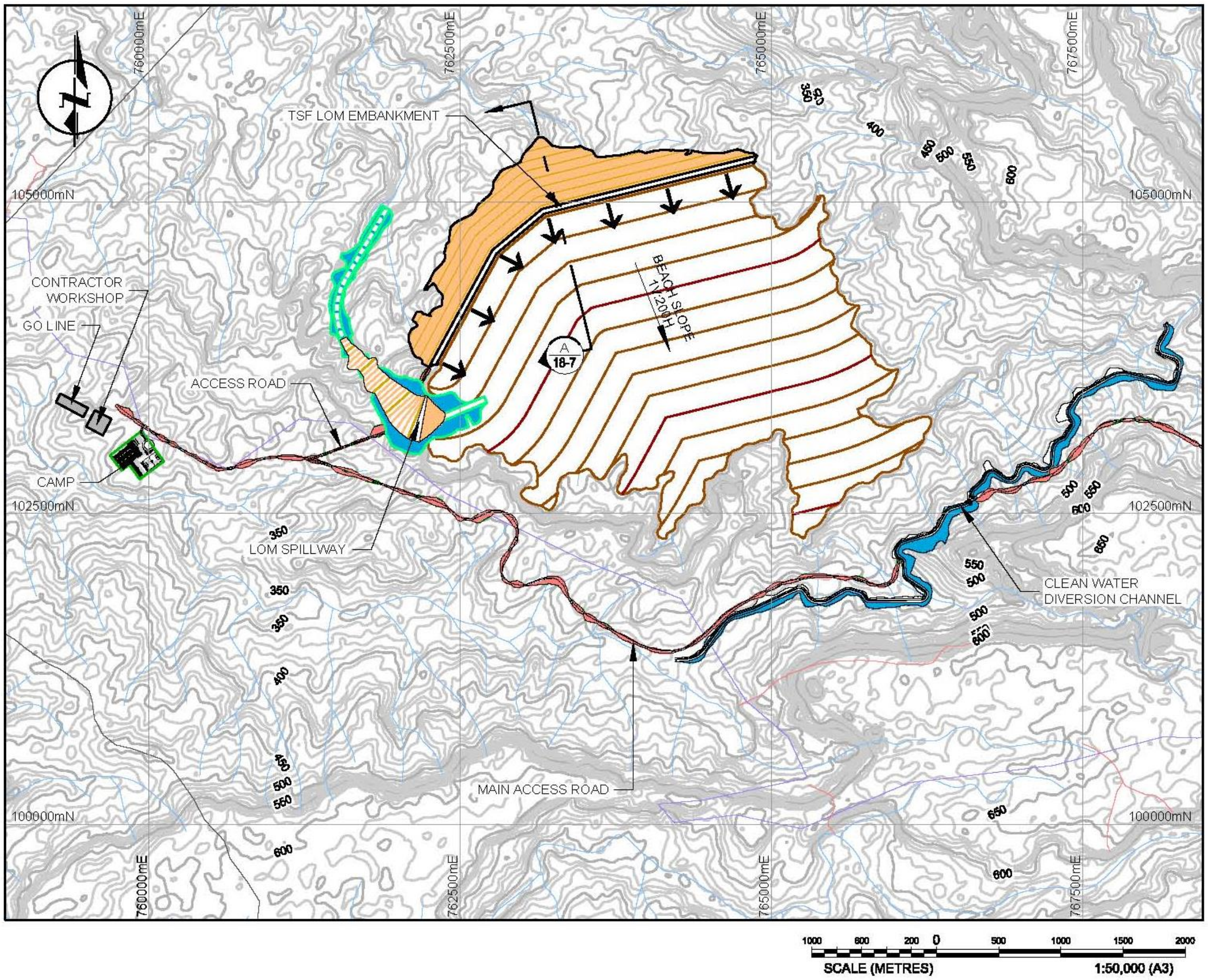
The embankment will be fully instrumented with real time monitoring of pore pressures and deformations. Additional real time monitoring of surface water flows, pond volumes, seepage outflows and pond volume will be proposed at the next phase of the design.

At closure, most of the tailings beach will be covered with rewon topsoil and borrow material and revegetated with the pond retained to maintain saturation of the cleaner tailings post closure.

Coastal Plains East TSF

The Coastal Plains East TSF is located distal to the proposed processing plant. The main embankment of the Coastal Plains Facility is approximately 36.5 km from the processing plant. The facility will comprise a single storage area with the embankment located along the same alignment as the Coastal Plains Main – Starter Embankment. The embankment height to store the full 529 Mt of tailings would be approximately 190 m from crest to downstream toe with an embankment length of 3.3 km. Minimal expansion capacity is available above the life of mine storage (529 Mt) within this facility.

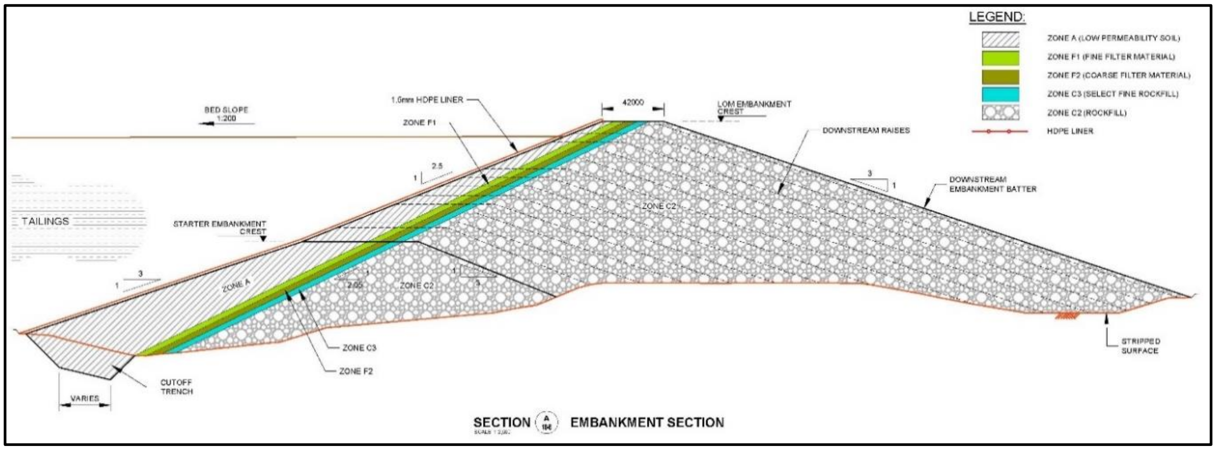
The starter embankment has been sized to match the optimal excavation elevation for the spillway which will be constructed at the initial stage, with the spillway excavated material employed to construct the starter embankment. The starter embankment would approximately 98 m high from crest to downstream toe with an embankment length of 1.9 km and would provide storage for approximately 35 Mt of tailings. The general arrangement of the Coastal Plains East TSF is provided in Figure 18-7.



Source: This study, 2024

Figure 18-7: Coastal Plains East TSF – general arrangement

The Coastal Plains East TSF embankment will be constructed as a zoned earth and rockfill embankment with a sloping upstream low permeability zone, two stage filter and downstream structural zone. Stripping of low strength material from the footprint of the embankment will be undertaken prior to construction of a filter compatible basal drainage blanket below the embankment. The upstream batter will be HDPE lined to reduce seepage and protect the low permeability zone from erosion when exposed to the high intensity rainfall at the site. The cross section of the Coastal Plains East Embankment is provided in Figure 18-8. Construction materials will be sourced from the spillway excavation, foundation stripping and a large borrow pits located along the ridgelines either side of the facility.



Source: This study, 2024

Figure 18-8: Coastal Plains East TSF – embankment section

To reduce the inflow of surface water, the facility will be equipped with a surface water diversion channel which will divert the majority of the catchment runoff around the facility. However, the facility will still have a positive water balance and will require active discharge of water from the facility.

The required pump capacity for the TSF have been calculated in the water balance model with a discharge pumping rate of up to 6.5 Mm^3 per month of pumping capacity required which will be provided by five HDPE pipelines operating parallel each with pontoon mounted pump station.

The volume of water discharged will be capped based on the flow in the upstream surface water diversion channel, into which the discharge will be released, such that sufficient mixing and dilution occurs within the diversion channel to ensure Ecuadorian water discharge criteria are met before the final release of the water (mixed surface runoff and discharge) to the receiving environment.

No water will be returned to the processing plant from the Coastal Plains East facility with water for processing to be sourced from mine dewatering and the Rio Mira adjacent to the processing plant.

The Coastal Plains East TSF will have the option of maintain the operational spillway at closure or construction of a low energy (flat) spillway channel through the ridgeline to the south at closure. There is a heritage listed rail alignment on the southern ridgeline and therefore permits to impact this feature would be required for the southern closure spillway alignment. Whichever spillway is adopted would be designed to safely pass the probable maximum flood.

The embankment will be fully instrumented with real time monitoring of pore pressures and deformations. Additional real time monitoring of surface water flows, pond volumes, seepage outflows and pond volume will be provided at the next phase of the design.

At closure, the majority of the tailings beach will be covered with rewon topsoil and borrow material and revegetated with the pond retained to maintain saturation of the cleaner tailings post closure.

Parambas TSF

The Parambas TSF is located proximal to the proposed processing plant (within 2 km) on the Cascabel concession. The facility will comprise a single storage area with a cross valley main embankment and an upstream diversion embankment to divert the upstream catchment runoff around the facility. The facility will have a maximum capacity of approximately 60 Mt with the facility only suitable for use at lower throughputs (i.e., ≤ 12 Mtpa) as such this facility could be employed as a starter facility for the early phases of operations. The embankment height at full capacity will be approximately 225 m from crest to downstream toe with an embankment length of approximately 1.1 km. The facility would be built in stages to match the deposition schedule with a spillway only installed at the final stage height, the facility was therefore sized to store the probable maximum precipitation event on top of the 1 in 100-year ARI storm event.

The Parambas embankment will be constructed as a zoned earth and rockfill embankment with a sloping upstream low permeability core, two stage filter and downstream structural zone. Stripping of low strength material from the footprint of the embankment will be undertaken prior to construction of a filter compatible basal drainage blanket below the embankment. The upstream batter will be HDPE lined to reduce seepage and protect the low permeability zone from erosion when exposed to the high intensity rainfall at the site. The facility has been designed with the downstream slope of 1V:3H.

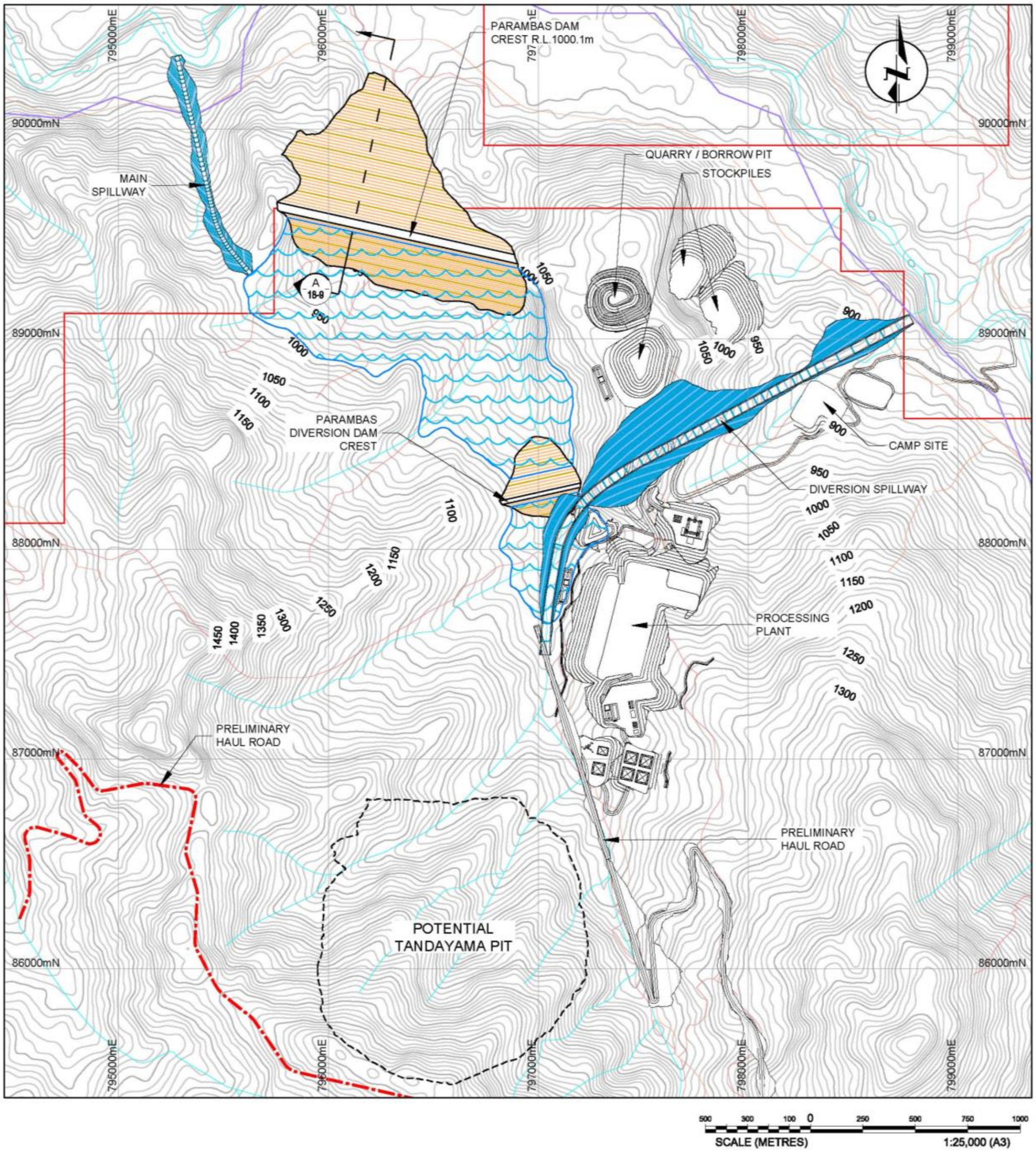
The closure spillway and the diversion embankment/diversion spillway were sized to pass the peak run-off generated from the probable maximum precipitation event. Water will be returned to the process plant from the Parambas TSF as such no discharge of water from the facility is envisaged during operation.

Embankment construction material would be sourced from borrow pits, spillway excavations, excavation for the process plant construction, excavation for the portals and from underground development waste. This option has poor embankment to storage efficiency (i.e., large embankment volumes) and therefore its economics are improved if the Tandayama Americana open pit is developed which would provide lower cost construction materials. Further design work is planned for this TSF option as the viability of the Tandayama Americana open pit Is assessed.

The general arrangement of the Parambas TSF is provided in Figure 18-9 with the embankment section shown in Figure 18-10.

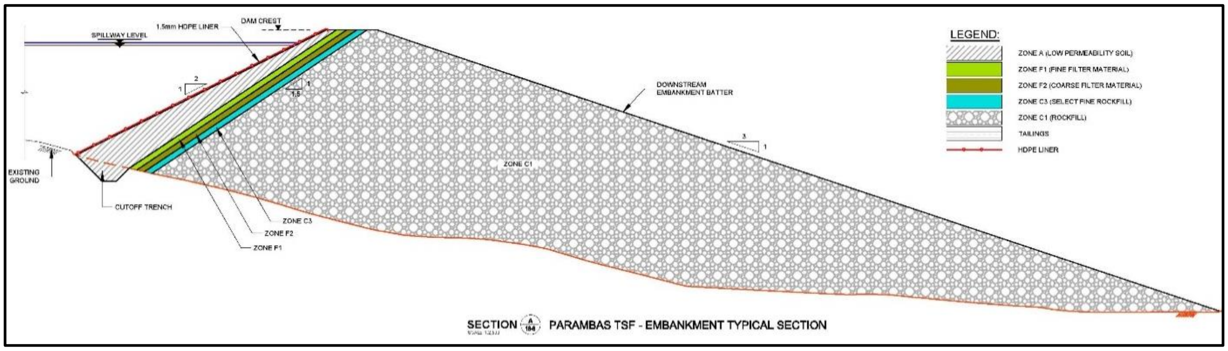
The embankment will be fully instrumented with real time monitoring of pore pressures and deformations. Additional real time monitoring of surface water flows, pond volumes, seepage outflows and pond volume will be undertaken.

At closure, the majority of the tailings beach will be covered with rewon topsoil and borrow material and revegetated with the pond retained to maintain saturation of the cleaner tailings post closure.



Source: This study, 2024

Figure 18-9: Parambas TSF – general arrangement



Source: This study, 2024

Figure 18-10: Parambas TSF – embankment section

Cachaco TSF

The Cachaco TSF is located proximal to the proposed processing plant (within 3 km) on the Cascabel concession and approximately 1.5 km west of the Tandayama Americana Deposit. The facility will comprise a single storage area with a cross valley main embankment. A diversion channel will be constructed upstream of the facility to direct catchment run-off around the facility.

The facility will have a maximum capacity of approximately 60 Mt with the facility only suitable for use at lower throughputs (i.e., ≤ 12 Mtpa) as such this facility could be employed as a starter facility for the early phases of operations. The embankment height at full capacity will be approximately 250 m from crest to downstream toe with an embankment length of approximately 1.3 km. The facility would be built in stages to match the deposition schedule with a spillway only installed at the final stage height, the facility was therefore sized to store the probable maximum precipitation event on top of the 1 in 100-year ARI storm event. The closure spillway was sized to pass the peak run-off generated from the probable maximum precipitation event.

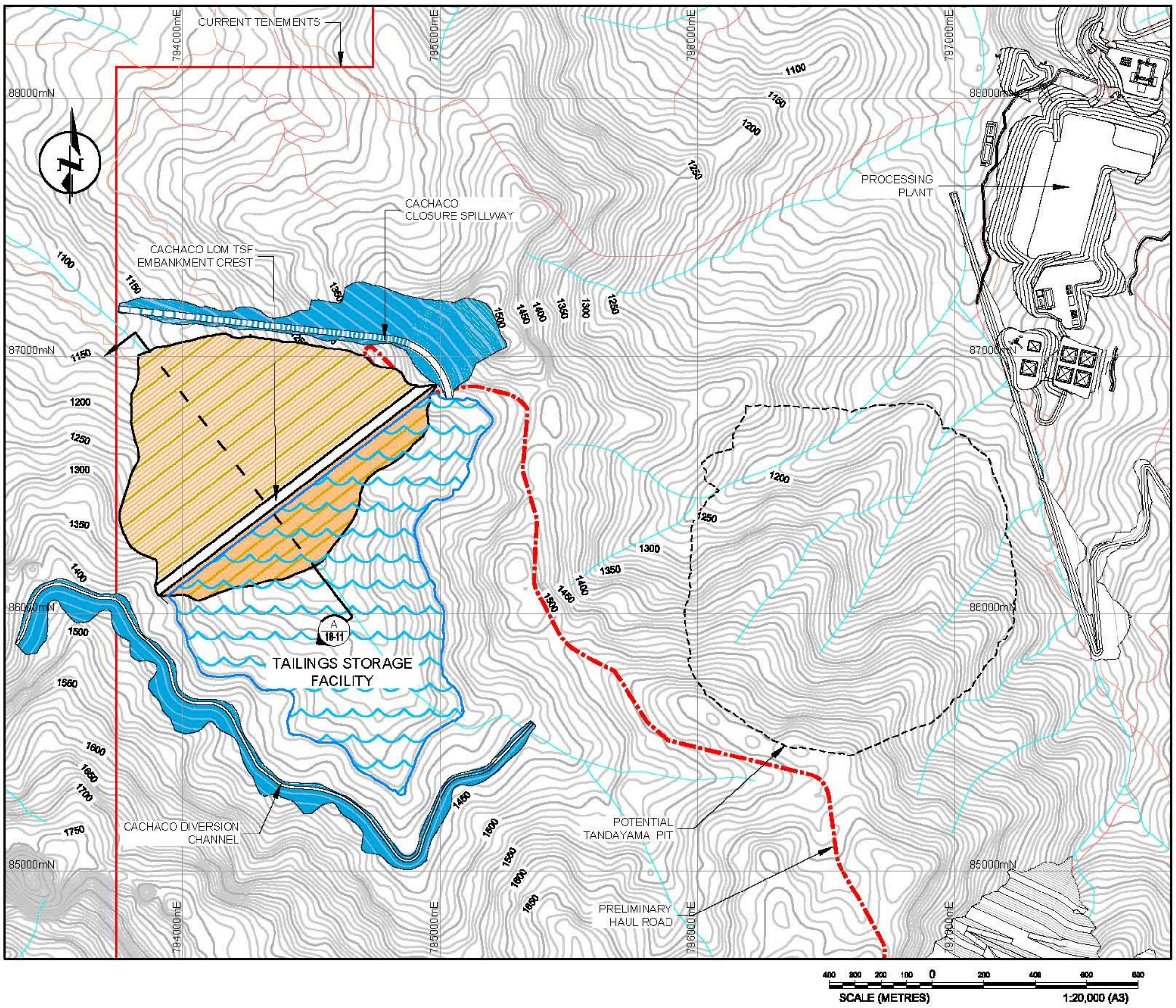
The Cachaco embankment will be constructed as a zoned earth and rockfill embankment with a sloping upstream low permeability zone, two-stage filter and downstream structural zone. Stripping of low strength material from the footprint of the embankment will be undertaken prior to construction of a filter compatible basal drainage blanket below the embankment. The upstream batter will be HDPE lined to reduce seepage and protect the low permeability zone from erosion when exposed to the high intensity rainfall at the site. The facility has been designed with the downstream slope of 1V:3H.

Embankment construction material would be sourced from borrow pits and spillway excavations. This option has poor embankment to storage efficiency (i.e., large embankment volumes) and therefore its economics are improved if the Tandayama Americana open pit is developed which would provide lower cost construction materials. Further design work is planned for this TSF option as the viability of the Tandayama Americana open pit is assessed.

The embankment will be fully instrumented with real time monitoring of pore pressures and deformations. Additional real time monitoring of surface water flows, pond volumes, seepage outflows and pond volume will be proposed at the next phase of the design.

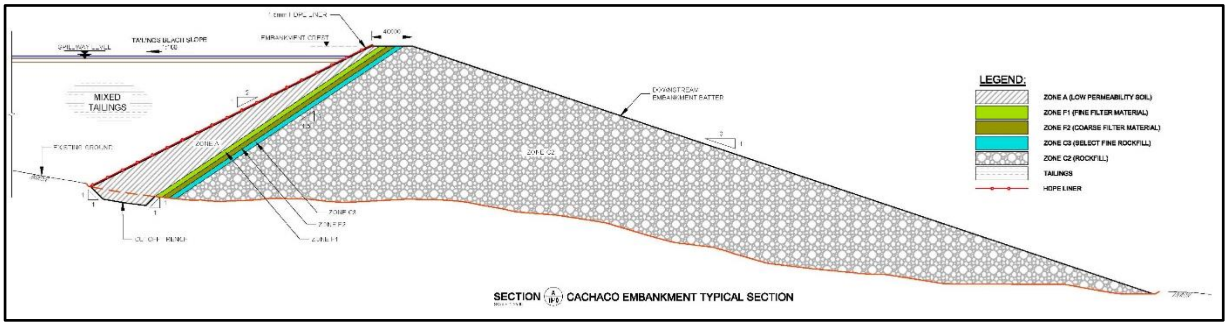
At closure, the majority of the tailings beach will be covered with rewon topsoil and borrow material and revegetated with the pond retained to maintain saturation of the cleaner tailings post closure.

The general arrangement of the Cachaco TSF is provided in Figure 18-11 with the embankment section shown in Figure 18-12.



Source: This study, 2024

Figure 18-11: Cachaco TSF – general arrangement



Source: This study, 2024

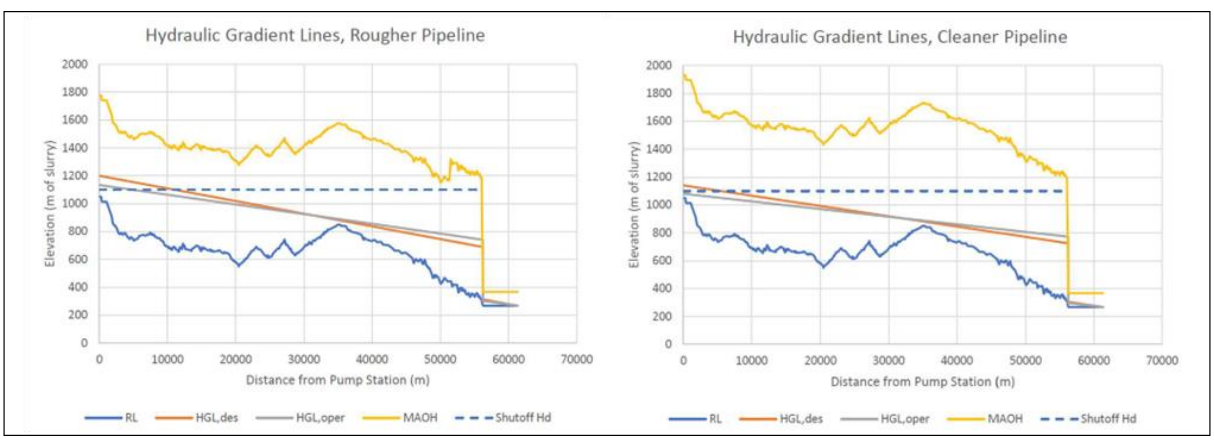
Figure 18-12: Cachaco TSF – embankment section

18.5.8 Tailings Pipelines

The tailings pipelines from the processing plant to the Coastal TSF have been designed by Slurry Systems Engineering considering separate rougher and cleaner pipelines. The pipeline route to the Coastal Plains TSFs will be approximately 57 km in length. The long section showing the hydraulic gradient line for normal operation (HGL-Operate), design maximum operation (HGL-design) as well as Maximum Allowable Operating Head (MAOH) of the preferred pipeline are provided in Figure 18-13.

The Slurry Systems Engineering has recommended use of carbon steel pipe fully welded, coated, trenched and buried. The pipes are assumed to be externally coated and protected by an impressed current cathodic protection system. The pipelines are assumed to be lined with an inserted HDPE liner.

