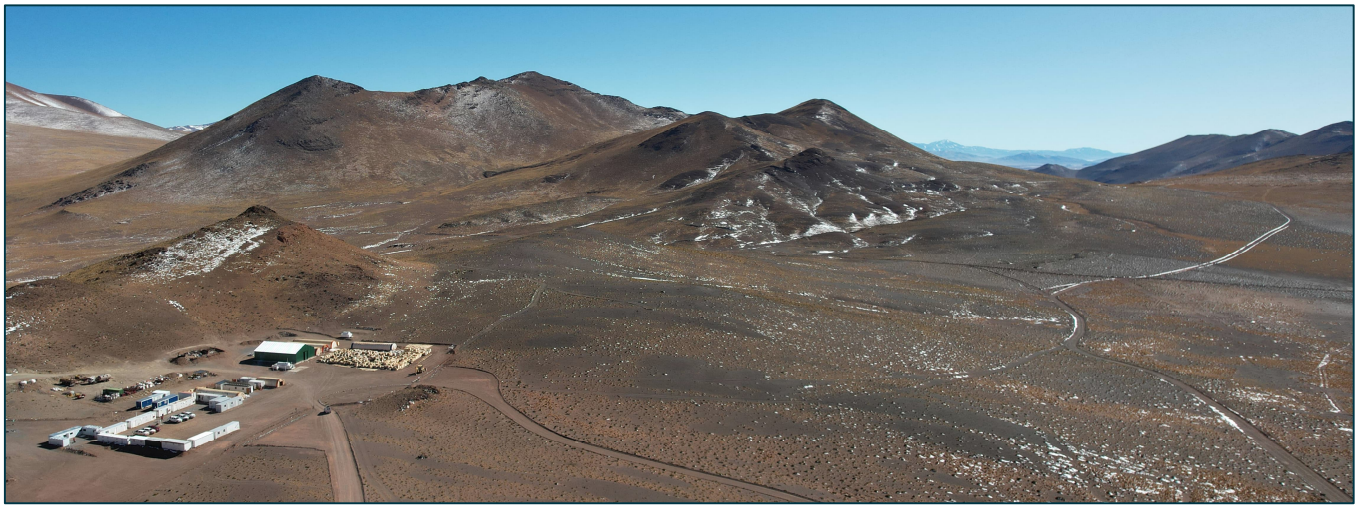
ABRASILVER RESOURCES CORP.

NI 43-101 Mineral Resource Estimate

Diablillos Silver-Gold Project



Effective Date: 06 May 2026

Report Date: 19 June 2026

Qualified Persons:

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Prepared for AbraSilver Resource Corp.

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Important Notice

This report was prepared as a National Instrument 43-101 Technical Report for AbraSilver Resource Corp. (“AbraSilver”) by a team of consultants contracted by AbraSilver (“the Team”). The quality of information, conclusions and estimates contained herein is consistent with the level of effort involved in the Team’s services, based on:

1. Information is available at the time of preparation.
2. Data supplied by outside sources as detailed herein.
3. The assumptions, conditions, and qualifications set forth in this report.

This report is intended for use by AbraSilver subject to the terms and conditions of its contracts with the consultants and the Team and to the relevant securities legislation. The contracts permit AbraSilver to file this report as a technical report with Canadian securities regulatory authorities, pursuant to National Instrument 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law or stock exchange rules, any other uses of this report by any third party are at that party’s sole risk. The responsibility for this disclosure remains with AbraSilver. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new technical report has been issued.

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Cautionary Statement

Certain information and statements contained in this report are “forward looking” in nature. Forward-looking statements include, but are not limited to: statements with respect to the economic and other parameters of the project; Mineral Resource and Reserve estimates; the cost and timing of any development of the project; the proposed mine plan and mining methods; dilution and mining recoveries; processing method and rates and production rates; projected metallurgical recovery rates; infrastructure requirements; capital, operating and sustaining cost estimates; the projected life of mine and other expected attributes of the project; the net value per block value (NVB); taxation and royalties; capital; future metal prices; the project location; the timing of the environmental assessment process; changes to the project configuration that may be requested as a result of stakeholder or government input to the environmental assessment process; government regulations and permitting timelines; estimates of reclamation obligations; requirements for additional capital; environmental risks; and general business and economic conditions.

All forward-looking statements in this report are necessarily based on opinions and estimates made as of the date such statements are made and are subject to important risk factors and uncertainties, many of which cannot be controlled or predicted. In addition to, and subject to, such specific assumptions discussed in more detail elsewhere in this report, the forward-looking statements in this report are subject to the following assumptions:

- There being no significant disruptions affecting the development and operation of the project.
- Exchange rate assumptions being approximately consistent with the assumptions in the Report.
- The availability of certain consumables and services and the prices for power and other key supplies being approximately consistent with assumptions in the report.
- Labour and materials costs being approximately consistent with assumptions in the report.
- Assumptions made in Mineral Resource and Reserve estimates, including, but not limited to, geological interpretation, grades, metal price assumptions, metallurgical and mining recovery rates, geotechnical and hydrogeological assumptions, capital and operating cost estimates, and general marketing, political, business, and economic conditions.

CERTIFICATE OF QUALIFIED PERSON

I, *Luis Rodrigo Peralta,* B.Sc. (Geo) FAusIMM CP (Geo), do hereby certify that I am author of the Sections; 1 to 12, 14, 18 to 20, 21, 23, 24, 25, 26 and 27 of the Technical Report titled "Mineral Resource Estimate For The Diablillos Ag-Au Project" prepared for AbraSilver Resource Corp. and dated June 19, 2026 (the "Technical Report").

1. My current work address is Virgen de Lourdes Oeste 1275, Capital, San Juan, Argentina, 5400.
2. I am an independent Senior Resource Geologist.
3. I graduated with a Bachelor in Geology Science from the National University of San Juan, San Juan City, Argentina in 2008.
4. I am a registered Fellow and Chartered Professional in good standing at the Australasian Institute of Mining and Metallurgy, since 2010. FAusIMM membership number 304480.
5. I have been practicing my profession continuously since 2008. My relevant experience includes over 15 years´ experience working in relevant open pit and underground mines in South America. I have advanced in position since exploration geologist, senior resource geologist to Technical Services Manager, overseen the Mineral Resource estimate at Casposo Mine, Cerro Vanguardia Mine, El Toqui Mine, Pirquitas Mine, Chinchillas Mine, and other projects. Also, I have worked as a geologist consultant evaluating projects in South America in all their levels of study: green field exploration, brownfield exploration to resource definition and mining production.
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
7. I am responsible for the preparation of sections 1 to 12, 14, 18 to 20, 21, 23, 24, 25, 26 and 27 of the Tehcnical Report.
8. I have previously participated in the preparation of six technical reports for this property, dated October 28th, 2021: January 3rd, 2022, and November 28th, 2022, April 30th, 2024, May 29th, 2024, January 17th, 2025 and September 12th, 2025, as an independent senior consultant.
9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
10. I am independent of AbraSilver Resource Corp. applying all the tests in section 1.5 of NI 43-101.
11. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
12. I visited the Property from April 24th to May 03rd, from October 2nd to October 8th, 2023, from October 18th to October 22nd, from December 6th to December 11th, 2024, and from April 8th to April 14th, from August 25th to August 29th, 2025, for the purposes of the Technical Report.

13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 19TH day of June in 2026.

“Signed and Sealed”

Luis Rodrigo Peralta, Bachelor in Geology Science, FAusIMM CP (Geo).
Fellow of the Australian Institute of Mining and Metallurgy – Membership Number 304480.

CERTIFICATE OF QUALIFIED PERSON

I, *Joseph M. Keane*, P.Eng. do hereby certify that:

1. I am an Independent Mineral Processing Engineer Consultant and a Registered Member of the SME. I contributed to the report entitled "Mineral Resource Estimate For The Diablillos Ag-Au Project" prepared for AbraSilver Resource Corp. and dated June 19, 2026 (the "Technical Report"), as an associate of the following organization: SGS North America Inc., 3845 North Business Centre Drive, Tucson, Arizona 85705, Telephone: 520-579-8315, Fax: 520-579-7045, E-Mail: Joseph.Keane@sgs.com
2. This certificate specifically applies to the Technical Report referenced above.
3. I graduated with a degree of Bachelor of Science in Metallurgical Engineering from the Montana School of Mines in 1962. I obtained a Master of Science degree in Mineral Processing Engineering in 1966 from the Montana College of Mineral Science and Technology. In 1989, I received a Distinguished Alumni Award from that institution. I have worked as a metallurgical engineer for a total of 60 years since my graduation from university.
4. I am a member of the Society for Mining, Metallurgy, and Exploration, Inc. (SME# 1682600) and am a registered professional metallurgical engineer in Arizona (#12979) and Nevada #5462).
5. I visited the property on 13th to 14th September 2022.
6. I have previously participated in the preparation of three technical reports for this property, dated May 29th, 2024, January 17th, 2025, and September 12th, 2025 as an independent senior consultant.
7. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
8. I am responsible for Section 13, 17, and 27 of the Technical Report, and I am the Qualified Person for matters relating to the information contained in those sections of the Technical Report.
9. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. As of the date of this certificate, to the best of my knowledge, information, and belief, the Technical Report sections for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites assessable by the public.

Dated this 19TH day of June in 2026.

“Signed and Sealed”

Joseph M. Keane, P.E., Q.P.
SME Membership number 1682600 and Registered Professional metallurgical engineer in Arizona #12979 and Nevada #5462.

CERTIFICATE OF QUALIFIED PERSON

I, *Miguel Fuentealba Vergara*, P.Eng. (Mining) MAusIMM, do hereby certify that I am author of the Sections 15, 16, and 27 of the Technical Report titled "Mineral Resource Estimate For The Diablillos Ag-Au Project" prepared for AbraSilver Resource Corp. and dated June 19, 2026 (the "Technical Report").

1. My current work address is Lote B8, Santa Ana, Cayumapu, Valdivia, Chile.
2. I am an independent Senior Reserve Mining Engineer.
3. I graduated with a Professional degree in Mining Engineering from the University of Santiago of Chile (USACH), Santiago City, Chile in 1998.
4. I am a registered Member of the Australasian Institute of Mining and Metallurgy, since 2007. MAusIMM membership number 226663.
5. I am a registered Member of the Chilean Resources and Reserves Mining Commission, since 2010. Membership number 100.
6. I have practiced my profession continuously since 1998. My relevant experience includes over 25 years´ experience working in relevant open pit and underground mines in South America. I have advanced in position since mine planning engineer to Technical Services Manager, overseen the Mineral Reserve estimate at Cerro Vanguardia and Gualcamayo Mine in Argentine, Carmen de Andacollo Mine from Teck and Mantoverde Mine in Chile, Minera San Gregorio Gold Mine from Orosur in Uruguay, and other projects. Also, I have worked as senior mining consultant evaluating projects in South America.
7. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
8. I am responsible for the preparation of sections 15, 16, and 27 of the Technical Report.
9. I have previously participated in the preparation of three technical reports for this property, dated May 29th, 2024, January 17th, 2025, and September 12th, 2025 as an independent senior consultant.
10. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
11. I am independent of AbraSilver Resource Corp. applying all the tests in section 1.5 of NI 43-101.
12. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
13. I visited the Property from October 02nd to October 8th, 2023, and from October 18th to October 22nd, 2024, and from August 25th to August 29th, 2025, for the purposes of this report.

14. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 19TH day of June in 2026.

“Signed and Sealed”

Miguel Fuentealba Vergara, Professional Mining Engineer, MAusIMM (Mining Eng.), Q.P.
Member of the Chilean Commission for Resources & Reserves, Member Number 100.
Member of the Australian Institute of Mining and Metallurgy, Membership Number 226663.

CERTIFICATE OF QUALIFIED PERSON

I, Shaida Miranda Gutierrez, MAusIMM CP, certify that I am employed as a Principal Mining Consultant within Mining Plus Canada Consulting LTD with an office address of Suite 504, 999 Canada Place, Vancouver BC.

1. This certificate applies to the technical report titled "Mineral Resource Estimate For The Diablillos Ag-Au Project" prepared for AbraSilver Resource Corp. and dated June 19th, 2026 (the "Technical Report").
2. I graduated from the Pontificia Universidad Católica del Perú - PUCP in 2009 with a B.S. in Mining Engineering and in 2016 with a M.Sc. in Corporate Finance.
3. I am a member of Australasian Institute of Mining and Metallurgy -AusIMM with CP #3002631.
4. I have been practicing my profession for 17 years, during which I have been directly involved in the development of mining plans in all stages of study, including operations. I have also conducted OPEX and CAPEX estimations in mining, as well as economic evaluations of open-pit mining projects and mining operations."
5. I have read the definition of "Qualified Person" set out in the 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 fulfil the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible.
6. I have not visited the Diablillos Ag-Au Project in Salta & Catamarca, Argentina.
7. I am responsible for Sections 22, and 27 of the Technical Report.
8. I am independent of AbraSilver Resource Corp. as defined in Section 1.5 of NI 43-101.
9. I have previously participated in the preparation of three technical reports for this property, dated May 29th, 2024, January 17th, 2025, and September 12th, 2025 as an independent senior consultant.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information so that those sections of the Technical Report are not misleading.
11. I consent to the use and publication of the Technical Report or any extracts or summaries thereof by AbraSilver Resource Corp.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication.

Dated this 19TH day of June in 2026.

"Signed and sealed"

Shaida Miranda, MAusIMM CP
Member of the Australian Institute of Mining and Metallurgy, Membership Number #3002631.

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1. Summary

This report was prepared as a Canadian National Instrument 43-101 (“NI 43-101”) Technical Report on a Mineral Resource Estimate (“MRE”) for AbraSilver Resource Corp. (“AbraSilver” or the “Company”) by Mr. Luis Rodrigo Peralta, FAusIMM CP (Geo), Principal Geologist and independent external consultant from INSA Consultora, for the Diablillos silver-gold project (“the Project”), located in Salta and Catamarca Provinces, Argentina.

The purpose of this Technical Report is to support the public disclosure of the updated Mineral Resource Estimate for the Diablillos Project, effective May 6, 2026. The Mineral Resource estimate has been prepared by Mr. Peralta, who is the Qualified Person (“QP”) responsible for the estimate and who considers that the available geological, drilling, sampling, analytical and density data are sufficient to support the current Mineral Resource Estimate.

The Diablillos Project is located within the Puna region of northwestern Argentina, approximately 160 km southwest of the City of Salta. The Project consists of a group of mining concessions controlled by AbraSilver and hosts several epithermal silver-gold deposits including Oculto, JAC, Fantasma, Laderas and Sombra.

Exploration work completed by several operators over the history of the Project now totals 170,368 m of drilling in 790 drill holes, including both diamond drilling and reverse circulation drilling. The current Mineral Resource Estimate incorporates drilling completed up to December 31, 2025, and includes approximately 13,270 m of additional drilling completed since the previous Mineral Resource estimate dated July 29, 2025.

The Diablillos deposits are classified as high-sulphidation epithermal silver-gold systems with strong supergene overprinting. Mineralization is principally controlled by structural corridors and favorable lithological horizons. Recent geological work has resulted in an updated three-dimensional geological, alteration and oxidation model that forms the basis of the current Mineral Resource Estimate.

Mineral Resources were estimated using Ordinary Kriging within a three-dimensional block model constrained by geological, alteration and oxidation domains. Mineral Resources have been classified in accordance with the CIM Definition Standards (2014) and are reported within a conceptual open pit shell demonstrating reasonable prospects for eventual economic extraction.

A significant advancement incorporated into the current Mineral Resource Estimate is the application of a dual-processing strategy consisting of a primary tank leach process and a secondary heap leach process. The tank leach component represents the higher-grade mineralized material, while the heap leach component captures a substantial inventory of lower-grade mineralization with potential economic significance.

Table 1-1: Total Diablillos Mineral Resource Summary (Tank & Heap Leach) – As of May 6th, 2026.

Zone	Zone	Category	Tonnes (000 t)	Ag (g/t)	Au (g/t)	AgEq (g/t)	Contained Ag (000 Oz Ag)	Contained Au (000 Oz Ag)	Contained AgEq (000 Oz Ag)
Tank Leaching	Oxides	Measured	41042	100	0.68	159	131 668	896	209 281
		Indicated	60978	41	0.58	92	81 060	1 143	180 078
		Measured & Indicated	102021	65	0.62	119	212 728	2 039	389 359
		Inferred	14400	25	0.57	74	11 468	262	34 187
Heap Leaching	Oxides	Measured	25469	13	0.09	19	10 997	76	15 425
		Indicated	104491	7	0.13	15	24 328	428	49 342
		Measured & Indicated	129960	8	0.12	16	35 325	503	64 767
		Inferred	34947	6	0.14	14	6 939	158	16 153
Total	Oxides	Measured	66512	67	0.45	105	142 665	971	224 706
		Indicated	165469	20	0.30	43	105 388	1 570	229 420
		Measured & Indicated	231981	33	0.34	61	248 053	2 542	454 127
		Inferred	49347	12	0.26	32	18 406	420	50 340

Notes for Mineral Resource Estimate:

1. Mineral Resources are not Mineral Reserves and have not demonstrated economic viability.
2. The formula for calculating AgEq is as follows: Silver Eq Oz = Silver Oz + Gold Oz x (Gold Price/Silver Price) x (Gold Recovery/Silver Recovery).
3. The Mineral Resource model was populated using Ordinary Kriging grade estimation within a three-dimensional block model and mineralized zones defined by wireframed solids, which are a combination of lithology and alteration domains. The 1m composite grades were capped where appropriate.
4. The Mineral Resource is reported inside a conceptual open pit shell derived using US$ 34.50/oz Ag price, US $3,200/oz Au price, 86.6% process recovery for Au, and 80.9% process recovery for Ag, for the tank leaching and 74.3% process recovery for Au, and 46.8% process recovery for Ag, for the secondary heap leaching.

5. Open pit optimization was constrained using a dual-process approach, with tank leaching as the primary process (total opex of US$32.58/t) and heap leaching as the secondary process (total opex of US$7.00/t).
6. The MRE has been categorized in accordance with the CIM Definition Standards (CIM, 2014).
7. A Net Value per block [NVB] calculation was used to constrain the Mineral Resource, determine the "Benefits = Revenue-Cost", where, Revenue = [(Au Selling Price (US$/oz) - Au Selling Cost (USD/Oz)) x (Au grade (g/t)/31.1035)) x Au Recovery (%)] + [(Ag Selling Price (US$/oz) - Ag Selling Cost (USD/Oz)) x (Ag grade (g/t)/31.1035)) x Ag Recovery (%)] and Cost = Mining Cost (US$/t) + Process Cost (US$/t) + Transport Cost (US$/t) + G&A Cost (US$/t) + [Royalty Cost (%) x Income]
8. The Mineral Resource is sub-horizontal with sub-vertical feeders and has a reasonable prospect for eventual economic extraction by open pit methods.
9. In-situ bulk densities were assigned to each model domain, according to samples averages for each lithology domain, separated by alteration zones and subset by oxidation.
10. All tonnages reported are dry metric tonnes and ounces of gold and silver contained are troy ounces.
11. Mining recovery and dilution factors have not been applied to the Mineral Resource estimates.
12. The Mineral Resource was estimated by Luis Rodrigo Peralta, B.Sc., FAusIMM CP (Geo), an INSA Consultora Managing Principal Geologist, and an Independent Qualified Person under NI 43-101.
13. Mr. Peralta is not aware of any environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues that could materially affect the potential development of Mineral Resource.
14. All figures are rounded to reflect the relative accuracy of the estimates. Minor discrepancies may occur due to rounding to appropriate significant figures.

The updated Mineral Resource Estimate demonstrates substantial growth compared to the previous estimate completed in July 2025. Total Measured and Indicated contained silver increased by approximately 25%, contained gold increased by approximately 48%, and contained silver-equivalent ounces increased by approximately 30%.

The most significant contribution to this growth comes from the Oculto deposit, where additional drilling successfully expanded mineralization and improved geological confidence. Additional growth was also achieved at JAC, Fantasma, Laderas and Sombra. Furthermore, the inclusion of the heap leach Mineral Resource has substantially increased the total mineralized material contained within the conceptual open pit.

1.1 Conclusions and Recommendations

Mr. Peralta considers that the Diablillos Project possesses reasonable prospects for eventual economic extraction through conventional open pit mining methods and precious metal processing technologies. The updated Mineral Resource Estimate confirms Diablillos as one of the largest undeveloped silver-gold projects in Argentina and provides a strong foundation for future mine planning, engineering studies and project development activities.

The principal recommendations arising from this report include continued exploration and resource definition drilling at Oculto East, the Oculto–JAC corridor and Sombra, additional metallurgical and engineering studies focused on heap leaching, and ongoing refinement of the geological and geometallurgical models. The estimated cost of the recommended work program ranges between US$17.5 million and US$21.0 million (Table 1-2).

Based on the results of the current Mineral Resource Estimate, the Qualified Person concludes that the Diablillos Project remains highly prospective and fully justifies continued investment in exploration, engineering and development activities.

Table 1-2: Recommended Work Program and Estimated Budget

Recommended Activity	Estimated Cost (US$M)
Additional Engineering Studies for Heap Leaching	3
Infill Drilling (15,000 m @ US$500/m)	7.5
Additional Step-Out Drilling (10,000–15,000 m @ US$500/m)	5.0 - 7.5
Additional Metallurgical test work for Heap Leaching	2.0 – 3.0
Total Estimated Budget	17.5 – 21.0

The engineering, capital cost, and operating cost parameters used in the Mineral Reserves classification in this report are based on AbraSilver's Pre-Feasibility Study, with an effective date of December 3, 2024, and report date of January 17, 2025 (the "PFS"). Subsequent to the PFS, AbraSilver is completing its Definitive Feasibility Study for the Diablillos Project (the "DFS"), which

will update and supersede the PFS. AbraSilver expects to announce the results of the DFS by the end of Q2 2026, with a supporting NI 43-101 Technical Report to follow shortly thereafter.

2. Introduction

This report was prepared as a National Instrument 43-101 Technical Report for AbraSilver Resource Corporation ("AbraSilver") by a team of consultants contracted by AbraSilver ("the Team") to complete an updated Mineral Resource estimate (MRE) of the Diablillos Project in conjunction with AbraSilver's geological, mining, process, and environmental consultants. The MRE was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and in accordance with the requirements of Form 43-101 F1.

2.1 Purpose of Report

The purpose of this report is to publish an updated Technical Report on the Mineral Resource estimate for the Diablillos Property summarizing:

- Land tenures, exploration history, and drilling.
- The Mineral Resource estimates for the Oculto, Laderas, Fantasma, Sombra and JAC deposits for both tank and heap leach mineralisation.
- The Mineral Reserve estimates for Oculto and JAC based on tank leach only as previously reported.
- A conceptual mine plan at a level to support a PFS.
- Recovery methods and process design at a level to support a PFS.
- The supporting infrastructure includes power, buildings, tailings management, and processing plant to support the conceptual mine plan.
- Environmental permitting requirements.
- Capital and operating expenditure estimates.
- Economic analysis.
- Recommendations.

2.2 Terms of Reference

AbraSilver engaged the services of the authors to write an independent NI 43-101 technical report on the Diablillos Property, located in the Puna region of Argentina. This report was prepared following NI 43-101 guidelines, Forms NI 43-101F1, and Companion Policy 43 101CP.

2.3 AbraSilver Resource Corp

AbraSilver's corporate office is located at 220 Bay St., Suite 550, Toronto, ON M5J 2W4. The Company is listed on the (TSX:ABRA; OTCQX: ABBRF).

"AbraSilver" or the "Company" is focused on rapidly advancing its 100%-owned Diablillos silver-gold project.

2.4 Qualification of Consultant

The consultants that prepared this Technical Report are specialists in the fields of geology, exploration, Mineral Resource and Reserve estimation, geotechnical engineering, open pit mining, environmental engineering, permitting, metallurgical testing, mineral processing, civil engineering, mechanical engineering, electrical engineering, mineral economics and financial analysis.

None of the Consultants or associates employed in the preparation of this report has any beneficial interest in AbraSilver. The Consultants are not insiders, associates, or affiliates of AbraSilver. The results of this Technical Report are not dependent upon any prior agreements, nor are there any undisclosed or future business dealings between AbraSilver and the Consultants. The Consultants are paid a fee for their work in accordance with a normal professional consulting practice.

2.5 Report Responsibility and Qualified Persons

The following individuals, by virtue of their education, experience, and professional association, are considered Qualified Persons ("QPs") as defined in NI 43-101. Below summarises the membership of the authors in professional institutions:

- Luis Rodrigo Peralta, Senior Resource Geologist, B. Sc., FAusIMM CP (Geo).
- Miguel Fuentealba Vergara, Senior Mining Engineer, MAusIMM.
- Joseph Keane, P.E., Independent Mineral Processing Engineer Consultant.
- Shaida Miranda, B.S. in Mining Engineering, MAusIMM CP. Senior Engineer.

The preceding QPs have contributed to the writing of this report and have provided QP certificates. The information contained in the certificates outlines the sections in this report for which each QP is responsible. Each QP has also contributed figures, tables, and portions of Sections 1 (Summary), 2 (Introduction), 3 (Reliance on other Experts), 25 (Interpretation and Conclusions), 26 (Recommendations), and 27 (References). Table 2-1 outlines the responsibilities for the various sections of the report and the name of the corresponding Qualified Person.