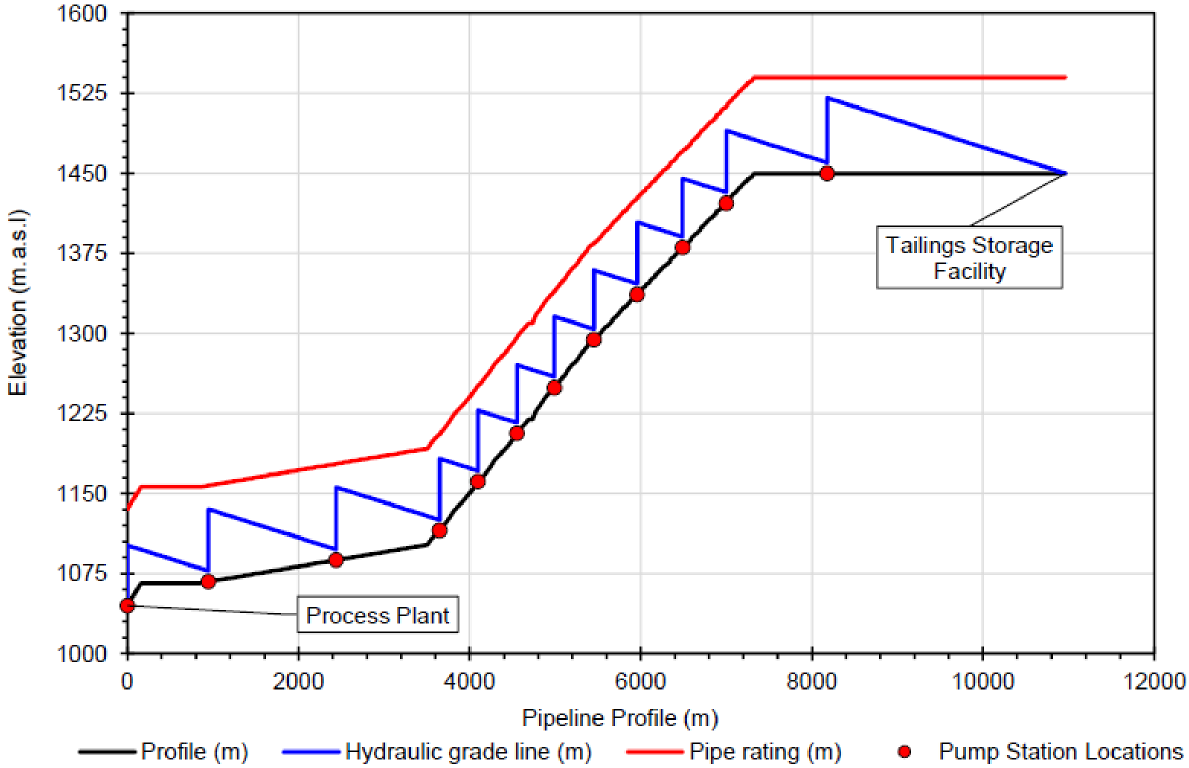
Under normal operating conditions both the rougher and tailings pipeline would flow by gravity from the process plant to the TSF with a single choke station installed in each line to prevent slack flow from occurring along the pipeline route. Tailings pumps would be required at the processing plants tailings hopper only for flushing of the rougher and cleaner pipelines and for tailings flow more than the normal operating rates.



Source: This study, 2024

Figure 18-13: Coastal Plains pipeline long sections and hydraulics

The tailings pipelines from the processing plant to the Cachaco TSF have been designed by Paterson and Cooke considering a combined tailings pipeline. The pipeline route to the Cachaco TSF will be approximately 10 km in length to maintain suitable grades with 12 pump stations included along the line. The long section showing the hydraulic gradient line for design maximum operation (HGL-design) as well as Maximum Allowable Operating Head (MAOH) of the preferred pipeline is provided in Figure 18-14.



Source: This study, 2024

Figure 18-14: Cachaco TSF pipeline long sections and hydraulics

The Paterson and Cooke has recommended use of UHMWPE pipes. The material reduced the pipeline wear rate, mitigates the risk of corrosion, and can manage the line pressures.

No detailed engineering has been conducted on the Parambas TSF pipeline design at this stage of study as the pipeline alignment is short and will not have high heads due to the elevation of the plant and this TSF being similar.

18.5.9 Quantities and Cost Estimates

Quantities and costs have been developed for the designs to match the proposed tailings production schedule. The capital, sustaining capital and operating costs for the Coastal Plain East TSF option have been incorporated into the overall Project financial models.

18.6 Water Management

18.6.1 Water Balance Modelling

Water balance models have been developed for the Coastal Plain Main and Coastal Plain East TSF options to determine pond volumes (and resultant embankment crest requirements), discharge requirements and estimate water chemistry. These models have been run as deterministic models with probabilistic models to be developed in the next phase of study. A detailed water balance model has not been established for the Parambas or Cachaco TSFs, which have very limited catchments and will have recycle to the process plant that will allow control of the pond volumes on these facilities.

Assessments have been undertaken of the flow data available for the Mira River to determine if sufficient flow is available to provide the process water demands for the plant. The minimum recorded flow in the river (1963 to 2008) always exceeds process water demand by a factor of ten or more, not accounting for other inputs into the process circuit, such as underground dewatering flows or recycle from the tailings dams (Parambas / Cachaco). Therefore, the assessment indicates that sufficient water will always be available for process water demands.

18.7 On-Site Infrastructure

18.7.1 Site Security and Access Control

A security station with access gate will be installed at the entrance to the Project site, with perimeter fencing restricting access only at this station. The site entrance will include a parking lot and staging area for all vehicles coming to site. Only approved vehicles will be allowed past the security entrance.

18.7.2 Camp and Accommodation

The existing site camp will be expanded for early works construction and throughout the construction phase of the Project. The construction camp at site will add one thousand beds in temporary facilities over the first 5 years using multi-person occupancy rooms.

A permanent camp with accommodations for 420 personnel will be built during the construction phase using modular constructed units. These will provide accommodations for EPCM and Owner’s team during construction and for the operating personnel once production commences.

Separate camp facilities will be located at the Coastal Plains TSF due to the distance from the Cascabel concession and the focused work area.

Facilities within the camps will include dormitories, health care facility, messing areas and recreation areas. The sizing and services of this camp will be reviewed once the Project execution strategy is further defined in the Feasibility study.

18.7.3 Water Supply

Water supply infrastructure includes process plant make up water, raw water, and potable water. The process plant make-up water will be mainly sourced from the site runoff collection pond. A reclaim pump and piping will provide water to the process water tank from the collection pond.

Raw water make up will be sourced from the Mira River via pipeline. Two vertical turbine pumps will be installed in the river and deliver water via a 100 mm pipe to the process plant facility.

A localised potable water treatment plant will be installed at the camp site and utilise the raw water from the Mira River. Additional bottled purified water will be delivered to site and available throughout the facilities for personnel to consume.

18.7.4 Administration Office and Auxiliary Facilities

The Project will include administrative and auxiliary buildings located in the proximity of the camp. The buildings will be designed and constructed to suit the local conditions consistent with the types of structures built within the region.

Administration Office

A site administration office for Owner's team will be in the vicinity close to the camp. The office will be adjacent to a medical center that will service the site with Owner's medical personnel.

Security Personnel Facility

A security personnel facility will be located near the camp area, with security equipment storage, personnel facilities, and control room for monitoring the site. This is a separate facility from the entrance control building with gate.

18.7.5 Mine Surface Infrastructure

The mine surface infrastructure is on the Portal Pad and includes the following facilities:

Mine Offices and Mine Personnel Facility

A building facility will be constructed for the mine management and technical staff and will include a personnel change and locker area for the UG personnel including shower facilities.

Emergency Response and First Aid

The emergency response facility will include a training room, emergency response equipment for the UG mine and process plant, and garage for the site ambulance and fire truck. It will include a first aid area as well as personnel area for first responders.

Vehicle Maintenance

The mine maintenance shop includes a heavy vehicle work area, light vehicle bay, tire bay and an electrical workshop. Servicing of site vehicles and major rebuilds of underground heavy vehicles will be undertaken in this facility. The shop is equipped for the storage and dispensing of lubricants and oils, as well as waste oil storage. The facility also includes an outdoor wash bay for vehicle cleaning and a tire bay equipped for both light and heavy vehicle tire changes. The vehicle bays will be equipped with an overhead crane.

This facility is a closed steel structure with concrete foundations and concrete floor slab construction. It will include the tools and equipment necessary, including a hydraulics servicing station, for site vehicles maintenance tasks.

Mine Warehouse and Laydown Areas

The mine warehouse is on the portal pad with the vehicle maintenance shop and other mine facilities. The warehouse is a large, enclosed steel frame and fabric covered building fitted out with suitable racking to store mine spares in a secure area. A secure laydown open area is adjacent to the main warehouse and will be used to store oversize items within a fenced compound.

The mine warehouse and yard will store the UG mine operation, process plant, and site services maintenance spare parts and consumables.

An office and front of warehouse area for inward and outward materials management are located in the warehouse.

Fuel Storage and Dispensing

A bulk diesel storage and dispensing facility service the mining fleet with six tanks of 60,000 liters of diesel fuel. A 5,000-litre gasoline tank will provide fuel for light vehicles. The facility will have 4 fuel pump filling stations for the diesel, including hi-flow capacity equipment. The area will be fenced, secured, and lined and bunded to provide adequate spill containment.

Batch Plant

A concrete mixing batch plant will service the UG operations, consisting of cement silo, aggregate and sand feed conveyor, and mixing plant. The UG equipment will be provided with shotcrete from this batch plant on the Portal Pad.

18.7.6 Process Plant Infrastructure

The following facilities will be located at the process plant:

Laboratory

A complete sample preparation and assay laboratory facility will be built adjacent to the process plant. It will include a complete lab equipment package that will comply with international standards and best practices.

Process Plant Offices & Change Facility

The mill office for plant staff and the control room will be located next to the process plant and locker room facilities for process plant personnel will also be provided at this facility.

Process Plant Maintenance Workshop

A fixed plant maintenance workshop will be near the process plant, on the platform between the ball mills and the fine ore stockpile. This facility is intended for process plant equipment maintenance. The workshop includes areas for mechanical equipment repair and an electrical/instrumentation workshop. The facility also includes stores for consumables and specialist parts as well.

A reagent storage warehouse will be adjacent to the process plant specially designed for the reagent storage in a bunded and contained facility. The reagents will be segregated by walls or curbs where required to avoid any potential cross contamination.

18.7.7 Fire Protection

Each area of the mine site will be provided with an appropriate fire protection system. Fire protection systems have been included in the mine infrastructure area, process plant, port, and administration areas. Each individual fire system will be for the exclusive use for that area and is not intended to have connections for use by other services, neither temporary nor permanent, or for uses not directly related to firefighting.

18.7.8 Laydown and Stockpile Areas

The following facilities are part of the site support infrastructure during construction, for expansions and for closure:

ROM Temporary Storage Facility

A temporary stockpile pad will be prepared by removing the topsoil and unsuitable material and preparing a level pad to accommodate preproduction ore. Contact water will be collected and treated via a passive wetlands treatment and settling pond before discharging to the environment.

Topsoil & Unsuitable Material Stockpile

The topsoil and overburden removed during the site earthworks for the Project will be stockpiled on two different stockpiles as shown in Figure 18-1. They will be used for the Project reclamation at closure to recover disturbed areas.

Aggregate Borrow Pit

An aggregate borrow pit will be developed to provide construction materials for the Phase 2 development of the earthworks platforms for the site infrastructure.

Aggregate Production Pad

The aggregate processing for construction materials will be set up South of the Portal pad. It is planned to be an area for contractor supplied equipment to process and screen the aggregates for the site construction.

18.8 Off-Site Infrastructure

18.8.1 Port

The proposed port location for the Project is the Port of Esmeraldas. The Port of Esmeraldas is approximately 208 km by road from the Cascabel concession and road access is via good quality sealed roads right to the mine entrance. The port itself is located within the city of Esmeraldas and is bordered by industrial and residential areas.

The Port Authority of Esmeraldas (*Autoridad Portuaria de Esmeraldas - APE*) is the current owner and operator of the port facilities at Esmeraldas. The port of Esmeraldas is the only port facility in Ecuador which is still managed by the Ecuadorian government.

The proposed port operation at the Port of Esmeraldas will receive copper concentrate delivered via ore haul trucks from the Alpala process plant. The ore haul will be by contractor. The facility will receive the concentrate, store the material prior to shipment and then load concentrate onto ships as required. The port operations will consist of the following facilities:

- Concentrate storage shed and reclaim equipment
- Mobile ship loading
- Power supply and distribution
- Offices, control room and workshop
- Utilities – water and air systems

18.9 Power and Electrical

18.9.1 Power Supply

The site power will be supplied from new hydroelectric power projects near the site. Multiple hydroelectric projects are currently in the advanced planning stage, with a total capacity of 200 MW having been identified in the local area. The Project plans to participate in these projects and secure the supply of power from them. Additional power from solar is being considered but is not developed enough to incorporate into this study.

Rates for electricity in Ecuador are set by the government's regulatory body, ARCONEL. Rates for self-generating projects are lower than standard commercial rates, influenced by factors like electricity production volume, consumption by the self-generator, potential surplus for public use, project location, type of electricity, and others. Exact rates can only be determined through formal discussions with ARCONEL after signing a Confidentiality Agreement (CA).

Recent private sector-funded projects have achieved reductions of \$0.01 to \$0.02 off the commercial rate of 0.075 cents /kWh. Participating in a self-generation project could achieve a rate between \$0.055 and \$0.065 cents, equating to a 13.33% to 26.67% reduction. A conservative case of \$0.065 /kWh is considered for the Project, and with a contribution of \$4 million of CAPEX to the hydroelectric projects to qualify for self-generation status with the government.

A single circuit 138 kV overhead (OH) transmission line from these new hydroelectric projects, within 20 km of site, will provide power to the Cascabel project site. The transmission line will be approximately 17 km long. It is anticipated that the Project will install a substation and appropriate switch gear at the hydroelectric generating station for the 138 kV power supply. The Project also contemplates a transmission line to the government grid in order to sell back excess power or to use the government grid if necessary, in an emergency.

18.9.2 Power Distribution

The Cascabel project will utilise the following voltages:

- 138 kV feed to the main substations at the Alpala substation
- 13.8 kV for reticulation to the mine infrastructure and process plant
- 13.8 kV for emergency generation
- 4.16 kV for medium voltage motors
- 480 V for process equipment and motors
- 220/480 V for general light and power

18.9.3 Emergency Power

The emergency generators will be either diesel or heavy fuel oil (HFO). The generators will deliver the rated power at 13.8 kV and the generators will be arranged to operate in parallel to produce a maximum of 18 MW. A load shedding scheme will be implemented when the main grid connection is offline due to a fault or maintenance that will progressively shut down noncritical areas and connect power to critical drives and systems.

18.10 Communications

18.10.1 Data Systems

A fibre optic cable network will be installed to operate as a backbone data communication system for the plant, mine, and administration areas.

This main network backbone will include redundant paths in different routes in order to ensure permanent connection between system nodes. Multi fibre optic cables will be specified to separate the dedicated fibre systems and ensure speed and reliability of data from each system.

The main paths specified for the fibre optic backbone are:

Plant/Primary Crushing: An overhead fibre optic cable installed next to the 13.8 kV transmission line going to the mine portal then onto the main decline conveyor to the primary crushing area.

Plant/MIA: A fibre optic cable installed next to the 23 kV transmission line going to the MIA connected into the mine control room.

The general data and telephone network will have dedicated fibre to allow all facilities to be integrated and to be able to use IP telephony. The data and telephone network will include wireless routers in the administration areas to provide a wireless data network (Wi-Fi).

18.10.2 CCTV

The process plant, TSF, port and mine will utilise a CCTV system to provide coverage of the operating areas of the site. A separate CCTV surveillance system will also be installed for use by the security personnel to monitor the periphery of the mine site.

The CCTV system will consist of movable PTZ cameras equipped with protective housings for outdoor industrial environments, and the dome type for indoor use.

All cameras will include native IP technology and high-definition resolution transmitting over optical fibre links in a video traffic network, with security protocols allowing access to this information only to authorised personnel.

18.11 Comments on Section 18

Several tailings storage facility locations have been identified as part of this study and previous studies, offering flexibility to the Project. These options could accommodate changes to the mine plan for the Project, most notably the on-concession tailings storage options, which would become more advantageous should open pit mining of either the Alpala or Tandayama deposits be considered in future design iterations.

The location of the distal tailings storage facility option included in this study has presented challenges in undertaking the study, as access to the site to collect baseline data was limited. Therefore, it is imperative that a clear pathway to land access be developed to allow for the collection of baseline data for the proposed site(s), enabling the designs to be developed in the next phase of study.

The restricted access to the coastal plains site has introduced risks to the designs. Engineering judgment had to be applied due to the lack of available physical site data. As a result, modifications to the design may be required in later stages. Further details on these risks are presented in Section 25.16.5.

19 Market Studies and Contracts

19.1 Introduction

The Project will produce copper-gold-silver containing concentrate. SolGold commenced engaging with potential off-takers in 2020 and engaged Wood Mackenzie to prepare an independent market report for sale of concentrate for the indicative terms applied in the financial model.

For the purposes of this PFS, SolGold did not update the terms discussed with potential off-takers, nor did it solicit an updated market report from Wood Mackenzie as the indicative terms have not materially changed.

19.2 Market Studies

Metallurgical testwork provided by SolGold for the Cascabel project concentrate indicates that it is a clean, precious metal enriched concentrate containing very low levels of deleterious elements; the project's concentrates can therefore be expected to be in strong demand globally. The key deleterious elements in the concentrate produced from the Cascabel project are expected to be lower than the average of Grasberg, Oyu Tolgoi, Collahuasi and Escondida, particularly for arsenic, mercury, and zinc. As such, it is expected that Cascabel project concentrate will not incur any penalties and will be viewed as desirable by custom smelters as the material can be used as a diluent to blend with more complex copper concentrates.

The gold content of the Cascabel project concentrate is likely to be attractive to smelters in Japan, Korea, and Europe where recoveries are greater than payables. This is particularly the case in high gold price environments, as currently experienced, since "free units" are of much higher value than in a low gold price environment. Indian smelters can also be expected to be interested due to the duty gains on gold in concentrates versus gold in bullion. All of this will heighten demand, as high recovery smelters look to replace Grasberg's (Freeport) large volumes and high gold concentrate if removed from the custom market when its new smelter at Gresik, Indonesia is built.

The difference in payability for silver (Ag) in concentrate between European and Asian markets suggests a preference for the sale of Cascabel project concentrate to Asian markets, given the expected silver content.

Concentrate shipping costs from 2030 are estimated at between $25 /t for a 30 kt parcel to $63 /t for a 10 kt parcel on a long-term basis. Alternatively, with large volumes of copper concentrate regularly shipped from the west coast of Latin America, particularly Chile, there could be an opportunity to leverage freight through parcel shipments. Parcels of 10 kt from Cascabel could be co-shipped with other exporters to make up the cargo for 20-30 kt Handysize or Handymax vessels. The geographic advantage of the Cascabel project, located to the north of the Chilean and Peruvian copper fields, means shipping could be at a discount to those southern operations.

For copper concentrate, typical payabilities are shown in Table 19-1.

Table 19-1: Cascabel project typical metal payabilities in copper concentrate

Content	China	Other Asia	Europe
<30% Copper	-1	-1	-1
Silver			
<30 g/t	None	None	None
30-99 g/t	90%	90%	Deduct 30 g/t Pay balance
Gold			
<1 g/t	None	None	None
1-3 g/t	90%	90%	Deduct 1 g/t Pay balance
3-5 g/t	94%	95%	Deduct 1 g/t Pay balance
5-10 g/t	95%	96%	Deduct 1 g/t Pay balance
>10 g/t	96%	97-98%	Deduct 1 g/t Pay balance

Source: SolGold, 2022

Long-term copper concentrate treatment and refining costs (TCRC) were analysed, and multiple scenarios investigated. In view of these scenarios, and in consideration of directional feedback provided to SolGold from the market, a long-term treatment cost of $79.00 /t and refining cost of $0.079 /lb were applied in the financial model.

19.3 Commodity Price Projections

In consideration of the specific attributes of the Cascabel project including size, geology, and a planned life of mine, and in following guidance outlined by the Canadian Institute of Mining, Metallurgy and Petroleum, appropriate long-term (2029+) prices of $3.60/lb for copper, $1,700/oz for gold and $19.90/oz for silver are supported as reasonable commodity prices for Cascabel project reserve modelling.

For the purposes of the financial analysis, the latest market and street consensus forecasts were used, resulting in long-term metal prices as follows: $3.85/lb for copper, $1,750/oz for gold and $22.50/oz for silver.

19.4 Contracts

19.4.1 Sales and Marketing Contracts

During the 2020 fiscal year, SolGold started engaging with leading copper smelters and established commodity traders to assess the extent of interest and commercial value of the Cascabel project concentrate which based on metallurgical test work to date contains:

- High copper
- High precious metals
- Low deleterious elements
- A well-balanced combination of sulphur, iron and copper

Based on the high level of interest in this quality concentrate, SolGold subsequently invited commodity traders to submit an initial Expression of Interest, covering:

- Their intended sales and marketing strategy for Alpala concentrate
- Proposed offtake volumes and tenure
- Comprehensive commercial terms
- Ability for short- and long-term financial support

SolGold has to date received ten qualifying Expressions of Interest (EOI). Demand for the Cascabel project concentrate from traders was significantly in excess of planned production volumes; as such, SRK remarks that it is possible that metal payabilities, precious metal refining charges and payment terms could be finalised on better terms than previous conceptual assumptions.

SolGold has also received material offers of funding in exchange for offtake from a number of traders. These include the provision of both short-term and longer-term capital with proceeds available for studies, mine construction and cost overruns as well as working capital during ramp-up.

No offtake agreements have been entered into at the time of this report and the QP has not reviewed any of the Expressions of Interest received by SolGold.

19.4.2 Other Contracts

No other contracts are currently in place for the Cascabel project.

19.5 Comments on Section 19

The QPs have reviewed the information provided by SolGold on contracts and note that the information provided is consistent with the source documents used.

20 Environmental Studies, Permitting, and Social or Community Considerations Impact

20.1 Introduction

SolGold has made considerable efforts to undertake environmental studies and community engagement in order to facilitate the advancement of the Project. Several environmental baseline studies have been initiated in anticipation of eventually permitting operational mine development with an Environmental and Social Impact Assessment (ESIA). The following presents a summary of the environmental aspects, permitting and social or community impacts of the work program to date.

20.2 Project Permitting

20.2.1 Legal Framework

Mining activities in Ecuador are mainly regulated by the Ministerio de Energía y Minas (Ministry of Energy and Mines) (MEM), the Agencia de Regulación y Control de Energía y Recursos Naturales no Renovables (Regulation and Control Agency of Energy and Non-Renewable Resources) (ARCERNNR), and the Ministerio del Ambiente, Agua y Transición Ecológica (Ministry of Environment, Water and Ecological Transition) (MAATE). The principal environmental laws that apply to the mining industry are the Constitution, the Ley de Minería (Mining Law), the Reglamento Ambiental para Actividades Mineras, or RAAM (Environmental Regulation for Mining Activities), the Texto Unificado de la Legislación Secundaria del Ministerio del Ambiente, or TULSMA (Unified Text of Secondary Environmental Legislation), the Ley Orgánica de Recursos Hídricos, Usos y Aprovechamiento Del Agua (Water Use and Exploitation Act), and the Código Orgánico del Ambiente, or COA (Environmental Code), which entered into force in April 2018 and encompasses all the environmental legislation in one single body of law.

The MAATE issues an environmental licence and water authorisation for mining following approval of an Environmental and Social Impact Assessment (ESIA) and Environmental Management Plan. The Ministry of Energy and Mines is responsible for mine planning in Ecuador, including the negotiation of contracts for the exploitation of minerals. ARCERNNR is responsible for supervising mining activities. Other permits required for mining activities include those for explosives use from the Ministerio de Defensa Nacional (Ministry of National Defense), special labour shifts from the Ministerio del Trabajo (Ministry of Labour), fire department, and construction permits (from ARCERNNR and the municipalities). A summary of the major permits required for the construction and operation of the Project is provided in Table 20-1.

Table 20-1: Schedule of major permits for construction and operation

Prior to Construction	
Environmental License (EA) - Exploitation	Ministry of Environment, Water and Ecological Transition (Quito)
Water License - Industrial Use	Ministry of Environment, Water and Ecological Transition (Quito)
Authorisation to Build a Tailings Storage Facility	Ministry of Energy and Mines
Explosive Transportation, Storage, and Use Permits	Armed Forces Department of Arms Control
Quarry / Borrow Permits	Local Government
Fuel Purchase Permit	Regulation and Control Agency of Energy and Non-Renewable Natural Resources
Approval of Camp Specifications	Regulation and Control Agency of Energy and Non-Renewable Natural Resources
Mining Contract	Ministry of Energy and Mines
Prior to Operations	
Registration as a Hazardous Waste Generator	Ministry of Environment, Water and Ecological Transition (Quito)
Possession and Use of Controlled Substances	Technical Secretariat of the National System of Professional Qualifications
Registration of Hazardous Chemical Storage and Use	Ministry of Environment, Water and Ecological Transition (Quito)

20.2.2 Environmental and Social Impact Assessment

In order to comply with the ESIA submission necessary to obtain mining permits, SolGold will prepare and submit an ESIA with the following components:

- Legal framework
- Detailed description of the Project, including an alternatives analysis
- Determination of the Area of Influence for all planned Project infrastructure on the environmental and social landscape
- Characterisation of the physical and biological baseline condition
- Characterisation of the socio-economic baseline condition
- Characterisation of the archaeological baseline condition
- Identification, prediction, and evaluation of environmental impacts
- Risk assessment
- Forest inventory and economic evaluation
- Citizen Participation Process

- Environmental Management Plan, which includes:
  - Mitigation Plan
  - Waste Management Plan
  - Communication, Training, and Environmental Education Plan
  - Community Relations Plan
  - Contingency Plan
  - Worker Health and Safety Plan
  - Monitoring Plan
  - Rescue and Protection Plan (for species of concern that need relocation)
  - Closure and Abandonment Plan
  - Rehabilitation Plan

The specific requirements for the Cascabel project ESIA will be elaborated by a qualified Environmental Consultant, following guidance from the MAATE.

The public has the right to participate in the environmental assessment of projects through consultations, public open houses, and other initiatives. The baseline environmental and social collection programs are "living programs" and expand along with enhanced project design. Additionally, all sampling will be conducted with transparent communications with communities, local residents and other stakeholders.

In addition to Ecuadorian requirements, SolGold will ensure that the ESIA is compliant with appropriate international standards. At a minimum, these would include the Equator Principles and the International Finance Corporation (IFC) Performance Standards and Environmental, Health, and Safety Guidelines.

20.2.3 Current Permitting

The Cascabel project conducts exploration under a valid Environmental License for Advanced Exploration activities (including drilling). The License was granted as Resolución 0618 by the Ministry of Environment in August 2013 following the submission of an EIA for the advanced exploration activities (Cardno Entrix, 2013). Subsequently, through a complementary Environmental Study, the environmental license has been updated and through Resolution No. MAATE-SCA-2023-0026-R of 31 October 2023, underground exploration activities are included.

The remainder of the concession was permitted as Initial Exploration under Resolución 757, which does not include drilling. However, approval for an EIA that expands the Advanced Exploration to the entire concession was issued by MAATE in Oficio MAATE-SCA-2023-3710-O in September 2023. The Environmental License associated with the EIA approval is expected in the coming months.

The current and forthcoming Environmental Licenses remain valid for the duration of the exploration and evaluation phases of the Project, subject to fulfillment of monitoring report submissions. The Ministry of Environment, Water and Ecological Transitions has similarly granted authorisation for the use and benefit of all the water used in exploration activities from local surface sources.

In fulfillment of the Environmental License requirements, the Project submits semi-annual reports to the MAATE that report on internal monitoring of water, soil, noise, air, flora, fauna, effluent, social, archaeology, training and occupational health and safety. These reports are supported by periodic environmental audits and site inspections by government authorities to demonstrate compliance.

SolGold has implemented a stringent Environmental Management Plan to conduct the exploration drill program. The construction of trails, roads, drill platforms or other infrastructure are assessed according to their potential impact on water courses, flora and fauna. Adjustments to the placement or alignment are conducted at the planning stage to minimise potential impacts, and where appropriate, sensitive plants and animals are relocated prior to clearing activities. The drill platform size is limited to 400 m^2 (or 480 $m^{2,}$ where a single platform shares more than one drill hole). It requires enhanced controls where infrastructure is located close to water or on a slope, including water diversion bunding, ditching, and blanketing with geomembrane. Trails are limited to a width of 1.5 m, and vehicle access roads are limited to a width of 6 m. A comprehensive monitoring plan of abiotic and biotic parameters is conducted semi-annually by qualified external consultants in order to confirm the efficacy of the mitigation program.

20.3 Environmental Studies

The Project is located in the biome unit of Bosque Siempreverde Piemontano de la Cordillera Occidental de los Andes (Evergreen Western Foothills Forest) (Ministerio del Ambiente del Ecuador, 2013). It is located within elevations of 300 m in the valley bottom to 2,200 m in the higher exploration zones. The topography is moderate to steep, incised by dendritic drainage complexes within the tributary watersheds of the Río Mira basin.

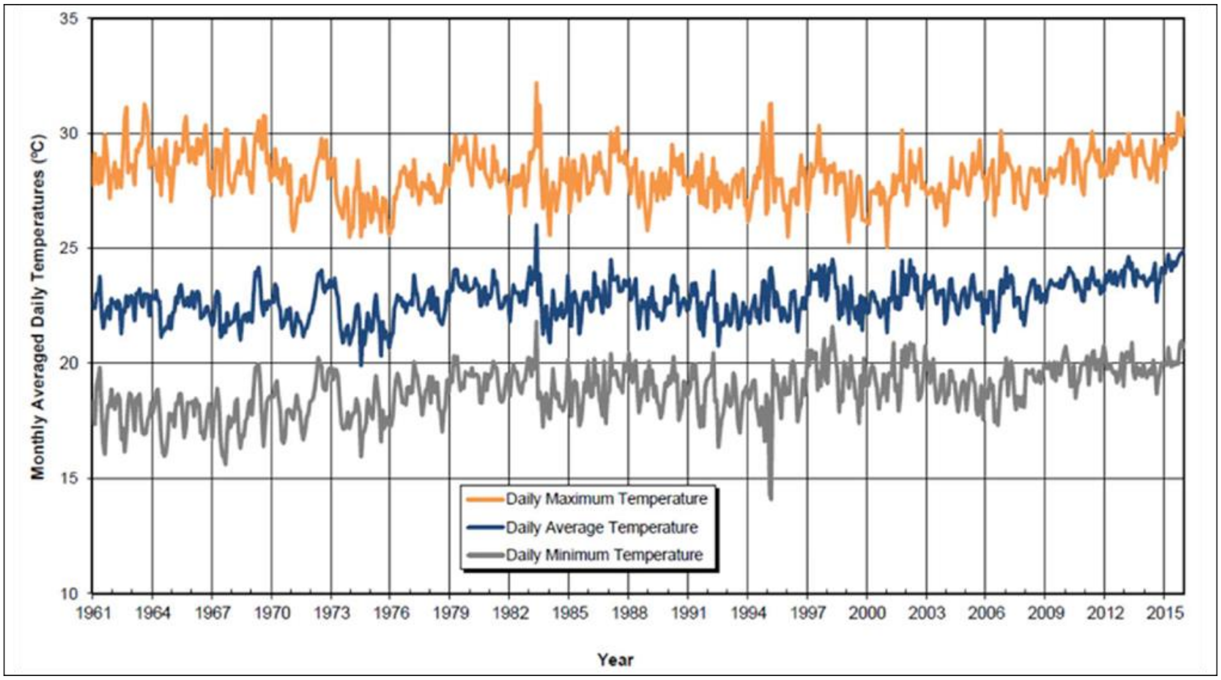
The ecosystem is formed by multi-strata evergreen forests with a canopy between 25 and 30 m in height. The area is under considerable pressure from agricultural activities, and much of the land has been cleared for farming, principally of cabuya, corn, naranjilla, citrus, guanabana, and varieties of bananas. Agricultural practices include the heavy use of pesticides, which has resulted in reduced soil capacities and effects to local drainages.

20.3.1 Meteorology

Two climate stations were installed at the site in April 2019 and continue to operate: one at the lower elevation Rocafuerte Camp and the second at the higher elevation Alpala Camp. A third station near the proposed tailings facility was installed in late September 2023 in Coastal Plains. Data from the most recent station has not yet been collated. Each station is compliant with World Meteorological Organisation (2010) standards for instrumentation and data collection, including automated data logging of temperature, relative humidity, wind speed, wind direction, and precipitation. Assessment of these stations was considered in the Knight Piésold, Cascabel Project Hydrometeorology Report, most recently updated in February 2020. Summary statistics include:

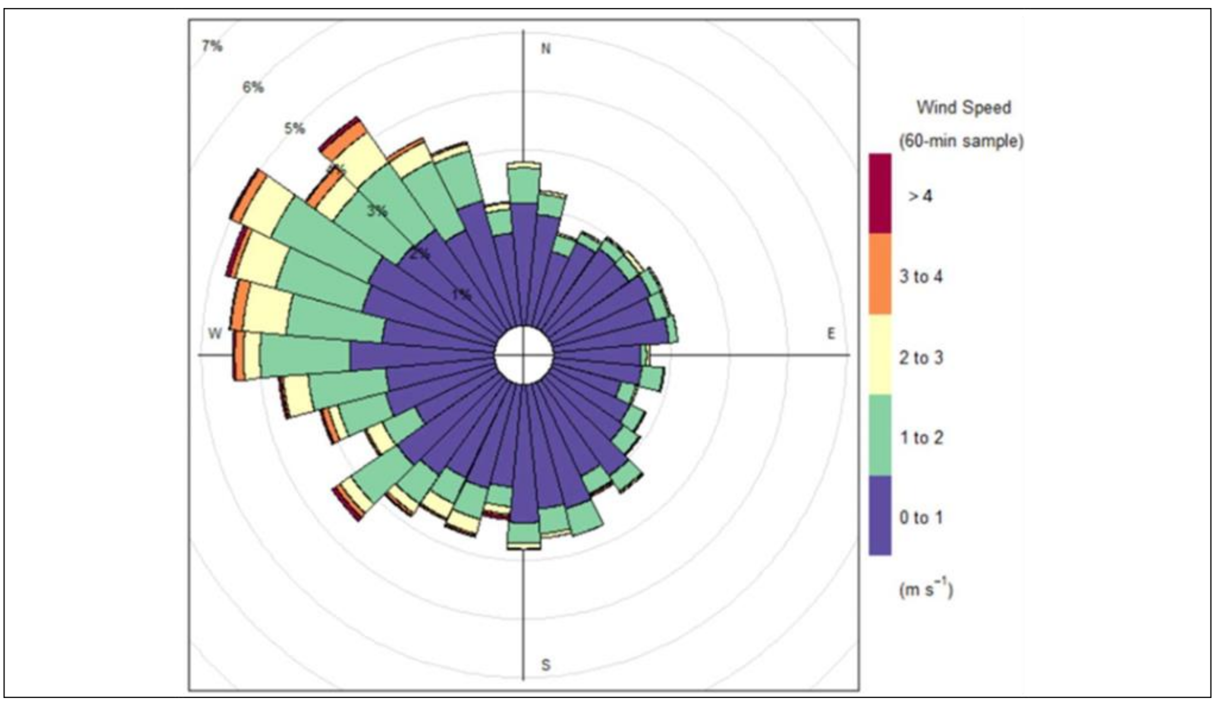
- The long-term mean annual temperature for the Rocafuerte station is estimated to be 22.8°C, with monthly mean temperatures ranging from a high of 26.0°C to a low of 19.9°C (Figure 20-1). The long-term mean annual temperature for the Alpala station is estimated to be 17.7°C, with monthly mean temperatures ranging from a high of 21.0°C to a low of 14.9°C.

- The monthly mean wind speed at the Rocafuerte station ranges from a low of 0.8 m/s to a high of 1.1 m/s, and at the Alpala station, it ranges from a low of 0.9 m/s to a high of 1.2 m/s. The prevailing measured wind direction at Rocafuerte is west and northwest, whereas at Alpala, it is northwest and southeast (Figure 20-2 and Figure 20-3).
- Maximum incoming solar radiation ranges from a high of 7.0 kWh/m^2 in May 2019 to a low of 4.2 kWh/m^2 in December 2019 at the Rocafuerte station and from a high of 6.0 kWh/m^2 in September 2019 to a low of 2.4 kWh/m^2 in October 2019 at the Alpala station.
- The monthly mean relative humidity at the Rocafuerte station varied between 87% in August 2019 and 95% in December 2019, while the monthly mean relative humidity at the Alpala station varied between 96% in August 2019 and 99% in December 2019. The relative humidity at the Alpala station is generally higher than at the Rocafuerte station.
- The recorded monthly total precipitation ranges from a low of 50.4 mm in June 2019 to a high of 502 mm in December 2019 at the Rocafuerte station and from a low of 138.6 mm in June 2019 to a high of 647 mm in December 2019 at the Alpala station (Figure 20-4 and Figure 20-5).
- Based on the available data, the estimated local orographic factor indicates an increase in precipitation of 4.2% per 100 m of elevation gain.
- The long-term average precipitation for the Rocafuerte station is estimated to be approximately 2,610 mm, with annual totals over a 55-year period ranging from a minimum of 1,574 mm to a maximum of 4,364 mm. Precipitation is unevenly distributed throughout the year, with most of the precipitation falling during the months of October to May and a pronounced drier season from June through September.
- The 100-year 24-hour precipitation at the Rocafuerte and Alpala stations are estimated to be 148 mm and 191 mm, respectively.



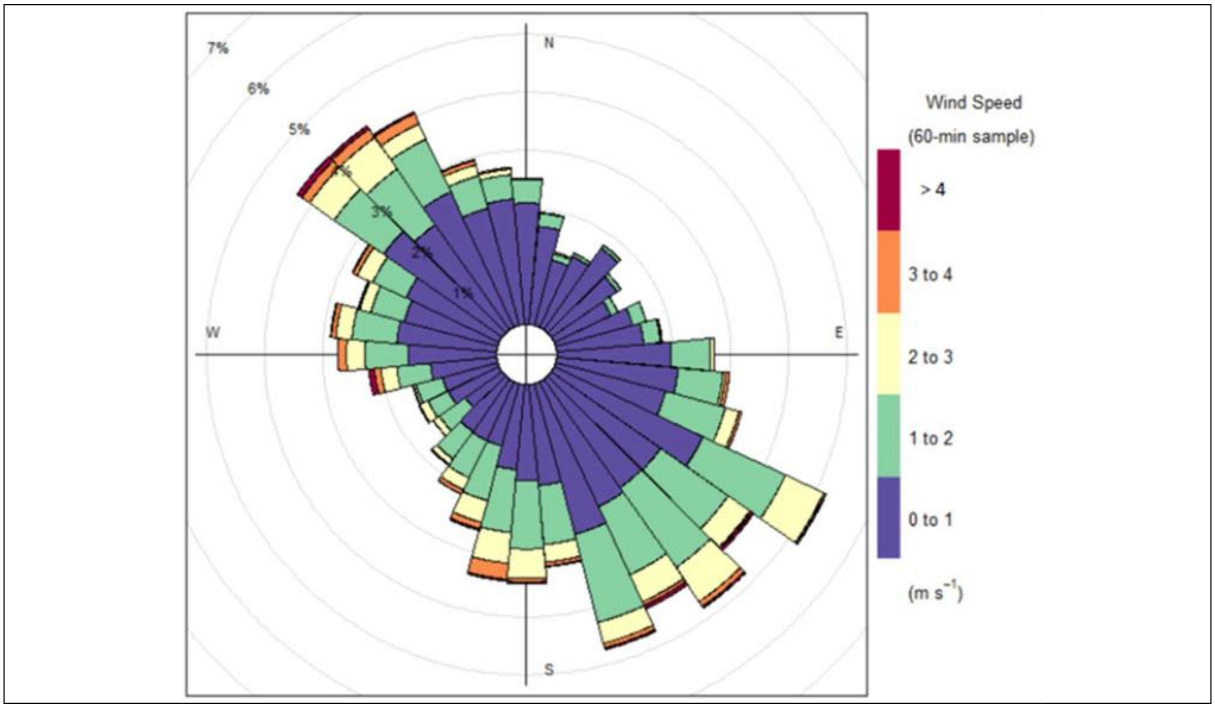
Source: Artica et al., 2022

Figure 20-1: Long-term synthetic monthly averaged temperatures at Rocafuerte station



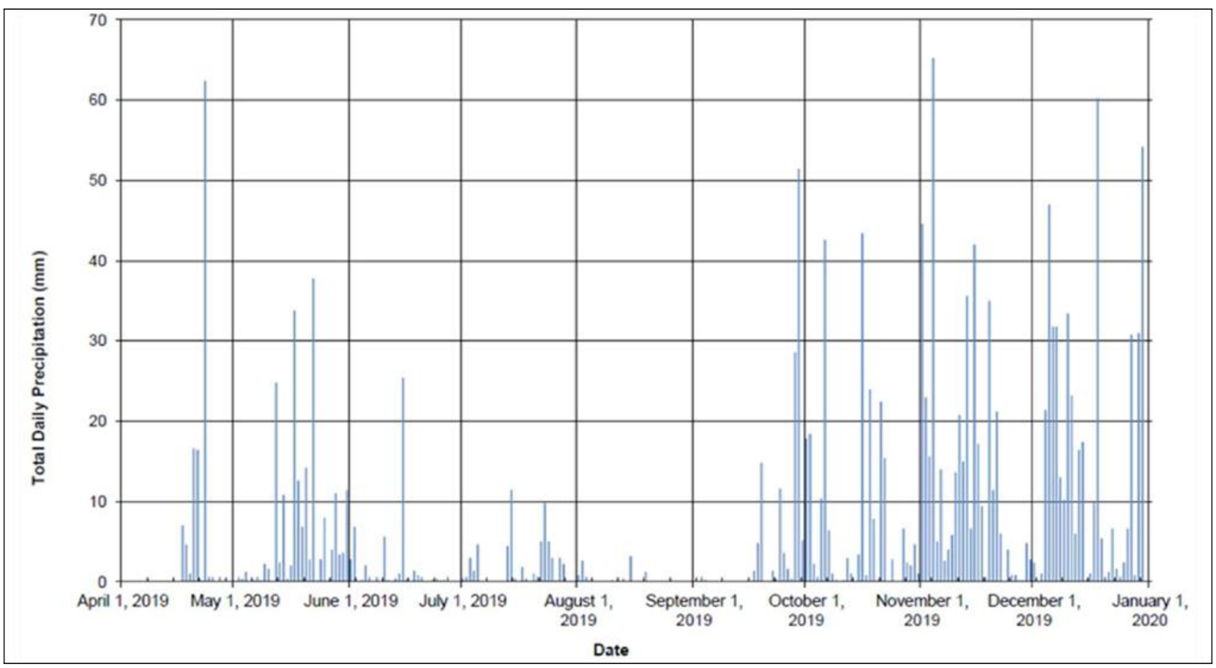
Source: Artica et al., 2022

Figure 20-2: Rocafuerte station wind rose



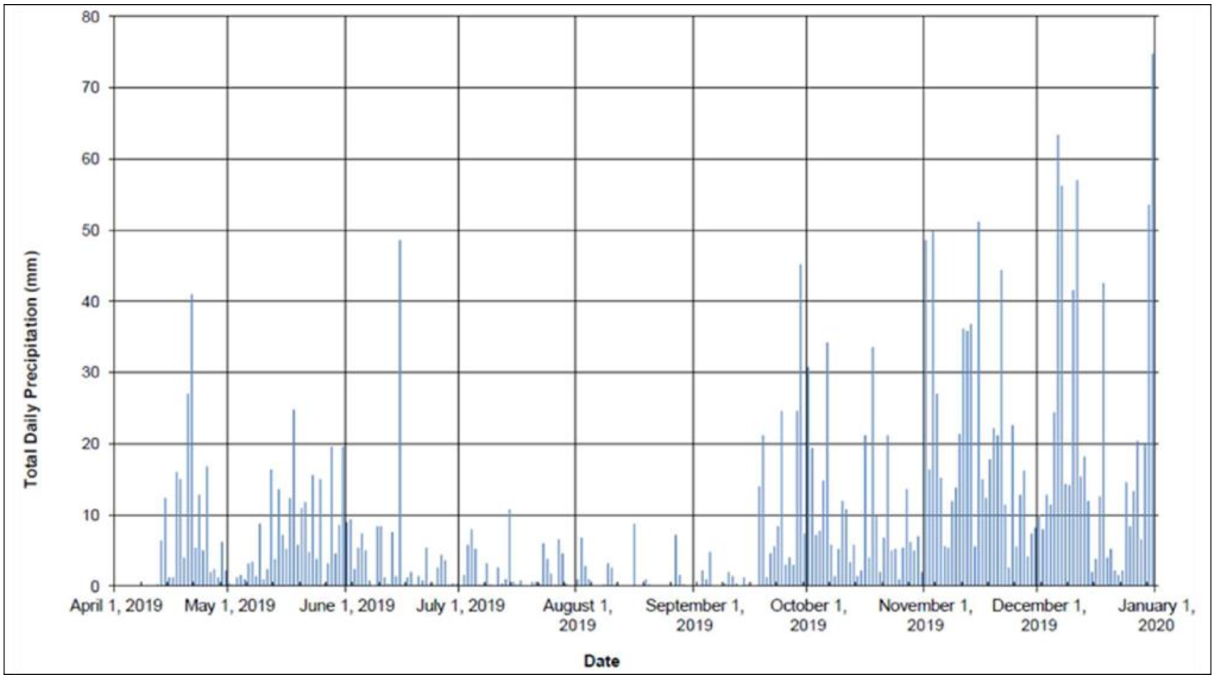
Source: Artica et al., 2022

Figure 20-3: Alpala station wind rose



Source: Artica et al., 2022

Figure 20-4: Total daily precipitation (Rocafuerte station)

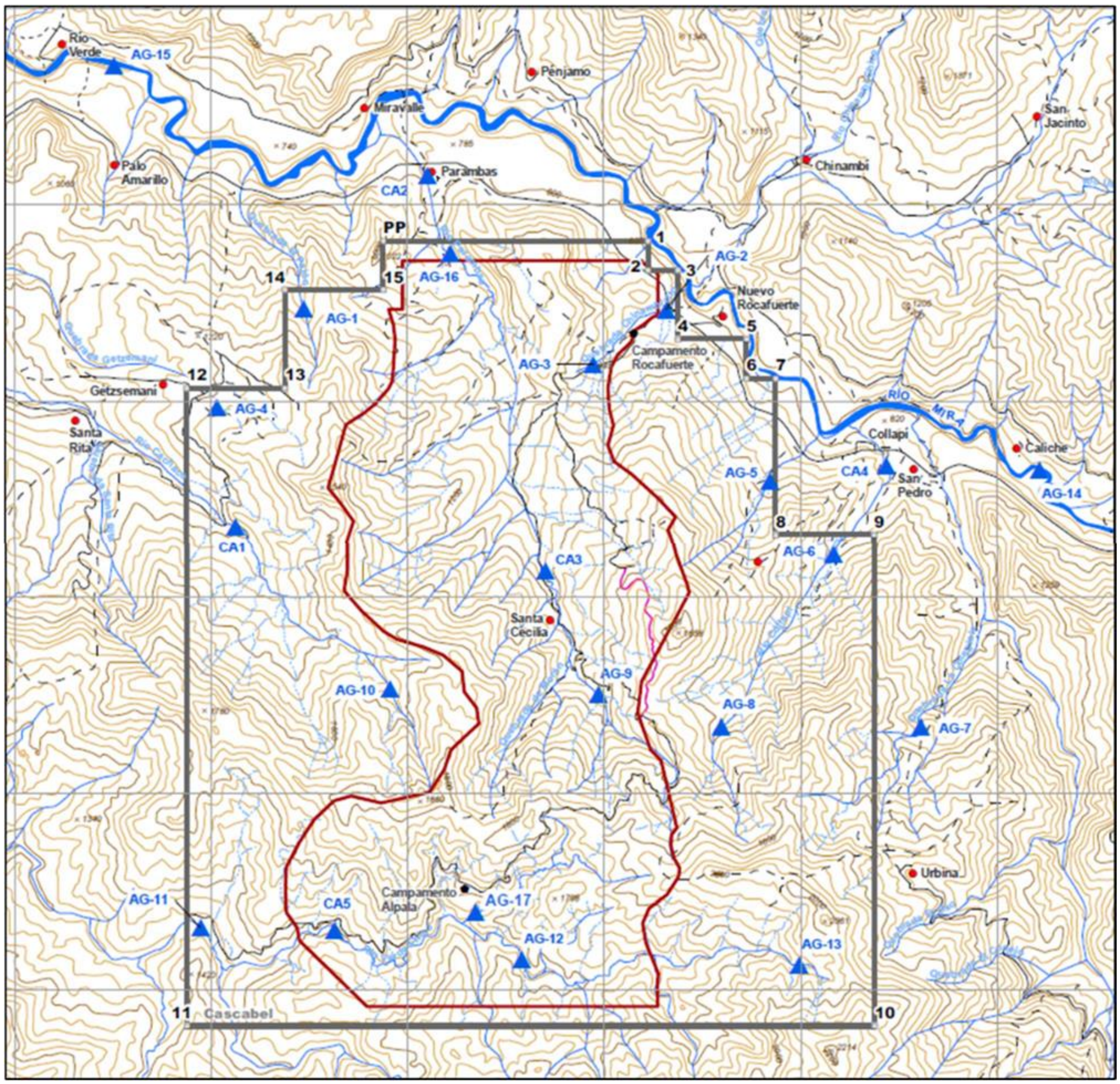


Source: Artica et al., 2022

Figure 20-5: Total daily precipitation (Alpala station)

20.3.2 Surface Hydrology

Primary drainages to the Río Mira include the Parambas, the Río Verde, and the Río Cachaco. Several other rivers and streams also drain the area of the concession and its surrounds (Figure 20-6). Flows in these catchments are bi-modal, with peaks seen in association with the rainy seasons of December and March. Average high flows range from 1 to 3 m^3/s, and during low flow or dry seasons can become dry or almost dry. The Río Mira, as the central drainage of the area, maintains flow year-round, with an average annual flow in excess of 40 m^3/s. During storms and periods of high rainfall, flows can peak at several orders of magnitude higher than their average high flow.



Source: Artica et al., 2022

Figure 20-6: Area drainages and water quality sampling locations

To complement the climate data studies, an enhanced surface hydrology program was implemented to develop a fulsome hydrometeorological dataset. This included the installation of automated hydrometric stations at five locations to record continuous water level data. The collection of stage level and discharge data at the hydrometric stations allows for the development of rating curves, which are necessary to develop a long-term synthetic flow series. The long-term flow series supports the water balance and water management design, water quality modelling, and subsequent effects assessment of the Project on fish and fish habitat.

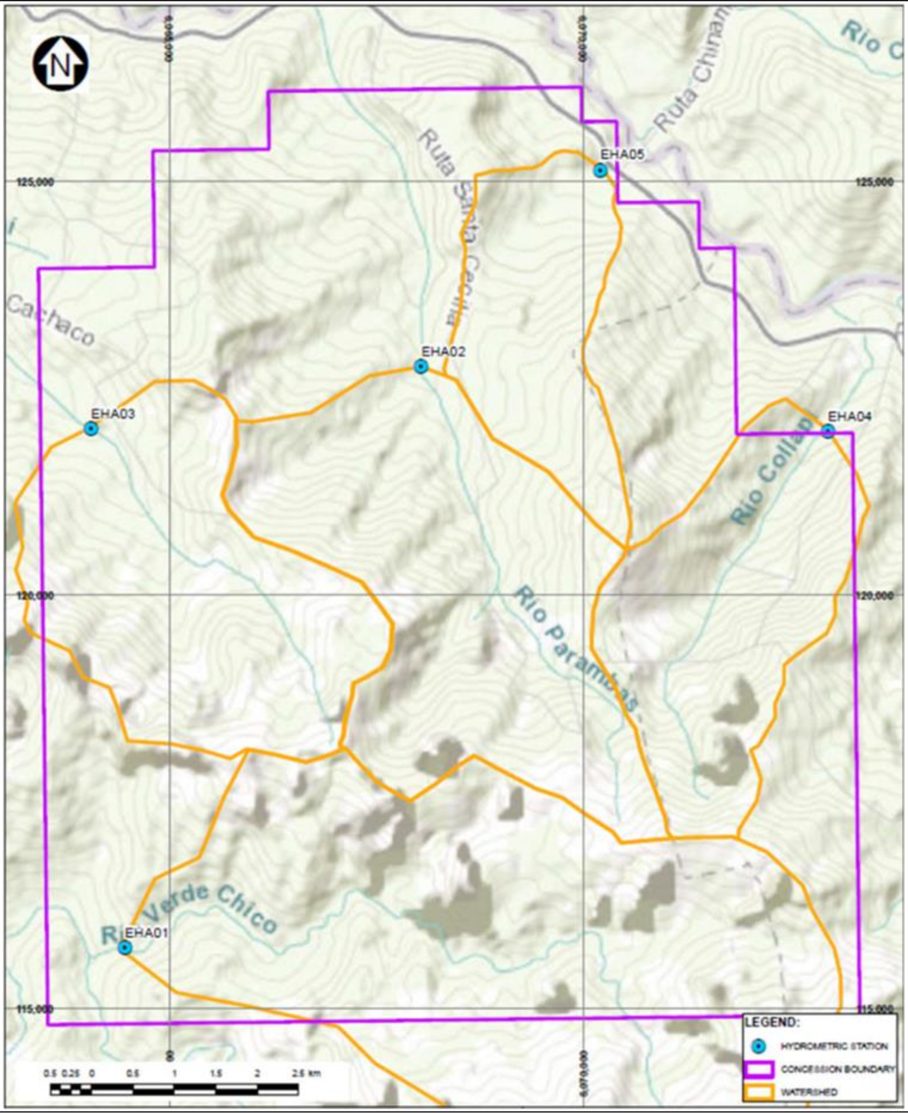
The current hydrometric monitoring network provides good spatial coverage of the concession area (Figure 20-7), including from the Rio Verde Chico watershed (Station EHA1), the Rio Parambas watershed (Station EHA2), the Rio Cachaco watershed (Station EHA3), the Rio Collapi watershed (Station EHA4), and the Quebrada Chinambicito watershed (Station EHA5). The five stations, constructed between 15 May and 11 July 2019, have collected water level data at 5-minute intervals since the installation. Additional hydrometric monitoring will be installed at the Coastal Plains tailings facility site. Available data collected to date and regressed against longer-term regional data from government sources suggest the following summary statistics:

- The mean annual unit runoff for watersheds within the Project area are estimated to be 54 L/s/km^2, which corresponds to a unit runoff depth of approximately 1,700 mm.
- The effective annual runoff coefficient for the basins within the Project area is estimated to be approximately 65%.

20.3.3 Surface Water Quality

As a requirement of the exploration permits, surface water quality samples were collected semi-annually at five sites (CA1 to 5) during early exploration. In anticipation of eventual ESIA development, surface water quality sampling was expanded in the Project area, starting with the upstream control and downstream effect sampling in and around the concession at 22 sites (Figure 20-6). Additional sites will be required to accommodate baseline study of the off-concession infrastructure, including the Coastal Plains tailings storage facility.

Water quality parameters analysed to date are generally found in concentrations below limits specified for human consumption, domestic use, and conservation of flora and fauna, as specified in the TULSMA. Some elevated levels of faecal coliforms were noted, especially near populated areas (e.g., at CA2 and CA3), where limited sewage treatment is available. Additionally, some peak values of aluminum, copper, iron, manganese, and zinc were also observed, and for specific samples, lead. It is important to note that these values are present in the baseline condition, and metals are associated with the nature of the deposit.



Source: Artica et al., 2022

Figure 20-7: Hydrometric monitoring locations

20.3.4 Geochemical Analysis

Geochemical analysis of representative waste rock has been planned to evaluate the potential for Acid Rock Drainage and Metal Leaching (ARD/ML). The results of these studies will aid in the development of mitigation and management plans. Test procedures will include static tests, total elements, acid-base accounting, mineralogy, and kinetic tests.