Table 2-1: Report Responsibility Table

Items	Description	Qualified Person	Company
Item 1	Summary	All QP's	All QP's
Item 2	Introduction	Luis Rodrigo Peralta	INSA
Item 3	Reliance on Other Experts	Luis Rodrigo Peralta	INSA
Item 4	Property Description and Location	Luis Rodrigo Peralta	INSA
Item 5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Luis Rodrigo Peralta	INSA
Item 6	History	Luis Rodrigo Peralta	INSA

Items	Description	Qualified Person	Company
Item 7	Geological Setting and Mineralization	Luis Rodrigo Peralta	INSA
Item 8	Deposit Types	Luis Rodrigo Peralta	INSA
Item 9	Exploration	Luis Rodrigo Peralta	INSA
Item 10	Drilling	Luis Rodrigo Peralta	INSA
Item 11	Sample Preparation, Analyses and Security	Luis Rodrigo Peralta	INSA
Item 12	Data Verification	Luis Rodrigo Peralta	INSA
Item 13	Mineral Processing and Metallurgical Testing	Joseph M. Keane	SGS
Item 14	Mineral Resource Estimates	Luis Rodrigo Peralta	INSA
Item 15	Mineral Reserve Estimates	Miguel Fuentealba Vergara	BMining
Item 16	Mining Methods	Miguel Fuentealba Vergara	BMining
Item 17	Recovery Methods	Joseph M. Keane	SGS
Item 18	Project Infrastructure	Luis Rodrigo Peralta	INSA
Item 19	Market Studies and Contracts	Luis Rodrigo Peralta	INSA
Item 20	Environmental Studies, Permitting and Social or Community Impact	Luis Rodrigo Peralta	INSA
Item 21	Capital and Operating Costs	All QP's	All QP's
Item 22	Economic Analysis	Shaida Miranda	MP
Item 23	Adjacent Properties	Luis Rodrigo Peralta	INSA
Item 24	Other Relevant Data and Information	Luis Rodrigo Peralta	INSA
Item 25	Interpretation and Conclusions	All QP's	All QP's
Item 26	Recommendations	All QP's	All QP's
Item 27	References	All QP's	All QP's

2.6 Site Visits

The following bulleted list describes the Qualified Persons who visited the Property:

- Luis Rodrigo Peralta, Senior Geologist, B. Sc. (Hons), Independent Consultant, INSA, visited the site from April 24th to May 03rd, from October 02nd to October 8th, 2023, from October 18th to October 22nd, from December 6th to December 11th, 2024, and from April 8th to April 14th, from August 25th to August 29th, 2025, from April 13th to April 19th, 2026.
- Joseph Keane, P.E., an Independent Mineral Processing Engineer Consultant, SGS, visited the site from September 13th to September 15th, 2022.
- Miguel Fuentealba Vergara, BME, MAusIMM CP, Independent Mining Engineer Consultant BMining Chile, visited the site from October 18th to October 22nd, 2024, from April 13th to April 19th, 2026.

2.7 Currency, Units of Measure, and Calculations

Unless otherwise specified or noted, the units used in this report are:

- metric.

- US dollars (USD or $).

This report includes technical information that required subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently a margin of error.

2.8 Effective Date

The issue date of this report is June 19th,2026. The effective date of the Diablillos Silver-Gold Project Mineral Resource Estimate is May 6th, 2026.

As of the effective date of this report, the authors are not aware of any material fact or material change with respect to the subject matter of this Technical Report that is not presented herein, or which the omission to disclose could make this report misleading.

3. Reliance on Other Experts

The QP, Mr Luis Rodrigo Peralta, has reviewed and analysed data and reports provided by AbraSilver, together with publicly available data, drawing conclusions.

This report relied on information provided by experts who are not QPs. The QP believes that it is reasonable to rely on these experts, based on the assumption that the experts have the necessary education, professional designation, and relevant experience.

- Mr. Luis Rodrigo Peralta relied upon Envis for matters pertaining to the design, capital, and operating expenditures of the AbraSilver Project tailings storage facility (TSF) as disclosed in Section 18.7.
- Mr. Luis Rodrigo Peralta relied upon data collected, summarized, and presented by AbraSilver for the areas of capital costs and operating costs.
- Mr. Luis Rodrigo Peralta has relied upon information provided by Pacific Rim Mining Corporation Argentina S.A., a wholly owned local subsidiary of AbraSilver and its Environmental Consultants Elisa Cozzi & Asociados for the Environmental Report as disclosed in Section 20.
- Mr. Luis Rodrigo Peralta has relied on land tenure information provided by AbraSilver. This includes two letters of legal opinion regarding the validity of the tenure from the legal firm, Perez Alsina Consultores Mineros, of Buenos Aires (Perez Alsina, November 13th, 2024) and Estudio Jurídico Ponferrada & Vila Melo, of San Fernando del Valle de Catamarca (Ponferrada & Vila Melo, November 11th, 2024).
- The QP, Mr. Joseph Keane has relied on the AbraSilver internal report – Diablillos Site Kinetic Humidity Cells test – 2023.

The QPs have assumed that all the information and existing technical documents listed in References Section 27 of this report are accurate and materially complete. While the QPs reviewed the information presented, they cannot guarantee accuracy and completeness. The QPs reserve the right, to revise the report and conclusions, if additional information becomes known after the date of this report.

Except for the purposes legislated under provincial securities laws, any use of this report by a third party is at that party's sole risk.

4. Property, Description, and Location

Mr. Peralta relied on land tenure information provided by AbraSilver.

The Diablillos Project consists of mining concessions and easements granted by the respective mining authorities of the Provinces of Salta and Catamarca, including overlapping concessions located in the border area between both provinces, and additional mining properties and easements in Salta and Catamarca that form part of the broader Diablillos regional land package, including properties acquired for regional exploration, logistics and/or potential water source purposes (the Diablillos Mining Properties). The Diablillos Mining Properties are held by AbraSilver through its Argentine subsidiary, Pacific Rim Mining Corporation Argentina S.A. (Pac Rim) (see Table 4-1 and Figure 4-1 to Figure 4-6).

The Diablillos Project lies within the border dispute zone between Salta and Catamarca. Both provinces have historically granted mining concessions in the same area, which has resulted in overlapping titles. The Argentine Supreme Court has consistently held that only the National Congress is competent to finally determine interprovincial borders (National Constitution, sec. 75.15) and has therefore declined to adjudicate jurisdictional disputes. In practice, until Congress resolves the conflict, the province whose authority first granted concessions may prevail for administrative and jurisdictional purposes, as recognized in the “Salar de Diablillos” Supreme Court precedent of 1985.

AbraSilver was already the registered holder of the concessions granted by the Province of Salta through Pac Rim, which they acquired from SSR Mining in 2016. In 2017, AbraSilver further strengthened its position by acquiring Minera Cerro Bayo S.A., an Argentine company that held the overlapping or potentially conflicting concessions granted by the then Mining Judge of the Province of Catamarca (now replaced by the Ministry of Mining of Catamarca, currently the granting authority in the province). In connection with the structuring of the Diablillos Project for purposes of the application to the Argentine Large Investments Incentive Regime (Régimen de Incentivo para Grandes Inversiones or “RIGI”), which requires the relevant project to be held and developed through a single project vehicle (Vehículo de Proyecto Único or VPU), Minera Cerro Bayo S.A. assigned and transferred the Catamarca concessions formerly held in its name and the overlapping the Salta concessions to Pac Rim. The relevant filings were submitted to the mining authority of Catamarca on November 10, 2025 and are currently registered in the name of Pac Rim. As a consequence of this consolidation, AbraSilver, through Pac Rim, became the owner of the mining concessions covering the Diablillos Project area, regardless of whether they were granted by Salta or Catamarca authorities, thereby eliminating title risks arising from multiple provincial grants.

Pac Rim’s application for adhesion to RIGI for the Diablillos Project was approved by Resolution No. 562/2026 issued by the Argentine Ministry of Economy on May 7, 2026 and published in the Official Gazette on May 11, 2026. The approval recognizes Pac Rim as the VPU for the Diablillos

Project and approves the related investment plan. RIGI provides eligible large-scale investment projects with certain tax, customs, foreign exchange, and regulatory stability benefits, subject to compliance with the approved investment plan and other applicable requirements. Under the approval resolution, Pac Rim must evidence, within the first two years from notification of the resolution, investments in computable assets for an amount equal to at least 40% of the minimum investment amount, representing US $80 million.

In March 2025, Salta and Catamarca executed a specific agreement for the Diablillos Project, following the precedent created in the “Sal de Oro” lithium project operated by POSCO Argentina S.A.U. Under this agreement, the provinces adopted a pragmatic approach to resolve jurisdictional, administrative, and fiscal issues: they agreed to share royalties and taxes equally, and to create an interprovincial authority to supervise environmental and operational matters. The agreement was supplemented by four protocols dealing with the area of application, the creation of an interprovincial management committee, mining royalties, and provincial taxes. In the Province of Salta, such agreement and its four additional protocols were approved by Law No. 8524, sanctioned on December 18, 2025 and published in the Official Gazette on January 9, 2026. In the Province of Catamarca, the agreement was approved by Law No. 5,940, sanctioned on May 14, 2026, and promulgated by Decree No. 626 of the Executive Branch of Catamarca, dated May 19, 2026.

Table 4-1 lists the Diablillos Mining Properties, which are comprised of the mining concessions and easements granted by both Salta and Catamarca. Figure 4-3 shows the principal Salta concessions, while Figure 4-4 shows the principal Catamarca concessions, in each case comprising the core mining properties of the Diablillos Project. Such figures do not include the additional regional mining properties and easements referred to above that form part of the broader Diablillos regional land package. Given the overlaps, adding these areas would be misleading. The correct overall property area is approximately 11,403 hectares, as depicted in Figure 4-5, which presents the principal Diablillos mining concessions.

Table 4-2 lists the concessions and easements related to water and roads.

Table 4-1: Mineral Tenure, Concessions, and Easements.

	Tenement ID		Type	Area (ha)	Date of Grant	Expiry Date
	File Number	Name				
	Diablillos - Catamarca Province					
	629/P/2009	Condor Yacu Este	Exploitation Concession	1880.14	12-03-10	N/A [1, 2]
	408/M/2003	Cerro Bayo	Exploitation Concession	1500.00	10-06-04	N/A [1, 2]
	550/M/2004	Cerro Bayo I	Exploitation Concession	1500.00	30-11-04	N/A [1, 2]
	220/A/2007	Dorotea	Exploitation Concession	718.07	27-02-08	N/A [1, 2]
	139/A/2013	Dorotea I	Exploitation Concession	2673.52	17-05-16	N/A [1, 2]
	38/1986	Condoryacu I	Exploitation Concession	100	23-08-83	N/A [1, 2]
	113/2010	Mi Belelo 3	Exploitation Concession	183	21-02-19	N/A [1, 2]
	303/2010	Natalia	Exploitation Concession	200	11-03-11	N/A [1, 2]
	Diablillos - Salta Province					
"Grupo Minero Diablillos" File 18,691	11749	Los Corderos	Exploitation Concession	598.65	13-02-84	N/A [1, 2]
	11750	Pedernales	Exploitation Concession	599.00	14-04-86	N/A [1, 2]
	11751	Renacuajo	Exploitation Concession	600.80	15-02-84	N/A [1, 2]
	11964	Relincho I	Exploitation Concession	624.66	29-04-85	N/A [1, 2]
	11965	Relincho II	Exploitation Concession	430.70	29-04-85	N/A [1, 2]
	11966	Relincho III	Exploitation Concession	668.10	29-04-85	N/A [1, 2]
	16031	Alpaca I	Exploitation Concession	300.00	03-07-98	N/A [1, 2]
	14840	Fantasma	Exploitation Concession	598.42	14-10-94	N/A [1, 2]
	19541	Alpaca	Exploitation Concession	3498.86	15-04-10	N/A [1, 2]
	745705	Alpaca III	Exploitation Concession	3149.54	25-08-21	N/A [1, 2]
	745714	Alpaca VI	Manifestation	3227.75	25-08-21	N/A [1, 2]
	745720	Alpaca VII	Manifestation	3426.75	25-08-21	N/A [1, 2]
	758823/21	Bianca X	Manifestation	2945.50	13-12-21	N/A [1, 2]
	747014/21	El Chañal	Manifestation	3500.00	07-09-21	N/A [1, 2]
	18,350	María Amalia	Exploitation Concession	221.35	09-03-06	N/A [1, 2]
	21,703	María Amalia I	Exploitation Concession	429.13	17-07-12	N/A [1, 2]

Notes:

[1]The Mining Concession does not expire as long as the concessionaire fulfils all maintenance conditions under the regulations.

[2]All Mining Concessions and Discovery Claims are in force.

Table 4-2: Concessions and Easements – Water and Roads

Tenement		Type	Area (ha)	Date of Grant	Expiry Date
ID	Name				
18927	Road Easement	Easement (Salta)	36.00	N/A	N/A
19332	Water Easement	Easement (Salta)	1.00	N/A	N/A
19333	Water Easement	Easement (Salta)	1.00	N/A	N/A
19334	Water Easement	Easement (Salta)	6.00	N/A	N/A
752594	Water Easement	Easement (Salta)	4.00	N/A	N/A
752595	Water Easement	Easement (Salta)	4.00	N/A	N/A
808224	Water Easement	Easement (Salta)	4.00	12-04-2023	N/A
808230	Water Easement	Easement (Salta)	4.00	12-04-2023	N/A
808233	Water Easement	Easement (Salta)	4.00	12-04-2023	N/A
808390	Water Easement	Easement (Salta)	4.00	12-04-2023	N/A
956772	Water Easement	Easement (Salta)	2141	06-02-26	N/A
16225	Camp Easement	Easement (Salta)	25	21-12-97	N/A
24312	Condoryacu Waste Rock Easement	Easement (Salta)	0.2	05-03-20	N/A
24312	Condoryacu Camp Easement	Easement (Salta)	0.35	05-03-20	N/A
24312	Condoryacu Plant Easement	Easement (Salta)	1.79	05-03-20	N/A
758016	Condoryacu Water Easement	Easement (Salta)	2.25	07-12-21	N/A
24312	Condoryacu Water Easement	Easement (Salta)	0.5	05-03-20	
EX-2025-02425541-CAT-SGAM#AM	Tailings Dam Easement	Easement (Catamarca)	515.7	N/A	N/A
EX-2025-00970600-CATSGAM#AM	Infrastructure and Services	Easement (Catamarca)	8284	10-04-26	N/A

Argentine mining concessions are granted in perpetuity, subject to compliance with certain conditions imposed on the concession holder. Among these conditions is the obligation to pay an annual canon to the province, in two semi-annual instalments payable in advance. The instalment corresponding to the first semester of each year is due by December 31 of the preceding year, and the instalment corresponding to the second semester is due by June 30 of the same year. According to AbraSilver, the canon was paid through the first semester of 2026 in respect of those properties that are subject to payment of canon. The second semester payment will be due on June 30, 2026.

AbraSilver does not own the surface rights over the concessions. However, under the Argentine Mining Code, mining concessions grant their holders easement rights over the concession area. Where the land is fiscal, as in the case of Diablillos, no compensation is required. AbraSilver either holds, or can readily obtain, all necessary permits to carry out proposed activities on the property. The biannual update of the Environmental Impact Report (EIR) covering drilling and other exploration activities (EIR for Exploration) is reported to be in good standing for the relevant properties. In Salta, the advanced exploration EIR for the Diablillos Project was approved by Resolution No. 63/24 and remains valid until July 2026. Certain environmental reports or updates for newly acquired or requested properties and easements remain pending approval, as applicable. In Catamarca, the advanced exploration EIRs for the Diablillos Project were reported to be in good standing. However, with respect to the mines named “Natalia” and “Mi Belelo 3”, which were recently acquired by Pac Rim, the corresponding EIRs remain pending submission.

The Environmental and Social Impact Report covering construction and operation (EIR for Exploitation) was submitted to the provincial authorities of Salta and Catamarca on September 30, 2024. In Salta, the EIR for Exploitation was approved by Resolution No. 048/2026 issued by the Mining Secretariat on April 24, 2026, and the corresponding Environmental Impact Declaration (DIA) was granted for such stage, subject to Pac Rim’s continued compliance with the terms, conditions, obligations, monitoring requirements, and other commitments set forth in such resolution and its annexes. In Catamarca, the EIR for Exploitation remains under review; according to AbraSilver, the required public hearing was already held and approval is expected in the near term.

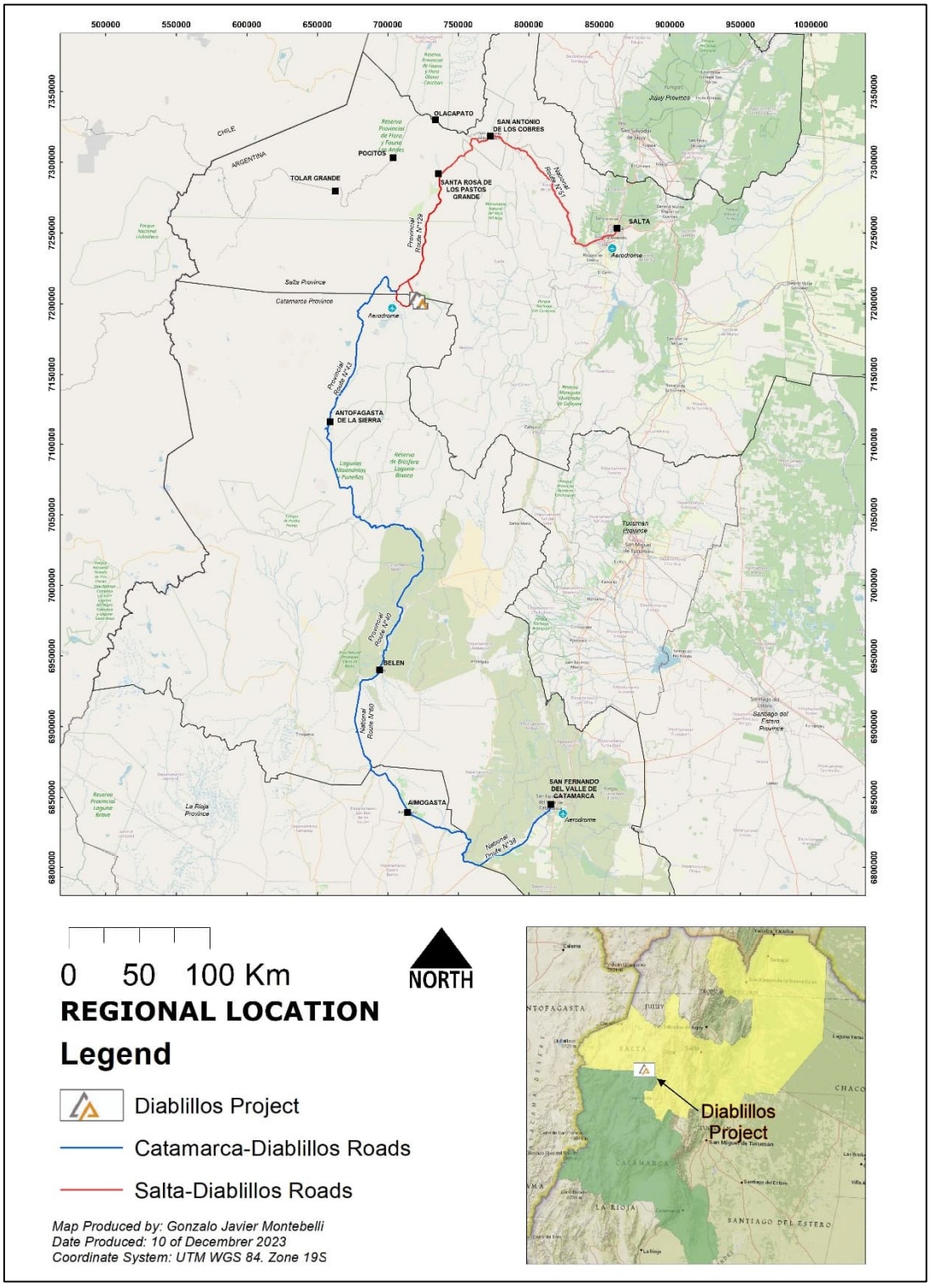
Pursuant to Resolution No. 562/2026 issued by the Argentine Ministry of Economy on May 7, 2026 and published in the Official Gazette on May 11, 2026, Pac Rim’s application for adhesion to RIGI and the investment plan for the Diablillos Project were approved. The resolution describes Pac Rim as the Single Project Vehicle (VPU) for the Diablillos Project and requires Pac Rim, within the first two years counted from notification of the resolution, to evidence investments in computable assets for an amount equal to or greater than 40% of the minimum investment amount, representing US $80 million.

Other than the matters described above, Mr. Peralta is not aware of any other significant factors or risks that could affect access, title, or AbraSilver’s ability to execute the proposed work program on the Diablillos property.