Tailings samples generated by the metallurgical laboratory program have been assessed for their geochemical properties. The University of Queensland completed testing on bulk (rougher) tailings material, the key findings of which were that they were classified as low Potentially Acid Forming, with copper being the main aqueous contaminant of potential concern.

Knight Piésold built on the University of Queensland assessment using two samples provided by SolGold (one rougher tailings sample and one cleaner tailings sample). The additional sampling determined that the rougher tailings are considered to be Non-Acid Forming, and no specific controls would be required with respect to the acid-producing potential of the rougher tailings sample. The cleaner tailings are, however, considered Potentially Acid-Forming and will require controls to prevent acid generation.

20.3.5 Air Quality

Air quality and noise data were collected as part of the 2013 EIA and have subsequently- been monitored as part of the exploration management plan at seven locations throughout the concession. Air quality measurements include PM_{10}, $PM_{2.5}$, and Total Settleable Particulates, as well as ozone, carbon monoxide, sulphur oxides (SOx) and nitrogen oxides (NOx). None of these parameters exceed acceptable government norms in the baseline condition.

20.3.6 Greenhouse Gas Emissions

Ecuador emits approximately 2.3 metric tonnes per capita of CO_2 equivalent and is shown to be increasing over the last decade (as of 2018, World Bank Data Store). The country's forests, particularly in the Amazon basin, may function as a carbon sink, offering some buffering capacity. However, as the conversion of forested land to agriculture and urban areas increases, the capacity of carbon capture of the forests is reduced. Ecuador has ratified the 2017 Paris Accord and agreed to target a 25% reduction of 2012 levels of Greenhouse Gases (GHG) by 2025.

The mining industry is energy-intensive and often a significant emitter of greenhouse gases. Vehicles account for between 50% to 80% of the total direct emissions within a mine. The technology associated with cleaner fuel sources and the electrification of fleets, coupled with behavioural protocols such as auto-shutoff vehicles and no-idling policies, can be implemented over time to reduce emissions.

Preliminary calculations based on projected diesel consumption of 100,000 litres per day at the Cascabel project indicate daily GHG emissions of approximately 1,000 tonnes of CO_2. This represents a conceptual assessment based on the US EPA Greenhouse Gases Equivalencies Calculator (EPA, 2023). A detailed GHG accounting will be completed as part of the ESIA and made public in due course.

20.3.7 Noise

Noise measurements collected at the site at nine locations throughout the concession are compared to TULSMA limits. In the baseline condition, the primary source of noise was determined to be flowing water, rainfall, or wildlife such as birds and insects. These sources produced peak noise measurements of up to 60 dB(A).

20.3.8 Flora and Fauna

Vegetation and wildlife of the Cascabel concession were initially described in the Environmental Impact Assessment prepared in 2013 for the exploration phase (Cardno ENTRIX, 2013) and continues to have regular surveys conducted every six months. Subsequently, expanded transects for flora and fauna were initiated within the concession area in anticipation of support for the EIA. The concession area is characterised as having a patchwork of remnant mature tropical forest interspersed with disturbed forest and cleared/agricultural land.

A total of 293 vegetation species have been identified within the concession, of which 15 are endemic. A total of 81 species of mammals were identified (of which 38 are bat species), 295 species of birds, 51 species of amphibians, 38 species of reptiles, and 28 species of fish. Many of these are endemic to the region. The highest concentration and diversity of species are found in the remnant patches of the original forest.

Ongoing monitoring of flora and fauna has identified species of conservation concern as defined nationally in the Listas Rojas De Especies De Vida Silvestre Del Ecuador (Red Lists of Ecuadorian Wildlife Species) and internationally as defined by the International Union for the Conservation of Nature (IUCN). A total of eight vegetation species, six mammal species, 11 bird species, and 29 amphibian and reptile species are listed as Vulnerable or Endangered. Additionally, the following species have been identified within the concession and are listed as Critically Endangered:

- *Ocotea pachypoda* (tree)
- *Pristimantis cf. loustes* (frog)
- *Pristimantis pyrrhomerus* (frog)
- *Astroblepus aff. ubidiai* (catfish)

It will be necessary to incorporate the habitat and conservation needs of these species into mitigation planning for the Project. If individuals or habitat of the Critically Endangered species are affected, compensation, in the form of preservation of habitat outside of the area of impact, may be required. Where compensation is required, international standards specify that a net gain must be demonstrated for the species of concern.

20.3.9 Environmental Program Development

Baseline Studies

Ongoing environmental studies contribute to the Project database and will inform continuous planning. As the Project definition increases, the environmental studies expand to ensure adequate baseline characterisation of all planned infrastructure. In preparation for the eventual ESIA and permitting requirements, SolGold expanded environmental studies in 2019.

A groundwater quality and quantity characterisation program has been initiated, which includes the determination of hydraulic properties of aquifer units and hydraulic head measurements to support the development of a water balance (e.g., water egress from the underground) and water management planning and identify surface water– groundwater interactions that could affect the use of water by local communities and aquatic biota.

The hydrogeological monitoring program was designed to have sufficient spatial and vertical coverage to characterise the three-dimensional groundwater flow regime at both the site and off-site in the receiving environment. Purpose-built monitoring wells were installed at ten sites to be used for water level monitoring and pump tests as required. Sampling at wells includes:

- Hydraulic testing (packer & slug tests)
- Water level monitoring
- Water quality sampling

The installation of the monitoring wells was initiated in early 2020 and completed later that year after an interruption during quarantine for COVID-19. Preliminary results are from wells in Sandy Silt to Clayey Silt Saprolite units. Hydraulic conductivity tests in saprolite indicate values of 10-6 m/s and near-surface bedrock values of 10^{-8} to 10^{-5} m/s. Water levels are near the ground surface in valleys and 20 to 50 m below the ground surface along topographic highs. A program to install additional water monitoring wells around other offsite infrastructure is ongoing.

The existing soil characterisation program will be expanded to include possible surface infrastructure footprint areas, such as the tailings storage facilities, processing plant, laydown areas, etc. The program will be designed to support assessment of general surface instability, erosion and/or sedimentation potential. Material characterisation of the soils will also be necessary to identify soil conservation measures (e.g., segregation, proper placement and stockpiling of clean soils and overburden material) for site remediation.

The spatial extent of the ongoing vegetation and wildlife studies will be expanded to assess natural habitats or species populations within the proposed footprint and to help direct reclamation planning. New baseline investigations will be conducted in areas proposed for infrastructure developments, such as the tailings facility, access road, concentrate shipping corridor, and transmission line. Fisheries and habitat surveys will be included in the faunal surveys within the rivers and creeks that will be within the Project's footprint and in areas downstream.

20.4 Social and Community Requirements

20.4.1 Local Socio-Economy

The mining concession is principally located within the Lita and La Carolina parishes within the Ibarra Canton, Imbabura Province.

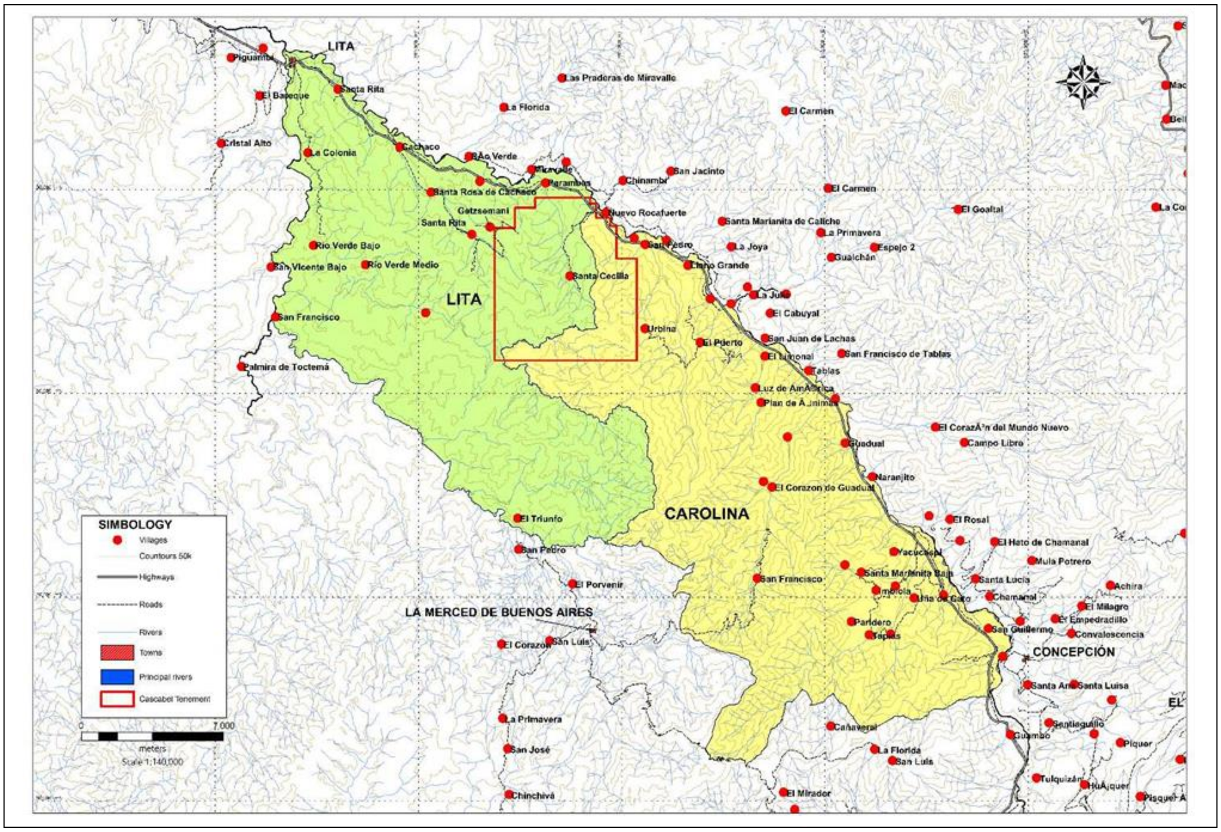
According to the 2022 census conducted by the National Institute of Statistics and Census (INEC, or Instituto Nacional de Estadísticas y Censos), the Lita parish has a population of 3,964, the La Carolina parish has a population of 3,258, and the Jijón y Caamaño parish has 1,961 inhabitants.

The primary access to the area is by Ecuador Highway 10 (E10, Transversal Fronteriza), which runs from Salinas to San Lorenzo. At Salinas, E10 joins with all other major highway systems in the country, connecting to major urban centres such as Ibarra and Quito to the south.

The communities most proximate to the Project are described in Table 20-2 and are illustrated, along with more distal communities in Figure 20-8.

Table 20-2: Communities in the Concession Area

Community	Characteristics
Santa Cecilia	Located in the centre of the concession, 280 people 74 families 75% of the families work for SolGold
Santa Rita	Located downstream of the Cachaco TSF, 80 people 36 families
Getzsemani	Access road located on Concession 324 people 91 families
Santa Rosa	Located several km downstream of Cachaco TSF, 140 people 42 families
Cachaco	Located near the Rio Mira at the base of the Rio Cachaco watershed, 166 people 65 families
San Pedro	Located downstream in the catchment of the waste rock storage area, 526 people 154 families
Parambas	Located immediately downstream of Parambas on E10, 422 people 132 families



Source: Artica et al., 2022

Figure 20-8: Communities in and around the concession

Each community in this area is relatively young, many having been founded in the 1900s. They are populated primarily with descendants of workers brought in the 1950s to construct the now defunct Ibarra – San Lorenzo railroad. Workers settled in the area's work camps, which now form local communities. The principal economic activity of the area is farming of naranjilla, a fruit-bearing shrub, and cabuya, an agave plant grown for the production of fibres used to make twine, as well as bananas and corn.

The Coastal Plains TSF is located in a remote area with minimal existing infrastructure. Residents are primarily subsistence farmers, with some reliance on artisanal mining. There are no towns or villages within the Coastal Plains TSF area.

20.5 Outreach Activities

SolGold operates a robust community engagement program within the concession area, with a permanent presence at the Project and a meeting room available to the public that is used as a base for outreach activities. Interviews with representatives of organisations, local governments, community leaders and members of the public in the local communities are conducted often to continually educate communities on the Project activities. Open houses and community presentations are conducted regularly, and community members are invited and trained to participate in ongoing environmental monitoring (Figure 20-9).

Source: Artica et al., 2022

Figure 20-9: Environmental monitoring with community member involvement

SolGold has made it a priority to emphasise good Corporate Citizenship, with a number of social programs implemented. This includes preferential employment and training for local residents. Over 300 Ecuadorian nationals are employed at the Project, representing approximately 97% of the total workforce. Financial support for school, health care and social events is provided, focusing on training, capacity building, and community strengthening.

SolGold constructed an agro-forestry nursery in 2013, which is managed by qualified agrologists. The nursery produces approximately 30,000 plants of native commercial and horticultural species, which are dispersed to the community along with professional advice to farmers, education on sustainable agriculture, and assistance with planting. Community members are also employed to work at the nursery.

Other recent initiatives include:

- Enhancement of chicken farming for communities to increase food quality and security (Figure 20-10)
- Food services contracts for SolGold camps and workers
- Improvements to health services, especially for children, pregnant women, elderly and disabled people
- Training for trades such as construction, mechanical, welding, etc.
- Post-harvest training and mechanization of coffee production and sales
- Pre-school education programming for young children
- Small business loan and mentoring program
- Transportation services of goods for sale (Figure 20-11)

These activities build on long-standing involvement in the community and successful programs developed in previous years.

Source: Artica et al., 2022

Figure 20-10: Program for chicken farming enhancements

Source: Artica et al., 2022

Figure 20-11: Assistance with transportation of market goods between the Cahuasquí and Jacinto Jijón y Caamaño parishes

As the COVID-19 pandemic developed in Ecuador and the Project area, SolGold was very proactive in assisting communities in planning and managing risk, sourcing and supplying safety equipment, and enhancing sanitation (Figure 20-12 and Figure 20-13).

Source: Artica et al., 2022

Figure 20-12: Distribution of 2,600 masks to the La Carolina parish

Source: Artica et al., 2022

Figure 20-13: Construction of sanitary washing stations in the Lita parish

These ongoing outreach activities will continue as part of the Environmental, Social, and Governance (ESG) activities and remain transparent as the Project develops, in keeping with public participation requirements of the Environmental Act.

In the near term, the expansion of engagement and outreach activities to the areas associated with the Coastal Plains TSF, the tailings pipelines, and concentrate shipping corridor is required to ensure that supporting studies and development proceeds uninterrupted.

20.6 Indigenous Communities

There are no identified indigenous individuals or communities within the concession or the immediate surroundings.

To the west and outside of the mine concession in the Lita parish and Esmeraldas province, there are settlements of the Awá people. The Awá are native to northern Ecuador and southern Colombia and have integrated mainly into modern agrarian society, yet still have unique customs and beliefs and are identified as Indigenous People. Currently, it is understood that the Coastal Plains TSF is not located within an area of influence for the Awá.

SolGold has plans to commission an anthropological study to examine the ability of the Indigenous People to conduct their way of life, including traditional customs and access to culturally significant sites. The planned study will include a participatory methodology incorporating criteria established by the Indigenous and Tribal Peoples Convention 169 of the International Labour Organisation (ILO, 1989).

20.7 Archaeology and Culture

An archaeological study was completed as part of the 2013 EIA, where sites pertaining to the pre-Incan Caranqui culture were identified in the Project area (Cardno Entrix, 2013). During the study, four sites containing ceramic fragments were found, further supporting the theory that there is archaeological value in the region.

In 2019, an archaeological survey was conducted on the entire concession, where areas of archaeological sensitivity were outlined and assessed as high, medium, low, or null. Within this framework, the company conducts exploration, monitoring and, if necessary, rescue with qualified archaeologists.

SolGold, through their community consultation program, has additionally identified culturally important sites in the region. Many of these are associated with their importance to the community, including water collection tanks, sports fields, and churches, but also include natural areas such as waterfalls.

To minimise potential impacts, all archaeological and cultural attributes will be incorporated into Project planning.

20.8 Closure Planning

20.8.1 Closure and Bonding Requirements

The Mining Law (Ley de Minería) and the Environmental Act specify that a Closure Plan is required as part of the ESIA and subsequently included in a Contrato de Explotación Minera (Mine Exploitation Contract) with the Ministry of Mining. The Closure Plan will consist of an estimated closure cost, upon which a financial guarantee or insurance policy in favour of the government will be required, which must remain in force until the final closure of operations.

The Reglamento Ambiental para Actividades Mineras (Environmental Regulation for Mining Activities, or RAAM) provides further detail on bonding requirements, specifically in Articles 35, 37, 124, and 129. The bond must be issued through an Ecuadorian financial institution and can be revised throughout the Project life to reflect changes to the closure and reclamation costs. A final and detailed closure and reclamation plan with an updated cost must be provided within two years of the final Project closure date.

The bond amount and financing mechanism will be negotiated by SolGold upon completion of the closure plan included in the ESIA and as part of the Mine Exploitation Contract. Costs associated with the bond are not included in this report.

20.8.2 Closure Planning

The detailed Closure Plan will be developed as part of the ESIA; the approach will be designed to ensure the long-term stability of the site's physical and chemical properties and to return the landscape to its pre-mining capability where possible. Specific closure items are considered below.

- Mine:
  - Access will be restricted to the subsidence zone with the use of berms, road closures, and warning signs to restrict access of personnel and vehicles and to prevent unsafe utilisation
  - Decommissioning and removal of mechanical and electrical equipment
  - Plugging of the main decline with reinforced concrete
  - Capping vent raises with reinforced concrete
  - Regrading and drainage management of the portal entrance area
- Process Plant and Surface Infrastructure:
  - Reagents and supplies will be removed and will be returned to suppliers, sold to other operations, disposed of in approved waste facilities, or transported to a certified company for disposal.
  - All foundations will be demolished and covered to approximate as closely as possible the pre-mining landscape topography.
- Tailings Storage Facilities:
  - The tailings facilities will have sufficient combination of flood storage and routing capacity to pass flood flows safely.
  - The tailings embankments will have an appropriate allowance to withstand settlement from a Maximum Credible Earthquake (MCE) event.
  - All Potentially Acid Generating (PAG) material will be isolated from possible oxidation, either in the underground, encapsulated in inert material, or stored sub-aqueously.
  - Tailings delivery pipelines will be decommissioned and removed.
  - Decommission and remove mechanical and electrical equipment
  - Decommission and remove the water return barges and pipelines
  - Access to the tailings facilities will be restricted with the use of berms, road closures, and warning signs to prevent access of personnel and vehicles.
- Port Facility:
  - Decommissioning and removal of mechanical and electrical equipment, as well as re-grading and revegetation of disturbed areas
- Electrical Transmission Line:
  - Equipment, conductors and other above-ground facilities for the electrical supply will be dismantled and removed.

The final closure methods will incorporate community involvement to ensure that end land use objectives are socially acceptable and in keeping with the broader land use planning of the area. Some Project infrastructure, such as roads, transmission lines, and port facilities, may be left in operable condition and transferred to government, communities, or private companies. The current closure cost has assumed that all infrastructure must be removed and rehabilitated or left in a stable, passive configuration.

Similarly, the current cost estimate does not provide an allowance for equipment sale upon closure or for salvage costs. These represent opportunities for future costing exercises.

Progressive rehabilitation is currently integrated into the exploration phase. It will be an important aspect of concurrent programs during operations in order to minimise final disturbance areas upon cessation of mining. The current program of successfully rehabilitating drill pads and other unused disturbance areas will be used as a basis of experience for revegetation during operations. The experience gained from the ongoing monitoring of rehabilitated plots and the utility of the agro-forestry nursery will be applied to the concurrent reclamation program.

Closure activities will include all active closures required to prepare the site for abandonment and a post-closure monitoring program. It is estimated that the active closure phase will require two years, and the post-closure monitoring will continue for a further ten years or until acceptable closure metrics are achieved.

20.9 Comments on Section 20

20.9.1 Baseline Studies

Baseline studies have been undertaken in the Cascabel concession only to date and are yet to commence for the remaining Project areas. The baseline studies at the Cascabel concession include meteorology, surface hydrology, surface water quality, geochemical analysis, air quality, greenhouse gas emissions, noise, flora, and fauna, archaeology, and social surveys.

20.9.2 Closure Plan

Closure planning has been undertaken to support the pre-feasibility study and will continue to be updated as the Project advances.

20.9.3 Environmental Considerations

Geochemical investigations indicate that the waste from tailings will be PAG, which has been considered in the design of the TSF.

Environmental and social investigations should be extended to all Project areas.

Water quality modelling should be extended to cover the likely closure scenario to confirm the flows and quality of water requiring treatment post-operations and to predict the number of years water treatment would likely be required once rehabilitation activities have been completed.

21 Capital and Operating Costs

21.1 Capital Cost Estimates

21.1.1 Basis of Estimate

The capital cost estimate meets the requirements for a PFS, consistent with AACE® International cost estimating guidelines for a Class 4 estimate. The estimate accuracy range of ± 25% is defined by the level of project definition, the amount of engineering inputs, the time available to prepare the estimate and the amount of project cost data available.

The capital cost estimate was compiled in United States dollars (USD) based on Q4 2023 prices and assumptions and is a combination of first-principles calculations, quotations from the mining contractors and equipment vendors, experience, and factored costs. The estimate has been compiled by SRK with inputs from consultants for their responsible scope of work:

- SRK Consulting: Mine development, production, mine services and maintenance, ventilation
- Allnorth: Dewatering, underground infrastructure, materials handling system
- Knight Piésold: Tailings storage, surface water management and closure costs
- Sedgman: Process plant
- JDS: Port facilities, surface infrastructure, tailings pipelines, power supply and distribution, and services and utilities
- SolGold: Owners' costs
- All: Indirect costs, EPCM, contingency

Table 21-1 below shows the consultants and their area of estimate contribution.

Table 21-1: Consultant contributions

Consultant	Scope
SRK Consulting	Mining: ■ Decline portal through to ore transfer into Primary Crusher ■ Mining specific surface and underground infrastructure – heavy vehicle workshop/s, fuel/lubrication storage, ventilation, air cooling, underground dewatering, utilities ■ Mobile equipment Mine roads
Allnorth	Underground Infrastructure: ■ Underground conveyors ■ Decline conveyors ■ Dewatering
Sedgman	Process Plant: ■ Secondary crushing ■ Stockpile ■ Tertiary crushing ■ Milling, flotation, thickening Tailings
JDS	Underground Infrastructure: ■ Primary crushing Surface Infrastructure: ■ Water (process, raw, storm, TSF decant, potable) ■ Power supply, switchyard, transformers, and distribution ■ Roads ■ Buildings – temporary and permanent ■ Mobile equipment ■ Camp TSF Infrastructure: ■ Pads and civil works ■ TSF power supply ■ Pipeline to TSF ■ Dam water return/discharge ■ Port Facilities: ■ Concentrate storage ■ Ship loading Utilities
Knight Piésold	Tailings Storage Facility (TSF) Site water management Site water dam Closure costs
SolGold	Owners Costs including: ■ Pre-production capital ■ Off-site workshop/warehouse/office facilities ■ Business readiness ■ Surface rights ■ ESIA process ■ Resettlement ■ Owners team ■ Security and medical services ■ Insurance Currency risk

Pre-production costs are all those costs before the ore is processed through the processing plant that has been assumed as upfront capital costs. Post-production costs are all those capital works after the commencement of ore processing that have been considered sustaining capital costs. Due to the long mine life, significant salvage values are not expected to be gained.

The following items/scenarios were not included in the capital cost estimate:

- Sunk costs incurred to date, including studies
- Geotechnical inconsistencies
- Changes to design criteria
- Work stoppages
- Scope changes or an accelerated schedule
- Changes in national law
- Changes in national duties
- Hydrological issues
- Environmental issues
- Hazardous waste issues

21.1.2 Mining

Basis of Estimate - Mining

The cost estimate was compiled in United States dollars (USD).

The underground mining capital costs were estimated using a combination of first-principles calculations, quotations from the mining contractors and equipment vendors, experience, and factored costs. The costs were developed for an owner operating scenario with the contractor use for vertical development and initial underground lateral development. No allowances for escalation, inflation factors or interest during construction were used in the estimates.

The mining capital cost estimate was based on the following:

- Preliminary project development plan
- Mining equipment list
- Budget quotes for the major mobile equipment obtained from equipment manufacturers
- Contractor equipment lease estimated based on the equipment budgetary prices and equipment depreciation periods
- Budget quotes from the mining contractors for vertical development
- SRK in-house database
- The regional cost information provided by SolGold, including:

  - Labour rates
  - Diesel fuel price of $0.85/litre for fuel delivered to the mine site
  - Electrical power price of $0.065/kWh as per all-in electricity cost on site

It was assumed that the following work will be done by a contractor:

- Initial mine development during the mine pre-production stage
- Vertical development, including ventilation raises, ore and waste passes
- Geotechnical and delineation drilling

The contractor will provide all labour, equipment and supplies except fuel and electric power, which will be supplied by the owner. A 10% markup has been applied to all contractor expenses, including labour, equipment, and materials, to account for the contractor's profit.

The estimated lease rates were applied to each unit in the contractor's equipment fleet required on-site during mine development on an annual period regardless of equipment utilisation.

The contractor's support equipment operating costs, such as grader, boom, fuel and mechanic trucks, personnel carriers, and supervisor vehicles, etc., as well as maintenance and site facilities, were not accounted for in the direct cost per unit of development. Those costs were estimated based on the assumptions for expected equipment utilisation and included in the contractor's overhead cost on an annual period.

Lateral development costs were estimated from the first principles, but vertical development costs were based on contractor quotes for raiseboring.

The raiseboring quote includes fixed costs for mobilisation to the Project site and demobilisation, drill setup and teardown, reamer installation, and variable costs, which include pilot hole drilling and reaming to the final size of a raise. Additional fixed costs were added for raise collar and drilling platform construction on the surface and underground excavation for the raiseborer chambers and reamers.

Underground capital development costs include contractor mobilisation and demobilisation costs and contractor's indirect costs. The contractor's indirect costs include equipment rental, auxiliary equipment operating costs, and indirect labour, including management, services, and maintenance.

The purchase of a permanent mining equipment fleet will be required for the mining activities performed by the owner. Mobile equipment costs were developed from estimated fleet requirements and vendor budgetary quotations. Unless provided by each vendor, the following assumptions were made for the additional expenses at the initial equipment purchase:

- 5% of equipment budgetary price to cover initial parts stock
- 4% of equipment budgetary price to cover freight and on-site assembly
- 2% of equipment budgetary price to cover equipment commissioning and training

Equipment life-cycle operating hours were based on manufacturer recommendations. The recommended life-cycle operating hours were used to calculate equipment replacement requirements.

Mining Capital

The capital cost for mining was estimated by SRK and is based on information provided in Section 16.

The cost model provides a first-principles estimate of the costs associated with accessing the Cascabel orebody and setting up a block caving mining operation. The cost model has been constructed based on physicals output from the project schedule.

The costs are based on the assumption that all construction phase works to access the mine and set up for a caving operation are completed by a mining contractor.

The decline advance rate and corresponding cost per metre to meet production requirements necessitate high-speed development rates for the initial access development. Advance rates to the 300 L will be 180 m per month.

21.1.3 Process Plant

The capital cost estimate for the process plant was undertaken by Sedgman and includes all costs associated with these areas.

A priced mechanical equipment list was the basis of the process plant estimate. Suppliers were approached for budgetary pricing of the major process equipment. Equipment quotes are based on global supply to maximise Project value.

Single unit rates have been used for material bulks and sourced from contractors and previous cost data for works performed in the region. Database rates were used as an alternative where site data was not available.

21.1.4 Port and Surface Infrastructure

The capital cost for the port and site infrastructure was undertaken by JDS and includes all costs associated with these areas.

Major mechanical and electrical equipment were identified for the infrastructure to support the Project. This equipment was estimated using various sources. Recent budgetary quotations, benchmark costs from similar recent projects, and values from the previous PFS where items did not change.

Single unit rates have been used for material bulks and sourced from contractors and previous cost data for works performed in the region. Database rates were used as an alternative where site data was not available.

Quantities for the estimate are taken off in the same categories as the commodity code listing. New commodities were only created for unique items or activities of significant cost. Where works were not yet engineered, quantities were factored or estimated using historical data for similar installations.

Quantities produced for the estimate are neat in place design quantities. Wastage was allowed for in the material rates.

Growth allowances by discipline have been applied to the estimate. The growth allowances applied to the estimate cover identified but undefined or poorly defined scopes of work in the estimate that, due to the preliminary nature of the engineering and lack of completed, accurate and detailed information, are not sufficiently defined or foreseen at the time the estimate was compiled.

Common earthworks rates were established for the Project and applied to all areas of the estimate. Clear and grub rates are based on heavily wooded and dense vegetation.

Construction material for roads and hardstands has been assumed to be available locally from borrow pits, reducing the requirement for imported material. Excavation rates were based on stockpiling of spoil on site.

Plant civil and earthworks quantities were measured from the engineered site layouts and 3D model. Cut/fill volumes and earthworks quantities were based on the topography and the following applied when developing them:

- Cut and Fill Slopes 1.5:1
- Topsoil Stripping and Grubbing Thickness 1m thickness
- Bedrock in area is 22m below surface based on INF-20-011 Borehole completed in 2021 - ENSA Report
- 40% of Saprolite Layer is Unsuitable and 60% Is Suitable for Fill Material
- Detailed excavation - 50% waste 50% usable backfill and 50% replacement with structural fill

The majority of the earthworks is required for the establishment of the roads, pads and water management structures. The quantities derived include for clear and grubbing of works area, striping of topsoil (1 m), bulk excavation for levelling of pads, drainage, and erosion protection. Roads have been designed and quantified. Process, raw water and run-off water pond excavations, sand bedding and HDPE liners have also been quantified. Limited geotechnical information was available during the Pre-Feasibility therefore common ground conditions are assumed. An allowance for rock excavation was included, including both rippable rock and with some requirement for drill and blast.

Concrete volumes were developed using site plan dimensions and database quantities for similar facility installations. The MTO's include total quantities of concrete and embedded metals. Concrete costs include labour, materials, and contractors' equipment. Mobilisation, demobilisation, and operation of a batch plant are estimated separately in contractor indirect costs.

Structural Steel and platework fabrication have been assumed to be sourced domestically therefore, local steel fabrication rates have been used.

Piping quantities were derived from site layouts and pump calculations. The piping MTO includes line sizing, material specification and length assigned to a work area, as defined in the work breakdown structure (WBS). Piping materials include HDPE, carbon steel and carbon HDPE lined. HDPE is used where possible. Small bore piping to 75 mm NB is included in the MTO. Pipe supports and anchoring are estimated where required. Current steel mill pricing from a pipe supplier was used for the tailings overland pipeline carbon steel material.

Major manual and control valves are allowed for in the estimate, based on recent budgetary quotations and purchases from other projects.

Major Electrical equipment for the power supply switchyard, Cascabel project main substation and 138 kV overhead transmission line was quantified and budget pricing obtained in the previous PFS and used for this estimate. All other electrical and instrumentation bulks were estimated based on recent projects of equivalent facilities. Minor electrical equipment related to vendor equipment has been included in the equipment cost.

All-in labour rate were used for installation based on local labour rates provided by SolGold and include labour and contractor consumables. The rate includes travel time, R&R leave, PPE, tooling, supervision, and home office support costs. Equipment rates were developed based on equipment fuel and usage costs for construction crews and applied based on labour hours required for the work.

21.1.5 Tailings and Water Management Facilities

The capital cost for the Coastal Plains TSF and water management was undertaken by Knight Piésold and includes all costs associated with the tailing's disposal and water management.

Quantities have been measured from the 3D models generated for the embankments, spillways, diversion channels and roads. No allowance has been included for growth of the quantities (escalation in volume above the design volume) as the design is slightly conservative and some optimisation of the design is expected in the next phase of study.

Unit rates were developed by Knight Piésold based on experience in design of similar projects.

21.1.6 Indirect Costs

Indirect costs for infrastructure include all costs needed to conduct the engineering procurement, and construction management (EPCM) services for the Project. These infrastructure and Project costs were calculated by JDS's estimating group. The main costs in this category are construction site services, equipment and crane rentals, temporary facilities and utilities, camp catering, third-party services, contractor indirect costs, spare parts, freight, and customs.

This estimate includes Owner's indirect costs calculated by SolGold.

Indirect costs have been included in each separate component of the capital cost estimate.

21.1.7 Sustaining Costs

Sustaining capital costs were estimated using the same estimation basis as outlined for the direct costs.

The sustaining capital is for planned future capital works for the Project, the tailings dam wall raises, mine equipment replacement and process plant expansion to a production rate of 24 Mtpa.

21.1.8 Closure Costs

The closure plan was organised into the main Project areas following the Work Breakdown Structure.

The timing of incurred expenses was estimated based on the expected reclamation and closure schedule and followed two main Project phases (Active Closure and Post Closure).

The schedule for this closure cost estimate includes the following:

- Active Closure phase (closure of the remaining mine infrastructure)
- Post Closure phase (monitoring of the mine infrastructure)

Estimates of ancillary costs for Mobilisation and Demobilisation, as well as Post Closure Monitoring and Maintenance were included in the cost model. These ancillary costs were proportioned into the overall costs for the main Project areas. The cost to perform each work item was estimated based on a quantity estimate multiplied by a unit cost estimate.

21.1.9 Capital Cost Summary

Initial capital costs are estimated to be US$1,554 million as summarised in Table 21-2 below.

Table 21-2: Initial capital cost summary

Area	Initial Capital Cost (US$M)
Mine	403
Process plant	262
Tailings storage facility	267
Port facility	17
Surface infrastructure	293
Owners costs	92
Contingency	221
Total	**1,554**

Expansion, sustaining capital and closure costs include the second process plant module, bringing the process plant capacity to 24 Mtpa, continued tailings storage facility development, equipment replacement and closure costs. Expansion, sustaining capital and closure costs are estimated to be $2,655 million. The total capital cost is therefore $4,209 million.

The LOM capital cost estimate is summarised in Table 21-3.

Table 21-3: Cascabel LOM project total capital cost summary

Area	Total Capital Cost (US$M)
Pre-Production Capital Cost	1,554
Sustaining/Expansion Capital Cost	2,573
Closure Cost	82
Total Capital Cost	**4,209**

21.2 Operating Cost Summary

21.2.1 Basis of Estimate

The overall Project operating cost estimate has been broken down into the following areas:

- SolGold: General and Administration
- SRK and Allnorth: Mine and underground infrastructure operations
- Sedgman: Process plant operations
- Knight Piésold: Coastal Plains Tailings Storage Facility construction, operations
- JDS: Surface Infrastructure (including camp) and Port operations

The operating cost estimate was compiled in United States dollars (USD) based on Q4 2023 prices and assumptions. The operating cost was prepared on the following basis:

- All equipment and materials will be new
- The labour rate build-up will be based on the statutory laws governing benefits to workers that were in effect at the time of the estimate

The following items were excluded from the operating cost estimate:

- Sales and marketing
- Foreign exchange, finance and interest charges
- Costs due to extraordinary currency fluctuations (e.g., materials sourced from overseas)
- Changes in Ecuadorian law
- Other duties and taxes (except as identified)
- Force majeure events
- Pre-operations training of personnel
- Transport and handling of concentrate and doré from the plant (included in the financial model)
- Freight estimates are based on vendor supplied freight quotations or in-house data
- No contingency is assumed
- No cost escalation (or de-escalation) is assumed

21.2.2 General and Administration

The General and Administration (G&A) Costs, as compiled by SolGold, encompass the following costs:

- G&A Costs: SolGold provided the basis for estimating the G&A costs; $0.80 per tonne milled has been applied to the production profile on a per year basis
- Camp Costs: JDS used SolGold's existing camp costs to arrive at the camp costs of $16.75 per man-day in camp. This cost includes accommodation, messing, transport to/from Ibarra, etc. The camp costs are based on an equal split roster, assuming each employee will spend 183 days in camp.

21.2.3 Mining

The underground mining operating costs were developed using a combination of first principles calculations, experience, and factored costs, in US dollars, based on the mining schedule developed for the Cascabel project.

The schedule includes:

- Capital development of the:
  - Materials handling system
  - Access to extraction and undercut levels
  - Haulage level
  - Declines to connect the production levels
- Preparation development to access the ore
- Ore development, including drawbells, undercut and any wall stripping
- Vertical development
- Draw column tonnes of ore
- Transportation of the broken rock from the production areas to the haulage level
- Transportation of the broken rock along the haulage level to the materials handling system
- Trucking the broken rock to surface during the early phases of footprint establishment

Underground Labour

A mining contractor will be used for the development during the pre-production period to allow time for the owner to recruit staff for the Project. During that period, the owner will provide overall Project management and contractor supervision.

The personnel requirements were estimated from first principles, based on the following:

- Mine development and production schedule
- A crew rotation of two 12-hour shifts per day with two crews working on site and one crew off

- 42:21 rotation for fly in/fly out workers and 5:2 rotation for residential workers

The final mine roster was estimated taking into account annual leave for vacation, sick days and training.

Summary Mining Operating Costs

Underground mining operating costs were developed based on an annual life of mine (LOM) schedule for the owner operating scenario. Productivity, equipment operating hours, labour and supply requirements, and costs were calculated for each cost activity, such as: mine operating development, production drilling and blasting, mucking, truck haulage, secondary breaking, ventilation, mine services, and maintenance. The cost of mine management and technical staff and operating and maintenance labour was estimated as separate cost items based on the staff roster.

The operating costs were estimated using a combination of first principles calculations, experience, and factored costs. Summary of the underground mining operating costs is presented in Table 21-4.

Table 21-4: Underground mining operating cost summary

Description	LOM Total US$M	Unit Cost US$/t
Production Drilling	$53.0	$0.10
Production Blasting	$38.5	$0.07
Production Mucking	$577.3	$1.07
Secondary Breaking	$349.6	$0.65
UG Truck Haulage	$40.5	$0.07
Crushing		
Conveying		
Ventilation	$117.1	$0.22
Dewatering		
Mine Services	$303.7	$0.56
Mine Rehabilitation	$43.9	$0.08
Definition and Geotechnical Drilling	$70.4	$0.13
Mine Management and Technical Staff	$220.9	$0.41
Mine Operating and Maintenance Labour	$243.8	$0.45
Total Mining Operating Costs	**$2,058.8**	**$3.81**

Notes: * without contingency. Totals do not necessarily equal the sum of the components due to rounding adjustments.

Basis of Estimate

The major assumptions used for the cost estimation include the following:

- Mine operating schedule:
  - For manual equipment operation: two 12-hour shifts per day including 2.0 hours of non-productive time accounting for shift change, equipment fueling and inspection, lunch and breaks during a shift, and 50 min as usable time in one operating hour.

  – For autonomous equipment operation: two 12-hour shifts per day including 1.0 hour of non-productive time accounting for shift change, equipment fueling and inspection, lunch and breaks during a shift, and 55 min as usable time in one operating hour.

- Labour: owner labour rates to be used for estimates were based on provided by SolGold and the similar underground mining projects in the countries of the same region
- Shift rotation: senior staff working 8-hour shifts, five days on and two days off and hourly staff working 12-hour shifts with three crews on rotation working 42 days on site and 21 days off site
- Diesel fuel price of $0.85/litre for fuel delivered to the mine site
- Electric power price of $0.065/kWh as per all-in electricity cost on site

The mine operating schedule input data used for estimation of productivities, equipment and labour requirements, and costs are presented in Table 21-5.

Table 21-5: Underground mining shift schedule

Description	Units	Quantity
Mine Operating Days	days/year	365
Shifts per Day	shifts/day	2
Shift Length	hours/shift	12
Shift Change	hours/shift	0.75
Equipment Inspection	hours/shift	0.25
Lunch / Coffee Breaks	hours/shift	0.75
Equipment Parking/Reporting	hours/shift	0.25
Subtotal Non-Productive Time / Shift	hours/shift	2.0
Usable Time / Shift	hours/shift	10.0
Shift Efficiency	%	83%
Usable Minutes per Work Hour	min/hour	50
Operational Efficiency (50 min in hour)	%	83%
Effective Work Time / Shift	hours/shift	8.3
Work Time Efficiency	%	69%

Material costs were based on estimated consumption of consumables and recent supplier's prices for drill and steel supplies, explosives, ground support and services supplies. Consumables costs were increased by 10% to account for miscellaneous use and material wastage. It was assumed that 25% of auxiliary ventilation and electrical cables will be recovered and reused, resulting in cost reduction.

Operating costs of major mobile equipment were estimated based on calculated operating hours and indicative costs for mobile equipment maintenance provided by vendors. The operating hours for auxiliary mobile equipment were estimated based on expected equipment utilisation.

Mobile equipment operating costs include maintenance consumables, tires, fuel, lube, and power. Those costs are part of mine development, production and services costs.

Underground Mining Operating Cost Estimate

Production drilling and blasting costs were estimated from first principles. These include drilling and blasting costs to create initial slots for undercut and costs for drilling and blasting of undercut and drawbell rings.

Two types of LHDs were selected for the following:

- 8.6 m^3 to be used for development and truck loading. Ore and waste from the faces of the development headings will be mucked to the remuck bays and from there will be loaded to the 51 t haulage trucks.
- 10.7 m^3 LHDs to be used for production mucking. Ore from the drawpoints will be trammed directly to the crusher dumping points. The mucking costs were estimated based on an average mucking distance on the extraction level from the drawpoints to the crusher's dumping points.

Secondary breaking costs were estimated based on the assumed amount of oversize material to be broken on the undercut level and in the drawpoints. The mobile fleet for secondary breakage activities includes water cannons, blockholers and mobile rockbreakers to deal with oversize material and drawpoint hangups. A stationary rockbreaker will be installed at the dumping point on the top of each crusher chamber and its operating cost is part of the primary crushing cost.

Trucking costs include waste trucking from development via the access decline to the portal. It also includes ore trucking from development when the underground crushing-conveying system cannot be utilised, as well as ore trucking from undercut level to a nearest crusher.

It was assumed that geotechnical and definition drilling will be performed by a drilling contractor. The geotechnical and definition drilling costs were estimated based on the assumptions made for the geotechnical and definition drilling requirements and the contractor cost per drilled meter.

An allowance for mine rehabilitation was made to account for drawpoint and general mine rehabilitation.

The mine services and maintenance costs include operating costs of the service mobile equipment fleet, allowance for mine services and maintenance supplies, and miscellaneous expenses.

Operating labour requirements were estimated for each mining activity. Indirect labour costs, which include mine management and technical staff, and mine services and maintenance labour were estimated based on the staff roster and labour rates provided by SolGold. The mine roster factor of 1.61 was estimated to account for additional labour based on shift days off and days of annual leave including vacation time, sick, training, and other duties. That factor was applied to the estimated on-site labour requirements to estimate the total mining labour requirements and labour costs.

A mining contractor would be used for the development during the pre-production period to allow time for the owner to recruit its own labour for the Project. Contractor labour and supervisory staff are not included in this section.

The labour rates for owner mine management and technical staff are presented in Table 21-6.

Table 21-6: Underground mining management and technical staff

Staff Description	Rotation	Hourly Rate ($)	Annual Rate ($)
Mine Management & Technical Staff			
Mining Team			
Mine Manager	5:2		$506,250
Mine Superintendent-Production	5:2		$135,000
Mine Superintendent- Development	5:2		$135,000
Technical Service Superintendent	5:2		$135,000
Senior Mining Engineer	5:2		$72,000
Mine Planning Engineer	42:21		$60,000
Mining Engineer	42:21		$36,000
Safety & Training Manager	5:2		$135,000
Safety & Training Officer	5:2		$60,000
Chief Ventilation	5:2		$114,000
Ventilation Engineer	42:21		$60,000
Ventilation Officer	42:21		$36,000
Chief Surveyor	5:2		$76,000
Senior Surveyor	5:2		$60,000
Mine Surveyor	42:21		$36,000
Surveyor Assistant	42:21		$14,400
Mine Clerk	5:2		$14,400
Geology/Geotechnical Team			
Chief Geologist	5:2		$135,000
Senior Geologist	5:2		$72,000
Resource Geologist	5:2		$60,000
Hydrogeologist	5:2		$60,000
Operational Geologist	42:21		$60,000
Grade Control Sampler	42:21		$18,000
Geotechnical Superintendent	5:2		$135,000
Senior Geotechnical Engineer	5:2		$72,000
Geotechnical Engineer	42:21		$60,000
Draw Control Technician	42:21		$36,000
Geotechnician	42:21		$36,000

The labour rates for owner operations management staff are presented in Table 21-7.

Table 21-7: Underground mining operations management staff

Staff Description	Rotation	Hourly Rate (US$)	Annual Rate (US$)
Mine Operational Management Staff			
Mining Operation Team			
Development Superintendent/Foreman	5:2		$135,000
Production Superintendent/Foreman	5:2		$135,000
Development Supervisor	42:21		$54,000
Production Supervisor	42:21		$54,000
Control Room Operator	42:21		$54,000
Mine Maintenance Team			
Maintenance Superintendent	5:2		$135,000
Maintenance Supervisor	42:21		$54,000
Maintenance Planner	5:2		$54,000
Chief Electrical	5:2		$114,000
Electrical Supervisor	5:2		$54,000
Clerk	42:21		$14,400
AutoMine System Manager	5:2		$114,000
AutoMine Technician	42:21		$54,000

The labour rates for owner operations and maintenance labour are presented in Table 21-8.

Table 21-8: Underground mining operations and maintenance labour

Labour Description	Rotation	Hourly Rate ($)	Annual Rate ($)
Mine Operations & Maintenance Labour			
Mine Operations Labour			
Jumbo Operator	42:21	$13.24	$36,000
Bolter Operator	42:21	$13.24	$36,000
LH Drill Operator	42:21	$13.24	$36,000
Chargeup-Development	42:21	$13.24	$36,000
Chargeup-Production	42:21	$13.24	$36,000
LHD Operator	42:21	$13.24	$36,000
Shotcreter Operator	42:21	$13.24	$36,000
Secondary Breaker Operator	42:21	$13.24	$36,000
Truck Driver	42:21	$6.62	$18,000
Transmixer Driver	42:21	$6.62	$18,000
Fuel/Lube Truck Driver	42:21	$6.62	$18,000
Personnel Carrier Driver	42:21	$6.62	$18,000
Grader Operator	42:21	$6.62	$18,000
Nipper	42:21	$6.62	$18,000
Service Crew Lead	42:21	$9.19	$25,000
Service Crew	42:21	$6.62	$18,000
Construction Worker	42:21	$6.62	$18,000
Magazine Keeper	42:21	$6.62	$18,000
Mine Maintenance Labour			
Fitter - Leading Hand	42:21	$14.71	$40,000
Fitter - Breakdown	42:21	$13.24	$36,000
Workshop Fitter	42:21	$13.24	$36,000
Light Vehicle - Fitter	42:21	$13.24	$36,000
Electrician - Leading Hand	42:21	$14.71	$40,000
Electrician - Underground	42:21	$13.24	$36,000
Auto Electrician	42:21	$13.24	$36,000
Tire Fitter	42:21	$13.24	$36,000
Boilermaker	42:21	$13.24	$36,000
AutoMine System Maintenance	42:21	$16.54	$45,000

21.2.4 Process Plant

The physical basis for this operating cost estimate has been based on the production schedule provided by SolGold, which was based on the mine production schedule.

Core physical data used in the estimate includes the ore feed rate based on the mine production plan, SMC and Bond comminution data and the metallurgical response of the ore. The metallurgical response of the ore determined the concentrate production and grade and tailings production rates along with reagent dosage rates.

21.2.5 Port and Surface Infrastructure

The physical basis for this operating cost estimate has been based on the mine production schedule provided by SRK.

Electrical power and labor costs were provided by SolGold and vetted against JDS benchmarked data. Fuel costs were benchmarked against other mining operations in Ecuador.

Other cost estimate information was produced from a variety of sources from within the Project, including calculations performed by the JDS QP, test work data, benchmarked data and internal JDS databases.

21.2.6 G&A

The physical basis for this operating cost estimate has been based on the production schedule provided by SolGold, which was based on the mine production schedule.

21.2.7 Operating Cost Summary

Overall operating costs are presented in Table 21-9. The process plant comprises two 12 Mtpa module-based concentrators, giving a combined capacity of 24 Mtpa.

Table 21-9: Operating cost summary

Area	LOM Total US$M	Unit Cost US$/t processed
Mine	3,319	6.15
Processing	3,993	7.40
TSF	79	0.15
Port	103	0.19
Surface infrastructure	182	0.34
G&A	551	1.02
Total	**8,227**	**15.24**

Note: Totals do not necessarily equal the sum of the components due to rounding adjustments.

22 Economic Analysis

22.1 Cautionary Statement

The results of the economic analysis represent forward-looking information that are subject to several known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. Forward-looking information includes Mineral Resource and Mineral Reserve estimates; commodity prices and exchange rate; smelter terms; the proposed mine production plan; projected recovery rates; use of a process method, infrastructure construction costs and schedule; mine capital and operating costs; and assumptions that Project environmental approval and permitting will be forthcoming from local, state, and federal authorities.

22.2 Financial Model Methodology

Financial analysis of the Cascabel project was carried out using a discounted cash flow (DCF) approach. This method of valuation requires projecting yearly cash inflows, or revenues, and subtracting yearly cash outflows such as operating costs, capital costs, and taxes. The resulting net annual cash flows are discounted back to the date of valuation and totalled to determine the net present value (NPV) of the Project at selected discount rates.

The internal rate of return (IRR) is expressed as the discount rate that yields a zero NPV.

The payback period is the time calculated from the start of production until all initial capital expenditures have been recovered.

This economic analysis includes sensitivities to variation in operating costs, capital costs, grade, and metal price. Note that grade and metal price are multiplicative; consequently, the two sensitivity lines are coincidental, with one overlying the other.

It should be noted that, for the sake of discounting, cash flows are assumed to occur in the middle of the year and are discounted to the start of Year -4, the start of construction.

All pricing and financial amount are stated in constant (real) Q4 2023 United States dollars (US$).

22.3 Financial Model Parameters

SRK prepared the financial model based on the following contributions and the assumptions outlined below:

- Resource model prepared by SolGold but verified by Dr. Arseneau of SRK
- Mine schedule, mine operating costs, and mine capital costs prepared by SRK, under the supervision of Mr. Jakubec of SRK
- Metal pricing provided by SolGold and agreed to by the QPs of SRK
- Smelter terms and refining costs previously provided by Wood Mackenzie and reviewed by Mr. Kottmeier of SRK

- Process recoveries provided by SolGold and reviewed by Mr. Adaszynski of Sedgman
- Process operating costs and capital costs prepared by Mr. Adaszynski of Sedgman
- G&A operating costs prepared by SolGold and Mr. Boehnke of JDS
- On-site infrastructure costs prepared by the QPs of JDS and Knight Piésold
- Off-site infrastructure costs prepared by the QPs of JDS and Knight Piésold
- Taxation based on the assumptions from the negotiated Term Sheet and reviewed by Mr. Kottmeier of SRK

22.3.1 Metal Prices

Copper, gold, and silver are saleable products. Metal pricing is based on street consensus analyst long-term price projections effective February 2024. The metal prices are shown in Table 22-1.

Table 22-1: Metal selling prices

Metal	Units	Price
Gold	US$/oz	1,750
Silver	US$/oz	22.50
Copper	US$/lb	3.85

22.3.2 Commercial Terms

The Cascabel concentrate is clean and low in deleterious elements, meaning it is capable of being treated globally. However, SolGold is prioritising those smelters capable of treating high gold concentrates such as Cascabel's. Therefore, it is expected that 50% of the concentrate will be shipped to Europe while the remaining 50% will be shipped to certain parts of Asia.

Concentrate Transportation Costs

Transportation costs to the port are estimated at $75.00 /wmt and the analysis was performed assuming FOB Port.

European Smelter Terms

Table 22-2 shows the European smelter terms provided by Wood Mackenzie and applied within the financial model to 50% of the concentrate.

Table 22-2: European smelter terms

Smelter Terms	Units	Value
Cu pay factor	%	96.6%
Cu minimum deduction	units	1.0%
Au pay factor	%	98.0%
Au minimum deduction	g/dmt	1.00
Ag pay factor	%	97.0%
Ag minimum deduction	g/dmt	30.0
Concentrate treatment charge rate	US$/dmt	79.0
Cu refining charge	US$/lb	0.079
Au refining charge	US$/oz	5.00
Ag refining charge	US$/oz	0.50

Asian Smelter Terms

Table 22-3 shows the Asian smelter terms provided by Wood Mackenzie and applied within the financial model to 50% of the concentrate.

Table 22-3: Asian smelter terms

Smelter Terms	Units	Value	lower end	higher end
Cu pay factor			**Cu grade**	**%**
	%	96.50%	20.00%	30.00%
	%	96.70%	30.00%	34.00%
	%	96.75%	34.00%	
Cu minimum deduction			**Cu grade**	**%**
	units	1.00%	20.00%	34.00%
	units	0.00%	34.00%	
Au pay factor			**Au grade**	**g/dmt**
	%	0.00%		1
		90.00%	1	3
		95.00%	3	5
		96.00%	5	10
		96.50%	10	15
		97.00%	15	20
		97.50%	20	40
		98.00%	40	60
		98.30%	60	
Ag pay factor			**Ag grade**	**g/dmt**
	%	0.00%		30
		90.00%	30	
Concentrate treatment charge rate	US$/dmt	79		
Cu refining charge	US$/lb	0.079		
Au refining charge	US$/oz	5		
Ag refining charge	US$/oz	0.5		

22.3.3 Royalties

Government Royalty

As outlined in Section 4.7, the Mining Law establishes that royalty rates shall be between 3% and 8%. The Mining Concessionaire, the State, and SolGold have agreed to the Government Royalty, a variable royalty on net smelter revenues for copper, gold, and silver, as outlined in Table 22-4, Table 22-5 and Table 22-6, respectively.

Table 22-4: Variable royalty on net smelter revenues for copper

Realised Copper Price (US$/lb)	Royalty % of Copper NSR Revenue
Copper Price ≤ $3.00	3.0%
$3.00 < Copper Price ≤ $3.50	4.0%
$3.50 < Copper Price ≤ $4.00	5.0%
$4.00 < Copper Price ≤ $4.50	6.0%
$4.50 < Copper Price ≤ $5.00	7.0%
Copper Price > $5.00	8.0%

Table 22-5: Variable royalty on net smelter revenues for gold

Realised Gold Price (US$/oz)	Royalty % of Gold NSR Revenue
Gold Price ≤ $1,400	3.0%
$1,400 < Gold Price ≤ $1,600	4.0%
$1,600 < Gold Price ≤ $1,800	5.0%
$1,800 < Gold Price ≤ $2,000	6.0%
$2,000 < Gold Price ≤ $2,200	7.0%
Gold Price > $2,200	8.0%

Table 22-6: Variable royalty on net smelter revenues for silver

Realised Silver Price (US$/oz)	Royalty % of Gold NSR Revenue
Silver Price ≤ $20.00	3.0%
$20.00 < Silver Price ≤ $22.50	4.0%
$22.50 < Silver Price ≤ $25.00	5.0%
$25.00 < Silver Price ≤ $27.50	6.0%
$27.50 < Silver Price ≤ $30.00	7.0%
Silver Price > $30.00	8.0%

SolGold and the Government of Ecuador have agreed to an advance royalty payment totaling $75 million, with $25 million due upon the concentrator construction start date. The remaining two payments, each of $25 million, will be made on the first and second anniversary, respectively, from the date of the first payment. The advance royalty will be deductible against the Government Royalty.