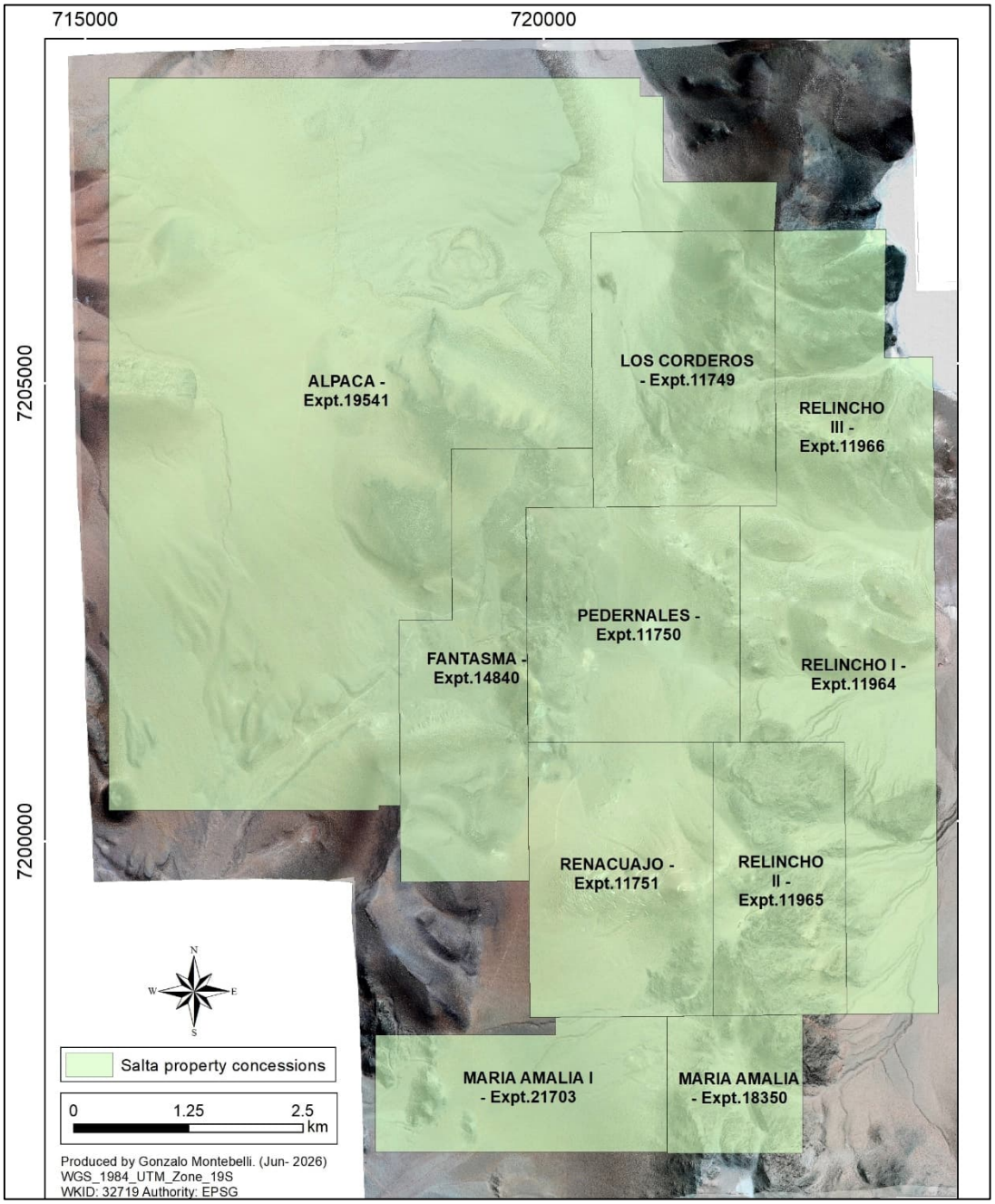
Source: AbraSilver Resource Corp., 2023

Figure 4-1: Property Location



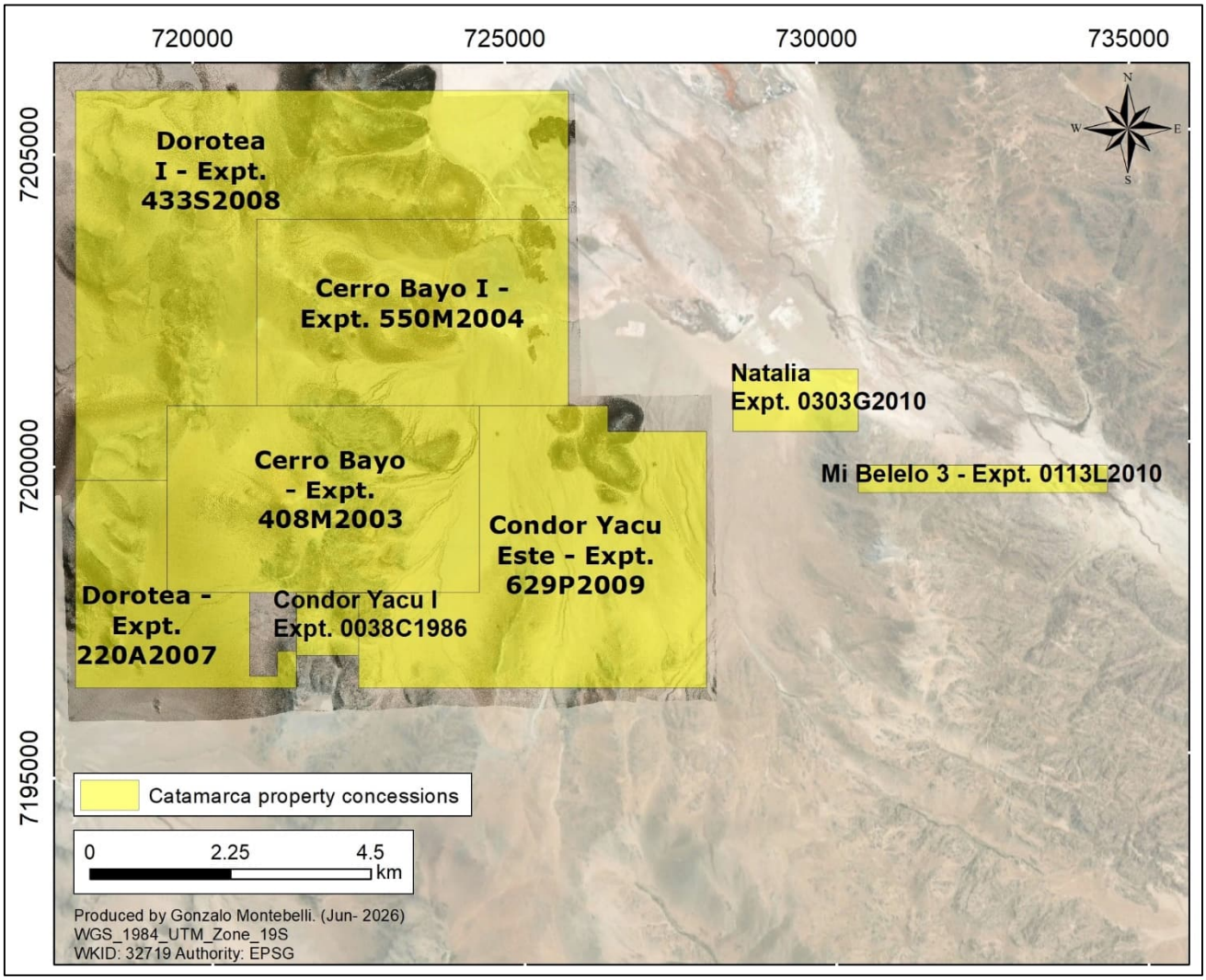
Source: AbraSilver Resource Corp., 2025

Figure 4-2: Detailed Property Location



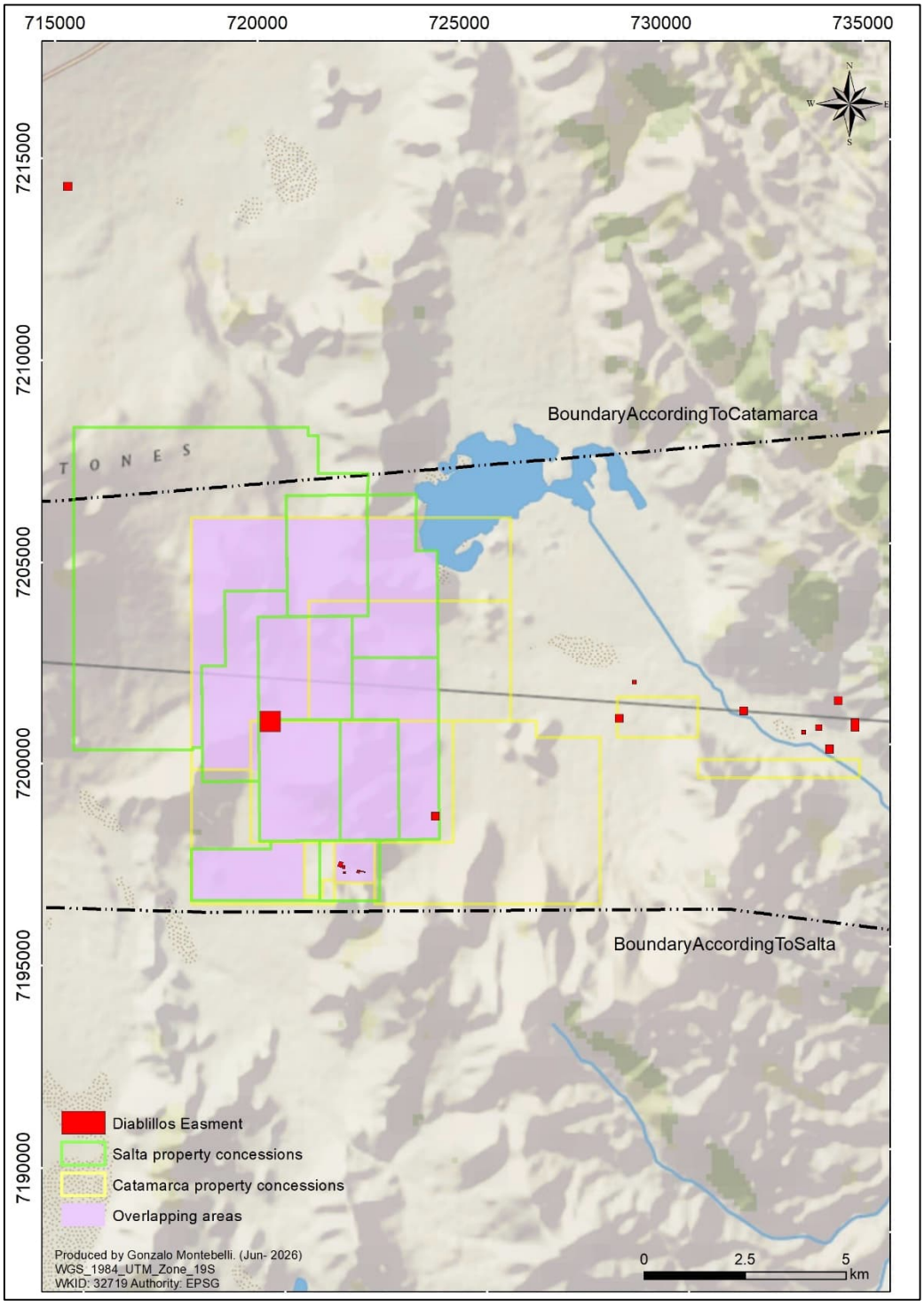
Source: AbraSilver Resource Corp., 2025

Figure 4-3: Salta Property Claims



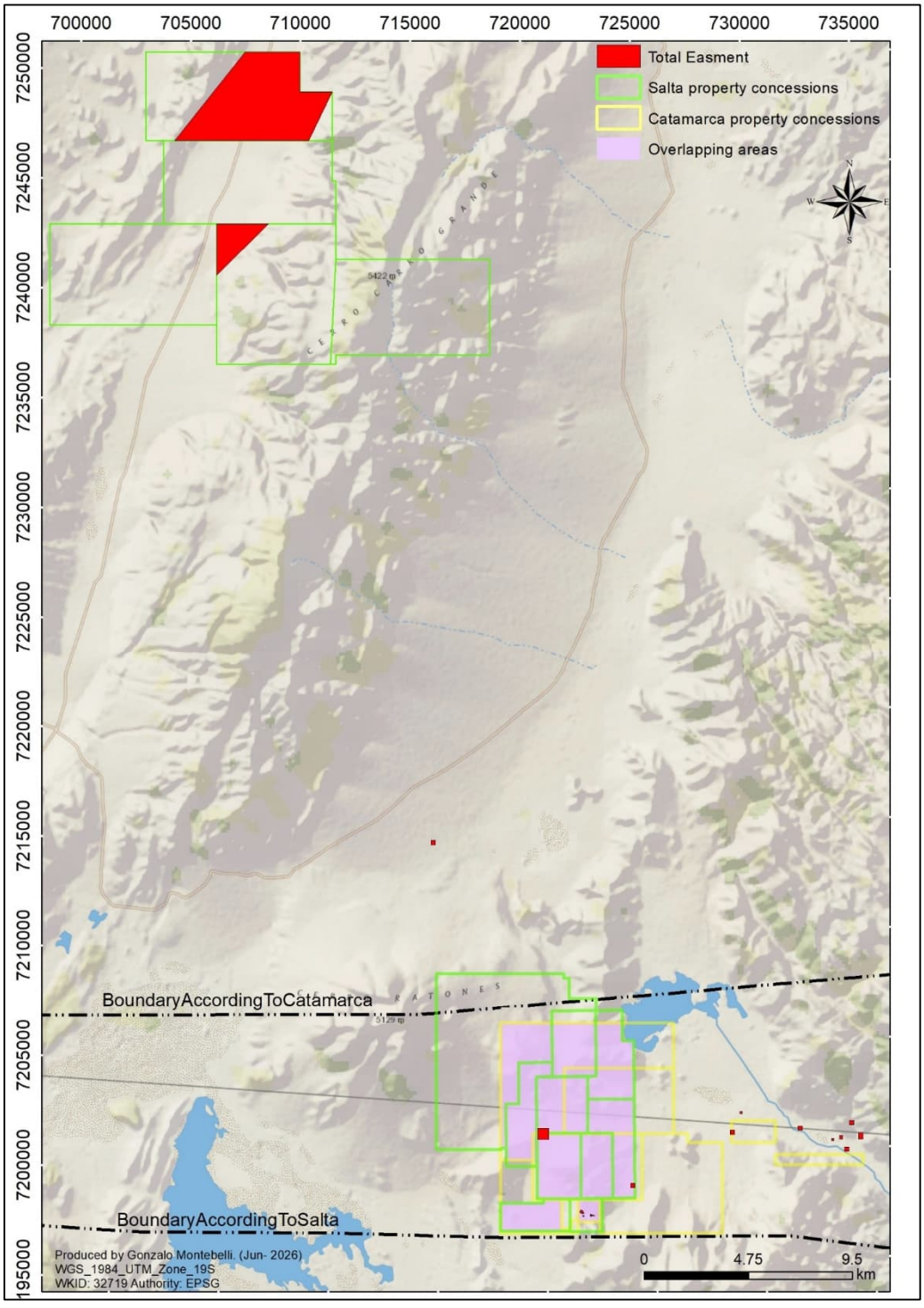
Source: AbraSilver Resource Corp., 2025

Figure 4-4: Catamarca Property Claims



Source: AbraSilver Resource Corp., 2025

Figure 4-5: Total Property Claims, Salta, and Catamarca Provinces



Source: AbraSilver Resource Corp., 2025

Figure 4-6: Total Water Easement, Salta, and Catamarca Provinces.

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

5.1 Accessibility

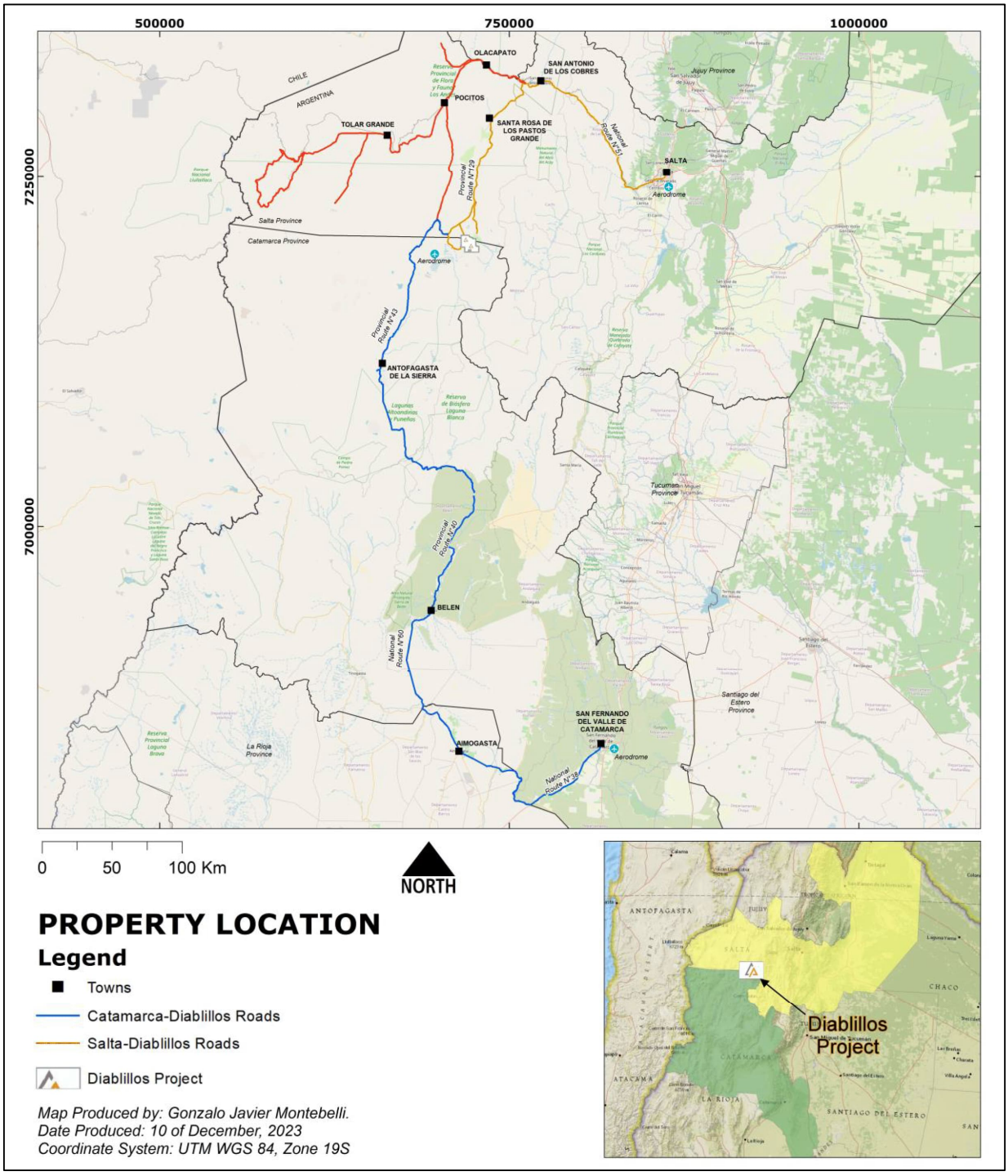
The Diablillos property is accessible from the City of Salta via the Town of San Antonio de Los Cobres along National Highway 51 (Figure 5-1). From there, a secondary all-weather gravel road leads south to Santa Rosa de Los Pastos Grandes and then to the property. Via this route, the distance from Salta to the property is approximately 320 km, totalling a driving time of five to six hours. Alternatively, the property can be reached via the Town of Pocitos on Provincial Route 17. This is the primary road access to Borax Argentina's Tincalayu operations, which are located a few kilometres southwest of the Diablillos property on the northeastern margin of the Salar del Hombre Muerto. A secondary route exists from the City of Catamarca via Provincial Route 60 to Aimogasta, National Route 40 to Hualfin and, finally, Provincial Route 43 to Antofagasta de la Sierra and a gravel road to Diablillos.

Most of the local roads are gravel and can be traversed by two-wheel drive vehicles with high clearances. However, during rainy periods, sections of the access roads are subject to flooding and small landslides. Four-wheel drive vehicles are required to access areas within the property.

Road maintenance is performed by the "Dirección de Vialidad de Salta" (Salta Province Highway Authority). Notably, a plan was recently announced to the Pastos Grandes community for Provincial Route 129. This road connects San Antonio de Los Cobres with the Salar del Hombre Muerto. The Project is approximately 19 km southeast of this road. If the Provincial Route 129 project comes to fruition, it will improve site access and reduce the length of road that will need to be maintained.

The closest good quality air strips are located at Arcadium Lithium (formerly "Livent Corporation"), which is 10 km southwest of the property, and at the Posco lithium mine, which are approximately 15 km northwest of the property.

It should be noted that the Salta government has expressed an interest in building a "Mining Logistic Centre" in the Town of Olacapato, which is located _ km from the property. This project seeks to improve the local mining infrastructure and includes an airport, industrial area, transportation, processing, service facilities, commercial premises, accommodation, parking facilities, and a health centre. While the Diablillos project does not rely on this infrastructure, there would be considerable benefits to the Project if it progresses. It is also a good sign of continuing commitment to mining projects by the regional authorities.



Source: AbraSilver Resource Corp. (2023)

Figure 5-1: Accessibility to the Property

5.2 Physiography

The property is located within the “Puna” physiographic region, an Andean highland with broad valleys separating mountain ranges exceeding 3,500 metres above seas level (masl). The Puna extends southwards from central Peru, across the altiplano of Peru and Bolivia, and south along the spine of the Andes that separate northern Chile and Argentina. Elevations on the property range from 4,100 masl to 4,650 masl. Although located at a high elevation, the local relief is moderate to gentle.

Vegetation is sparse and typically comprised of upland grasses and stunted shrubs.

5.3 Climate

The climate is arid, with annual precipitation less than 200 mm per year. However, in some years, no precipitation was registered.

The only reliable meteorological data comes from the Fenix meteorological station, which is owned by Minera del Altiplano S.A. and is located in the western basin of the Salar del Hombre Muerto, approximately 45 km southwest of the Project. According to historical data, the mean annual precipitation was 82.2 mm/year between 1992 and 2020.

Rain falls mainly during February and March. Temperatures measured in the project area range from a minimum of -26°C to a maximum of 32°C, with an annual average of 5.1°C. Strong northwestern and western winds of more than 45 km/h are common in the area, particularly during winter and spring.

5.4 Local Resources

Salta is the largest city in the region, and is serviced by daily commercial flights, major highways, and a narrow-gauge railway to Antofagasta, Chile. It is the principal source of supplies, fuel, and equipment. The nearest permanent communities are San Rosa de Los Pastos Grandes and San Antonio de Los Cobres, with estimated populations of 150 and 1,500 inhabitants, respectively. Limited basic supplies and fuel may be purchased in San Antonio de Los Cobres.

The Town of Pocitos is located approximately 100 km north of the property and is the nearest access point to the railway and electrical power grid. Two solar plants are located approximately 130 km north of the property and are operated out of Pocitos and Olacapato.

A gas pipeline was recently completed that runs from Pocitos towards the Salar de Hombre Muerto Lithium mine. It is located approximately 24 km from the Diablillos property.

Furthermore, the Government of Salta is planning to construction second pipeline (the Gasoducto Productivo Salteño (GPS) (Salta Productive Gas Pipeline)) pursuant to Decree N°248/21, issued on March 23, 2021. The Government of Salta declared that this pipeline is in the public interest and empowered the Energy and Mining Resources of Salta (Recursos Energéticos y Mineros de

Salta SA - REMSA) to move forward with the project. AbraSilver is in close contact with the government to secure future use of natural gas from this pipeline if sufficient capacity becomes available.

Recently, a joint venture between Central Puerto and IFC announced the technical feasibility of building a 140 km powerline with a 400MW capacity from Olacapato to San Antonio de Los Cobres and Salar de Pocitos. Several mining projects will connect to this powerline. The Diablillos Project assumes onsite self-generation until the viability of alternate energy sources is assessed. This new powerline is one potential option.

AbraSilver identified an aquifer near Oculto in the upper part of the Barranquillas valley basin. It is believed that this aquifer holds sufficient water to support operations at the current production rate. At this time, the Project has permission to use this water for exploration.

5.5 Infrastructure

There is a small exploration camp at the property that can accommodate approximately 80 people. The property has good access to local resources, including power, water, and personnel.

There are large areas adjacent to the Diablillos deposit that could potentially serve as areas for tailings impoundment, waste rock disposal, and plant facilities.

6. History

This section builds upon the prior reports, AbraSilver PFS Update (2025), AbraSilver PFS (2024), AbraSilver MRE (2023), AbraSilver PEA (2022), RPA (2018), Ronning (1997) and Stein (2001). In the property's exploration history, particularly before 1980, the property extents and locations of work do not appear clearly known. Consequently, it should be noted that some of the work reported from those early years may not have been conducted within the current boundaries of the Diablillos property.

6.1 Prior Ownership

Exploration in the area surrounding Diablillos began in the 1960s, when the Directorate General of Military Manufacturing (Dirección General de Fabricaciones Militares), an arm of the Argentine military, evaluated the Argentine Puna for porphyry-style deposits of copper and molybdenum. Exploration directed specifically at Diablillos began in 1971, when the National Mining Secretary (Secretaría de Minería de la Nación "SMN") undertook geological and geochemical reconnaissance work at a scale of 1:50,000. On December 31, 1971, the property was included in a federal government mineral reserve area for copper-molybdenum porphyry deposits. This status however expired in 1984 (Stein,2001).

Ronning (1997) reported that Abra de Mina, an Argentinean prospecting partnership, acquired what now constitutes the Diablillos property in the late 1970s. Stein (2001) and Wardrop (2009), however, report that this occurred in 1984. Stein further reported that, at that time, rights to the adjacent Condor Yacu property were held by Manfredo Arheit, of Buenos Aires.

Shell C.A.R.S.A, a joint venture between Shell and Billiton, explored in the area from 1984 to 1987, and optioned Diablillos in 1985.

The Ophir Partnership Ltd. ("Ophir"), a U.S. limited partnership, optioned the property in early 1987. Minera Utah International Ltd., a subsidiary of Broken Hill Proprietary Ltd. ("BHP"), began preliminary reconnaissance exploration in the area the following year. By late 1989 they concluded agreements with Ophir and Abra de Mina. The property was held by BHP until September 1991, when the option agreement with Abra de Mina was terminated.

In 1992, Pacific Rim optioned the property from Abra de Mina and acquired 100% of the property on July 1, 1997 (Stein, 2001). Pacific Rim conducted exploration work until 1996, when Barrick Exploraciones Argentina S.A., a wholly owned subsidiary of Barrick Gold Corporation ("Barrick"), obtained an option on the shares of Pacific Rim Mining Corporation Argentina S.A. Barrick continued exploration and initiated preliminary environmental impact and metallurgical studies.

SSRM Mining ("SSRM") acquired all assets of Pacific Rim Mining Corporation Argentina S.A. in December 2001, for a staged total of US$3.4 M, paid as a combination of cash and shares.

On November 1st, 2016, AbraSilver Resource Corp. (“AbraSilver’), formerly AbraPlata Resource Corp. (“AbraPlata”) and Angel BioVentures Inc. originally acquired the mining concessions granted by the Government of Salta through an agreement with SSRM Mining (“SSRM”) and Pacific Rim Mining Corporation Argentina S.A. (an Argentinian company and the registered owner of the Diablillos property). Under this agreement, AbraSilver acquired, through a merger with Huayra Minerals Corporation, certain subsidiaries of SSRM, including Pacific Rim Mining Corporation Argentina S.A.

As consideration for the payment concessions, SSRM received USD 6.35 million in cash payments and 24.15 million in AbraSilver common shares comprising 17.65% of the issued shares at that time.

To fulfil the agreement terms, AbraSilver paid a cash payment of USD 7 million to SSRM on construction start-up or at the fifth anniversary. (July 31st, 2025).

In addition to these payments, SSRM is entitled to receive 1.0% net smelter return (“NSR”) royalty on production from the project.

As of September 6th, 2017, AbraSilver completed the definitive documentation necessary to acquire a 100% equity interest in Minera Cerro Bayo SA (“Cerro Bayo”). The owner of the conflicting mineral rights granted by the government of Catamarca, thereby indirectly acquiring ownership and control of any conflicting mineral interests. As consideration, AbraSilver will pay US$3.325 million in cash (USD 0.96 million paid) and issue 500,000 (Issued) common shares to the shareholders of Cerro Bayo in instalments over a five-year period.

On September 11, 2019, AbraPlata and Aethon Minerals Corporation (“Aethon”) entered into a binding agreement whereby AbraPlata acquired all the issued and outstanding shares of Aethon. The transaction value was approximately $10.9 million on a fully diluted in-the-money basis, and Aethon and AbraPlata shareholders received approximately 46% and 54% of the combined entity, respectively.

On March 4th, 2021, AbraPlata formerly changed its name to AbraSilver Resource Corp.

On July 29th, 2021, SSRM announced the sale of their royalty portfolio to EMX Royalties. This transaction included the 1% NSR held on Diablillos project as well as the remaining USD 7 million payment which is due in 2025 (or upon commencement of commercial production).

On March 24th, 2025, and April 10th, 2025, the purchase payments for Minera Cerro Bayo SA and Pacific Rim Mining Corporation Argentina SA were completed; consequently, no further payments are due, and AbraSilver now owns 100% of the mining concessions through these two subsidiaries.

On November 13th, 2025, EMX Royalties merged with Elemental Altus to become Elemental Royalty Corporation.

6.2 Exploration and Development History

Work completed on the property throughout its history is summarized Table 6-1.

Table 6-1: Exploration and Development Work Conducted

Diablillos Project History		
Year	Operator	Description
Pre 1983	Secretaría de Minería de la Nación	1,409 rock chip samples (includes 190 outcrop and 271 slope debris samples from Diablillos Sur)
1984-1987	Shell C.A.R.S.A	Rock geochemical survey; three Winkie drill holes
1987	Ophir Partnership	34 rotary drill holes (approximately 30 m deep) in the Corderos, Pedernales, Laderas, and Jasperoide areas
1988-1991	BHP	Geological mapping (1:1,000 to 1:7,500 scale); 380 rock chip samples; 1,200 m of bulldozer trenches; 56 air RC holes (6,972m)
1991	BHP	"Mineral Resource and Reserve Estimate" (see below)
1992-1993	Pacific Rim Mining Corporation	Five diamond drill holes (1,001.8 m) in the Oculto zone
1994	Pacific Rim Mining Corporation	148 km of chain and compass grid; geological mapping; 122 line-km of ground magnetic survey; 34 line-km of induced polarization (IP) survey; 213 hand auger samples; 2.5 km of trenching; 250+ rock chip samples; 12 diamond drill holes (2,016 m)
1996-1999	Barrick Gold Corp.	Geological mapping; surface sampling; RC drilling; CSAMT survey; mag survey; environmental impact study; metallurgical test work
1999	Pacific Rim Mining Corporation	Mineral Resource estimate
2001	D. M. Stein (Barrick)	MSc thesis
2001	Pacific Rim Mining Corporation	Mineral Resource estimate (see below)
2003	Pacific Rim Mining Corporation (for Silver Standard)	20 diamond drill holes (3,046 m)
2005	Pacific Rim Mining Corporation (for Silver Standard)	Five diamond drill holes each at Renacuajo and Alpaca, with a total of 10 diamond drill holes with 1,772m
2007	Pacific Rim Mining Corporation (for Silver Standard)	54 diamond drill holes (10,324 m) on Oculto; one hole (203 m) at Laderas; three holes (unknown length) at Pedernales; five holes (unknown length) at Los Corderos; four HQ-size diamond drill holes sampled for metallurgical tests
2008	Pacific Rim Mining Corporation (for Silver Standard)	52 diamond drill holes (7,971 m), three of these for geotechnical studies; additional metallurgical studies
2009	Silver Standard Resources Inc.	Mineral Resource Estimate
2011-2012	Silver Standard Resources Inc.	Internal Preliminary Economic Assessment, rock chip sampling, 1,679 m diamond drilling (19 holes)

Diablillos Project History		
Year	Operator	Description
2015	SSR Inc.	Internal Mineral Resource Estimate for Oculto. (Executed by MFW)
2016	SSR Inc.	Mineral Resource Estimate for Oculto. (Executed by RPA, 2016)
2017	AbraSilver	28 drillholes and a total of 3,148.5m (Fantasma), and redefining the geology and mineralization concepts
2018	AbraSilver	Preliminary Economic Assessment (PEA 2018) including updated Mineral Resource Estimate
2019	AbraSilver	Phase I Drilling campaign with 2 diamond drill holes (844 m) (Oculto deposit)
2020-2021	AbraSilver	Phase II Drilling campaign of 55 drillholes and a total of 15,143 m expanding the Oculto deposit to the north, west and east
2021	AbraSilver	Preliminary Economic Assessment (MP PEA 2021) including Mineral Resource Estimate
2021-2022	AbraSilver	Phase II - Part B - Drilling campaign of 84 drill holes and a re-logging campaign totalling 106 drill holes. Drilling extended the west and north breccias at the Oculto deposit and drilled the discovery hole at the JAC zone. Infill drilling converted “Indicated” resources to the “Measured” category at Oculto.
2022	AbraSilver	Updated Mineral Resource Estimate (MRE22)
2022-2023	AbraSilver	Phase III - Drilling campaign of approximately 110 drillholes at JAC zone, focused on defining a new high-grade deposit located southwest of Oculto.
2024	AbraSilver	Pre-Feasibility Study in junction with updated Mineral Resource estimate and Mineral Reserve estimate. Start of Phase IV drilling campaign.
2024	AbraSilver	Phase IV - Drilling campaign of approximately 48 drillholes at JAC zone, focused on defining its limits and mineral classification. A new cover target was found “Sombra”. Some peripherical target were also drill as Laderas, Fantasma, Oculto NE, Alpaca among others. Phase IV has not been included in this report as still in progress.
2025	AbraSilver	Updated Pre-Feasibility Study. An update to the PFS (2024) was done to capture the new law RIGI (Régimen de Incentivo para Grandes Inversiones, Ley 27.742).
2025	AbraSilver	Phase V - Drilling campaign of approximately 80 drillholes at Oculto NE, focused on extending and defining its limits and mineral classification. Some peripherical target were also drill as Cerro Blanco and Cerro Viejo among others.
2025	AbraSilver	Updated Mineral Resource Estimate including both tank and heap leaching process. NI 43-101 Technical Report, Mineral Resource Estimate for the Diablillos Ag-Au Project. Salta & Catamarca Provinces, Argentina. Report date: September 12th, 2025.