Santa Barbara Resources Limited Royalty

A 2% Net Smelter Royalty (NSR) is payable to Santa Barbara Resources Limited, the previous owners of the Cascabel concession. These royalties can be bought out by paying a total of $4 million. Fifty percent (50%) of the royalty can be purchased for $1 million, 90 days following the completion of a feasibility study and the remaining 50% of the royalty can be purchased for $3 million, 90 days following a production decision. It is assumed that the Santa Barbara royalty will be bought out for $4 million ahead of construction; therefore, the royalty is considered a sunk cost and not included in the financial analysis.

Franco-Nevada Royalty

SolGold entered into a Net Smelter Returns Financing Agreement with Franco-Nevada Corporation (FN) on 11 May 2020, with the following key terms (NSR Financing Agreement between FRANCO-NEVADA CORPORATION and SOLGOLD PLC, 11 May 2020):

- Royalty Terms: A perpetual 1.0% NSR against metal production within the "Concesion Minera Cascabel"
- Buyback: A 50% buy-back option exercisable at SolGold's election for 6 years from closing at a price delivering Franco-Nevada a 12% IRR ("FN Buyback")
- Gold conversion: Option in favour of Franco-Nevada to convert the NSR interest into a gold-only NSR interest (6 years from Year 2 of operations) can be exercised in a period of 8 months following the signing of the NSR

The financing agreement with FN stipulates that for the first seven years of operations, if actual production is less than 85% of minimum metal production as stipulated in the agreement, then FN is entitled to a "top-up" payment to this level with a further permanent "top-up" to the royalty rate if production in Years 5, 6, or 7 is less than minimum metal production. Because the PFS production plan does not meet the 85% minimum metal production in Years 5, 6, and 7, the "top-up" is triggered, and the LOM NSR increases from 1.0% to 1.71%.

Osisko Gold Royalties Royalty

SolGold entered into a royalty agreement with Osisko Gold Royalties Ltd. on 30 November 2022, with the following key terms (Royalty Agreement between OSISKO GOLD ROYALTIES LTD and SOLGOLD PLC, 30 November 2022):

- Royalty Terms: A perpetual 0.6% NSR against metal production within the "Concesion Minera Cascabel"
- Buyback: One-third buy-back option exercisable at SolGold's election for four years from closing subject to, among other things, the terms of the FN Buyback and exercise of the FN Buyback

Total royalties payable by the Project are estimated at $2,357 million (see Table 22-7).

Table 22-7: Royalties payable

Description	US$M
Franco-Nevada NSR Royalty	552
Osisko Gold Royalties Royalty	194
Government Royalty Copper	1,064
Government Royalty Gold	538
Government Royalty Silver	9
Total Royalty Payments	**2,357**

22.3.4 Capital and Operating Costs

The capital and operating costs are reported in Section 21 Capital and Operating Costs.

The Total Cash Costs (TCC) average $0.26 per lb payable copper, all-in sustaining costs (AISC) average $0.69 per lb payable copper, and all in costs (AIC) average $0.96 per lb payable copper (Table 22-8).

Table 22-8: Total cash cost summary

Description	US$/lb Cu payable
Operating costs on-site	
G&A	0.09
Mine	0.55
Process plant	0.66
TSF processing	0.01
Port	0.02
Surface infrastructure	0.03
Total operating costs on-site	**1.37**
Operating costs off-site	
Concentrate treatment and refining charges	0.43
Royalty	0.39
Local Tax	0.02
Total operating costs	**2.21**
By-product credits	
Au premium	(1.91)
Ag premium	(0.04)
Total by-product credits	(1.95)
Net direct cash cost (TCC)	**0.26**
Capital & Closure costs	
Sustaining costs	0.43
Capital Cost	0.26
Post-production Costs	0.01
All-in sustaining cost (AISC)	**0.69**
All-in cost (AIC)	**0.96**

22.3.5 Corporate Tax Regime

The taxation modelled within the financial analysis is based on the terms and conditions of term sheet agreement of the Exploitation Agreement, the expected impact of the new Investment Protection Agreement once approved by the Government of Ecuador, and tax advice provided by an in-country tax expert contracted by SolGold. Based on the foregoing, the expected Ecuadorian tax regime is summarised below:

- The Corporate Income Tax (CIT) is calculated using the expected rate of 20% that will be in effect once the Government of Ecuador approves the new Investment Protection Agreement. For CIT purposes, capital expenditures are accumulated in tax pools that can be deducted against mine income at different prescribed rates, depending on the type and mining phase of the capital expenditures.

- Pre-production, preparation and mine development expenditures are considered capitalised investments that are accumulated and deducted against taxable income in accordance with the rates and years prescribed by Ecuadorian tax and accounting regulations applicable for large mining scale concessions.
- Capital costs are amortised in accordance with Ecuadorian tax and accounting regulations.
- For Project modelling purposes, closure costs are not amortised or expensed until the final years of the Project, during the closure phase.
- Profit sharing is calculated and deducted for CIT purposes using the current applicable rate of 15% over pre-income tax annual profits. Of that 15%, 12% will be paid to the Federal and local governments (GAD).
- Tax losses are calculated and amortised considering the 5-year carry forward rule and 25% limitation based on annually generated taxable profits.
- Other taxes expected to be applicable are summarised below:
  - Value Added Tax 12% (VAT) is calculated over a certain portion of CAPEX and OPEX. Most input VAT is considered as tax credit eligible for refund once mine exports begin. Input VAT not eligible for refund is considered as a deductible expense for CIT purposes.
  - Regarding Sovereign Adjustment, the Constitution of Ecuador establishes that the State will participate in the benefits of non-renewable resource projects, in an amount that will not be less than those of the company that exploits them. Pursuant to the Term Sheet in connection with the Exploitation Agreement, the State participation rate of cumulative discounted benefits will not be less than 50%. Based on the annual discounted free cash flow of the base case financial model, the benefits received by the Ecuadorian Government resulting from the Project would not trigger the calculation and payment of Sovereign Adjustment, given that the Government's benefits would be higher than the benefits received by the concessionaire.

Over the LOM, the Project pays $5,549 million in taxes. Sovereign Adjustment would not be paid because the Project's benefit does not exceed the Government's benefit. The $52 million paid in VAT is for the non-recoverable VAT charges on G&A costs (Table 22-9).

Table 22-9: Corporate taxes

Tax Description	US$M
Profit share - government	2,061
Profit share - employees	515
Corporate tax	2,920
Sovereign adjustment	-
VAT	52
Total tax	**5,549**

In addition to the corporate taxes paid, $7 million is paid for the Conservation Patent Fee and $95 million is paid in various local taxes.

22.3.6 Reclamation and Closure Costs

The Mining Law specifies that a Closure Plan is required as part of the environmental management plan submitted as part of the EIA. The Closure Plan is to include an estimated closure cost, upon which a financial guarantee or insurance policy in favour of the government is required, which must remain in force until the final closure of operations. SolGold recognises that a closure cost guarantee is required, but at this stage it has not been negotiated with the Ecuadorian government. For consistency with other recently filed Ecuadorian Technical Reports, closure cost guarantees are left out of the financial model. Only LOM closure costs of $82 million are included in the model. Closure costs are not amortised or expensed until the final years of the Project during the closure phase.

22.3.7 Financing

The Project is 100% equity financed. No debt is included within the financial model.

22.3.8 Inflation

The financial analysis assumes constant Q4 2023 dollars (real). No escalation or inflation are accounted for in the financial model.

22.3.9 Working Capital

Working capital is the capital required to fund operations prior to receiving revenue from the finished product. It is defined as the current assets minus the current liabilities. The financial model estimates working capital by subtracting 30 days of direct operating costs from 45 days of revenue. Over the Project life, working capital nets to zero.

22.3.10 Salvage Value

No salvage value is included within the financial model.

22.4 Financial Model Results

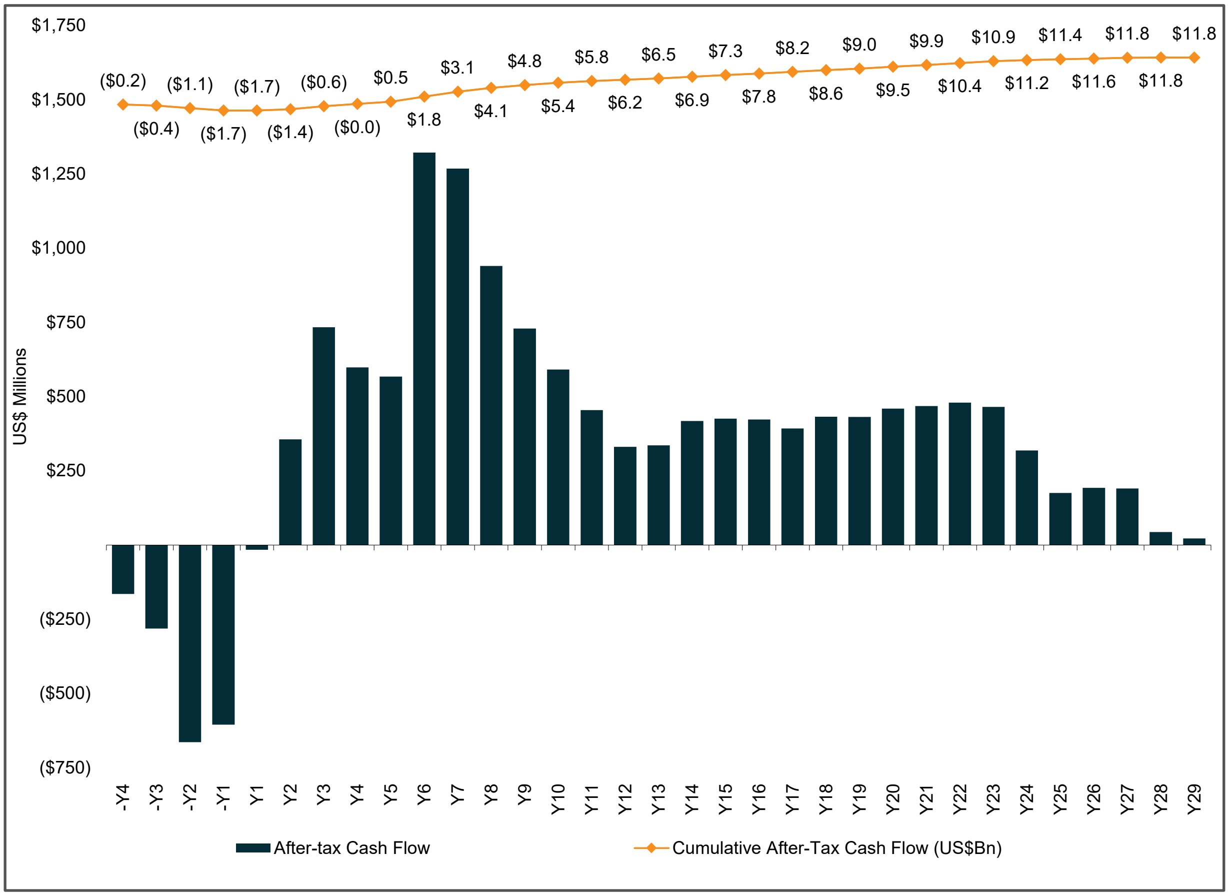
Based on SRK's financial evaluation, the Cascabel project generates positive before and post-tax financial results. Post-tax IRR is 24% and the post-tax NPV_8 is $3,221 million. Post-tax payback is achieved four years following the start of production. Table 22-10 and Table 22-11 provide the pre- and post-tax financial results, respectively and Figure 22-1 shows the distribution of post-tax cash flows. The full Project production schedule is shown in Table 22-12 and the cash flow is shown in Table 22-13.

Table 22-10: Pre-tax financial results

Before-tax	Units	Value
Cash flow	US$M	17,384
NPV @ 5%	US$M	8,139
NPV @ 8%	**US$M**	**5,372**
NPV @ 10%	US$M	4,115
Payback period	years	3.1
IRR	%	33%

Table 22-11: Post-tax financial results

Post-tax	Units	Value
Cash flow	US$M	11,835
NPV @ 5%	US$M	5,159
NPV @ 8%	**US$M**	**3,221**
NPV @ 10%	US$M	2,354
Payback period	years	4.1
IRR	%	24%



Source: This study, 2024

Figure 22-1: Post-tax cash flow distribution

Table 22-12: Project production

	Year	Total / Average	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28
Mining Summary																																		
Ore mined	(000s Tonnes)	539,724	0	4	377	771	3,370	9,266	12,441	12,119	12,115	24,031	23,969	23,958	23,953	23,922	23,899	23,233	23,862	23,956	22,366	23,939	23,914	23,851	23,824	23,814	23,825	23,843	23,845	17,722	11,918	11,918	11,900	7,797
Cu Grade	(%)	0.60%	0	0.21%	0.36%	0.68%	0.73%	0.83%	0.96%	0.94%	0.99%	0.97%	0.89%	0.76%	0.66%	0.59%	0.54%	0.48%	0.47%	0.53%	0.55%	0.52%	0.49%	0.50%	0.50%	0.52%	0.52%	0.51%	0.50%	0.46%	0.44%	0.43%	0.42%	0.41%
Au Grade	(g/t)	0.54	0	0.14	0.29	0.69	0.75	1.05	1.43	1.26	1.18	1.18	0.95	0.74	0.62	0.55	0.48	0.40	0.39	0.45	0.46	0.40	0.36	0.35	0.36	0.36	0.35	0.34	0.33	0.29	0.26	0.27	0.26	0.26
Ag Grade	(g/t)	1.62	0	0.29	1.12	1.52	1.55	1.65	2.14	2.35	2.18	2.28	2.10	1.85	1.71	1.60	1.49	1.29	1.24	1.59	1.69	1.59	1.39	1.41	1.44	1.48	1.47	1.43	1.40	1.38	1.48	1.55	1.58	1.57
CuEq. Grade	(%)	0.97%	0	0.30%	0.56%	1.15%	1.24%	1.54%	1.93%	1.80%	1.79%	1.77%	1.53%	1.27%	1.09%	0.97%	0.87%	0.76%	0.74%	0.84%	0.87%	0.80%	0.74%	0.74%	0.75%	0.77%	0.76%	0.75%	0.73%	0.67%	0.62%	0.62%	0.61%	0.59%
Recovered Production by Metal																																		
Copper	(000s lbs)	6,314,438	--	--	--	--	61,126	154,262	242,999	232,301	244,785	475,467	429,028	361,851	312,727	275,992	248,510	213,738	214,276	245,792	236,355	240,246	226,122	227,560	229,809	236,150	237,029	233,054	228,016	156,626	98,785	97,656	94,486	59,688
Gold	(000s oz)	6,875	--	--	--	--	75	242	458	391	373	734	576	431	348	298	255	201	200	237	229	211	183	183	185	189	185	177	168	109	66	66	65	42
Silver	(000s oz)	18,418	--	--	--	--	144	324	562	601	558	1,159	1,065	938	864	808	752	632	627	802	800	805	704	708	725	743	738	719	707	515	373	390	397	258
CuEq. Production	(000s lbs)	**9,547,109**	--	--	--	--	**95,835**	**266,318**	**454,429**	**413,739**	**417,498**	**815,723**	**696,903**	**563,383**	**475,785**	**416,048**	**369,000**	**308,718**	**308,713**	**358,232**	**345,043**	**340,697**	**313,457**	**314,871**	**318,072**	**326,242**	**325,389**	**317,783**	**308,591**	**209,394**	**130,934**	**129,895**	**126,285**	**80,130**
Copper Production	(000s t)	2,864	--	--	--	--	28	70	110	105	111	216	195	164	142	125	113	97	97	111	107	109	103	103	104	107	108	106	103	71	45	44	43	27
CuEq. Production	(000s t)	4,330	--	--	--	--	43	121	206	188	189	370	316	256	216	189	167	140	140	162	157	155	142	143	144	148	148	144	140	95	59	59	57	36
Payable Production by Metal																																		
Copper	(000s lbs)	6,015,363	--	--	--	--	58,231	146,956	231,490	221,298	233,192	452,947	408,708	344,712	297,915	262,920	236,740	203,615	204,127	234,150	225,161	228,868	215,412	216,782	218,925	224,965	225,802	222,016	217,216	149,208	94,107	93,031	90,011	56,861
Gold	(000s oz)	6,550	--	--	--	--	71	234	443	378	359	707	553	412	331	284	243	190	189	224	217	199	172	172	174	177	173	166	157	102	61	61	60	39
Silver	(000s oz)	11,196	--	--	--	--	76	154	292	339	287	627	584	530	508	492	467	387	382	517	524	524	443	445	459	470	464	450	444	333	256	273	283	186
CuEq. Production	(000s lbs)	**9,057,947**	--	--	--	--	**91,048**	**254,105**	**434,673**	**395,139**	**398,182**	**778,118**	**663,534**	**535,094**	**451,514**	**394,684**	**349,854**	**292,173**	**292,143**	**339,129**	**326,657**	**322,205**	**296,203**	**297,518**	**300,544**	**308,242**	**307,364**	**300,104**	**291,338**	**197,588**	**123,501**	**122,533**	**119,147**	**75,613**

Table 22-13: Project cash flow

	Year	TOTAL	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29
NET SMELTER RETURN (NSR)																																			
NSR																																			
Metal Payable - Europe	US$M	17,284	-	-	-	-	174	486	832	756	761	1,487	1,267	1,022	862	753	667	557	557	647	623	614	564	567	573	587	585	571	555	376	235	233	227	144	-
Metal Payable - Asia	US$M	17,590	-	-	-	-	177	493	841	765	772	1,508	1,288	1,038	877	766	680	568	568	659	634	626	576	579	585	600	598	584	567	385	240	239	232	147	-
TC/RCs	US$M	(1,540)	-	-	-	-	(15)	(38)	(60)	(57)	(60)	(117)	(105)	(88)	(76)	(67)	(61)	(52)	(52)	(60)	(58)	(58)	(55)	(55)	(56)	(57)	(57)	(56)	(55)	(38)	(24)	(24)	(23)	(14)	-
Transport	US$M	(1,055)	-	-	-	-	(10)	(26)	(41)	(39)	(41)	(79)	(72)	(60)	(52)	(46)	(42)	(36)	(36)	(41)	(39)	(40)	(38)	(38)	(38)	(39)	(40)	(39)	(38)	(26)	(16)	(16)	(16)	(10)	-
Total NSR	**US$M**	**32,279**	-	-	-	-	**325**	**915**	**1,573**	**1,425**	**1,432**	**2,799**	**2,378**	**1,911**	**1,610**	**1,406**	**1,245**	**1,037**	**1,037**	**1,205**	**1,161**	**1,142**	**1,048**	**1,052**	**1,063**	**1,090**	**1,086**	**1,060**	**1,028**	**697**	**435**	**432**	**420**	**267**	-
NSR ROYALTY																																			
Royalty																																			
Franco-Nevada NSR Royalty	US$M	(552)	-	-	-	(10)	(10)	(25)	(33)	(37)	(31)	(35)	(36)	(30)	(25)	(22)	(19)	(16)	(16)	(19)	(18)	(18)	(16)	(16)	(17)	(17)	(17)	(17)	(16)	(11)	(7)	(7)	(7)	(4)	-
Osisko NSR Royalty	US$M	(194)	-	-	-	-	(2)	(5)	(9)	(9)	(9)	(17)	(14)	(11)	(10)	(8)	(7)	(6)	(6)	(7)	(7)	(7)	(6)	(6)	(6)	(7)	(7)	(6)	(6)	(4)	(3)	(3)	(3)	(2)	-
Government Royalty Copper	US$M	(1,064)	-	-	-	-	(10)	(26)	(42)	(40)	(42)	(81)	(73)	(61)	(53)	(47)	(42)	(36)	(36)	(41)	(40)	(40)	(38)	(38)	(39)	(40)	(40)	(39)	(38)	(26)	(17)	(16)	(16)	(10)	-
Government Royalty Gold	US$M	(538)	-	-	-	-	(6)	(19)	(37)	(31)	(30)	(58)	(46)	(34)	(27)	(23)	(20)	(16)	(15)	(18)	(18)	(16)	(14)	(14)	(14)	(14)	(14)	(14)	(13)	(8)	(5)	(5)	(5)	(3)	-
Government Royalty Silver	US$M	(9)	-	-	-	-	(0)	(0)	(0)	(0)	(0)	(1)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	-
Advanced Royalty (Paid) / Credit	US$M	-	-	-	(25)	(25)	(17)	15	15	15	15	7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total NSR royalty	**US$M**	**(2,357)**	-	-	**(25)**	**(35)**	**(45)**	**(61)**	**(106)**	**(101)**	**(96)**	**(185)**	**(169)**	**(137)**	**(115)**	**(101)**	**(89)**	**(74)**	**(74)**	**(86)**	**(83)**	**(82)**	**(75)**	**(75)**	**(76)**	**(78)**	**(78)**	**(76)**	**(74)**	**(50)**	**(31)**	**(31)**	**(30)**	**(19)**	-
OPERATING COSTS																																			
Operating Costs																																			
G&A	US$M	(551)	-	-	-	-	(5)	(9)	(13)	(12)	(12)	(25)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(23)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(18)	(12)	(12)	(12)	(8)	-
Mine	US$M	(3,319)	-	-	-	-	(37)	(63)	(94)	(99)	(101)	(141)	(132)	(133)	(132)	(142)	(149)	(150)	(150)	(149)	(143)	(148)	(145)	(134)	(134)	(134)	(134)	(134)	(133)	(107)	(84)	(82)	(76)	(60)	-
Process Plant	US$M	(3,993)	-	-	-	-	(34)	(70)	(94)	(92)	(92)	(177)	(177)	(177)	(177)	(177)	(176)	(171)	(176)	(177)	(165)	(177)	(176)	(176)	(176)	(176)	(176)	(176)	(176)	(131)	(88)	(88)	(88)	(58)	-
TSF Processing	US$M	(78)	-	-	-	-	(1)	(1)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(2)	(1)	(1)	-
Port	US$M	(103)	-	-	-	-	(1)	(3)	(3)	(3)	(3)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(3)	(3)	(3)	(2)	-
Surface Infrastructure	US$M	(182)	-	-	-	-	(15)	(4)	(5)	(5)	(5)	(7)	(7)	(7)	(7)	(7)	(7)	(7)	(7)	(7)	(7)	(7)	(7)	(7)	(7)	(7)	(7)	(7)	(7)	(6)	(5)	(5)	(5)	(4)	-
Total Operating Costs	**US$M**	**(8,226)**	-	-	-	-	**(93)**	**(151)**	**(212)**	**(214)**	**(216)**	**(357)**	**(348)**	**(349)**	**(348)**	**(358)**	**(364)**	**(360)**	**(364)**	**(365)**	**(344)**	**(364)**	**(360)**	**(349)**	**(348)**	**(348)**	**(348)**	**(348)**	**(348)**	**(269)**	**(195)**	**(191)**	**(185)**	**(132)**	-
NET OPERATING EARNINGS																																			
Net operating earnings																																			
Operating Revenue (NSR)	US$M	32,279	-	-	-	-	325	915	1,573	1,425	1,432	2,799	2,378	1,911	1,610	1,406	1,245	1,037	1,037	1,205	1,161	1,142	1,048	1,052	1,063	1,090	1,086	1,060	1,028	697	435	432	420	267	-
Total Royalty	US$M	(2,357)	-	-	(25)	(35)	(45)	(61)	(106)	(101)	(96)	(185)	(169)	(137)	(115)	(101)	(89)	(74)	(74)	(86)	(83)	(82)	(75)	(75)	(76)	(78)	(78)	(76)	(74)	(50)	(31)	(31)	(30)	(19)	-
Operating Cost	US$M	(8,226)	-	-	-	-	(93)	(151)	(212)	(214)	(216)	(357)	(348)	(349)	(348)	(358)	(364)	(360)	(364)	(365)	(344)	(364)	(360)	(349)	(348)	(348)	(348)	(348)	(348)	(269)	(195)	(191)	(185)	(132)	-
Conservation Patent	US$M	(7)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	-
Local Tax	US$M	(95)	(0)	(1)	(3)	(4)	(4)	(4)	(4)	(4)	(5)	(5)	(4)	(5)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(3)	(3)	(3)	(2)	(2)	(1)	(1)	(0)	(0)	(0)	(0)	-
Net Operating Earnings	US$M	**21,593**	**(1)**	**(1)**	**(28)**	**(39)**	**183**	**698**	**1,250**	**1,105**	**1,115**	**2,252**	**1,856**	**1,421**	**1,142**	**943**	**787**	**599**	**594**	**749**	**729**	**692**	**609**	**625**	**636**	**661**	**658**	**634**	**606**	**377**	**208**	**209**	**205**	**115**	-
TAXES																																			
Taxes																																			
Profit Share - Government	US$M	(2,061)	-	-	-	-	(15)	(71)	(136)	(118)	(119)	(256)	(209)	(157)	(122)	(98)	(78)	(55)	(52)	(70)	(67)	(60)	(49)	(49)	(50)	(52)	(50)	(46)	(41)	(21)	(7)	(6)	(5)	(1)	-
Profit Share - Employees	US$M	(515)	-	-	-	-	(4)	(18)	(34)	(30)	(30)	(64)	(52)	(39)	(31)	(24)	(19)	(14)	(13)	(17)	(17)	(15)	(12)	(12)	(12)	(13)	(13)	(11)	(10)	(5)	(2)	(1)	(1)	(0)	-
Corporate Tax	US$M	(2,920)	-	-	-	-	(21)	(101)	(192)	(167)	(168)	(363)	(296)	(222)	(173)	(138)	(110)	(77)	(74)	(99)	(96)	(85)	(69)	(70)	(71)	(73)	(71)	(65)	(59)	(30)	(10)	(8)	(7)	(2)	-
Total Taxes	**US$M**	(5,497)	-	-	-	-	(40)	(189)	(362)	(315)	(317)	(684)	(558)	(418)	(326)	(260)	(208)	(146)	(139)	(186)	(180)	(161)	(130)	(132)	(133)	(138)	(134)	(122)	(110)	(56)	(20)	(15)	(14)	(3)	-
Net Income	**US$M**	**16,096**	**(1)**	**(1)**	**(28)**	**(39)**	**143**	**508**	**888**	**790**	**798**	**1,568**	**1,298**	**1,004**	**816**	**683**	**579**	**453**	**454**	**563**	**549**	**532**	**479**	**493**	**502**	**523**	**525**	**512**	**496**	**322**	**189**	**194**	**191**	**112**	-
Sovereign Adjustment	**US$M**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Net Income after Sovereign Adjustment	**US$M**	16,096	(1)	(1)	(28)	(39)	143	508	888	790	798	1,568	1,298	1,004	816	683	579	453	454	563	549	532	479	493	502	523	525	512	496	322	189	194	191	112	-
CAPITAL COST, CLOSURE AND WORKING CAPITAL																																			
Capital & Closure Cost																																			
Capital Costs																																			
Mine Underground	US$M	(1,571)	(61)	(67)	(86)	(88)	(140)	(98)	(55)	(80)	(88)	(40)	(11)	(19)	(20)	(74)	(95)	(109)	(76)	(84)	(81)	(63)	(41)	(15)	(21)	(21)	(12)	(6)	(5)	(9)	(6)	(0)	(0)	(0)	-
Infrastructure - Mine Underground	US$M	(278)	-	(48)	(125)	(8)	-	(57)	(8)	-	-	-	-	(25)	-	(3)	(1)	(2)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mineral Processing	US$M	(528)	-	-	(151)	(151)	-	-	-	(114)	(113)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tailings Storage	US$M	(1,214)	(1)	(63)	(112)	(131)	-	-	(15)	(15)	(28)	(45)	(57)	(54)	(75)	(39)	(46)	(34)	(40)	(38)	(43)	(47)	(53)	(41)	(46)	(38)	(42)	(27)	(27)	(27)	(32)	-	-	-	-
Port Facility	US$M	(20)	-	(0)	(4)	(15)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Surface Infrastructure	US$M	(409)	(73)	(66)	(91)	(107)	(8)	-	-	(0)	(0)	(2)	(11)	(21)	(26)	(0)	(1)	(1)	(1)	(0)	(0)	-	(0)	(0)	(0)	-	(0)	(0)	-	(0)	(0)	-	(0)	-	-
Owners Costs	US$M	(106)	(20)	(21)	(33)	(32)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Capital Costs	**US$M**	**(4,127)**	**(155)**	**(265)**	**(602)**	**(532)**	**(147)**	**(155)**	**(77)**	**(209)**	**(229)**	**(87)**	**(78)**	**(119)**	**(121)**	**(116)**	**(143)**	**(145)**	**(117)**	**(123)**	**(125)**	**(111)**	**(95)**	**(56)**	**(67)**	**(58)**	**(54)**	**(33)**	**(32)**	**(36)**	**(38)**	**(0)**	**(0)**	**(0)**	-
Closure Costs																																			
Closure Costs	US$M	(82)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(82)	-
VAT Paid	**US$M**	**(52)**	**(9)**	**(16)**	**(36)**	**(32)**	**21**	**71**	**(1)**	**(1)**	**(1)**	**(2)**	**(2)**	**(2)**	**(2)**	**(2)**	**(2)**	**(2)**	**(2)**	**(2)**	**(2)**	**(2)**	**(2)**	**(2)**	**(2)**	**(2)**	**(2)**	**(2)**	**(2)**	**(2)**	**(1)**	**(1)**	**(1)**	**(1)**	-
Working Capital																																			
Working Capital																																			
Change in Accounts Receivable	US$M	-	-	-	-	-	40	73	81	(19)	1	169	(52)	(58)	(37)	(25)	(20)	(26)	0	21	(5)	(3)	(11)	1	1	3	(0)	(3)	(4)	(41)	(32)	(0)	(1)	(19)	(33)
Change in Accounts Payable	US$M	-	-	-	-	-	8	5	5	0	0	12	(1)	(0)	0	1	0	(0)	0	0	(2)	2	(0)	(1)	(0)	(0)	0	0	(0)	(7)	(6)	(0)	(1)	(4)	(11)
Change in Working Capital	**US$M**	-	-	-	-	-	**(32)**	**(68)**	**(76)**	**19**	**(1)**	**(157)**	**51**	**58**	**37**	**26**	**20**	**26**	**0**	**(21)**	**4**	**4**	**11**	**(1)**	**(1)**	**(3)**	**0**	**3**	**4**	**35**	**26**	**0**	**1**	**15**	**22**
VALUATION INDICATORS																																			
Before-tax																																			
Cash Flow	US$M	17,384	(155)	(266)	(630)	(571)	3	475	1,097	915	885	2,009	1,829	1,360	1,058	853	665	479	477	606	608	586	525	567	567	600	604	604	578	376	196	209	206	48	22
Discount Factor		-	0.96	0.89	0.82	0.76	0.71	0.65	0.61	0.56	0.52	0.48	0.45	0.41	0.38	0.35	0.33	0.30	0.28	0.26	0.24	0.22	0.21	0.19	0.18	0.16	0.15	0.14	0.13	0.12	0.11	0.10	0.10	0.09	0.08
NPV 8.00%	**US$M**	**5,372**	(150)	(237)	(520)	(436)	2	311	665	514	460	967	815	561	404	302	218	145	134	158	146	131	108	108	100	98	92	85	75	45	22	22	20	4	2
Payback Period	**years**	**3.1**																																	
IRR Before-Tax	%	**33%**																																	
After-tax																																			
Cash Flow	US$M	11,835	(165)	(282)	(666)	(603)	(16)	356	734	598	567	1,322	1,268	940	729	591	455	331	336	418	426	423	393	433	432	459	468	480	465	318	175	193	190	44	22
Discount Factor		-	0.96	0.89	0.82	0.76	0.71	0.65	0.61	0.56	0.52	0.48	0.45	0.41	0.38	0.35	0.33	0.30	0.28	0.26	0.24	0.22	0.21	0.19	0.18	0.16	0.15	0.14	0.13	0.12	0.11	0.10	0.10	0.09	0.08
NPV 8.00%	**US$M**	**3,221**	(158)	(251)	(550)	(461)	(11)	233	445	336	295	637	565	388	279	209	149	100	94	109	103	94	81	83	76	75	71	67	61	38	20	20	18	4	2
Payback Period	**years**	**4.1**																																	
IRR After-Tax	%	**24%**																																	

22.5 Sensitivity Analysis

A sensitivity analysis was completed over the range of ± 30% for capital costs, operating costs, grade, and metal prices. Note that sensitivity to grade and metal price are coincidental and follow the same general trend.

The Project's NPV_8 is most sensitive to changes in copper price and grade, followed by changes in gold price and grade. The Project's NPV_8 is least sensitive to changes in silver price and grade. Sensitivity tables showing sensitivity to copper price and gold price and discount rates are shown in Figure 22-2 and Figure 22-3.

Post-tax $NPV_{8\%}$ Sensitivity to Copper Price and Discount Rate (US\$Bn)

Discount Rate (%)	Copper Price: -30%	-20%	-10%	0%	+10%	+20%	+30%
5%	$3.2	$3.9	$4.5	$5.2	$5.8	$6.3	$6.9
6%	$2.7	$3.3	$3.9	$4.4	$4.9	$5.4	$5.9
7%	$2.3	$2.8	$3.3	$3.8	$4.2	$4.7	$5.1
8%	$1.9	$2.3	$2.8	**$3.2**	$3.6	$4.0	$4.4
9%	$1.6	$2.0	$2.4	$2.8	$3.1	$3.5	$3.8
10%	$1.3	$1.7	$2.0	$2.4	$2.7	$3.0	$3.3

Source: This study, 2024

Figure 22-2: Post-tax NPV sensitivity (Cu price and discount rate)

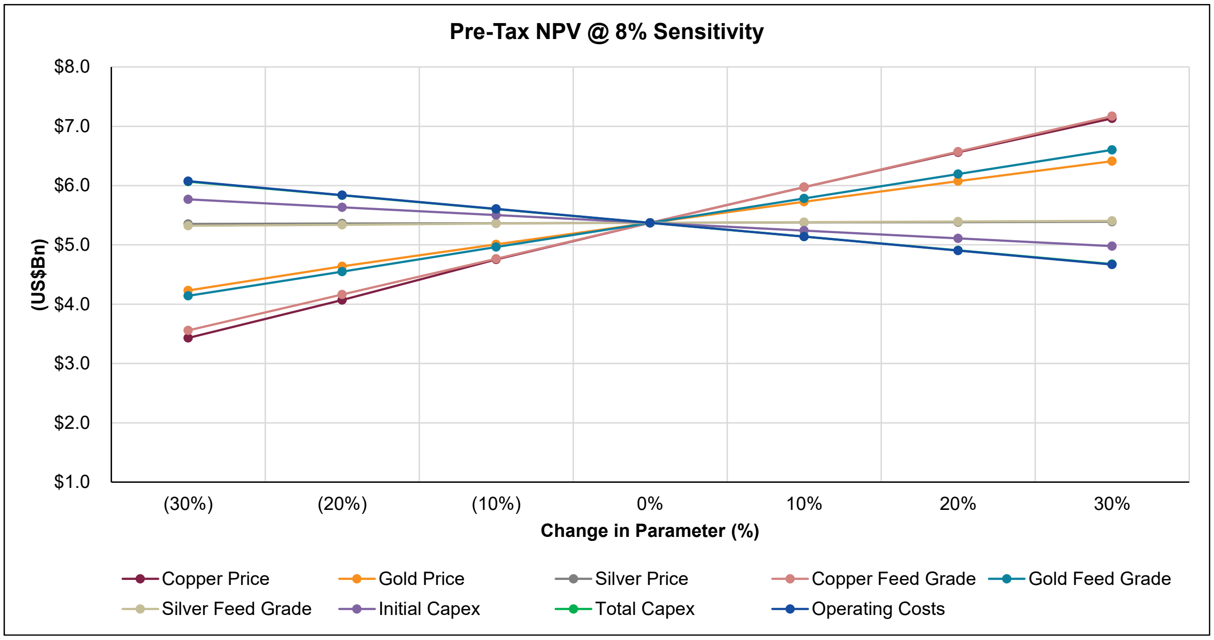
Post-tax $NPV_{8\%}$ Sensitivity to Gold Price and Discount Rate (US\$Bn)

Discount Rate (%)	Gold Price: (30%)	(20%)	(10%)	0%	10%	20%	30%
5%	$4.1	$4.5	$4.8	$5.2	$5.5	$5.8	$6.2
6%	$3.4	$3.8	$4.1	$4.4	$4.7	$5.0	$5.3
7%	$2.9	$3.2	$3.5	$3.8	$4.0	$4.3	$4.6
8%	$2.4	$2.7	$3.0	**$3.2**	$3.5	$3.7	$3.9
9%	$2.1	$2.3	$2.5	$2.8	$3.0	$3.2	$3.4
10%	$1.7	$2.0	$2.2	$2.4	$2.5	$2.7	$2.9

Source: This study, 2024

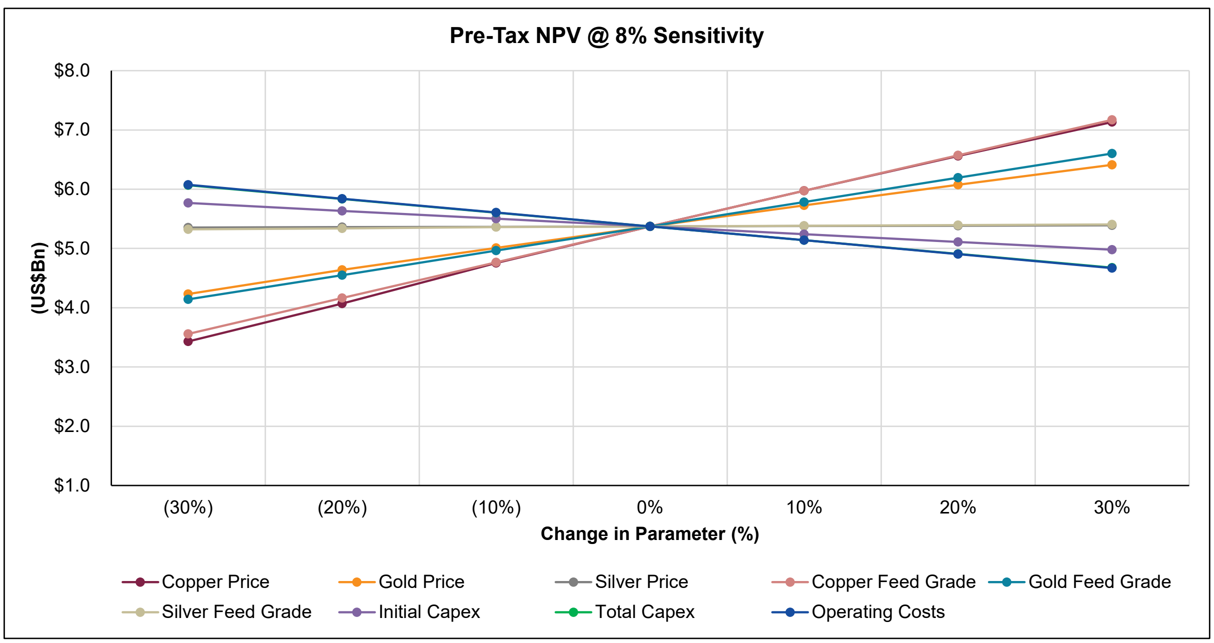
Figure 22-3: Post-tax NPV sensitivity (Au price and discount rate)

Sensitivity to a variety of cost drivers were completed for the pre- and post-tax NPV_8 and IRR. The sensitivity graphs are shown in Figure 22-4 to Figure 22-7.



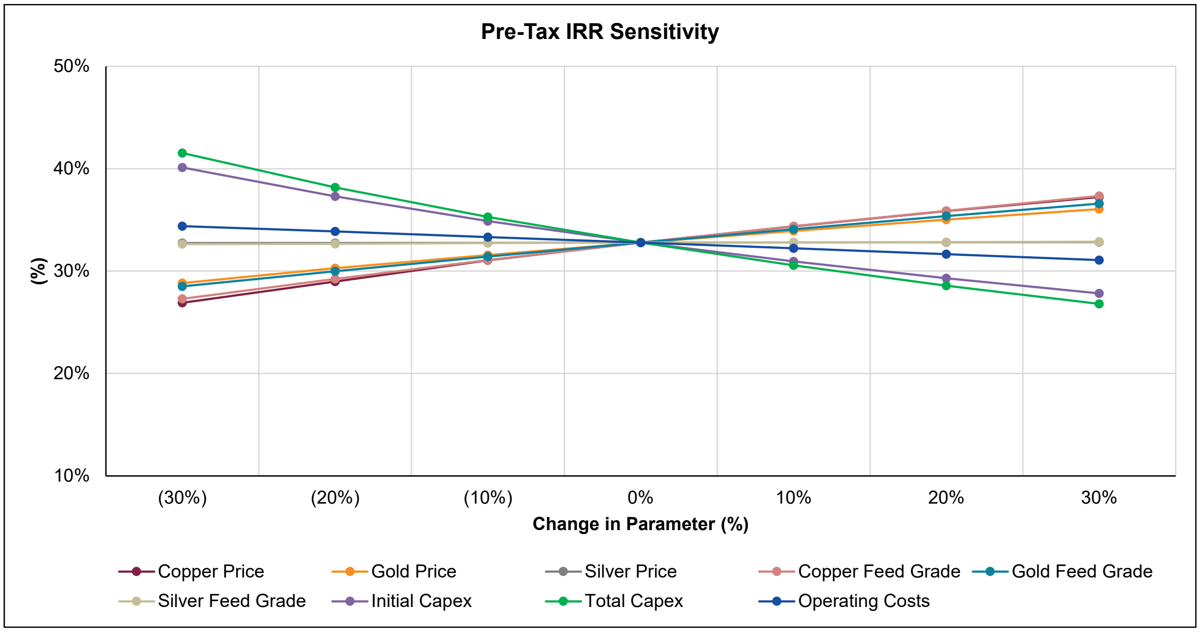
Source: This study, 2024

Figure 22-4: Pre-tax NPV$_8$ sensitivity



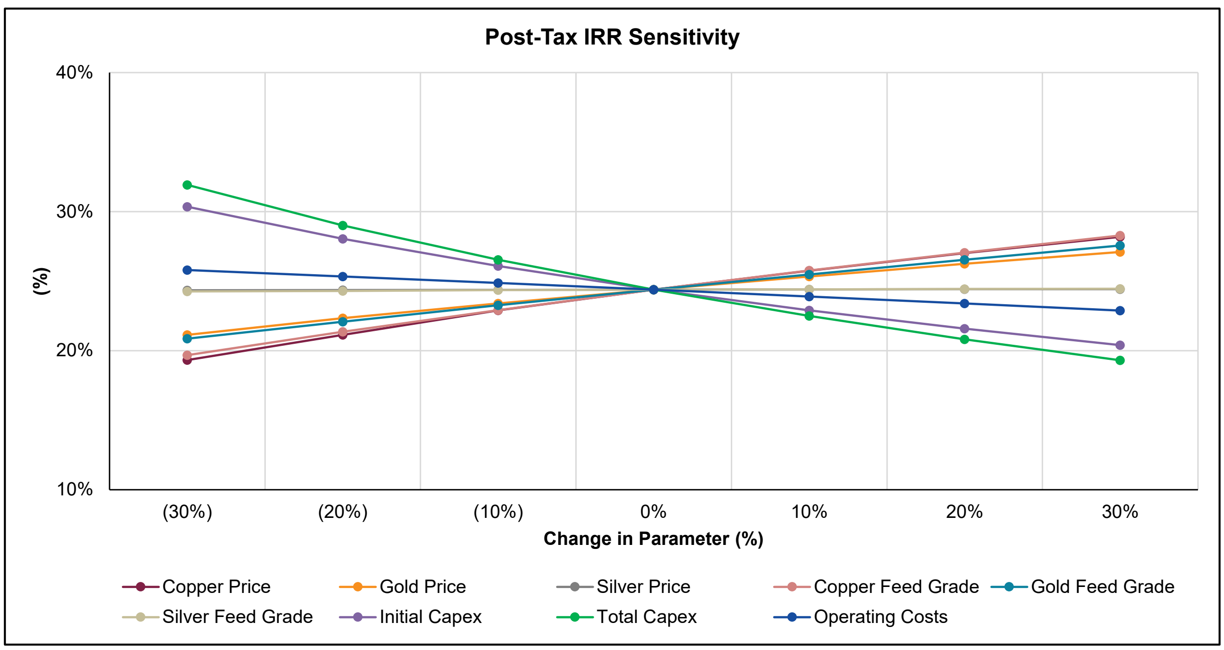
Source: This study, 2024

Figure 22-5: Post-tax NPV$_8$ sensitivity



Source: This study, 2024

Figure 22-6: Pre-tax IRR sensitivity



Source: This study, 2024

Figure 22-7: Post-tax IRR sensitivity

23 Adjacent Properties

The authors have no information to report from adjacent properties.

24 Other Relevant Data and Information

24.1 Workforce Considerations

24.1.1 Personnel Requirements

The personnel required to construct and operate the Cascabel project will comprise principally local personnel with the support of expatriates in key positions where expertise is required.

The peak personnel requirements will occur as the parallel construction of the second process plant module in parallel with the ongoing mine development and process plant operations, as shown in the Figure 24-1.

Source: Artica et al., 2022

Figure 24-1: Personnel requirements

24.1.2 Human Resources

SolGold aims to maximise employment directly (and indirectly through local businesses contracted services) in the following order of precedence:

- Immediate area of interest
- Region
- Province
- National

- Internationally (when skills and/or experience are not readily available in the Ecuadorian market. When this happens, we will look to building skills/experience in Ecuador over the longer term and recruit with a preference to South America to secure capability in the shorter term).

SolGold will continue to work closely with government, industry, education providers and community to promote and develop professional and skilled labour capability requirements for the Project.

24.1.3 Training

In Ecuador, current regulations state that no more than 20% of the workforce can be expatriate. For both the professional and skilled labour capability requirements, there is an expected minimum lead time to educate and train workforce members of three years. To ensure the required workforce quantities are available for each phase of the Project, SolGold will time the start of its respective training programs to ensure that trained workers are available for each phase of work.

Training will be designed around a cascading system whereby SolGold will focus on training a selected group of in-country workforce who will in turn train others (i.e., a “train the trainer” approach).

Block cave specific skills and experience development for Ecuadorian workers will ideally incorporate on-the-job training in another block cave mining operation. However, opportunities to place workforce at other operations will be limited.

24.2 Risk Management

24.2.1 Risk Management Strategy

A risk assessment of the Project was undertaken as part of the PFS update. The risk assessment process was used to identify key design, operational, safety, financial and environmental risks of the Project, and establish potential control measures to mitigate the identified risks to acceptable levels.

The risk process that was adopted utilised a facilitated workshop and discussions to develop the risk register for the study. The methodology followed for the risk assessment process was as follows.

The risk assessment methodology that was adopted for each identified risk considered the current risk rating including any existing controls, and then assessed the residual (target) risk rating, assuming the successful implementation of the additional proposed control measures.

The risk rating for the Project has been based on the SolGold risk matrix provided, where ‘red’ is extreme or high, ‘yellow’ is moderate and ‘green’ is low. The ‘likelihood’ and ‘consequences’ rankings used the definitions and criteria from the SolGold risk assessment methodology.

Each risk in the register is assigned a current and target (residual) rating, which are defined as:

- Current risk rating: The level of risk remaining after taking into account the controls that are already in place and have had their effectiveness tested
- Target Risk Rating: The level of risk remaining after risk treatment

The risk manager will review the risk at regular intervals to evaluate the effectiveness of the proposed mitigation strategies.

24.2.2 Risk Workshops

A series of internal risk assessment workshops were initially held with discipline-based groups to identify risks specific to the area being covered. Workshops were held for environment and community, mine geotechnical and hydrogeological works, mining, process plant and infrastructure and tailings dams.

Preliminary risk registers were generated in each workshop and then consolidated into a single preliminary risk register.

The combined workshop was split into key areas as follows:

- Project Management Risks:
  - Project / Study Management
  - Permitting and Approvals
  - Environment
  - Community
  - Health and Safety
- Technical Risks:
  - Mining
  - Tailings
  - Processing

24.2.3 Risk Matrix

The Cascabel project risk register contains one hundred (100) active items, the current and target ratings of which are displayed in Figure 24-2.

Source: Artica et al., 2022

Figure 24-2: Risk rating summary

The high residual risk ranked items are shown in the table below.

Table 24-1: Project risk register summary

Issue No.	Description of Threat or Opportunity	Current Risk Rating	Proposed Actions (To mitigate further, or to reduce chance of recurrence)	Residual Risk Rating
28	Negative pressure from Community or NGO once Pre-Feasibility is public if not appropriately socialised. In particular: Open Pit and new TSF location.	15	Planning for strong community engagement before public announcement of locations. Develop specific management strategies for stakeholders identified in mapping exercise. Announcement of OP must be socialised carefully by community relations team. Strategy to take community leaders, stakeholders, government representatives to Mirador OP to experience the impact. Provide appropriate level of budget to deliver social strategy. Buy all land that is required for infrastructure. Build social team and resource appropriately to allow proper public socialisation. Slow sufficient time to get appropriate information to the communities. Do not publish the location of any controversial infrastructure (e.g. Tailings, Open Pit),	12
69	Ground Falls in underground workings (seismic incident, insufficient geotech investigations/ understanding of	15	Install appropriate monitoring equipment. Run numerical models to determine support demand and incorporate into support design requirements across the footprint. Undertake drilling and testwork through Pre-Feasibility, FS, ongoing to develop and validate geotech models. Enhanced ground control training for mine workers, evaluation of support during operations, establishment of a ground control QA/QC team during operations. Ensure	12

Issue No.	Description of Threat or Opportunity	Current Risk Rating	Proposed Actions (To mitigate further, or to reduce chance of recurrence)	Residual Risk Rating
	ground including in extraction drives and production levels including footprint collapse)		structural investigation is carried out. Ensure tunnel mapping is carried out during development. Ensure that planned bore holes reflect level of confidence required for each level of study. Carry out in situ stress measurements. Develop a design undercut sequence based on geotechnical information (e.g., matches the capacity of the rock mass).	
213	Seismic event within the mine	15	Drilling, in situ stress testing, structural model; seismic monitoring. Numerical stability modelling integrated with support design. Assess structural stability and potential for large seismic events. Ensure critical infrastructure offset from structures with seismic potential and abutment stress interactions with appropriate support design. Minimise undercut front stress/damage concentrations and potential for bursting. Investigate hydrofracturing and de-stress blasting practices to reduce seismic potential during development of the undercut.	12

25 Interpretation and Conclusions

25.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on a review of the data available for this report.

25.2 Mineral Tenure, Surface Rights, Royalties and Agreements

- Information from legal experts supports the validity of the mining tenure and is sufficient to support the declaration of Mineral Resources and Mineral Reserves.
- SolGold, through its subsidiary ENSA, has secured or is in the process of securing the surface rights necessary to support the development of the Cascabel project. SolGold is aware of the avenues available to secure surface rights for the Cascabel project and is preferentially pursuing the options that ensure benefit to all parties. The QP considers it a reasonable assumption that with continued negotiation, all necessary surface rights to support the Cascabel project can be obtained.
- The Cascabel project proposes to secure water within the concession catchment areas.
- It is expected that there may be environmental contamination in areas where artisanal workings have been undertaken within the proposed Project areas.
- On 19 November 2021 SolGold plc and SolGold Finance AG (together the "Investors"), and Exploraciones Novomining S.A. and SolGold-Ecuador S.A. (together the "Receiving Companies") signed a preliminary commitment declaration with regard to an Exploration Investment Protection Agreement (IPA) with Ecuador Ministry of Production, Foreign Trade, Investments and Fisheries (MPCEIP).
- Additional permits are required for Project development, the most important being the Mine Exploitation Agreement and ESIA.
- There are no outstanding operations or any other kind of agreements that may limit ENSA's right to conduct mining activities.

25.3 Geology and Mineralisation

- The Alpala and Tandayama-America deposits, at the Cascabel project in northern Ecuador, occur near the overlap of Eocene and Miocene Andean porphyry belts that extend from Colombia through Ecuador and Peru into Chile and Argentina.
- The mineralisation observed at surface and in the drill core at the Cascabel concession is considered as a classic porphyry Cu-Au system on the basis of:
  - Porphyritic texture of causative intrusions, where feldspar, quartz and mafic phenocrysts are contained in a fine-grained to aplitic groundmass
  - Several stages of hydrothermal alteration associated with each mineralising intrusion

- Extensive development of fracture-controlled alteration and mineralisation in both porphyritic intrusions and adjacent wall rock
- A progression from early, discontinuous and irregular veins and veinlets (A-veins), through transitional, planar veins (B-veins) to late, through-going veins (D-veins) and breccia bodies
- A progression in hydrothermal alteration from early, central potassium silicate and distal propylitic styles to late sericitic/phyllic, advanced and intermediate argillic alteration types
- Sulphide and oxide minerals, which vary from early bornite-magnetite through transitional chalcopyrite-pyrite to late pyrite-hematite, pyrite-enargite or pyrite-bornite

25.4 Exploration, Drilling and Analytical Data Collection in Support of Mineral Resource Estimation

- The Cascabel Property has been the focus of intense exploration and drilling activity since the discovery of the Alpala deposit in 2013.
- World-renowned copper porphyry experts have studied the deposit, provided advice and guided exploration.
- Best practices in geological data capture, storage and interpretation were implemented early in the Project's development and have been maintained diligently.
- Throughout the Project, the QAQC results have demonstrated the reliability of the sampling and assaying procedures.
- The QP for the Alpala and Tandayama-America deposits undertook audits of the database underpinning the respective Mineral Resource Estimates and reviewed the site procedures. The geological and assay data are considered to be robust and suitable for inclusion in a Mineral Resource Estimate.

25.5 Metallurgical Testwork

- Testwork undertaken and reported here is relevant to the Project, supporting the process recoveries and plant design.
- Acceptable copper, gold and silver recoveries were achieved in the flotation process.
- Using the LCT results, recovery relations were developed for the flotation process for copper, gold, silver, and mass to concentrate.
- The concentrate is saleable without major penalties.