Diablillos Project History		
Year	Operator	Description
2026		Phase VI - Drilling campaign of approximately 50 drillholes at Oculto NE and JAC, focused on extending and defining its limits and mineral classification. Some peripherical target were also drill as Cerro Blanco and Cerro Viejo among others. Phase VI has not been included in this report as still in progress. Continuity of a DFS study at feasibility-level engineering, including a new mine plan.

Source: AbraSilver Resource Corp, 2026.

6.2.1 1970s to 2012

Throughout the Diablillos Project, several prospecting and exploration works have been carried out (Table 6-1). Prospecting was carried out by the National Mining Secretary (Secretaría de Minería de la Nación) and Shell C.A.R.S.A, including geochemical rock sampling and geology mapping.

Various exploration activities were subsequently followed up by Ophir Partnership, BHP, Pacific Rim Mining Corporation, Barrick Gold Corp, and Silver Standard Resources Inc ("SSRI") between 1987 until 2012. This included:

- Geological mapping.
- Rock chip sampling.
- Trenching.
- Geophysical studies: induced polarization (IP) survey, ground magnetic survey, CSAMT survey; mag survey.
- Drilling with diamond drill holes and rotary drill holes.
- Mineral Resource estimation and metallurgical test work.

In 1990, BHP Utah drilled a single RC hole in the Fantasma zone.

In 2010, SSRI commissioned M3 Engineering and Technology Corporation ("M3") to conduct an internal Preliminary Economic Assessment ("PEA") on the Oculto zone. This was completed in June 2011.

In 2011, SSRI re-sampled historical trenches at Fantasma. The following year they then drilled four diamond holes at Fantasma. These holes intersected mineralization, but drilling was not extensive enough to result in a Mineral Resource Estimate for Fantasma.

6.2.2 2015

SSRI retained MFW to update the Mineral Resource estimate for Oculto.

6.2.3 2016

SSRI commissioned to RPA the audit of the estimate and prepared a Technical Report, issued in November 2016 (RPA, 2016). This Technical Report was filed on SEDAR and is publicly available.

6.2.4 2017

AbraPlata executed a drilling campaign that was designed to explore the Fantasma deposit, totalling 28 drill holes and 3,148.5 m.

6.2.5 2018

Abraplata commissioned RPA to produce a PEA. This Technical Report was filed on SEDAR and is publicly available.

6.2.6 2019 to 2021

A drill campaign was designed to expand the Oculto deposit to the north, west and east. Targets were selected to track mineralized structures identified through geochemical, lithological/alterations analysis and structural maps.

Mineralised areas were re-interpreted based on the relation of vertical feeder structures to sub-horizontal permeability zones. Particular attention was given to the intersection of the Main and Cross breccias. Emphasis was also placed on defining zones of shallow mineralization shown on maps prepared by Nick Tate (2018).

Infill drilling was carried out to increase the confidence level of the Oculto zone enabling an expansion of Mineral Resources in the Measured category.

Overall, the 2019-2021 campaign contributed to advances into the geological model and a better understanding of the areas, behaviour, and continuity of mineralization.

6.2.7 2021 to 2022

An updated Mineral Resource estimate was performed in October 2021 with an effective date of October 28th, 2021. A subsequent PEA was then completed with an effective date of January 13th, 2022. Both Technical Reports were performed by Mining Plus.

A Phase II drilling campaign was designed to extend the North and West breccias and to recategorize the Indicated Mineral Resources to Measured. A total of 143 drillholes were included in Phase II, totalling 35,827 meters, from DDH-20-001 to DDH-22-040. In July 2022, AbraSilver decided to update the Mineral Resource estimate (MRE22) with effective date as November 28th, 2022.

The reports are filed on SEDAR.

6.2.8 2022 to 2023

Phase III drilling campaign was designed to extend some known areas and re-categorize Mineral Resources. As part of this drilling campaign a set of brownfield exploration drillholes were performed in the surrounding area of Oculto. Hole DDH-22-019 was one of those, which lead to the discovery of the JAC zone. The company then decided to concentrate in this zone for the second half of 2022 (Phase III) until July 2023.

A total of 110 drillholes were drilled in the JAC zone. In August, AbraSilver decided to update the Mineral Resources of the Diablillos project and produced a maiden Mineral Resource estimate for the JAC zone.

6.2.9 2024

The company decided to advance to a PFS, incorporating the recent maiden Mineral Resource from JAC and produce a maiden Mineral Reserve for the Diablillos Project. A Technical Report was prepared with an effective date of March 7th, 2024.

In May 2024, a Phase IV 20,000m exploration drilling campaign was started, ending in March 2025. The results of this drilling, of which 24,055 metres had been completed as of the date of this report.

Phase IV drilling campaign has been designed to define limits of the JAC orebody, as some margins were still open. Also, a few re categorization drillholes were made to migrate inferred resources to indicated or measured. In addition, peripherical targets as Alpaca, Fantasma, Laderas, the junction between Fantasma were drilled. It's important to mention that a new zone has been discovered named as "Sombra", currently under drilling definition.

6.2.10 2025

The company decided to advance to an Updated Pre-Feasibility Study, incorporating the recent RIGI law for the Diablillos Project. A Technical Report was prepared with an effective date of January 20th, 2025.

In April 2025, a Phase V Drilling Campaign of 20,000m of exploration drilling was started. Phase V drilling campaign has been designed to define limits of the JAC orebody, as some margins were still open. Also, a few re categorization drillholes were made to migrate Inferred Mineral Resources to Indicated or Measured at the Oculto deposit. In addition, peripherical targets as Oculto NE will be drilled, related to the new high grade area at the northeast of the Oculto zone.

The result of this drilling campaign is described in the report "NI 43-101 Technical Report, Mineral Resource Estimate for the Diablillos Ag-Au Project. Salta & Catamarca Provinces, Argentina. Report date: September 12th, 2025". This estimate includes both tank and heap leaching process.

In July 2025, the company decided to advance to a Feasibility-level engineering, including new mine plan. In parallel, the Company has commenced early-stage planning for engineering, procurement and construction activities

6.2.11 2026

In January 2026, a Phase VI Drilling Campaign of 20,000m of exploration drilling was started. Phase VI drilling campaign has been designed to define limits of the JAC orebody, as some margins were still open. Also, a few re categorization drillholes were made to migrate Inferred Mineral Resources to Indicated or Measured at the Oculto deposit. In addition, peripherical targets as Oculto NE will be drilled. Phase VI has not been included in this report as still in progress.

In March 2025 new mineral resource estimate commenced and is the basis of this report.

6.3 Past Production

No prior production has been reported from the property.

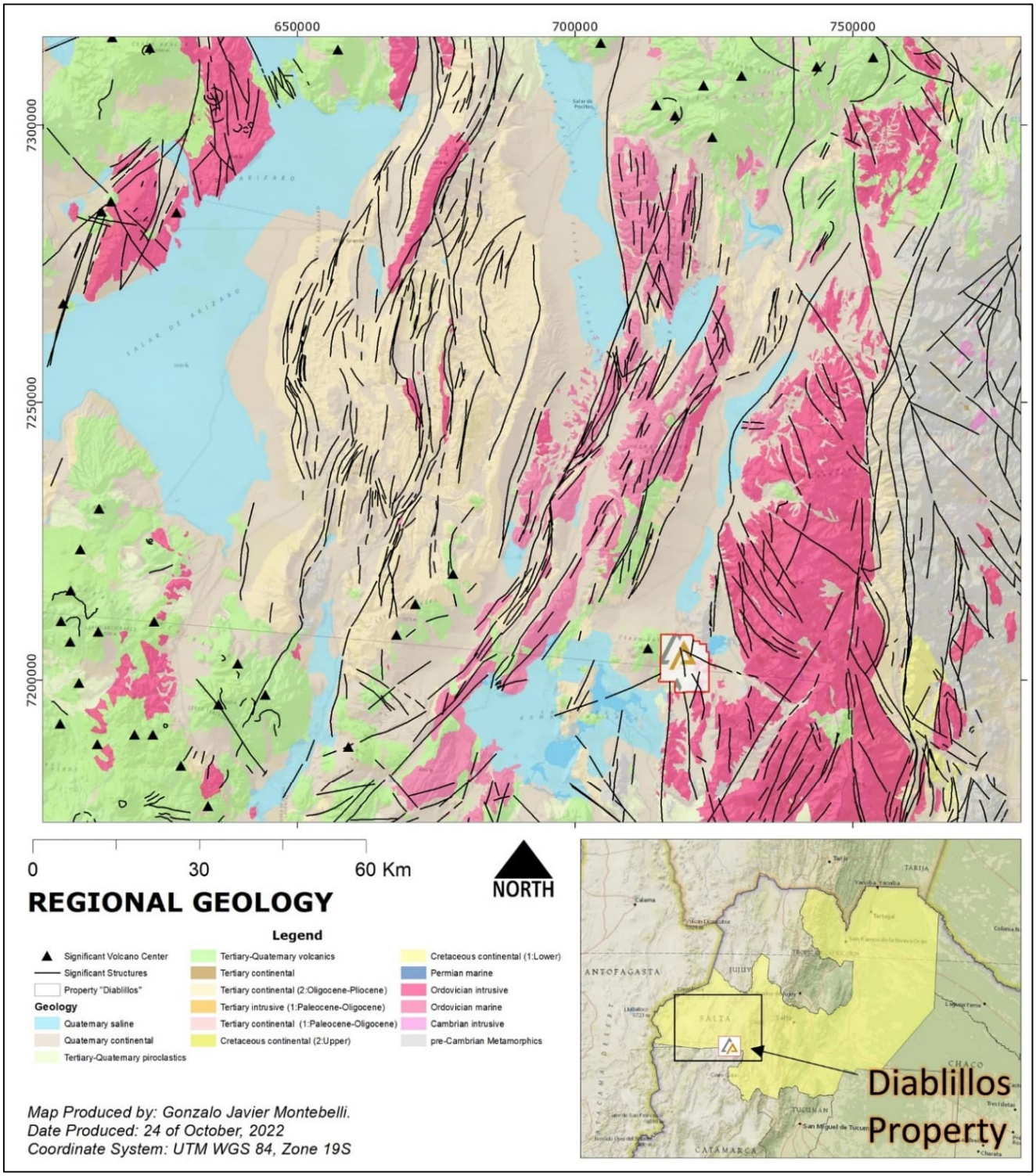
7. Geological Setting and Mineralization

7.1 Regional Geology

The Diablillos Property is located in the Argentine Puna region. The region is the southern extension of the Altiplano of southern Peru, Bolivia, and northern Chile and a plateau, separating the Cordillera Oriental to the east and the Andean Cordillera (Cordillera Occidental) to the west.

The Cordillera Occidental is a modern volcanic arc formed by the subduction of the Nazca Plate below the continental South American Plate. The Cordillera Oriental, or Precordillera, is an older north-south trending mountain chain extending 1,000 Km from the Argentina-Bolivia borders to Neuquén. These domains are separated by north-south trending regional scale faults (Figure 5-1).

During the mid-Miocene Quechuan Orogeny, the subduction zone beneath the Puna gradually steepened as the South American plate overrode the Nazca plate. Extensive late Miocene to Pliocene volcanic activity occurred along the western margin of the Puna Plateau and along northwest-southeast conjugate structures. Easterly to northwest-southeast directed compression resulted in the creation of reverse fault-bounded intra-arc basins. Uplift began in the Early Miocene accelerating in the Middle Miocene resulting in an elevation change in the order of 2,500 m. The current elevation in southern Puna averages 4,000 masl.



Source: AbraSilver Resource Corp., 2022

Figure 7-1: Regional Geology, including Faults

7.2 Local Geology

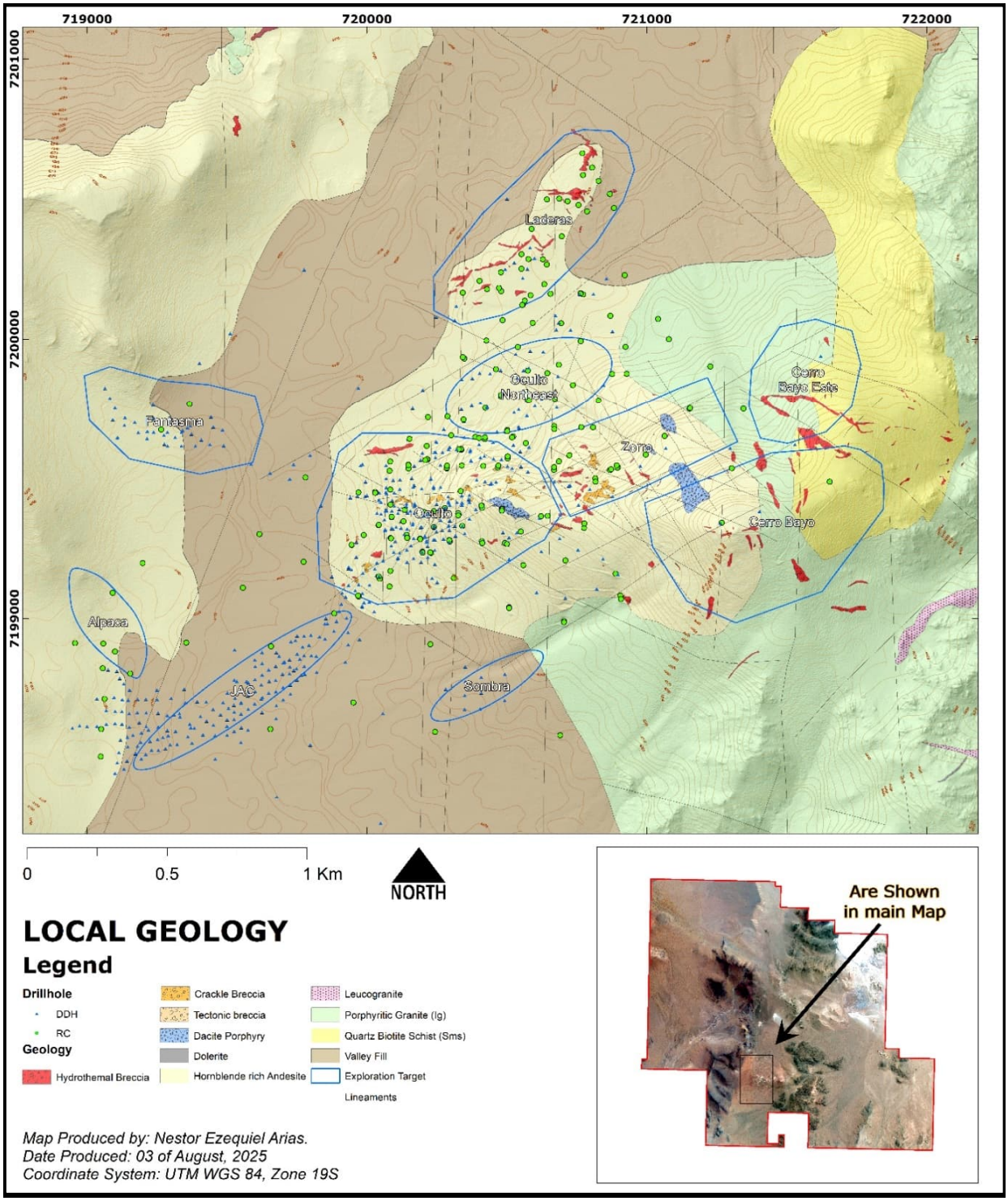
The Diablillos deposit lies close to the eastern margin of the Puna and close to the intersection of the north-south trending Diablillos-Cerro Galán fault zone with the north-westerly trending Cerro Ratones lineament (Figure 7-2). The Diablillos-Cerro Galán fault structure is one of several major north south brittles to ductile shear zones in the Puna. These were formed during neoproterozoic and lower Palaeozoic tectonism and then reactivated during the Mesozoic and Cenozoic.

Ronning (1995) lists the following regional lithologic units occurring in the vicinity of the property:

- Stocks and Extrusive Domes:
  - 12 to 15 Ma-old sub-volcanic intrusive and extrusive, frequently associated with tephra deposits from low volume, plinian to phreatomagmatic eruptions. They are generally K2O-rich dacitic rocks with biotite and occasional amphibole mafic phenocrysts, and accessory apatite, ilmenite, allanite, and tourmaline.
- Cerro Ratones Volcanics:
  - Reportedly of Oligocene age (30 ± 3 Ma), but a recent 40Ar/39Ar age of approximately 7 Ma for biotite from a flank unit at Cerro Ratones indicates a possible wider age range.
- Faja Eruptiva Granitoids:
  - Magmatic rocks of broadly Ordovician age, widespread in north-western Argentina, including a belt known as the Faja Eruptiva de la Puna Oriental, or simply the Faja Eruptiva. This belt extends from approximately 27° South latitude in Argentina to approximately 22° South latitude in southernmost Bolivia. In the Diablillos area, the Faja Eruptiva is spatially coincident with the Diablillos–Cerro Galán fault zone.
  - Rocks of the Faja Eruptiva form large and elongate bodies of porphyritic and equigranular, partly hypabyssal granitoids rich in sedimentary xenoliths. Near Diablillos, rocks assigned to the Faja Eruptiva contain feldspar phenocrysts up to 4 cm long. They follow a calc-alkaline differentiation trend and are peraluminous. Based on five U-Pb age determinations, the igneous rocks of the Faja Eruptiva are believed to be middle Ordovician.
- Ordovician Sediments:
  - The Faja Eruptiva intrudes and is folded with a sequence of Ordovician metasedimentary rocks. Near Diablillos, these rocks are phyllites, metasiltstones, and quartzites. Farther north, the Ordovician metasedimentary rocks contain late Ordovician fossils, in contradiction to the middle Ordovician radiometric ages for the Faja Eruptiva.
- Precambrian Units:
  - The pre-Ordovician basement of the eastern Puna has been termed the Pachamama Igneous-Metamorphic Complex. It consists of three subparallel north south belts 200 km long. The Diablillos property is situated near the western margin of the eastern belt, which comprises metamorphosed pelitic, psammitic, and granitic rocks that have been intruded by younger granitoids of the Faja Eruptiva.

Disseminated and vein occurrences of northern and central Puna are characterized by base metal, gold, silver, tin, and antimony mineralization. They are commonly associated with small, potassic-

rich, Tertiary stocks and extrusive domes. These intrusive/extrusive features have been dated at 15 ± 2 Ma (Sillitoe, 1977, in Coira et al., 1993, quoted in Ronning, 1997). Elsewhere, the salars (salt flats) in the vicinity of Diablillos host borate and lithium.



Source: Modified by AbraSilver, based on Grosse y Guzmán (2017) and on geology maps from SEGEMAR and Schnurr et al. (2006).

Figure 7-2: Simplified Geology of Diablillos Project

7.3 Lithology Units

The Diablillos Property hosts several zones of high-sulphidation epithermal alteration and mineralization with a strong supergene overprint. The main zone of mineralization, the Oculto zone, is hosted by a subaerial volcanic sequence, ranging in composition from pyroxene–hornblende to biotite-hornblende andesite (Figure 7-3). These volcanic rocks have been age dated by Stein (2001) and assigned to the Middle Miocene Tebequincho Formation. Basement rocks comprise Ordovician-age alkali-feldspar, porphyritic granite of the Complejo Eruptivo Oire and Neoproterozoic to Cambrian age metasedimentary rocks of the Complejo Metamorfico Rio Blanco. Small altered dacitic bodies have also intruded the basement and andesitic sequence (Stein, 2001).

The volcanic rocks are spatially restricted to areas west of the Pedernales Fault. These rocks are divided into two groups by the Jasperoide Fault Younger andesite flows and tuffs to the west, older pyroclastic and apron-bedded breccias to the east. Hydrothermal breccias form pipes and dikes throughout the area from the Jasperoide Fault in the west to the Demonio Fault east of the property boundary. The basement complex is exposed in many areas, except to the west of the Jasperoide Fault.

Basement phyllites are restricted to the far north-western corner of the map area and to the east of the Demonio Fault. The phyllites contain approximately 2% by volume quartz boudinage with molybdenum and iron oxide staining.

The Faja Eruptiva granite of the basement complex occupies a 1.5 km wide north-south strip through the centre of the map area. The granite contains numerous xenoliths of the quartz mica schist and locally is sheared to ultra-mylonite's. Subsequently pervasively silicified and injected with sheeted quartz veins. The largest of these shear zones forms a prominent ridge on Morro Eco, in the vicinity of the Cerro Viejo prospect (Figure 7-3).

The Faja Eruptiva granite is hosted in a quartz mica schist, located primarily west of the Pedernales Fault, east limited by the Demonio Fault. The schist exhibits substantial deformation denoted by tight small-scale folding, enhanced on weathered surfaces by differential weathering. Where altered, the schist changes in appearance. White in colour, with alteration of dark micas to light-coloured clays. In more intensely altered zones, the schist is completely silicified. Imparting a sugary quartzite appearance on broken surfaces, the relic folded texture is however maintained including on weathered surfaces.

The basement complex is intruded by tertiary stocks and dikes and mantled by their extrusive equivalents. The stratigraphically lowest unit of the tertiary volcanic units exposed between the Jasperoide and Pedernales faults consists of fragmental andesites Generally strongly clay altered and not forming natural exposures. The best artificial exposures observed are located at field station (fs) DW 38 on the DAR 6 drill platform. At this location, a fault, oriented at 000°/62°E, limits alteration to the west. This has preserved a pod of fresher andesite fragmental. The fragmental is believed to be overlain by a lithic pyroclastic like on top of the Oculto zone. This

pyroclastic unit is relatively rare and has only been found in outcrop in one locality. Observed resting on top of the andesite fragmental.

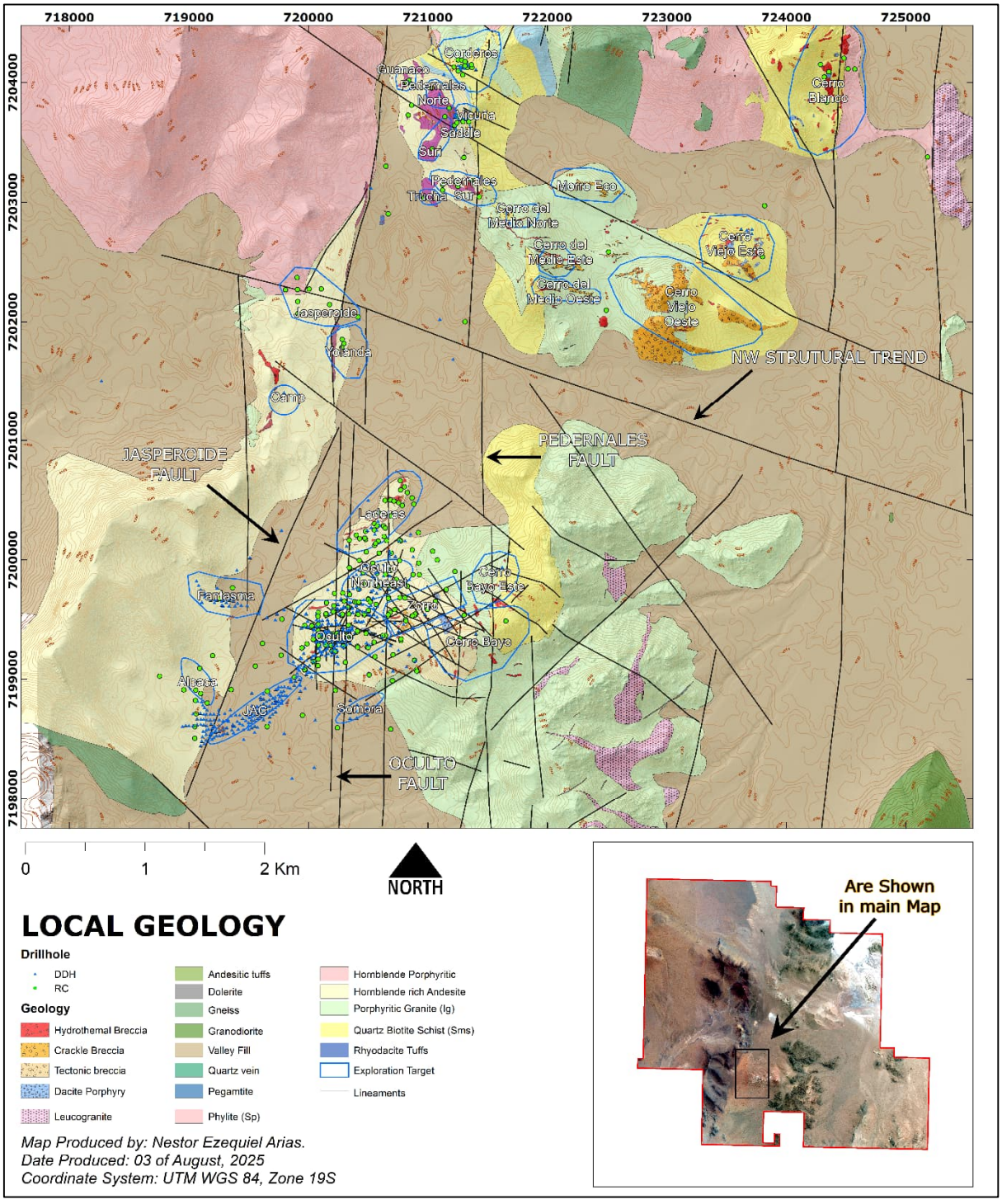
The uppermost rocks in the volcanic stratigraphic column are apron breccias. These are heterolytic breccias which form prominent exposures and are locally well bedded. The strike and dip of the bedding ranges from 110°/05°SW at la Trucha to 237°/22°NW at Guanaco. Indicating a source to the east. At least two distinct phreatic eruption events have occurred. The first dominated by clasts of andesite composition, followed by a more heterolytic clast event from the earlier andesite. Locally, the apron breccias exhibit evidence of sedimentary reworking with channels and cross bedding.