25.6 Mineral Resource Estimates

- The Mineral Resource Estimates for Alpala and Tandayama-America were prepared following industry best practice as described in the "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" (CIM, 2019).
- The estimation for both deposits was completed with conventional methods including implicit modelling of geology and mineralisation and estimation of grades into a 3D block model.

- Block model validation showed good agreement between the input composite grades and the estimated block grades in both Alpala and Tandayama-America.
- The MREs have been classified in accordance with the CIM Definition Standards (CIM, 2014) and were both informed by a drill hole spacing study based on Alpala assumptions.
- The reasonable prospects for eventual economic extraction (RPEEE) criteria were met by reporting inside optimised pit shells and/or underground optimisation shapes, assuming block caving methods.

25.7 Mineral Reserve Estimates

- The mine plan is based on Measured and Indicated Mineral Resources converted to Proven and Probable Mineral Reserves. The Inferred Mineral Resources grades were set to zero for the purposes of Mineral Reserve estimation due to the proposed mining method.
- The current Mineral Reserve estimates are based on the most current knowledge, permit status and PFS-level engineering and operational constraints. Mineral Reserves have been estimated using standard practices for the industry and conform to the 2014 CIM Definition Standards
- Factors that may affect the Mineral Reserves include long-term commodity price assumptions, long-term exchange rate assumptions, and long-term consumables price assumptions. Other factors that can affect the estimates include changes to: Mineral Resources input parameters, constraining stope designs, shut-off grade assumptions, geotechnical and hydrogeological factors, metallurgical and mining recovery assumptions, and the ability to control unplanned dilution. Operations will require the grant of the exploitation licence.

25.8 Mining Plan

- The following elements were considered for the development of the mine plan:
  - In total, 62 structures were interpreted with one principal, 52 major, and 9 intermediate structures categorised.
  - The faults were imported and modelled into the geotechnical studies on caveability and extraction level stability.
  - The geotechnical properties for the Alpala deposit were determined from the intact lab strengths, investigation of the core logging database, rock mass statistics, and empirical databases where no information was available.
  - The upper (above 650 MASL) and lower databases (below 650 MASL) were separated to account for the differences noted in rock masses. For each database, the low, middle, and high cases were determined to reflect the variation within the rock strengths and rock masses.
  - Based on all available hydrogeologic data and the mine plans for underground infrastructure (including ramps, drifts, and shafts) and block caving, Itasca developed a conceptual hydrogeologic model and the three-dimensional numerical groundwater flow model.
  - The numerical groundwater flow model was calibrated to the groundwater levels measured during hydraulic testing performed in boreholes drilled across the Project.
  - Hydrological conditions are not considered to materially affect mining development rates or pumping costs relative to other block caving operations.

- Mining is planned to be a block caving mining method, whilst all horizontal development will be undertaken utilising conventional drill and blast practices.
- Access to the Alpala underground mine is expected to be via twin declines commencing from a boxcut located near the surface and the first lift near the 300 mRL.
- Vertical development for the main ventilation raises will be excavated using raisebore methods.
- The extraction level has a haulage level running underneath it that will be utilised for material handling during the production. The haul drives allow trucks to transfer the ore from ore passes to the level crushers where the crushed ore will be transferred to the main decline conveyor via transfer conveyors. These drives also form part of the ventilation circuit for the level development.
- In addition to the initial access shaft and the access and conveyor declines, the PFS design includes 13 shafts for ventilation (7 exhaust shafts and 6 intake shafts).
- The costs are based on the assumption that all construction phase works to access the mine and set up for a caving operation are completed by a mining contractor.
- Although specific equipment manufacturers and equipment models are described in this Report, no final equipment selection has been made. The equipment listed is indicative of the type and size of the equipment planned and was used as the basis for the capital cost estimate. Equipment selected is conventional within the industry.

25.9 Recovery Plan

- A conventional comminution and flotation process was selected to maximise the recovery of copper and as much gold as may be recovered to a concentrate.
- The flotation circuit was sized based on metallurgical testwork, flotation kinetic curves and standard scale-up parameters.
- The grinding circuit is designed at the 80th percentile.
- As the underground mine ramps up over a period of years, the process plant has been configured as two modules, with the second module online six years after the first module commences operation.
- LOM recoveries to concentrate are expected to be 88.4% for copper and 70.8% for gold. The concentrate itself is expected to have a copper grade of 22.0% Cu and a gold grade of 15.9 g/t Au.
- Concentrate production in the first five years is estimated to range from 126 to 505 ktpa as mine production increases. Over the Project life, concentrate production is estimated to average 465 ktpa.

25.10 Infrastructure

- Primary access to the Cascabel project process plant and mine will be via a new section of road starting from the E10 highway (two lane sealed road) that runs along the Rio Mira valley.
- Waste management facilities include sewage, solid and liquid waste and will be located at Project areas including: Cascabel concession, Coastal Plains TSF, and Port.

- On-site infrastructure includes:
  - On-site buildings will include security, mine office building with change house, process plant office, electrical rooms and control rooms, workshops, warehouses, tire bay, wash bay, canteen, first aid station and fire station, laboratory, and camp.
  - On-site non-process services such as the camp, greenhouse, sewage treatment plant and mobile equipment will support the operation. There will be fresh water, domestic water and process water systems and a fire detection and protection system. Mobile equipment for maintenance, operations services and transportation is included.
  - The permanent camp facilities will be located close to the Cascabel concession and at the Coastal Plains TSF. The camp will accommodate personnel as required for construction and operations within the Project areas. The camp facilities will be constructed early in the Project construction phase to accommodate construction and operations personnel.
- Some support facilities such as administration offices will be located off the main Project site.
- The Coastal Plains TSF is proposed for the Project:
  - Based on the Global Industry Standard on Tailings Management the Coastal Plains TSF has been assigned an "extreme" consequence category at the PFS stage and design criteria reflective of this consequence category were adopted.
  - A total of 460 Mt Rougher tailings and 69 Mt Cleaner tailings will be pumped to the TSF at 55% and 45% solids, respectively.
  - The facility will be located distal to the mine and processing plant being located on the edge of the Ecuadorian coastal plain approximately 40 km from the processing site. Separate handling and disposal of rougher and cleaner tailings will occur within the Coastal Plains East TSF.
  - The Coastal Plains East TSF Embankment will be a cross-valley embankment with a crest length of 3,300 m at final height with a height from crest to downstream toe of 190 m.
  - A geochemical assessment has been conducted on both the cleaner and rougher tailings. The rougher tailings were classified as Non-Acid Forming with a low number of enriched elements in solids. The cleaner tailings classified as Potentially Acid Forming with a moderate number of enriched elements in solids. Assessments are proposed to continue in subsequent study phases to confirm initial conclusions.
- A site-wide water balance model was developed for each of the production cases assessed as part of the PFS. The model was used to size the TSF embankment stages and to determine the amount of water that would require discharge as well as assessing the availability of water for processing operations and the required capacity of the Parambas water storage facility.
- The Port at Esmeraldas is proposed for the location of the concentrate storage and ship loading facilities. Location options are being evaluated.
- The Project includes a transmission line from the proposed hydroelectric generating stations to the Cascabel concession.

25.11 Environmental, Permitting and Social Considerations

- Baseline studies have been undertaken in the Cascabel concession only to date and are yet to commence for the remaining Project areas. The baseline studies at the Cascabel concession include meteorology, surface hydrology, surface water quality, geochemical analysis, air quality, greenhouse gas emissions, noise, flora and fauna, archaeology, and social surveys.
- Closure planning has been undertaken to support the PFS and will continue to be updated as the Project advances.
- Geochemical investigations indicate the rougher tailings will be non-acid generating and the cleaner tailings will be potentially acid generating, which has been considered in the design of the TSF.
- ENSA currently holds permits required for exploration, including the Advanced Exploration permit and associated ESIA for the Cascabel concession area.
- SolGold operates a robust community engagement program within the Cascabel concession area, with a permanent presence at the Project, and a meeting room available to the public, which is used as a base for outreach activities.
- There are no currently known environmental issues that could materially impact the ability of SolGold to extract the mineral resources or mineral reserves.

25.12 Markets and Contracts

The Project will produce copper-gold-silver containing concentrate. No contracts are currently in place for any production from the Project.

25.13 Capital Cost Estimates

The initial capital cost is estimated to be $1,554 million. The sustaining capital is estimated to be $2,655 million, and includes expansion, sustaining and closure costs. The total capital cost for the Project is $4,209 million.

25.14 Operating Cost Estimates

The total life-of-mine site operating cost is estimated to be $15.24/t processed.

25.15 Economic Analysis

- Based on SRK's financial evaluation, the Cascabel project generates positive pre- and post-tax financial results. Post-tax IRR is 24% and the post-tax NPV_8 is $3,221 million. Post-tax payback is achieved 4.1 years following the start of production.
- The Project's NPV_8 is most sensitive to changes in copper and gold price, followed by changes in copper and gold grade. The Project's NPV_8 is least sensitive to changes in silver price and grade.
- It is assumed that the 2% NSR Santa Barbara royalty will be bought out for $4 million ahead of construction and is considered a sunk cost and therefore not included in the financial model.

- SolGold recognises that a closure cost guarantee is required, but at this stage it has not been negotiated with the Ecuadorian government; consequently, closure assumptions within the financial model could vary from what is modelled.

25.16 Risks and Opportunities

25.16.1 Geology and Mineral Resource

Mineral resources are estimated based on drill hole sampling points and grades are interpolated between sampled points. Any interpolation is subject to basic assumptions of continuity of grade and geology between sampled points. As such, the mineral resource estimates are subject to some risks. Conversely, opportunities to expand the mineral resources in areas of assumed geological continuity where grade has not been confirmed because of insufficient sampling or drilling also exist.

Alpala Mineral Estimation Risk

The drilling density at Alpala is good but most drill holes were drilled from few drill platforms because of the steep topography. As a result, most drill holes are steep to vertical. Mineralisation is hosted in steeply dipping to structures. The risk that the continuity of some mineralised intervals is exaggerated because the drill holes are sub-parallel to the mineralisation does exist.

Drilling shallower holes from underground when access is permissible would help reduce the risk of possible over-estimation caused by the steep drill holes. The QP evaluated the proportion of near vertical holes and compared them with the angles holes that do exist in the database and found that in general that there was a good agreement between the vertical and incline drill holes and concluded that the risk of over-estimation was small and not likely material.

The Alpala deposit remains open at depth. Additional deeper drilling could expand the mineralisation deeper than is currently modelled.

Tandayama-America Mineral Estimation Risk

The geological interpretation at Tandayama-America is derived from surface mapping and drill hole logging. The drill hole spacing at Tandayama-America is wider than for Alpala, as such, the geological domains are less well-defined as for the Alpala deposit. Since copper and gold grades vary by geological domains, there exists the possibility that some grades were incorrectly assigned to the wrong geological domain. The QP evaluated the variability of grades between domains and concluded that this risk is not material to the global resource estimate but could have an impact on the local grade distribution. Because the deposit is likely to be mined by open pit or underground block cave, the QP doesn’t believe that this is a significant material risk.

The Tandayama-America deposit remains open for expansion both at depth and to the northwest. Additional mineralisation has been identified in some shallow drill holes to the northeast of the main deposit.

Tandayama-America Economic Potential

The Tandayama Americana deposit outcrops at surface. The QP envisions that the TAM deposit could be exploited by open-pit mining methods. An initial assessment of TAM's economic potential was performed by the QP, however, due to gaps in the technical data, the assessment does not meet the required standards of a PEA. Additional geotechnical, hydrologic work, waste rock characterisation, metallurgical testing and mine planning are required to accurately determine the viability of the TAM deposit at a PEA or PFS level.

The QP performed preliminary mine planning work, utilising Whittle pit optimisation shells and selected the TAM ultimate pit. Annual production schedules were developed and utilised to determine mine equipment requirements. Haulage profiles were developed on an annual basis to determine haulage fleet requirements. Equipment and fleet quotes were obtained to estimate capital requirements, and estimates were used for infrastructure requirements. The open pit at the TAM deposit may have the potential to provide ore to the mill as the underground block cave at Alpala is being developed. The QP recommends conducting the testwork required for a PEA to be completed.

25.16.2 Mining and Mineral Reserves

Mining technical risks in implementing cave mining of the Alpala orebody include:

- Delays in access development
- Difficulties in undercutting and ramp-up delays
- Stability and seismicity issues during cave development and propagation
- Airblast during cave development and propagation
- Fragmentation and material handling issues, including oversize, fines, and hang-ups
- Poor drawpoint interaction
- Caving block interaction
- Overdraw, dilution, and recovery issues
- Mudrushes during cave mining

These risks can be mitigated by taking the following actions:

- Acquiring comprehensive orebody knowledge through the implementation of data collection and analysis programmes for rock mass, structures, and water for cave access and development.
- Developing standards and procedures.
- Implementing comprehensive surface and underground monitoring and instrumentation programmes.
- Designing the mine to reflect industry standards and the latest learnings from deep caves.
- Undertaking numerical stability modelling and benchmarking.
- Developing a "plan B" for aspects such as cave initiation and undercutting direction.

Risks listed above can be grouped into two major periods:

1. Risks during cave development, including development stability, airblast, and seismicity.

The QP recommends the following mitigation measures:

   a. Implement an undercutting strategy that minimises damage to the rock mass due to induced abutment stress through advanced undercutting.
   b. Develop and implement a comprehensive cave monitoring and instrumentation programme prior to cave commissioning. This will enable the operator to adjust activities such as the rate and direction of undercutting and draw strategy to eliminate airblast risk and minimise potential seismic risk.
   c. Develop and implement Standard Operating Procedures (SOP) and Trigger Activity Response Plan (TARP) in advance so they can be implemented before undercutting takes place. Develop and implement a Cave Management Plan (CMP) and Ground Control Management Plan (GCMP).
   d. Consider establishing an independent Technical Review Board to advise on compliance with established procedures and industry standard practices.

2. Risks during cave production, including dilution, fragmentation oversize and hang-ups, production level stability, drawpoint brow stability, and mudrush.

The QP notes that orebody knowledge impacts all aspects of technical cave mining risks and recommends the following mitigation measures during cave production:

   a. Develop and implement comprehensive material flow instrumentation, including "Smart Markers" and "Cave Trackers" or similar tools, to enable adjustments to the draw plan.
   b. Develop a comprehensive draw management and draw compliance plan and procedures within the CMP to minimise dilution and mudrushes.
   c. Undertake a mudrush risk study, including an assessment of the physical properties of the caved material and the development of a surface water management strategy.
   d. Consider managing the subsidence zone, including water diversion, subsidence slope management, and potential pre-stripping.

25.16.3 Metallurgy and Processing

The following risks and opportunities have been identified:

- The study currently considers a radial stacker added during the first phase capable of producing a stockpile for both processing lines. Late in the study the civil pad was redesigned to reduce the earthworks and now considers a mine pad and process plant pad on different tiers. Potential alternatives such as an elevated tripper conveyor utilising the height difference of the mine pad and process plant pad may provide a better option for stockpile loading.
- The previous PFS considered a primary grind of 150 µm prior to rougher flotation. A review of the metallurgical test work indicated that, where tested, a primary grind of 200 µm provided equivalent

metal recovery although at a higher mass yield to concentrate. This study has considered the costs and designs associated with a 200 µm primary grind. Further metallurgical testing is required, over a broad range of composites and variability samples, to confirm the viability of this approach.

- The testing for the Specific Grinding Energy, which is used to size the regrind mill, was based on a receiving a top size of 150 µm and will need to be repeated for 200 µm following new flotation testing. Current values will be underestimated.
- The study considered conventional mechanically agitated tank cells for flotation. Alternative cell types may provide benefits of increased recovery, improved grade, reduced power consumption and reduced overall footprint and should be considered going forward.
- Metallurgical testing considered MIBC as a frother for the rougher stage flotation but observed that in the cleaner stages that a stronger polyglycol frother (W31) performed better. W31 was not tested in a rougher capacity but could potentially be an improvement to the flotation performance. For the purposes of estimating operating costs W31 was assumed to be utilised in both the rougher and cleaner flotation circuits.
- To reduce the overall footprint and associated earthworks the study considers shared concentrate and cleaner tails thickeners for each phase. This approach also provides a lower overall Project cost but puts additional cost into the first phase.
- As the previous PFS considered a separate gold recovery circuit which was removed from this study there may be opportunities to further improve gold recovery. Consider additional metallurgical testing focused on gold recovery, which may include alternative flotation chemistries and/or gravity concentrators in the grinding circuit that may be able to produce a gold concentrate.

25.16.4 Surface Infrastructure

There are noted risks with the Project infrastructure that are listed as follows:

- There are potentially acid forming (PAF) materials from the mine being used as fill material in the earthworks pads construction. A buffer of inert non acid generating material 1 m thick is assumed to be placed on top of the PAF material on the outer slopes of the pads. No inert material is used to cover the pads where contact water is collected in the holding pond. It is assumed this water will be treated via a passive water treatment system before discharge to the environment of that water. If there are further environmental requirements to address the use of this PAF material, it can impact the capital cost of construction.
- Hydrogeneration for power is also in the PFS stages of planning, and could potentially not be available, which may increase the supply price of power for the Project.
- The right of way for the TSF pipeline to the TSF has not been secured. It is assumed to follow the highway for the majority of its route, however, changes to the route could increase the cost due to access for construction.
- The land and location for the port facility has not been secured. A different location at the port site would add cost to construction for wharf access and could require additional equipment at the port site.

Opportunities that are potentially available to the Project include:

- An additional reduction of power cost from the hydro electric generating plan is possible, which would be a significant benefit to the operating costs.
- A tailings facility closer to the mine site would reduce the cost of the TSF overland pipe installation.

25.16.5 Tailings Management

During the course of the study several opportunities and risk have been identified which need to be addressed in the next phase of study. Key opportunities identified are as follows:

- Potential to steepen the embankment slopes as the designs are optimised.
- Potential to steepen the spillway excavation slopes as the designs are optimised.
- Cyclone sand dam construction may be a viable option for Coastal Plains Main Facility, but additional testing will be required.

Key risks identified are as follows:

- Land access and permitting has not been completed, although a memorandum of understanding is in place to purchase the land where the TSF will be located, and therefore there is a risk that access to the site may not be achievable.
- Minimal geotechnical data is available, and this could impact the designs.
- Sterilisation drilling has not been completed at the sites.
- Material availability for dam construction has not been confirmed and the geotechnical properties of the construction materials could impact the designs.
- Limited rainfall and stream flow data are available within the Coastal Plains TSF catchments, and this could impact the designs. Climate and stream flow monitoring should be established within the Coastal Plains TSF catchment.
- Streamflow data for the Mirra River is available downstream of the proposed extraction point but significant gaps are present in the dataset, stream flow monitoring should be established at the proposed extraction point.
- A probabilistic water balance model should be developed at the next stage of study to assess the variability in water demands, availability of supply and water excesses over a range of probabilities.
- The Ecuadorian guidelines prescribe pseudo static analysis for seismic design of the embankments. This method is not considered a preferred method in high seismic zone as it can be extremely conservative. Use of this analysis method may require flattening of the slopes or reduction in the height of the embankments to obtain permitting.
- The tailings management system requires discharge of water, permits to discharge water may not be granted.
- Limited tailings samples have been geochemically tested additional samples need to be tested to confirm representativeness of the current data.

- Security of remote tailings pump stations, chole stations and public access to tailings pipelines should be reviewed and included in the Project risk register.
- The Project is unique to Ecuador in scale, so a limited cost database is available for projects of this size, therefore the cost estimates may be inaccurate.

25.16.6 Cost Estimates

- There is a risk that the capital required to build and operate the project may be higher than that forecast in this study. The QP recommends that the precision of the estimates be refined before commitment to project construction is made.
- Opportunities to reduce or defer capital expenditure may be realised in future studies. Care should be taken when considering the relationship between lower capital opportunities and technical risk to the project.
- There is a risk that the operating costs incurred to operate the project may be higher than that forecast in this study. SRK notes that variability in the operating cost drivers (productivity, input costs and labour costs) over time is expected. The analysis assumes constant conditions but is best thought of as reflecting an expectation of average costs. The QP recommends that the precision of the estimates be refined before commitment to project construction is made.
- Operating costs may be lower than forecast for the purposes of this study.

25.16.7 Economic Analysis

- There is a risk that commodity prices, and therefore revenues, may be lower than assumed in this study.
- There is a risk that the schedule to build the project may vary from that assumed in the study. This is an asymmetrical risk, with significantly more downside scope than upside. This risk is exacerbated by the seasonality of the location, with more difficult construction conditions occurring in rainy months. Small delays have the potential to be more significant than might otherwise be the case if they push critical path activities into rainy months, thereby incurring a disproportionately longer delay.
- There is a risk that achieved recoveries, and therefore revenues, could be lower than estimated.
- The risk of a longer-than-anticipated permitting timeline will reduce the project value as it is considered from “today” forward.
- Higher commodity prices, both realised and forecast, should lead to re-optimisation of the mine and processing plans with a potential to create additional value beyond that shown by the sensitivity analysis.
- Further metallurgical testwork could allow for optimisation of the process flow sheet and plant design for the Feasibility Study. Recoveries, and therefore revenue, better than the current planning assumptions may be possible.

25.16.8 Environmental and Permitting

During the course of the study several risks have been identified which need to be addressed in the next phase of study. Key risks identified are as follows:

- Environmental baseline studies to date have been limited to the Cascabel concession and are yet to commence for the remaining project areas. The outcomes of these baseline studies could impact aspects of the project design.
- Water quality modelling needs to be further developed from both the on-concession mining activities and the tailings management to confirm the flows and quality of water requiring treatment post-operations, and to predict the number of years water treatment would likely be required once rehabilitation activities have been completed. This aspect could change the closure allowance required for the project.
- Social baseline studies to date have been limited to the Cascabel concession and the surrounding area but are yet to be expanded out to the broader project areas. The outcomes of these social baseline studies could impact aspects of the project design.
- Further work is required to develop a robust ESIA for the Cascabel project and may need to include resettlement action plans to be developed for the small populations located within the impact areas of the broader project development footprint.

26 Recommendations

26.1 Introduction

Two key technical work fronts have been proposed for the Cascabel project:

1. Early mine access to conduct technical investigations.
2. Continued technical works and completion of a Feasibility Study.

Continued activities associated with securing surface rights for the Cascabel project and advancement of the ESIA and mine exploitation contract will also continue within Ecuador.

26.2 In-Country Works

Continued advancement of permitting associated activities include:

- Baseline studies and investigations for Advanced Exploration and Mine Exploitation
- Preparation and submission of an ESIA for the Cascabel project
- Submission for mine exploitation contract with the Ecuadorian government

The budget for advancement of permitting activities is $4 million.

It is also recommended that securing surface rights for the proposed Cascabel mine, process plant and infrastructure continues. The budget for securing surface rights is $23 million.

26.3 Technical Works

26.3.1 Early Mine Access

The scope of the Early Mine Access project is to advance the development decline to support technical investigations for the Feasibility study. The Early Mine Access engineering works will also provide direct inputs to the Advanced Exploration permit ESIA modification.

The budget for the Early Mine Access project is $410 million.

26.3.2 Feasibility

The Feasibility study will be based on the recommended case following review of the PFS. The study will utilise the additional technical investigations to improve the capital and operating cost estimate accuracy and provide increased confidence in the financial model outcomes.

The budget for the supporting testwork and Feasibility study is $25 million.

26.4 Additional Data Collection

26.4.1 Exploration

Minor in-fill drilling works will be undertaken to investigate the Tandayama-America deposit. The budget for the exploration works is $5 million.

26.4.2 Hydrogeology

Recommendations for hydrogeology include:

- Complete additional pumping tests for the underground mine
- Continue/commence collecting water flow and quality information from the catchment areas within the Cascabel project areas
- Conduct hydraulic testing for planned geotechnical holes
- Update the surface and underground water models based on flow results
- Update surface and underground water management plans based on the results of the water models

The budget for the hydrogeology works is $2 million.

26.4.3 Geotechnical

Feasibility study and surface and UG mine geotechnical investigations include the following physical investigations:

- Diamond drilling at various lengths depending on location and sterilisation requirements.
- Test pitting where drilling is not required.

The geotechnical investigations also include testwork and reporting, as well as site visits by Knight Piésold to audit the physical investigations and logging by SolGold personnel.

Areas where geotechnical investigations are proposed include:

- Process plant and infrastructure major infrastructure areas
- Parambas valley dam wall location
- On-site roads at Cascabel
- Mining areas:
  - Underground extensions to current model
  - Underground access locations including decline
- Power corridor
- Coastal Plains TSF and infrastructure locations
- Port infrastructure locations

- Pipeline corridor

The budget for the geotechnical investigations is included in the Feasibility study budget.

26.4.4 Geochemistry

Continue sample selection and testwork across the relevant Project facilities in accordance with MEND guidelines and study phase to enable commensurate run off/inflow water quality assessments. Facilities include:

- All mine areas
- Coastal Plains TSF
- Process plant and infrastructure areas

The budget for the geochemical investigations is included in the Feasibility study budget.

26.4.5 Process

Recommendations and further testwork planned for the Feasibility study is summarised below:

- Comminution variability data set will be increased through additional sampling and testwork
- Pilot plant materials handling and HPGR testwork for key Alpala underground deposit samples
- Additional variability testwork for flotation concentrate and gold recovery
- Further testwork to confirm gold and silver recoveries to doré through the pyrite flotation and CIP processes should be undertaken
- Vendor dynamic thickening tests for rougher, cleaner tailing and final concentrate, including flocculant screening and dosage
- Final vendor concentrate filtration tests
- Threshold moisture limit (TML) tests
- Concentrate quality
- Concentrate transport tests, including: rheological tests, corrosion/erosion tests, operability tests, characterisation tests, abrasivity tests
- Tailings transport tests, including rheological and corrosion/erosion tests

The budget for the process investigations is included in the Feasibility study budget.

26.4.6 Infrastructure

- Develop run of river hydroelectric project plans with supplier and secure these projects for power supply to site
- Secure right of way for tailings pipeline to the TSF

26.4.7 Environment, Permitting and Social Considerations

- Environmental and social investigations should be extended to all Project areas.
- Water quality modelling should be extended to cover the likely closure scenario to confirm the flows and quality of water requiring treatment post-operations, and to predict the number of years water treatment would likely be required once rehabilitation activities have been completed.
- Social activities for the wider Cascabel project are in the very early stages and should commence in order to enable the proposed Project development plan.
- Further works are required to progress the mine exploitation contract and ESIA for the Cascabel project.

26.4.8 Tailings Management

Recommendations in support of TSF design and tailings management include:

- Hydrogeological study of the TSF area to determine the baseline hydrogeological regime.
- Geophysical studies to investigate the subsurface structural geology.
- Geotechnical studies to investigate dam foundation and borrow material sources.
- Establish monitoring of site-specific weather station within the TSF catchment area.
- Establish monitoring of all major streams in the tailings area for flow rates and water chemistry.

The budget for the tailings management investigations is included in the Feasibility study budget.

26.4.9 Economic Analysis

- Complete negotiations with the Ecuadorian government to establish tax rates for the government mining royalty, currency outflow tax, sovereign adjustment tax, and other negotiable taxes.
- Refine the depreciation pools between assets and between development and operations to better estimate depreciation allowances.
- Complete offtake agreements or LOIs with smelters at the next level of study to better establish final product terms and potential credits for the Cascabel ore which is clean and low in deleterious elements.
- Negotiate the Project’s closure guarantees with the Ecuadorian government.

The budget for this work is included in the advancement of permitting activities budget.

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28 Date and Signature Page

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All data used as source material plus the text, tables, figures, and attachments of this document have been reviewed and prepared in accordance with generally accepted professional engineering and environmental practices.