Hydrothermal breccias crosscut all lithologies except for the younger andesites west of the Jasperoide Fault and basement phyllites. The clasts in the hydrothermal breccias strongly reflect the host rock into which they were injected. They nearly always contain clasts of Faja Eruptiva porphyritic K-spar granite. It is this cross-cutting of the andesite fragmental that was the primary criterion to originally differentiate the hydrothermal breccias from the apron breccia, which they can closely resemble. The hydrothermal breccias form isolated round to elongate pipes and dike structures. The largest of the exposed pipes measures 70 m by 150 m and is located at the north end of Cerro del Medio (Figure 7-3). The largest of the dike-like hydrothermal breccias is discontinuously exposed over a strike length of 550 m. These dikes form three sub-populations with respect to their orientation including:

1. Striking 076° with strong silica-alunite alteration.
2. Striking 100° with strong silicic alteration.
3. Striking 167° with mixed silica and silica-alunite alteration.

Groups “a” and “b” are concentrated in the lower central part of the property. Group “c” is the least common and is restricted to the east.

The Tertiary intrusives are largely quartz-feldspar porphyry and form small dikes and stocks on Cerro Viejo Este in the south-east. The porphyry exhibits a close spatial relationship to hydrothermal breccia; however, no porphyry clasts have been observed even where enveloped by the porphyry.



Source: Internal mapping from AbraSilver Resource Corp., 2026

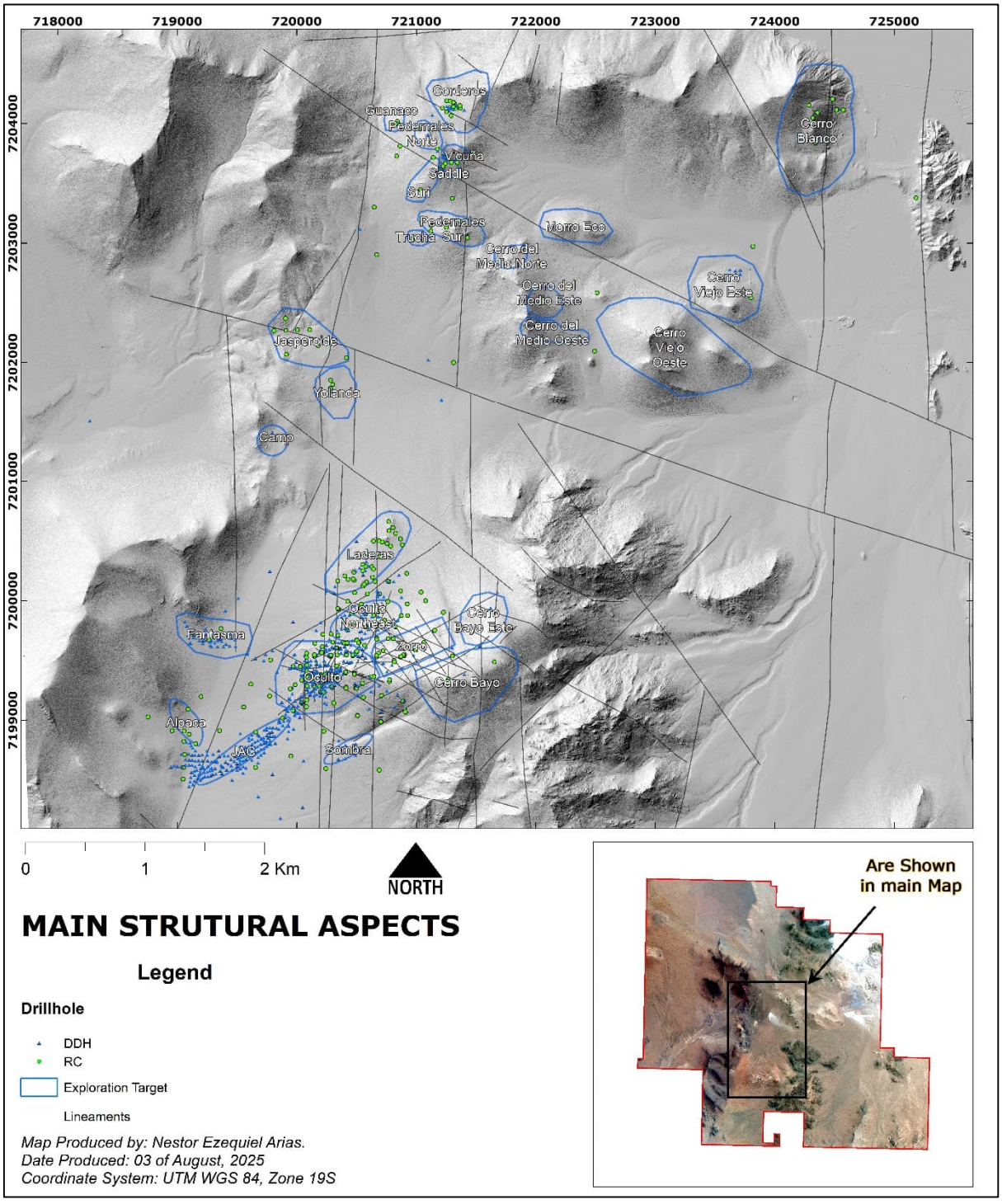
Figure 7-3: Main Geologic Aspect and Lineaments of Diablillos Project

7.4 Structural Features

The Diablillos Property lies near the intersection of two regional structures. Comprising the north south Diablillos-Cerro Galán Fault, and the northwest trending Cerro Ratones lineament. Within the project area there are two north-trending faults, the Pedernales, in the central portion, and the Jasperoid in the west (Figure 7-4). These faults bracket a wedge-shaped graben which hosts most of the altered volcanic rocks. The graben range are 2.7 km wide at Oculto with a maximum of 4.5 km to the north and a minimum at Pedernales where it is 800 m wide.

Numerous east-west and northwest-southeast structures branch from the main Diablillos - Cerro Galán corridor. These structures are considered to have channelled local magmatic and hydrothermal activity. The northwest-trending structures appear to be related to regional movement along the Cerro Ratones lineament.

The Tertiary stratigraphy is generally flat lying with gentle dips. The underlying Ordovician and Precambrian rocks have been strongly deformed and metamorphosed resulting in a wide range of structural orientations.



Source: Internal mapping from AbraSilver Resource Corp., 2026

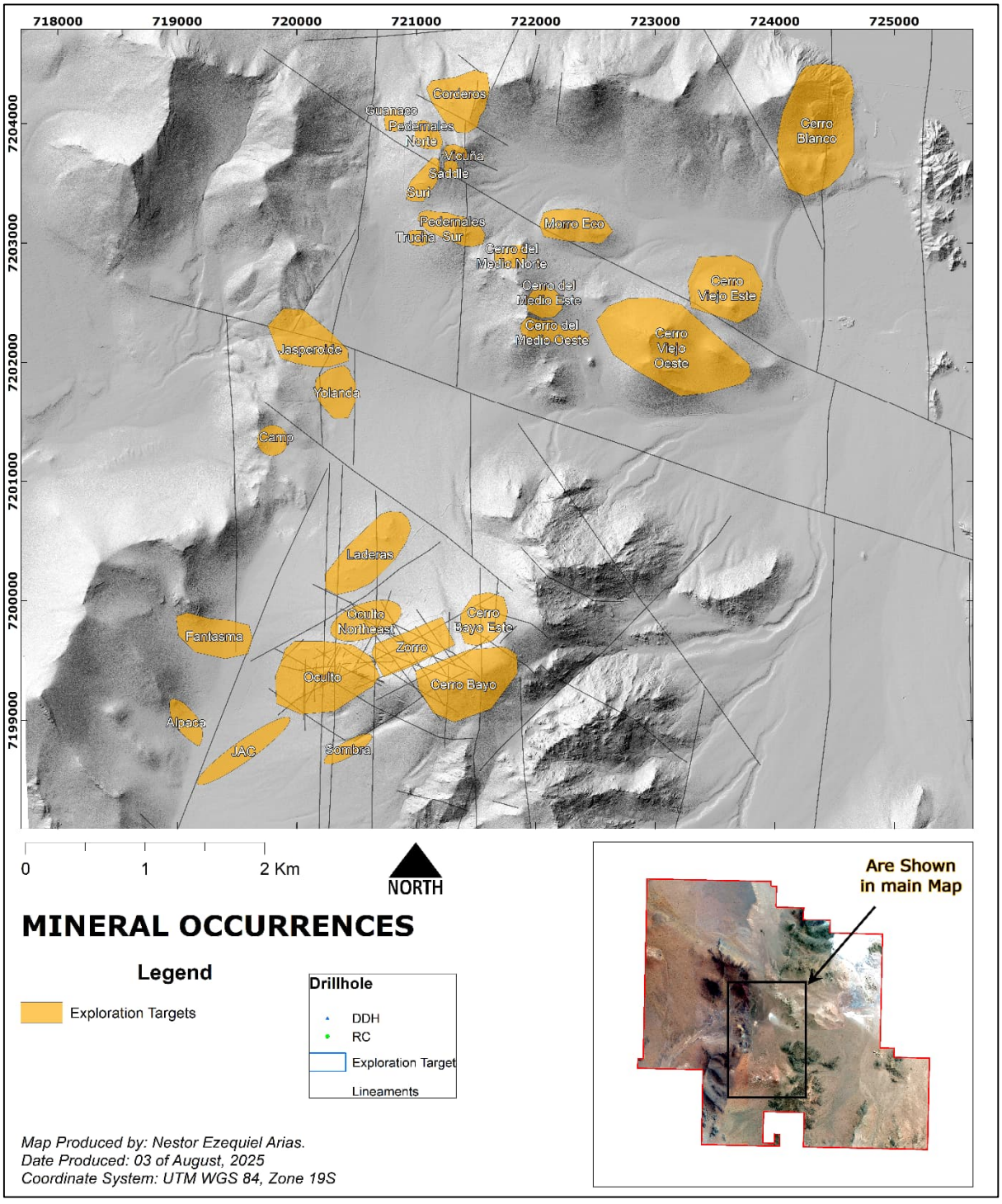
Figure 7-4: Main Structural Aspects and Lineaments of Diablillos Project

7.5 Mineralization

There are several mesothermal, and epithermal precious and base metal occurrences situated along the trend of the Diablillos-Cerro Galán Fault zone. Within the northern and central Puna, including Diablillos, Incahuasi, Cóndor Yacu, Inca Viejo, and Centenario (Figure 5-1 and Figure 7-2). Many of the mineral occurrences are spatially, and probably genetically, related to small Tertiary stocks and extrusive domes. They can be hydrothermally altered with disseminated and vein - hosted lead, zinc, silver, and gold (± tin, antimony, copper, and molybdenum) mineralization (Coira et al., 1993, quoted in Wardrop, 2009 and RPA, 2018).

There are several known mineralized zones on the Diablillos property. Oculto, JAC, JAC North, Fantasma, Laderas and Alpaca zones are the most important known to date (see Figure 7-5). The known mineralized zones are:

1. Oculto (including the Oculto NE and Deep)
2. JAC and JAC North
3. Fantasma
4. Laderas
5. Alpaca
6. Sombra
7. Pedernales including the Pedernales Sur subzone (including Truchas and Saddle showings) and Pedernales Norte subzone (including Vicuña, Corderos, Suri, and Guanaco showings).
8. Cerro Bayo
9. Cerro del Medio
10. Cerro Viejo
11. Cerro Viejo Este



Source: Internal mapping from AbraSilver Resource Corp., 2026.

Figure 7-5: Diablillos Project Mineral Occurrences

The majority of the mineral resources on the Property are hosted by the Oculto and JAC deposits. They are components of These were formed as high-sulphidation epithermal silver-gold deposits following Tertiary-age local magmatic and volcanic activity. At surface there is a broad zone of intense acid leaching on the flank of Cerro Bayo. However economic mineralization notably does not appear in outcrop. Host rocks at surface are hornblende porphyritic andesite, intruded by a dacite porphyry. Tate (2018) hypothesized these to be the thermal driver for the mineralization. The andesites overlie a basement assemblage of phyllites and granitic rocks. At the contact between the andesite and basement, there is a paleo-surface occupied by a discontinuous breccia unit of widely ranging thickness. Recent drilling review suggests that this unit thickens along a trend corresponding to one of the predominant controlling structures. This zone is coincident with broader lateral extent of mineralization. Tate (2018) suggests that the conglomerate filled a paleo-trough related to that structure. Later reactivating and providing a conduit for ore-forming fluids.

The deposits are strongly oxidized to approximately 300 m to 400 m below surface. In the oxide zone, precious metal mineralization consists of native gold and chlorargyrite. Less common is iodargyrite, and locally common bismuthinite (Stein, 2001). These minerals can be found as fine-grained fracture fillings and vugh linings associated with quartz, jarosite, plumbojarosite, hematite, and goethite. Other accessory minerals include alunite, barite, native sulphur, and bismoclite.

Stein (2001) reported the occurrence of a high-grade zone at Oculto comprised of native gold, native silver, and acanthite. Accessory minerals included chlorargyrite, iodargyrite, and jalpäite in the southwest Oculto extremity. Gangue minerals in this zone included quartz, alunite, jarosite, iron oxide, and intergrowths of barite.

Hypogene mineralization comprises vein or breccia-hosted sulphides underlying the oxide zones. Primary sulphide and sulphosalt minerals include pyrite, galena, enargite, chalcopyrite, sphalerite, tennantite, and matildite. Accessory minerals include barite and alunite. Incipient supergene enrichment was observed by Stein (2001), where covellite partially replaced chalcopyrite and polybasite replaced tennantite. A review of drilling results conducted by Tate (2018) outlined a flat-lying zone of very high silver grades located between 100 and 120 m below surface. This zone has no apparent relationship with any contact or geological unit and is viewed as a possible zone of supergene enrichment.

The precious metal mineralization throughout the mineralized system occurs as extremely fine grains along fractures and in breccias or coating the inside of vugs and weathered cavities. Mineral grains are very difficult to identity in core or hand specimen, and much of the identification of these minerals was done using electron microscope or microprobe.

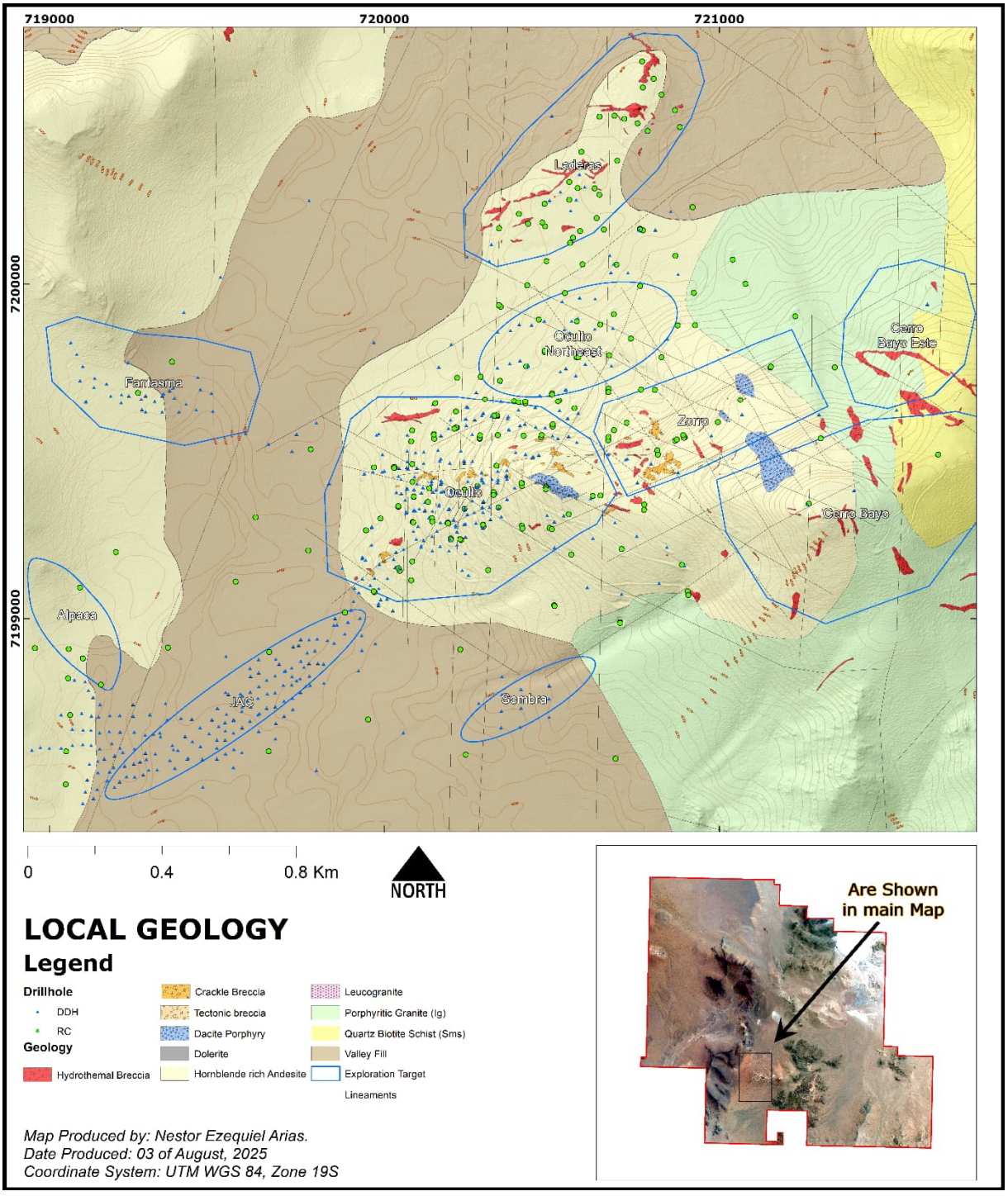
The main controls on alteration and mineralization are structural with some lithology influence (Figure 7-6 and Figure 7-7). Fluid flow propagated along predominantly east-north-easterly and

north-easterly trending steep fractures. Flow was nevertheless present along the unconformable contact between basement granites and phyllites and the overlying Tertiary andesitic pile.

Gold-silver mineralization is observed to occur in tabular silica veins, disseminations in wall rocks, and siliceous hydrothermal breccias. Mineralization propagated laterally along the trend of the conglomerate and the Tertiary-Ordovician contact. This has imparted a complex geometry, with a broadly north-easterly trend consisting of steeply dipping, structurally hosted zones and horizontal tabular bodies. Mineralization occurs within a vertical range of 3,965 masl and 4,300 masl, predominantly between elevations of 4,050 masl and 4,250 masl.

The source of the mineralization in the epithermal system at the Oculto and JAC zones is interpreted as being a porphyry copper-gold intrusion at depth. Evidence for this is the occurrence of porphyry intrusive style veining in rock fragments incorporated in silica breccias. Precious metal mineralization in the oxide zone at Oculto includes both silver and gold. At JAC, oxide mineralization is silver-dominant, with very little gold. This zonation of metals is also evident within the underlying copper sulphide mineralization, with copper-gold at the Oculto zone and copper-silver at the JAC zone.

In the central and eastern portions of the property, up to an elevation of approximately 4,350 masl, the upper Tertiary rocks exhibit evidence of a late, shallow steam-heated alteration. This overprints the earlier hypogene alteration (MDA, 2001, quoted in Wardrop, 2009). Late stage altered rocks have a light grey colour and porous texture with abundant kaolinite and white, finely crystalline alunite, minor opal, and occasional native sulphur. Hypogene alteration of the volcanic rocks differs from that of the intrusive rocks at Diablillos, due to the different host mineralogy.

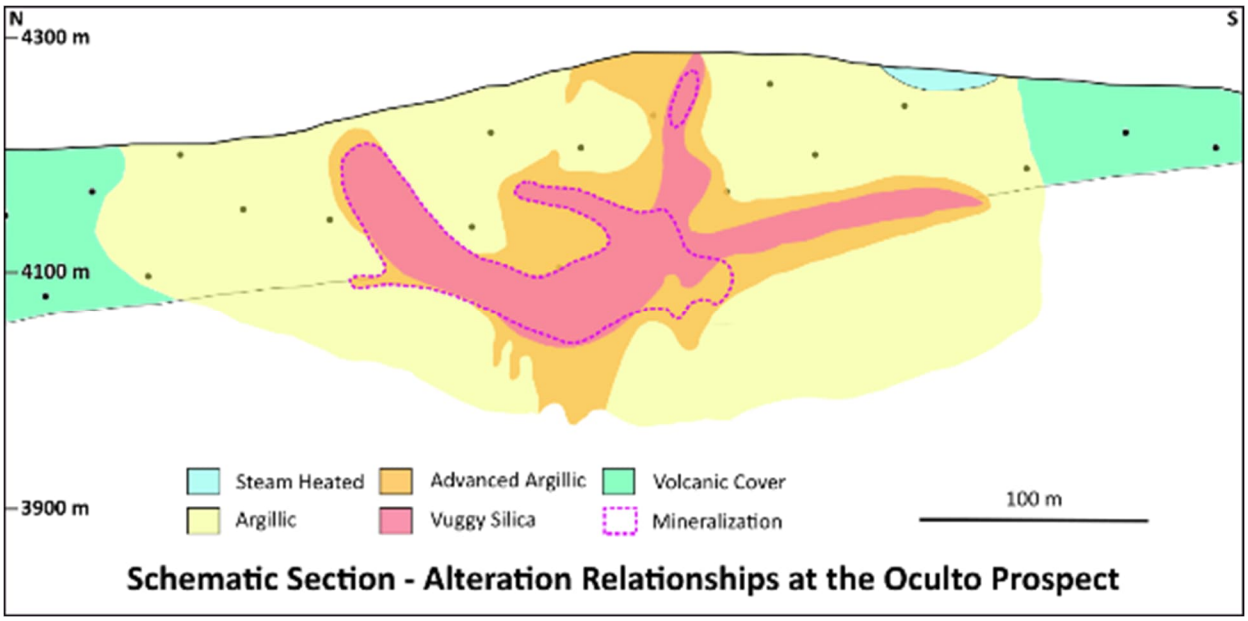


Source: updated from Ristorcelli and Ronning, 2001, with internal mapping from AbraSilver Resource Corp. 2025.

Figure 7-6: Oculto Geology Map

Alteration facies of the volcanic and intrusive rocks mapped at Diablillos include:

- Alteration Facies in Upper Volcanic Rocks
  - Propylitic: Mainly characterized by chlorite, usually with significant development of clay minerals. Propylitic alteration has been observed on the surface at the Pedernales Sur zone and subsurface at Laderas and Oculto zones.
  - Intermediate Argillic: More abundant than propylitic alteration with clay minerals being dominant.
  - Advanced Argillic: Argillic alteration occurs in many mineralized zones and typically comprises clay minerals. At Oculto and Pedernales some alunite is present.
  - Quartz-Alunite: Alunite is typically the dominant or sole alteration mineral, sometimes completely replacing the protolith. Associated minerals identified in PIMA studies include dickite, pyrophyllite, and diaspore.
  - Vuggy Silica: The central core of the Oculto deposit consists of strongly developed vuggy silica, probably temporally related to late stage boiling epithermal fluids and steam alteration. Vughs may be lined or partly filled by pyrophyllite, dickite and diaspore, or by alunite.



Source: Ristorcelli and Ronning, 2001

Figure 7-7: Oculto Geology Map

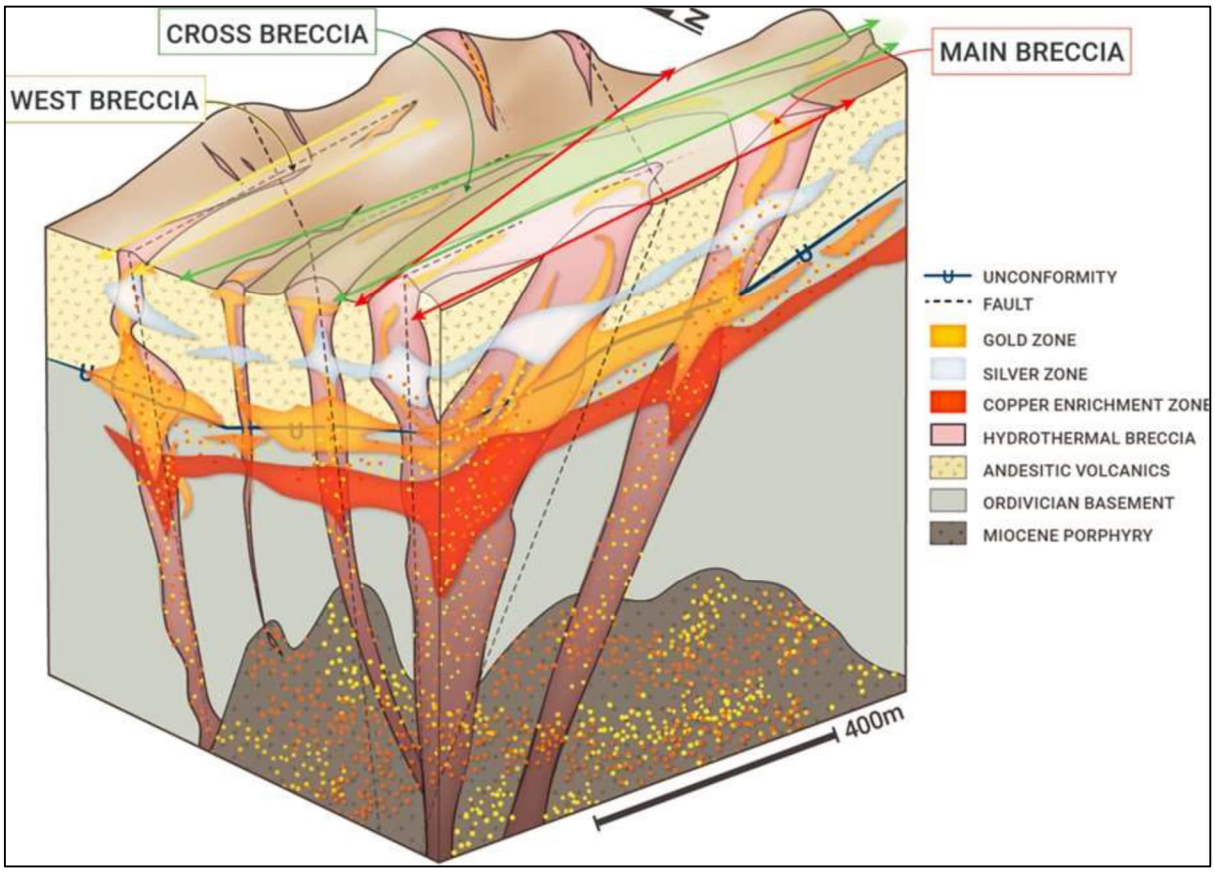
- Alteration Facies in Intrusive Rocks
  - Argillization: Occurs away from the locus of hydrothermal activity as clay alteration of feldspars and biotitization of mafic minerals.
  - Alunitization: Alunite occurs as fine-grained or microcrystalline masses replacing feldspars and mafic minerals in the granitic rocks. Alunite also occurs with quartz as veinlets and occasionally with jarosite.

- Silicification: Silicification is most pronounced adjacent to main hydrothermal fluid channels. Tabular bodies of silica have the appearance of quartz veins or veinlets but are silicified granitoid rocks.

Figure 7-7 and Figure 7-8 show the conceptual mineralization model and the property-wide distribution of alteration facies.

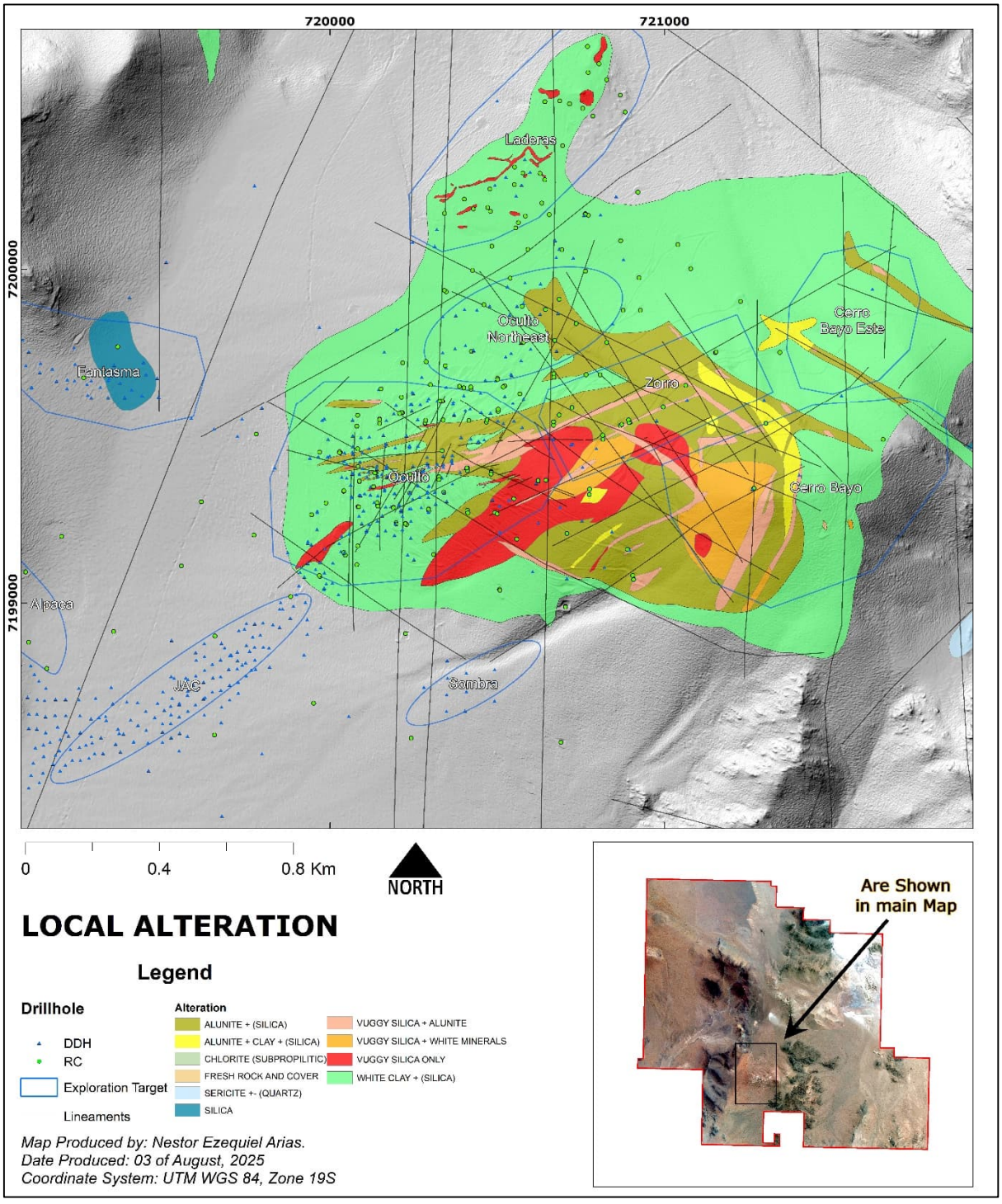
Alteration at the Oculto deposit is similar in style and mineralogy to many high sulphidation epithermal systems with concentrically zoned assemblages (Figure 7-9).

). The core of the deposit is predominantly vuggy silica alunite, surrounded by a zone of pervasive alunite and clay alteration. In turn these grades outwards into kaolinite with illite, smectite, and chlorite (Stein, 2001). Pervasive chlorite alteration underlies the mineralization in the southwest. A steam-heated zone of alunite-clay-opal is preserved above 4,330 masl and outcrops in the central portion of the deposit.



Source: AbraSilver Resource Corp., David O'Connor, 2019

Figure 7-8: Conceptual Mineralization Model for the Oculto Deposit



Source: Modified from Ristorcelli and Ronning, 2001 with internal mapping of AbraSilver Resource Corp., 2026

Figure 7-9: Alteration Map for the Oculto Deposit

8. Deposit Types

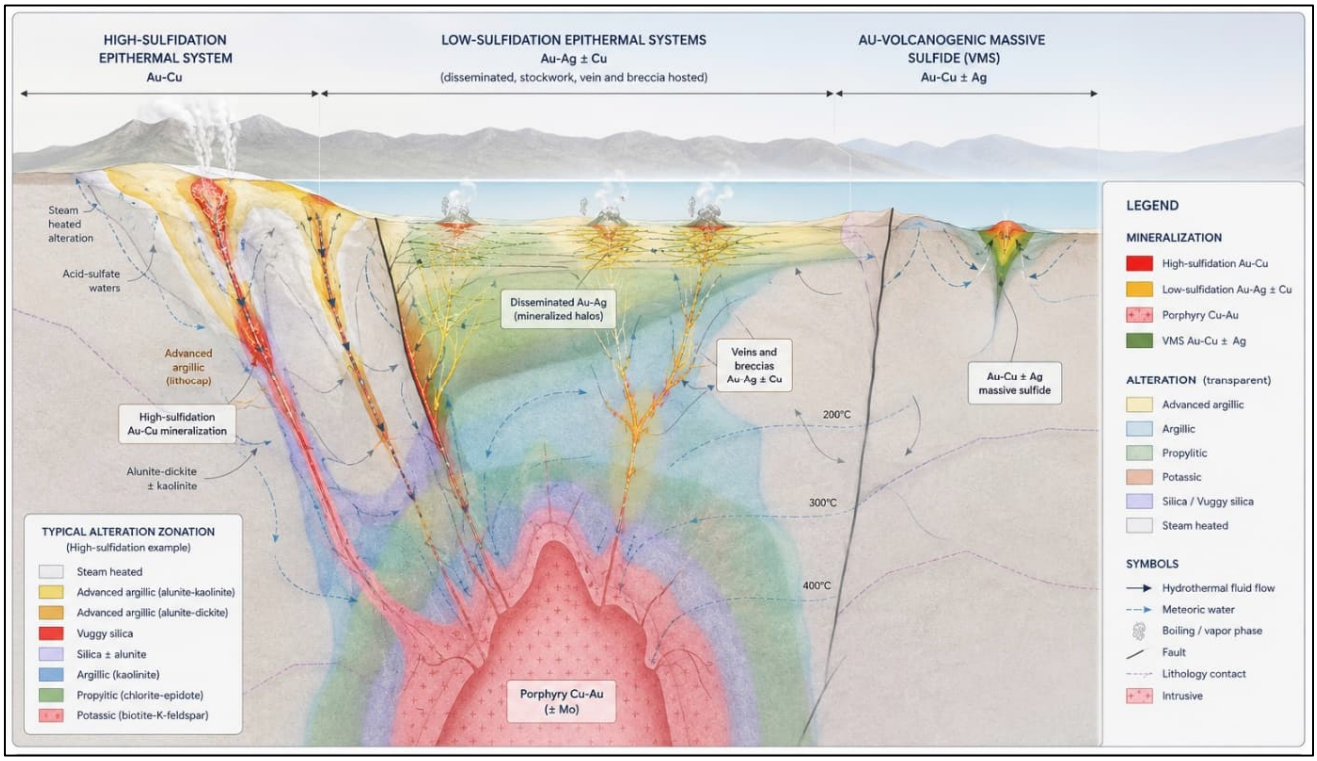
The Diablillos deposits are examples of high-sulphidation epithermal silver-gold mineralization driven by hydrothermal fluid activity at shallow depth. The principal mineralizing process is by convective flow of meteoric waters driven by remnant heat from intrusive activity at depth and often related to copper porphyry systems. The term “high-sulphidation” refers to the dissociation of magmatic SO2 in aqueous solution into H2SO4 and H2S resulting in a highly acidic environment responsible for the diagnostic assemblage of alteration facies. Mineral occurrences are structurally and hydrostatically controlled. Deposition occurs as open space filling at or near the level at which boiling occurs or where hydrothermal fluids mix with meteoric waters. They characteristically subtend a limited vertical range, except where cyclical healing and failure of fractures results in up and down migration of the boiling zone.

High-sulphidation epithermal mineral deposits form in subaerial volcanic complexes of intermediate to felsic composition. Often associated with shallow porphyry intrusions in island arc, back arc, or trans tensional tectonic regimes. Volcanic host rocks are typically andesitic to rhyodacitic flows, pyroclastic rocks and their subvolcanic intrusive equivalents. The age of many deposits is close to that of the volcanic rocks and typically ranges from Tertiary to Quaternary, although older examples are known.

Principal economic minerals include native gold, acanthite, electrum, chalcocite, covellite, bornite, and enargite/luzonite. Accessory minerals include pyrite, chalcopyrite, sphalerite, tetrahedrite/tennantite, galena, marcasite, arsenopyrite, silver sulphosalts, and tellurides. Dominant gangue minerals are quartz and pyrite, occasionally with barite. Alteration is characterized by lateral and vertical zonation of silicic, advanced argillic, argillic, sericitic, and phyllitic facies. Rocks typically have a bleached appearance owing to the acidity of the mineralizing solutions. These deposits can encompass a wide range of geometries from large, low-grade variants to smaller, higher-grade narrow vein types.

Following hydrothermal mineralization, the upper 200-300 meters was oxidized by meteoric weathering. Cu and Ag sulphides may be partially remobilized to form a thin silver enriched oxide layer. An enriched copper sulphide layer forms at the base of oxidation, with secondary chalcocite coating and replacing primary pyrite and chalcopyrite mineralization.

Examples of high - sulphidation epithermal deposits from the Andes include Yanacocha (Peru); El Indio (Chile); Lagunas Nortes / Alto Chicama (Peru); Pierina (Peru); Veladero (Argentina); Del Carmen / Alturas (Chile) and Filo del Sol (Argentina).



Source: modified from Hedenquist 1994, Corbett & Leach 1998, Sillitoe 2010 and Hedenquist 2020 by Peralta, 2026.

Figure 8-1: Geochemical Zonation, Quartz Type and Alteration Patterns of Hydrothermal System

9. Exploration

This section builds upon the prior Technical Reports RPA (2018); MRE (2021); PEA (2022) and INSA MRE (2022), INSA MRE (2023), PFS (2023) and Update PFS (2025), INSA MRE (2025).

There are several known mineralized areas in the Diablillos Project, with the Oculto and JAC zones the most significant. Mineral Resources were additionally estimated at the Fantasma zone based on historical drilling and on the Laderas zone. A location map of the various exploration targets is shown in Figure 9-1. Many of these targets have been mapped, trenched, and drilled by former operators.

Since acquiring the property, AbraSilver has continued exploration. In 2017 this included reconnaissance, geological mapping, and diamond drilling (at Fantasma). Geological mapping and an overall review of exploration data was carried out by AbraSilver consulting geologist, Nick Tate.

Targets remote from the existing Mineral Resources are considered longer-term exploration projects while closer targets have potential to add resources. Close-range, near-term targets include the Oculto, Fantasma and Sombra deposits themselves, Laderas, Alpaca and JAC. Most longer-term distal targets, except Yolanda, are aligned along a curving trend collectively known as the Northern Arc zones (Figure 9-1). These zones include Cerro Viejo Este and Oeste, Cerro del Medio Norte, Pedernales, and Corderos. This group of prospects lies three to four kilometres north-northeast of the Oculto deposit. All encompass epithermal silver-gold targets similar in style to Oculto. Cerro Viejo shows potential for porphyry mineralization.

9.1 Extensions to Existing Deposits

The Oculto and JAC deposits have been the most explored prospects in the project area. Further drilling aims to confirm and upgrade classification. Several open zones additionally have the potential for expansion.

Tate (2018) observed that a broad horizontal zone of higher-grade gold mineralization occurs near the contact of Tertiary volcanic rocks and the Ordovician basement assemblage. This Deep Gold zone (“DG”) as termed by Tate, is approximately 30 m thick. In parts this correlates with the erosive breccia / conglomerate contact. This contact represents a target for exploration. Recent drilling shows precious metal mineralisation extending beyond the base of the Tertiary volcanic rocks into the underlying Ordovician basement assemblage.

Tate (2018) has also observed that a high-grade zone of silver (“SE”) measuring approximately 40 m in thickness occurs at a depth of between 100 m and 140 m. While this zone is not co-incident with a specific stratigraphic horizon, it may represent supergene enrichment close to a paleo water table. This zone represents a target horizon at Oculto.

Two satellite bodies have been intersected in the east (Oculto Northeast) and north-east (Cerro Bayo Este & Zorro) (see Figure 9-2). These zones have limited drilling but are coincident with surface exposures of breccia and the two mineralized horizons proposed by Tate. As such, AbraSilver considers these targets to have significant potential to add Mineral Resources alongside potential expansion to the northeast. Mineral Resources for this Oculto Northeast Zone is currently categorized as Indicated and Inferred, while no Mineral Resources have been modelled for Cerro Bayo Este & Zorro due to limited quantity of drilling.

Tate (2018) also noted potential along strike of two of the principals controlling structures in the Oculto deposit. Potential exists to the southwest (JAC) along the northeast southwest and east-northeast striking fracture zones that traverse the deposit (see Figure 9-2).

The Fantasma zone located one kilometre west of Oculto is silver dominant mineralization with minimal gold. It is interpreted as a distal part of the epithermal system. The westerly-striking fault system at Oculto trends toward Fantasma (Figure 9-2) and has potential to expand the Fantasma deposit eastwards to connect with Oculto.

The recently discovered JAC deposit is located 500 meters to the southwest of the Oculto zone and is also silver dominant. The JAC mineralization strikes parallel to breccia zones at Oculto and is an extension of the epithermal system. Additional drilling is planned to verify an extension.

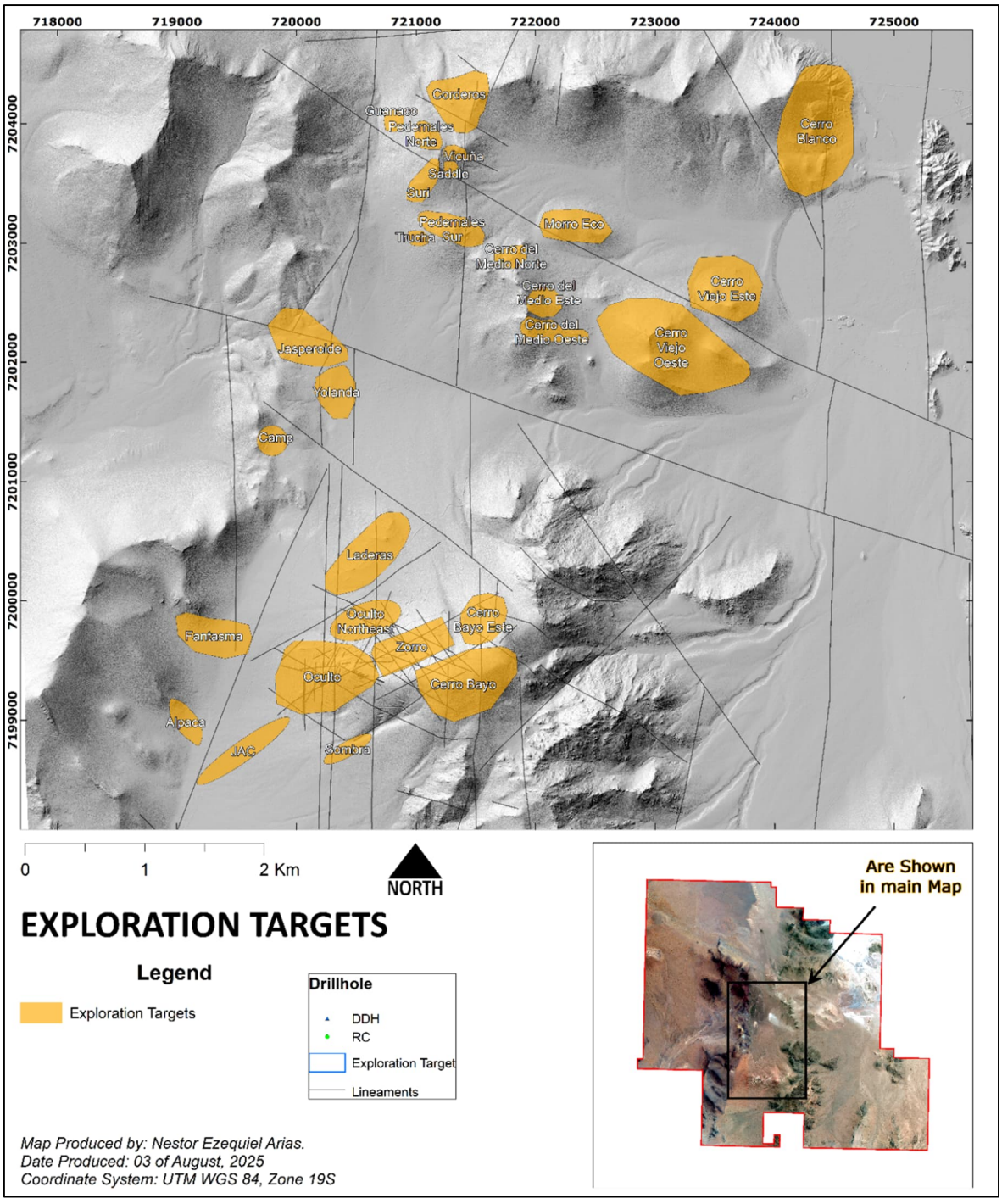
Drilling completed in the 2024 campaign testing for southwest extensions of the JAC Zone, which was discovered in the 2023 drilling campaign, included the following intercepts:

Drilling completed in the 2024 campaign testing for southwest extensions of the JAC Zone, which was discovered in the 2023 drilling campaign, included the following intercepts:

- Hole DDH 24-003 - 33.0m @ 87.0 g/t Ag
- Hole DDH 24-004 - 33.4m @ 245.0 g/t Ag
- Hole DDH 24-005 - 26.0m @ 113.0 g/t Ag
- Hole DDH 24-014 - 48.8m @ 70.0 g/t Ag
- Hole DDH 24-016 - 53.5m @ 110.0 g/t Ag
- Hole DDH 24-018 - 31.5m @ 277.0 g/t Ag
- Hole DDH 24-020 - 53.0m @ 58.0 g/t Ag
- Hole DDH 24-023 - 24.0m @ 76.0 g/t Ag
- Hole DDH 24-033 - 50.0m @ 250.0 g/t Ag
- Hole DDH 24-038 - 62.0m @ 175.0 g/t Ag
- Hole DDH 24-039 - 52.0m @ 42.0 g/t Ag
- Hole DDH 24-044 - 37.0m @ 152.0 g/t Ag
- Hole DDH 24-045 – 12.0m @ 107.0 g/t Ag

- Hole DDH 24-046 – 12.0m @ 107.5 g/t Ag

The results of these holes are included in this Mineral Resource estimate.



Source: AbraSilver Resource Corp., 2025

Figure 9-1: Exploration Target Areas at the Diablillos Project

9.2 Near-Term Exploration Prospects

Drilling has not fully explored potential targets around Oculto and JAC deposits. Opportunities may exist to expand the Mineral Resource within existing deposits and explore for new mineralized zones.

AbraSilver has established a priority sequence with a focus on areas close to the Oculto and JAC deposits. Targets include linear zones of low-magnetic intensity interpreted as where hot acidic hydrothermal fluids have destroyed host rock magnetite. This style of target definition led to the discovery of the JAC deposit, as long as the newly zone Sombra.

In addition, drilling has shown that the Oculto and Oculto NE mineralised system extends both east and north of the margin of the conceptual open pit margin where silver and gold are associated. Intercepts in these areas include:

- Hole DDH 24-011 –21m @ 131.3 g/t Ag and 0.36 g/t Au.
- Hole DDH 24-015 – 6m @ 19.9 g/t Ag and 0.31 g/t Au.
- Hole DDH 24-017 – 14.5m @ 160.3 g/t Ag.
- Hole DDH 24-021 – 5m @ 31.7 g/t Ag and 0.19 g/t Au.
- Hole DDH 24-024 – 11m @ 47 g/tand 0.19 g/t Au.
- Hole DDH 24-027 – 19m @ 39.9 g/t Ag and 0.14 g/t Au.
- Hole DDH 24-031 – 15m @ 495.6 g/t Ag and 0.28 g/t Au.
- Hole DDH 24-034 – 5m @ 29.8 g/t Ag and 0.61 g/t Au.
- Hole DDH 24-040 – 9m @ 52.3 g/t Ag and 2.10 g/t Au.
- Hole DDH 24-043 – 4.3m @ 0.52 g/t Au.
- Hole DDH 24-064 – 22.0m @ 2.78 g/t Au. – Oculto Zone.
- Hole DDH 24-057 – 3.0m @ 0.66 g/t Au. – Oculto NE Zone.
- Hole DDH 24-064 – 9.0m @ 5.35 g/t Au. – Oculto NE Zone.
- Hole DDH 24-070 – 17.0m @ 1.80 g/t Au. – Oculto NE Zone.
- Hole DDH 24-023 – 2.0m @ 1.58 g/t Au. – Sombra Zone.
- Hole DDH 25-024 – 31.0m @ 9.96 g/t Au. – Oculto NE Zone.

The results of these holes are included in this Mineral Resource estimate.

The mineralisation in this area is strongly structurally controlled, and a detailed structural map was prepared during 2024 to guide drilling for 2025 drilling campaign.

Intercepts in these areas include:

- Hole DDH 25-060A – 60.0m @ 1.05 g/t Au. – Oculto East Zone.
- Hole DDH 25-067 – 16.0m @ 1.74 g/t Au and 17.3 g/t Ag. – Oculto East Zone.

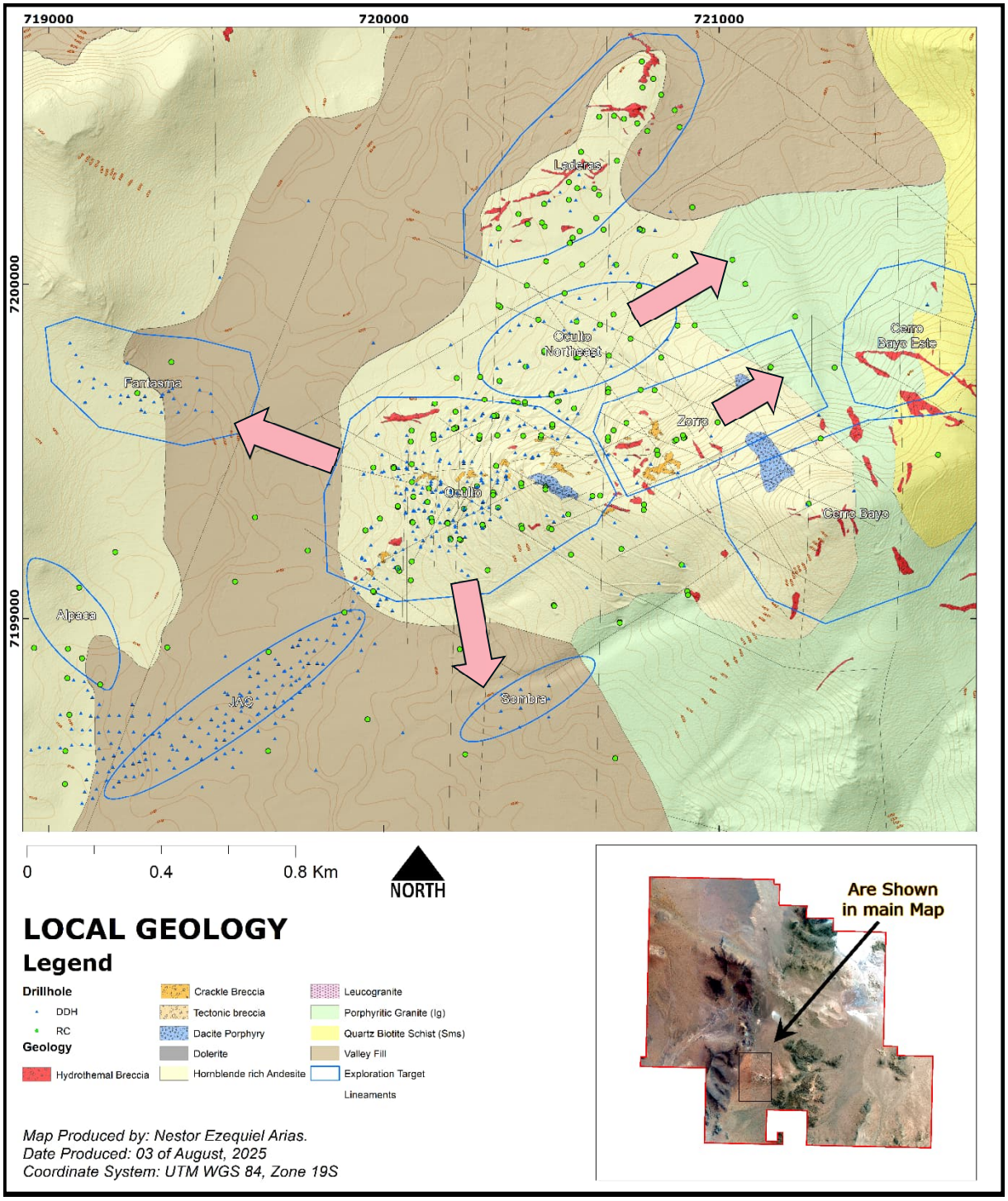
- Hole DDH 25-068 – 21.0m @ 1.51 g/t Au and 49.5 g/t Ag. – Oculto East Zone.
- Hole DDH 25-066 – 10.0m @ 108.9 g/t Au. – Oculto East Zone.
- Hole DDH 25-069 – 24.0m @ 1.02 g/t Au. – Oculto East Zone.
- Hole DDH 25-071 – 33.0m @ 0.42 g/t Au. – Oculto East Zone.
- Hole DDH 25-072 – 140.0m @ 0.57 g/t Au. – Oculto East Zone.
- Hole DDH 25-074 – 31.0m @ 0.59 g/t Au. – Oculto East Zone.
- Hole DDH 25-075A – 73.0m @ 0.55 g/t Au. – Oculto East Zone.
- Hole DDH 25-077 – 44.0m @ 0.65 g/t Au. – Oculto East Zone.

The results of these holes are included in this Mineral Resource estimate.

The Laderas target lies immediately north of Oculto, along the trend of a prominent east west ridge (Figure 9-2). Geological mapping and review of Laderas drill results conducted in 2017 indicated that gold and silver mineralization occurs where those vertical structures intersect favourable horizons (like the paleo-surface). Like Oculto, there is potential for mineralization at depth within structurally controlled breccias hosted in Tertiary sedimentary and volcanic rocks (Tate, 2017). Controlling structures are steeply dipping and strike in various orientations. The northwest, west-northwest, and westerly striking structures dip at 75° to 85° to the south to southwest. The east-northeast and northeast striking structures appear to dip north-westerly. Mineralized zones are accompanied by silica-alunite alteration which rapidly grades outward to alunite at the breccia walls.

At the Fantasma-Oculto zone a small, shallow silver mineralization occurs at the northwest end of a linear zone of low magnetic intensity. This extends for about one kilometre from Oculto. AbraSilver drilled five reconnaissance holes in the centre of this magnetic anomaly and intersected shallow silver mineralization. This demonstrates potential for continuity between the Oculto and Fantasma deposits.

Historical drilling in the Alpaca zone intersected moderate grade silver mineralization (see Figure 9-2). This is located approximately 200 meters northwest of JAC following a northwest trending zone of low magnetic intensity. Whereas the JAC resource follows an east-northeast trending magnetic low, the northwest anomaly extending from JAC to Alpaca is interpreted as being conjugate structures into which hydrothermal fluids migrated. Drilling between Alpaca and JAC intersected broad silver mineralization within granitoid basement rocks. This indicates a possible extension of the mineralizing system beneath andesitic flows.



Source: AbraSilver Resource Corp., 2025

Figure 9-2: Near Term Exploration Targets at the Diablillos Project

9.3 Planned Exploration

While there are various exploration targets at Diablillos, priority has been placed on upgrading Inferred Mineral Resources and defining the recent extension area to Oculto NE, and on targets considered to have potential to add high-grade mineralization. Target areas for drilling include:

- Oculto NE.
- Sombra zone.
- Extension and Mineral Resource definition of Oculto surroundings
  - Shallow mineralization (in Fantasma direction)
  - Shallow mineralization (in Alpaca direction)

Planned work includes the re-logging of historical core to maintain consistency with the new geological and alteration model. Also, it is important the fact that many assays are still pending from Oculto NE at the time this report was written, which can re drive the direction of planned exploration.

In addition to target areas peripheral to the Oculto and JAC zones, there is outcropping porphyry style alteration and veining in the Cerro Blanco and Cerro Viejo areas located approximately 4 km northeast of Oculto (see Figure 9-1). A TITAN ground geophysical survey was completed over this porphyry camp to assist in drill target definition. An initial four-hole drilling program was completed in 2024 and two extra drillholes during 2025 drilling campaign.

Intercepts in these areas include:

- Hole DDH 24-056 – 36.0m @ 1.91 g/t Au.
- Hole DDH 24-059 – 81.0m @ 0.25 g/t Au.
- Hole DDH 24-071 – 106.0m @ 0.15 g/t Au.
- Hole DDH 25-050 – 128.0m @ 0.24 g/t Au.
- Hole DDH 25-056 – 200.0m @ 0.32 g/t Au.

9.4 QP Discussion

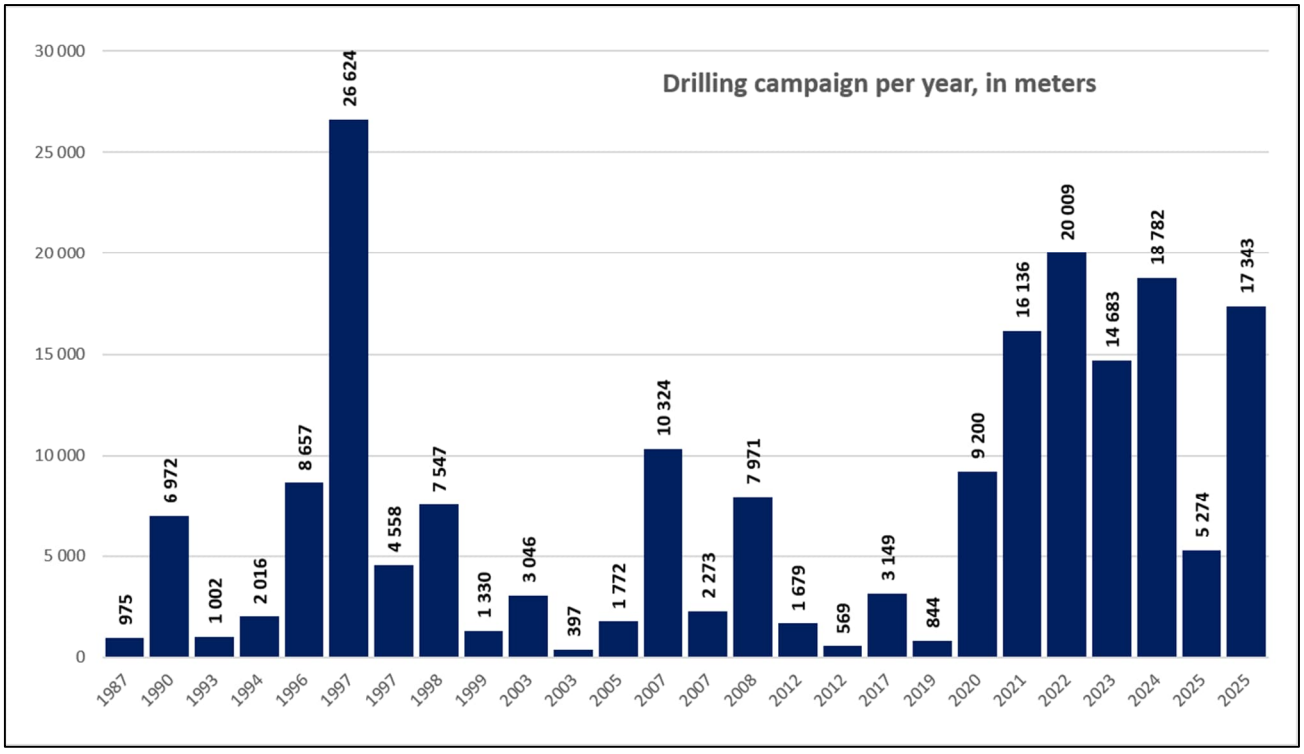
In the author's opinion exploration targets defined by AbraSilver geologists at Diablillos have been established using sound geological observations and interpretations. The author recommends the planned exploration work to be undertaken.

10. Drilling

This section builds upon the prior Technical Reports RPA (2018), MP (2021), MP PEA (2022) and INSA MRE (2022), INSA MRE (2023), PFS (2024) and Updated PFS (2025), INSA MRE (2025).

Prior to AbraSilver's acquisition, previous operators had completed 476 RC and diamond drillholes and 26 trenches totalling 87,594.55 meters. BHP Utah drilled a single RC hole in 1990. Details of this drilling can be found in Wardrop (2009), MDA (2001), M3 (2011) and RPA (2018).

Much of the earlier drilling was completed in the Oculto zone. In 2016, AbraSilver completed further drilling at the Oculto and JAC zones. Additional reconnaissance holes were drilled in other areas including Laderas, Fantasma, Alpaca and the Northern Arc. Figure 10-1 shows the location of drillhole collars while Table 10-1 provides a list of holes by year, type and length.



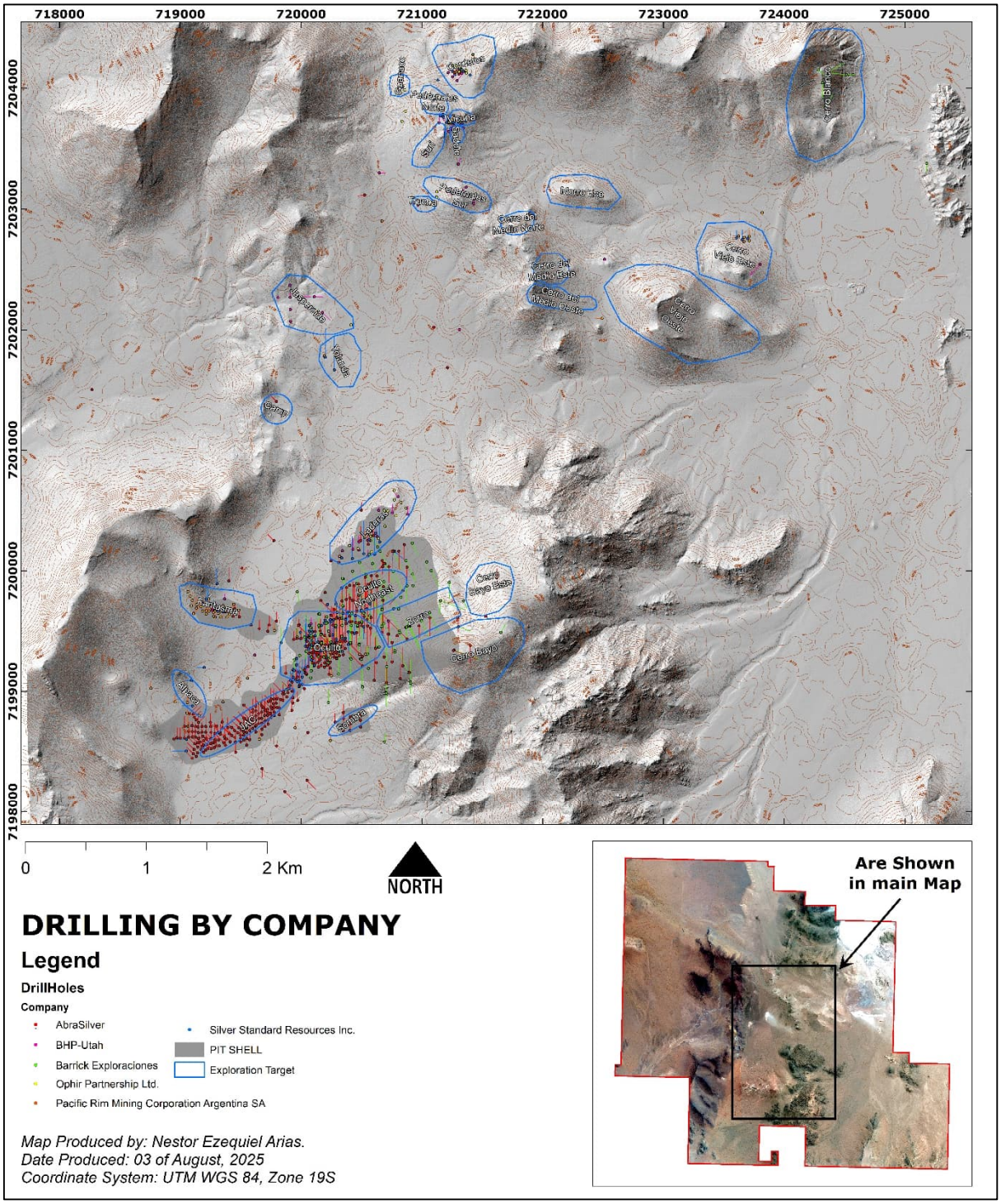
Source: AbraSilver Resource Corp., 2026..

Figure 10-1 : Summary of Drilling Campaign per Year at Diablillos Project

Table 10-1: Summary of Drilling Campaign by Year, AbraSilver Resource Corp. – Diablillos Project

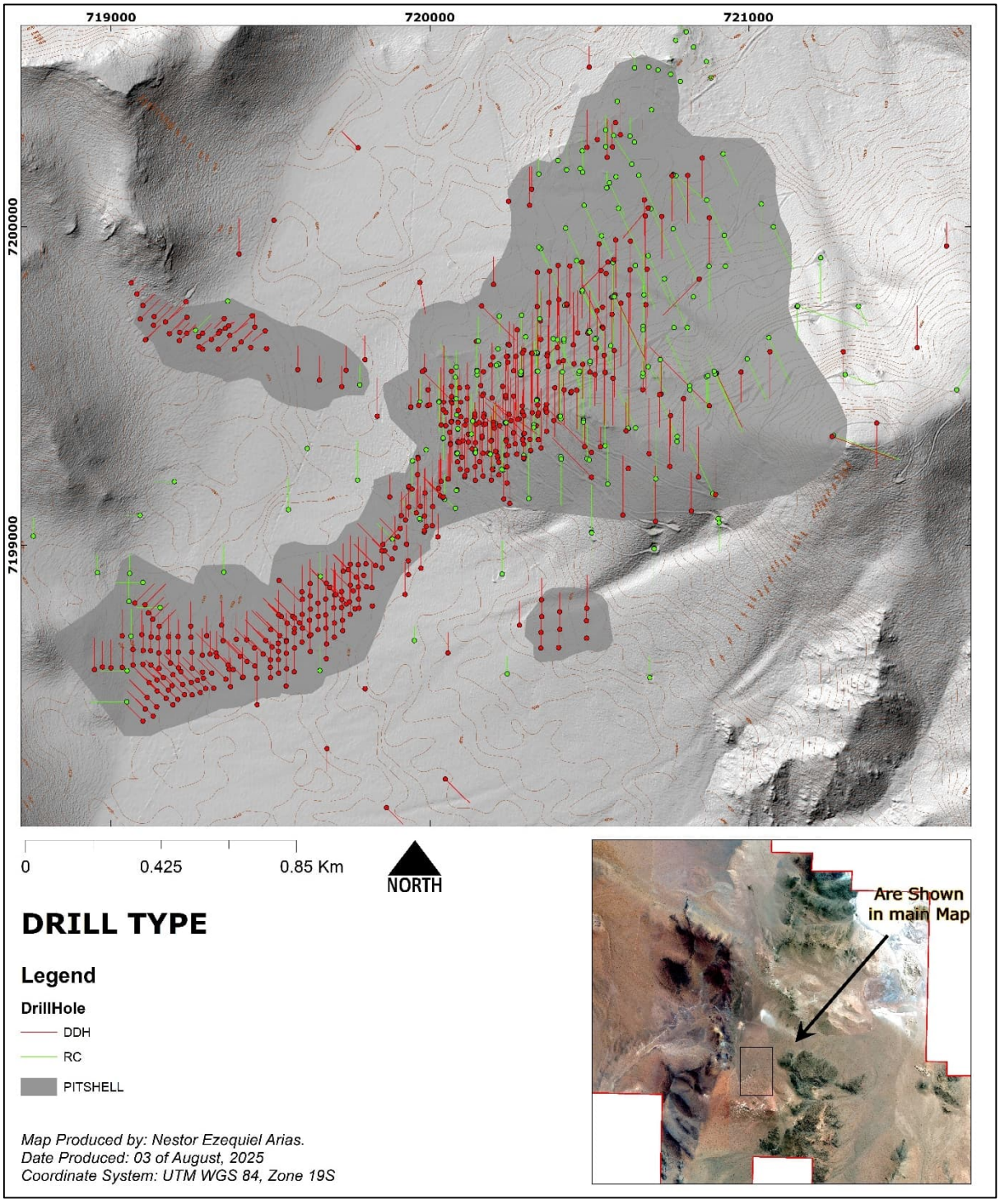
Drilling Campaign	Type of Hole	Number of Holes	Meters Drilled	Average Meters Drilled	Min Metters Drilled	Max Metters Drilled
1987	RC	34	975	29	3	34
1990	RC	56	6,972	125	50	250
1993	DDH	5	1,002	200	146	254
1994	DDH	12	2,016	68	25	255
1996	RC	32	8,657	271	140	400
1997	RC	102	26,624	261	49	413
1997	DDH	19	4,558	240	31	380
1998	RC	24	7,547	314	220	70
1999	DDH	5	1,330	266	191	450
2003	RC	20	3,046	152	48	282
2003	DDH	6	397	66	46	76
2005	RC	10	1,772	177	101	252
2007	DDH	54	10,324	191	31	365
2007	Trenche	20	2,273	114	38	284
2008	DDH	52	7,971	153	40	355
2012	DDH	19	1,679	88	41	126
2012	Trenche	6	569	95	47	145
2017	DDH	28	3,149	112	40	327
2019	DDH	2	844	422	38	464
2020	DDH	34	9,200	271	50	610
2021	DDH	69	16,136	233	50	451
2022	DDH	89	20,009	227	22	411
2023	DDH	82	14,683	179	30	245
2024	DDH	90	18,782	211	89	749
2025	DDH	28	5,274	181	101	525
2025	DDH	80	17,343	216	29	1147
Subtotal	RC	278	55,593	190	87	243
Subtotal	Trenche	26	2,842	105	43	215
Subtotal	DDH	674	134,299	204	60	445
Grand total		978	193,131	187	65	358

The Oculto, JAC, Sombra, Fantasma and Laderas zones are shown in Figure 10-2 and Figure 10-3, including drilling used in the Mineral Resource estimate.



Source: AbraSilver Resource Corp., 2026

Figure 10-2: Diablillos Drill Hole Locations by Company



Source: AbraSilver Resource Corp., 2026.

Figure 10-3: Oculto Drill Hole Locations, Coloured by Type.

10.1 Drilling Campaigns by Year

10.1.1 1987

Ophir drilled 34 shallow RC holes with an aggregate length of 975 m, mostly at the Laderas zone. No drilling was conducted in the Oculto zone. Drilling was carried out by contractor Dresser Atlas. No technical information could be found in the database regarding hole sizes, surveys, or equipment.

10.1.2 1990

In 1990, BHP drilled 56 RC holes totalizing 6,972 m, six of which were in or around the Oculto zone. The drilling contractor for this work was also Dresser Atlas.

10.1.3 1993 - 1994

During 1993-1994, Pacific Rim contracted Connors Argentina to perform 3,018 m of DDH drilling in 17 holes. Holes were generally collared as HQ and subsequently reduced to NQ. The program was entirely focused on the Oculto zone. Holes were oriented along sections aligned north-south as well as at approximately 155°/335°. All holes were inclined at dips between -45° and -65°. Drilling conditions were reportedly poor, with several holes failing to reach target (Wardrop, 2009). Holes DDH-094-008 and DDH-094-008b were abandoned at 24 m and 57 m, respectively, and holes DDH-094-006 and DDH-094-011 were terminated due to rods twisting off in the holes (M3, 2012). There does not appear to have been routine downhole surveys, although reportedly acid dip tests were performed on holes DDH-094-001 and DDH-094-004.

10.1.4 1996 - 1997

Barrick drilled 134 RC holes totalling 35,281 m and 19 diamond drill holes totalling 4,558 m, in the Oculto zone. Drilling was conducted along both north-south and 155° section planes. The program included twinning four RC holes with diamond holes. Boytec Boyles Bros. was the drilling contractor, with RC holes drilled using Drillteck D40K and Ingersoll Rand TH75 machines. Hole diameters were 5 ¼ in. (13.34 cm) and oriented at inclinations ranging from -47° to vertical. Most holes encountered water, which required collection of wet samples. Samples were collected every meter down the hole, while composites were collected every five metres for PIMA analysis. A truck mounted Longyear 44 rig was used for diamond drilling. Holes were collared as HQ and reduced at 200 m downhole to NQ. Downhole surveys were done either with a Reflex Maxibor or simply acid dip tests. Acid tests were conducted every 50 m, while Maxibor readings were every ten meters.

10.1.5 1998

Barrick drilled 24 RC holes totalizing 7,547 m. Drilling was conducted along both north-south and 155° section planes. Boytec Boyles Bros. was the drilling contractor, RC holes were drilled using Drillteck D40K and Ingersoll Rand TH75 machines with hole diameters of 5 ¼ in. (13.34 cm).

10.1.6 1999

Barrick drilled 5 diamond holes in the Oculto zone totalling 1,330 m. Drilling was aligned both north-south and on 155° section planes. A truck mounted Longyear 44 rig was used.

10.1.7 2003

Pacific Rim, on behalf of SSRI, drilled 3,443.2 m in 26 holes, 20 RC holes and 6 diamond holes primarily on the Oculto zone, as well as at Corderos, Alpaca and Pedernales. The drilling contractor was Patagonia Drill Mining Services (Patagonia). Most holes were inclined along north direction. No information was available in terms of the survey methods.

10.1.8 2005

Ten reverse circulation drill holes totalling 1,772 m were drilled by Pacific Rim/SSRI. Five targeted Oculto and the other five Alpaca. The holes were drilled by Patagonia Drilling. All holes were inclined ranging from -60° to -70°, with south direction for Oculto and west for Alpaca.

10.1.9 2007

Pacific Rim/SSRI drilled 54 diamond holes, totalling 10,324 meters. Drilling was carried out by Major Drilling. Eight of these holes, the LC and PN series, were not drilled at Oculto. The balance was drilled along north-south oriented section planes, at inclinations ranging from vertical to -45°. The inclined holes were directed both north and south. Four of the Oculto holes provided sample material for metallurgical test work.

Drill collars were surveyed by differential GPS, with downhole surveys taken at 50 m intervals. The downhole survey instrument type was not reported. As both azimuth and dip information were recorded, the author infers that an instrument such as the Maxibor was used.

Eight holes were reportedly abandoned or terminated due to difficult drilling conditions.

10.1.10 2008

In 2009, a total of 7,971 m of HQ diamond drilling was completed at Oculto. 52 holes as contracted by Pacific Rim/SSRI to contractor Major Drilling. All but two holes were drilled along the north-south orientation. DDH-08-067 and DDH-08-067A, were however oriented at azimuth 335° (i.e., the 155° section planes). Three holes, the KP series, were drilled for geotechnical purposes. The remainder were intended to further define Oculto. Collar locations for holes DDH-08-063 to DDH-08-071 were surveyed by differential GPS. The balance, DDH-08-072 to DDH-08-108, were surveyed by compass and tape from existing collars. AbraSilver additionally re-measured every hole with differential GPS, updating collar coordinates during the 2020 drilling campaign. Downhole surveys were collected at 50 m intervals, again presumably with a Maxibor or similar instrument.

10.1.11 2012

Pacific Rim/SSRI drilled 19 holes, totalling 1,679 m in the Fantasma, Laderas, Cerro Viejo, and Pedernales zones. The work was conducted under contract by CAP S.A. No holes were drilled at Oculto.

10.1.12 2017

AbraPlata drilled 28 diamond holes at Diablillos in 2017, totalling 3,149 meters. All were in the Fantasma zone. BHP Utah also drilled a single RC hole in 1990. Barrick excavated six trenches, but the sampling results have been lost. In 2011, SSRI cleaned out and re-sampled the trenches, then drilled four diamond holes (see Table 10-1). These holes intersected mineralization, but the drilling was not extensive enough to permit an estimate of Mineral Resources for Fantasma. The 2017 drilling program was successful in expanding and confirming the extent and tenor of the silver mineralization.

10.1.13 2019

AbraSilver drilled 2 diamond holes in Oculto in 2019, totalling 844 meters. A new interpretation of the project was given following Tate's visit to the Diablillos project. These two holes were drilled to test for vertical gold and silver mineralizing fluids migrating laterally.

10.1.14 2020

AbraSilver drilled 34 diamond holes in 2020, totalling 9,200 meters. Two at Laderas, three at Oculto Northeast and the remainder in Oculto itself. All were designed to test the conceptual basis of the deposit. The holes were oriented north-south, dipping between 60° to 65°. Almost all holes intercepted mineralization. The first five holes were executed by drilling contractor FORACO, the others Hidrotec Perforaciones. All holes were drilled in HQ core size. Collar locations were surveyed using differential GPS with an RTK system. Collar orientations were determined with a compass. Downhole surveys were conducted at intervals of 20 meters, using a Gyro Champ tool.

10.1.15 2021

AbraSilver drilled 69 diamond holes in 2021, totalling 16,136 meters. Two were drilled at Corderos, three at Fantasma, one at Jasperoide, two at Laderas, two at Pedernales and two at the valley geophysics anomaly. The remainder were drilled at Oculto. All were aligned north-south dipping between 60° to 65°. Almost all intercepted economic mineralization. All holes were drilled by drilling contractor Hidrotec Perforaciones. All holes were drilled in HQ core size. Collar locations were surveyed using differential GPS with an RTK system, collar orientations were determined with a compass. Downhole surveys were conducted at intervals of 20 meters, using the Gyro Champ tool.

10.1.16 2022

During the first half of 2022 AbraSilver drilled 40 diamond holes, totalling 10,491.2 meters. Five were drilled for Oculto geotechnical purposes, four for condemnation one hole (DDH-022-019) discovered the JAC deposit. All other holes were at Oculto and aligned north-south, dipping between 60° to 85°. Almost all holes intercepted mineralization. All holes were drilled by drilling contractor Hidrotec Perforaciones. Holes were drilled in HQ core size. Collar locations were surveyed using differential GPS with an RTK system, with collar orientations determined with a compass. Downhole surveys were conducted at intervals of 20 meters, using the Gyro Champ tool.

During the second half of 2022 AbraSilver drilled 48 diamond holes, totalling 9,536 meters. Two were condemnation, one for water exploration and the balance at JAC. Most were oriented in a north-south orientation, dipping between 60° to 85°. Almost all holes intercepted mineralization. All holes were drilled by drilling contractor Hidrotec Perforaciones. Core size for all holes was HQ. Collar locations were surveyed using differential GPS with an RTK system. Collar orientations were determined with a compass. Downhole surveys were conducted at intervals of 20 meters, using the Gyro Champ tool.

10.1.17 2023

AbraSilver drilled 82 diamond holes in 2023, totalling 14,683 meters. All but four were drilled at JAC. Holes DDH-23-076 and DDH-23-077 were drilled at Alpaca, DDH-23-078 and DDH-23-079 were drilled at Fantasma. Most were oriented north-south, dipping between 60° to 85°. Almost all holes intercepted mineralization. All holes were drilled by drilling contractor Hidrotec Perforaciones. Core size for all holes was HQ diameter. Collar locations were surveyed using differential GPS with an RTK system. Collar orientations were determined with a compass. Downhole surveys were conducted at intervals of 20 meters, using the Gyro Champ tool.

10.1.18 2024

AbraSilver drilled 90 diamond holes in 2024, totalling 18,782 meters. 40 drillholes were drilled at JAC, 21 drillholes were drilled at Oculto and Oculto Northeast, 4 drill holes at Cerro Viejo, 3 drillholes at Fantasma, 4 drillholes at Jasperoide, 6 drillholes at Sombra and 6 geotechnical drillholes with oriented core. Most were oriented north-south, dipping between 60° to 85°. Almost all holes intercepted mineralization. All holes were drilled by drilling contractor Hidrotec Perforaciones. Core size for all holes was HQ diameter. Collar locations were surveyed using differential GPS with an RTK system. Collar orientations were determined with a GPS-RTK lining. Downhole surveys were conducted at intervals of 5 meters, using the Gyro Champ tool.

10.1.19 2025

AbraSilver drilled 28 diamond holes in 2025, corresponding to Phase 4 drilling campaign, totalling 5,117 meters. 14 drillholes were drilled at JAC, 5 drillholes were drilled at Oculto and Oculto Northeast, 5 drill holes at Sombra and 2 at Fantasma target.

In addition, 80 diamond holes were drilled in 2025, corresponding to Phase 5 drilling campaign, totalling 17,342 meters. 23 drill holes were drilled at Oculto NE, 33 drill holes were at JAC target, 5 at Cerro Viejo and Cerro Blanco, 2 at Sombra and the rest 16 were geotechnical drill holes dedicated to process plant foundations and tailing storage facilities.

Most drill holes were oriented north-south, dipping between 60° to 85°. Almost all holes intercepted mineralization. All holes were drilled by drilling contractor Hidrotec Perforaciones. Core size for all holes was HQ diameter. Collar locations were surveyed using differential GPS with an RTK system. Collar orientations were determined with a GPS-RTK lining. Downhole surveys were conducted at intervals of 2 meters, using the Gyro Champ tool.

10.1.20 2026

AbraSilver completed a total of 21 diamond drill holes during the 2026 Phase VI drilling campaign, for an aggregate of 6497 metres. All drill holes were collared within the Oculto Northeast target area as part of the ongoing exploration program aimed at expanding and further delineating mineralization.

At the effective date of this Technical Report, analytical results, geological interpretations and associated evaluations from the Phase VI drilling campaign remain in progress. Consequently, the results of this program have not been incorporated into the Mineral Resource estimate or considered in the geological interpretations and conclusions presented herein.

10.2 Discussion and Conclusions

The author of this section, Mr. Peralta, visited the site while AbraSilver was drilling at the Oculto East and JAC zone and Oculto Northeast from April 24th to May 03rd and from October 02nd to October 8th, 2023, also from December 6th to December 11th, 2024, from April 8th to April 14th, from August 25th to August 29th, 2025 and from April 13th to April 19th, 2026 for the purpose of this report.

The author reviewed drilling procedures, core samples and methodologies of collaring, surveying, logging, sampling, and chain of custody for the drilling campaigns from 2020 to 2026.

In the author's opinion, all drilling conducted by AbraSilver was completed in an appropriate manner consistent with common best industry practice.

11. Sample Preparation, Analyses, and Security

Information regarding sample preparation and analysis procedures prior to AbraSilver have been taken from Ristorcelli (1997), Barrick Exploraciones SA (1998), MDA (2001) and an internal Technical Report M3 (2011). This section builds upon the prior Technical Reports RPA (2018); MRE (2021); PEA (2022) and INSA MRE (2022), INSA MRE (2023), PFS (2023) and Update PFS (2025), INSA MRE (2025).

11.1 Pre-1996

The core and chip logging, sampling, and analytical protocols used for holes drilled prior to 1996 were not documented in the information provided.

11.1.1 RC Drilling

Cuttings from every meter was collected, logged and archived.

11.1.2 Diamond Drilling

Core was logged on site for lithology, alteration, mineralogy, and geotechnical data. This was then marked by the logging geologist for sampling. Sample intervals were typically 1.0 m and ranged from 0.5 m to 1.5 m in length with breaks for lithology or structure.

Samples comprised half-cores, cut by diamond saw, with the remaining half retained for storage. The core and photograph archive were stored in Salta. In 2017 cores were transported to site and stored in new wooden boxes.

11.1.3 Analyses

Historical reporting by Ristorcelli (1997) indicated original samples were sent to SGS. In 1997, check sampling was carried out to validate the database. Both rejects and pulps were sent to Chemex Labs for the check analysis. The pulp sample results were reported to be 4.5% higher for gold and 2.9% higher for silver.

11.2 1996 – 1999 (Barrick)

11.2.1 RC Drilling

Cuttings were collected every meter, logged and archived. Five metre composite samples were collected and submitted for PIMA scans.

Dry samples were split at the drill with a cyclone, one quarter was sent for analysis and the remainder stored on site. Most holes encountered water, requiring wet sampling. Initially, wet

cuttings were split using a wet splitter, however, this was found unsatisfactory due to the volume of sample material collected. Barrick personnel considered samples inadequate if recovery was less than 25% of recovered cuttings or if total recovery was less than 50%. From hole RC-096-022 onward, if the volume was too low, all cuttings were sent to the laboratory to be split after drying.

11.2.2 Diamond Drilling

Core was logged on site for lithology, alteration, mineralogy, and geotechnical data. This was then marked by the logging geologist for sampling. Sample intervals were typically 1.0 m and ranged from 0.5 m to 1.5 m in length with breaks for lithology or structure. Marked core was photographed and sent for sampling. Samples comprised half-cores, cut by diamond saw, with the remaining half retained for storage. The core and photograph archive were stored in Salta. In 2017 cores were transported to site and stored in new wooden boxes.

11.2.3 Analyses

Bondar Clegg Ltda. in Coquimbo, Chile (“Bondar Clegg”) analysed samples from drill holes RC-96-001 through RC-97-53 for gold and silver.

Samples from RC-97-54 through RC-97-122 were analysed by SGS, Minerals Division, in Santiago, Chile (“SGS Santiago”).

The 1998 samples, RC-98-123 through RC-98-146 were analysed by SGS, in their laboratory in Mendoza, Argentina. Barrick’s quality control program uncovered problems with the precision of results from the Mendoza analyses and the majority of the 1998 samples were re-analysed by SGS, Santiago. Corrected data was incorporated into the final database.

At the laboratory, samples were dried at a maximum of 60°C, crushed to 90% passing a Tyler 10 mesh screen, and split to a 1,000 g sub-sample. The entire 1,000 g sample was pulverized to 95% passing a Tyler 150 mesh sieve. The pulp was riffled to a 250 g pulp sample for assay. The remaining 750 g of pulp material was returned to Barrick.

Gold and silver analyses were generally by fire assay (“FA”) with a gravimetric finish. Some of the analyses were done by ICP Atomic Emission Spectroscopy (“ICP-AES”). It is not known what accreditations were held by Bondar Clegg or SGS in the period in question. In the author’s opinion these laboratories were, and continue to be, recognized in the industry as reputable. Bondar Clegg has since been acquired by ALS Chemex in Mendoza, Argentina (“ALS Chemex”), which has ISO 9001:2000 certification.

11.2.4 Metallurgical Sampling

Holes DDH-097-012 to DDH-097-016, inclusive, were sampled in their entirety and sent to Lakefield Research Chile S. A. (“Lakefield”) in Santiago, Chile for metallurgical testing.

11.3 2007 - 2008 (Pacific RIM for Silver Standards Resources)

11.3.1 Logging

During 2007, a total of 54 diamond drillholes were completed totaling 10,324.3 meters and an additional 2,272.4 meters of trenches. The data from the trenches was not considered for Mineral Resource estimates.

In 2008, 52 diamond drillholes were drilled, totalling 7,970.95 meters.

For both drilling campaigns core was transported by truck to the onsite logging facility where it was washed and photographed. Digital images were uploaded to the on-site computer.

Core was logged for recovery and RQD. Artificial breaks in the core caused by drilling or handling were ignored for the RQD determination. Marbled sections were lightly tapped with a hammer. If they remained intact, they were included as intact intervals for RQD measurement.

Logging was conducted for lithology, structure, alteration, and mineralogy. The data was then transcribed onto spreadsheets for entry into a Gemcom database.

The logging geologist marked core for sampling. Sample intervals were limited to a minimum of 0.5 m and a maximum of 2.0 m with breaks for lithology and mineralization. An attempt was made to constrain the samples to 1.5 meter lengths and extend them to the 2.0 meter maximum only where contacts were encountered.

11.3.2 Sampling

Samples were split using a manual blade splitter. One half was retained for archiving and one half sent for assay. Samples were placed in plastic bags, sealed with plastic straps, and stored within a locked area prior to shipment. Samples remained under the supervision of the project geologist while in storage. Individual sample bags were placed in woven nylon rice bags for shipment by truck to ALS Chemex in Mendoza.

The remaining core was cross stacked in chronological order, then shipped to the SSRI warehouse in Salta. In 2017, cores were transported to site and stored in new wooden boxes if required.

11.3.3 Sample Preparation and Analyses

Upon arrival at the ALS Chemex laboratory, core samples were logged into a database, placed into a stainless-steel tray, then dried for four to eight hours at 100°C (depending on moisture content). Samples were processed through primary and secondary crushers to 70% passing a 2 mm (Tyler 10 mesh) screen. Standard crushing practice included cleaning the equipment prior to and after each sample. This was done using coarse quartz material, and air cleaning the crushers after each sample. Crushed material was riffle-split to 250 g to 500 g, depending on the

requested analysis, and the remaining coarse reject material was returned to Pacific Rim for storage.

The 250 g to 500 g sub-sample was processed in a disk pulveriser to 85% passing a 75 µm (Tyler 200 mesh) screen. A 250 g pulp sample was collected and sent for analysis. All samples were analysed by ICP mass spectroscopy ("ICP-MS") for 48 elements, after digestion in nitric, perchloric, and hydrofluoric acids.

Gold analyses was conducted by FA on a 30 g aliquot with an atomic absorption finish. For assays above 10 g/t Au, FA with a gravimetric finish was employed. Silver samples with ICP-MS assays greater than 200 g/t Ag were re-run by FA with a gravimetric finish.

11.4 2017 – 2023 (AbraSilver Resource Corp.)

11.4.1 Logging

Core was delivered daily to the camp logging area. AbraSilver geologists inspected and re-aligned core, photographed each box, and measured the recovery, RQD and PLT(Point Load Test). Logging was conducted for lithology, alteration, and mineralogy. All information logged was captured in spreadsheets.

11.4.2 Sampling

Sampling was conducted at 1.5 meter intervals in weakly mineralized zones, reducing to one metre where mineralization was more intense. Breaks were introduced at obvious contacts. The core was split using a core diamond saw, with one half taken for assay and the other placed back in the box for storage.

The samples were then bagged, placed into larger rice bags with QA/QC materials and shipped to SGS Argentina SA in Salta ("SGS Salta"). Each shipment was accompanied by a manifest listing contents and laboratory instructions. A copy of the manifest was retained at site, another sent to AbraSilver's main office in Buenos Aires. The core and samples were continuously in the custody of AbraSilver personnel.

11.4.3 Sample Preparation and Analyses

All samples received at SGS Salta, Campo Quijano were processed to a pulp sachet and directly dispatched to their facility in Lima, Peru for analysis. The samples were dried at 100°C, then passed through a jaw crusher to 90% passing a -10-mesh screen. A 250 g split was processed in a ring and puck pulveriser to 95% passing -140 mesh.

All samples were analysed using a multi-element technique consisting of a four-acid digestion followed by ICP/AES detection. Gold was analysed by 50g Fire Assay with an AAS gravimetric

finish. Silver results greater than 100g/t were re-analysed using four acid digestion with an ore grade AAS gravimetric finish.

Lima and Campo Quijano laboratories returned the pulps and the rejects to Diablillos where they are stored.

11.5 Quality Assurance/Quality Control

Assay precision was assessed by reprocessing duplicate samples. Considering each stage of the analytical process from the primary stage of sample splitting, through sample preparation stages of crushing/splitting, pulverizing/splitting, and assaying.

11.5.1 Pre-AbraSilver QA/QC

The QA/QC programs conducted since the beginning of the project have been reported by Wardrop (2009), AMC Consultants Pty Ltd. in M3 (2011), RPA (2018), MP (2021) and MP PEA (2022). All these reports refer to internal third-party studies, including Bruce Goad (1994), Ristorcelli (1997) and Barrick (1998). This is summarised below.

11.5.1.1 1993 Drilling Campaign

BHP-Utah implemented a protocol for field duplicates to be taken for ten percent of total population. A selection of high-grade pulp samples was additionally sent to Canada (Chemex Lab, Vancouver). Original and duplicate reject samples were assayed at SGS Chile Ltd.

The entire 1993 drilling campaign was reviewed by Bruce Goad from Inukshuk Exploration Inc. on 18th March 1994. An internal report summarised logging, sampling, and assaying protocols. The report included original certificates for each hole and a comparative table for duplicates. In the author's opinion, the sampling and analytical work for this program was conducted in an appropriate fashion and used industry standard methods. Assaying was done by well-known independent commercial laboratories.

11.5.1.2 1996 – 1999 Drilling Campaign

Barrick initially implemented a protocol for field duplicates to be taken every ten samples, with selected samples re-assayed at a secondary laboratory. In 1998 revised procedures for the RC drilling were implemented based upon recommendations by Smee and Associates Consulting Ltd. These procedures included the following:

- Every 20 m, a field duplicate was collected, assigned a new sample number, and inserted into the sample stream.
- One standard and one blank were inserted after every 40th sample.

The standard material was obtained from Barrick's Pascua Project in Chile. The blank comprised gneiss from a bulk material supplier. Five samples of the blank material were sent to each of three laboratories to confirm that they were not mineralized.

11.5.1.3 2007 – 2008 Drilling Campaign

Assay QA/QC protocols were established by Pacific Rim, on behalf of SSRI. One control sample, consisting of either a blank, standard, or field duplicate, was inserted every 20th sample. Check assays were sent to a secondary laboratory, Assayers Canada in Vancouver. These were also conducted at a rate of no less than one in twenty samples.

A total of 6,561 duplicates or repeats, representing 11.54% of the database compiled during the period, were collected up to 2007. A further 600 duplicates representing 7.23% of the database, were taken during 2007 and 2008. During 2007 and 2008, 952 standards and blanks were inserted into the sample stream, representing 11.47% of the total.

Wardrop (2009) reported that C. Vallat reviewed the 2007-08 QA/QC data for SSRI. No concerns were reported, and the database was declared suitable for Mineral Resource estimation.

11.6 Discussion of QA/QC Assays prior to AbraSilver

In the QP's opinion, the sampling and analytical work between 1996 and 2008 was conducted in an appropriate fashion. Assaying used conventional, industry standard methods by well-known independent commercial laboratories.

The number of holes, orientation, and sampling methods employed were considered appropriate for the mineralization. Drill cuttings, core, and samples were handled solely by operator personnel or their contractors and securely stored. The site is remote and was attended continuously during the drilling and sampling operations reducing any opportunity for tampering.

The author notes that a manual blade splitter has been used for much of the sampling. These devices, if used properly, can perform satisfactorily. However, a diamond saw is generally superior in producing unbiased samples. A noted risk is that the diamond blade can wash out fine material. The QP recommends that future drilling programs utilise a diamond saw splitter.

The QP considers that the QA/QC protocols for the majority of the drilling meets a suitable standard. The only concern noted was with the precision of some assay results of SGS in Mendoza. The insertion rate for control samples appears adequate, however, detailed reports of QA/QC results should be produced in the future. The information is currently spread in different internal paper documents. It is recommended these reports be retained as a reference for future technical reports and audits.

In the author's opinion, the sampling and analytical work on Oculto prior to AbraSilver is acceptable for use in Mineral Resource estimation.

11.7 AbraSilver QA/QC

11.7.1 2017 to 2021

AbraSilver assay QA/QC protocols included insertion of blanks, standards (two types), and core duplicates into the sample stream. Blanks were inserted at a rate of one for every 25 samples, core duplicates were taken every 25 samples.

Two standards were taken from the 2012 drilling and inserted at a rate of one in 25 samples. Standards PM 1122 SR-I and STRT-04, were commercial reference materials prepared by WCM Minerals, of Burnaby, BC, Canada, and Smee and Associates Consulting Ltd., of North Vancouver, B.C., Canada respectively.

Reference values for these standards have been provided in Table 11-1.

Table 11-1: Certified Reference Material Specifications

Certified Reference Material						
Parameter	PM-1122			STRT-04		
Element	Au	Ag	Cu	Au	Ag	Cu
Unit	[g/t]	[g/t]	[ppm]	[g/t]	[g/t]	[ppm]
Expected Value	1.37	168	6,500	0.861	26.8	24,740
Two Standard deviation	0.08	11	162	0.026	2.8	480

A total of 926 blanks, representing 5.9%, 450 standards (2.9%), 750 core duplicates (4.8%) and 74 reject duplicates were submitted during the program. From a total of 15,750 samples the overall QA/QC samples represent 13.52% of the total population of samples. Only 4 samples were detected with no description, representing no significant quantity. Industry best practice recommends at least 10% of the total population.

A summary of the QA/QC sample numbers can be found in Table 11-2.

Table 11-2: Summary of AbraSilver QA/QC Sample Counts

Sample Type	Count	Percentage	STRT-04	PM 1122 (SR-I)	Core	Reject	Pulp
Number of samples	15,750	100%					
Original	13,620	87%					
Blank	926	5.9%					
CRM	450	2.9%	262	188			
Duplicate	750	4.8%			676	74	0
Validation	4	0%					

Analysis detection limits for the ICP-AES method were 0.5 g/t Ag and 5.0 ppb Au. AbraSilver protocol for a blank failure is ten times the detection limit. None of the blanks returned values that met this definition. Only two blank samples returned a gold value greater than five times the detection limit and one blank sample returned a silver value greater than five times the detection limit.

The gold performance and silver performance for blanks has been presented in Figure 11-1 and Figure 11-2.

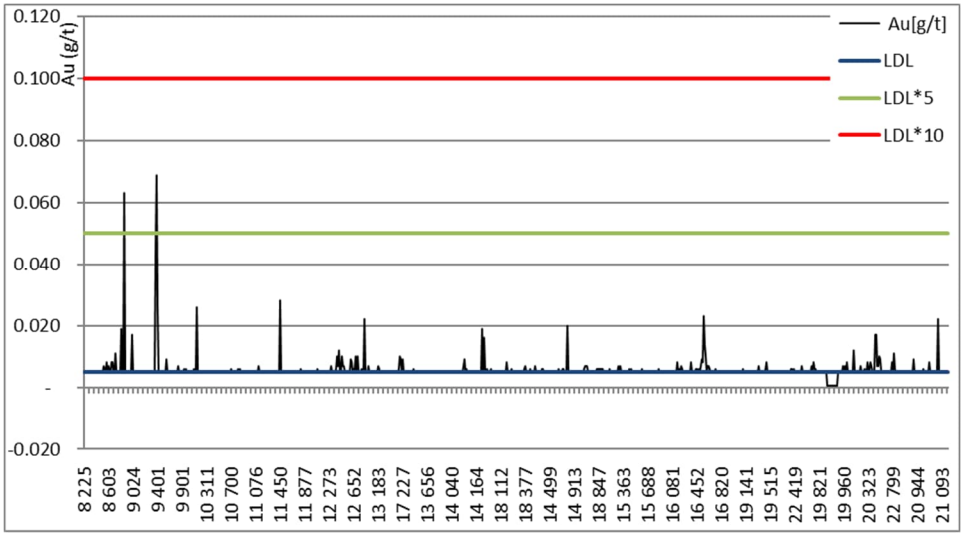


Figure 11-1: Blank Sample Performance for Gold

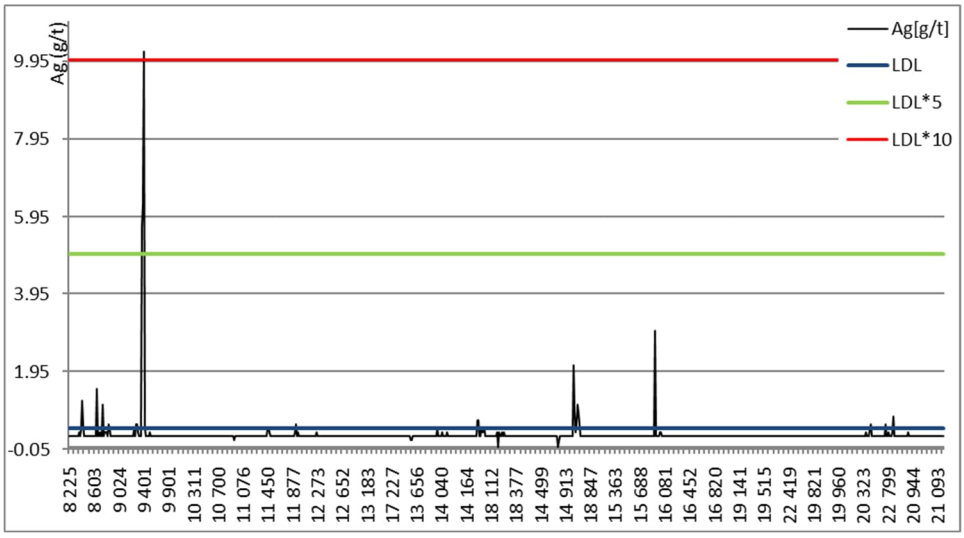


Figure 11-2: Blank Sample Performance for Silver

AbraSilver defines a reference material (“CRM”) failure as a value that differs from the recommended value by more than 5%, equivalent to three standard deviations.

Four standards returned values outside this error limit for CRM STRT-04. Three for gold and one for silver. For CRM PM11 all values were within the error limit.

The gold performance and silver performance for CRM STRT-04 can be seen in Figure 11-3 and Figure 11-4. CRM PM 1122 can be seen in Figure 11-5 and Figure 11-6.

Core duplicates were obtained from splitting half cores into two separate samples equivalent to 1/4 core, each one bagged and labelled separately. Core duplicates reflect all levels of errors from its first splitting to analytical error. These features are evidenced in Figure 11-7 and Figure 11-8, which show the moderate to high variability. The core duplicates were observed to agree quite closely with the original assays for gold and silver.

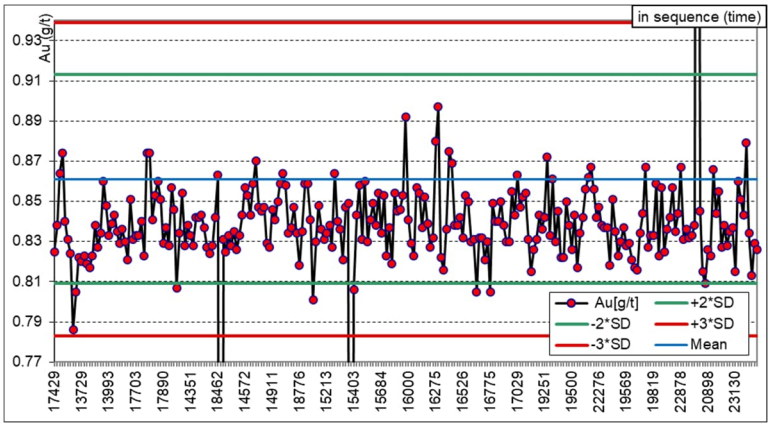


Figure 11-3: Certified Reference Material STRT-04, Gold Performance

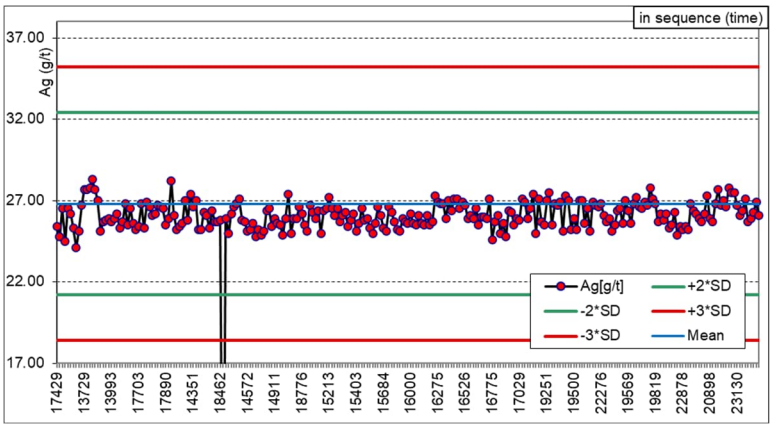


Figure 11-4: Certified Reference Material STRT-04, Silver Performance

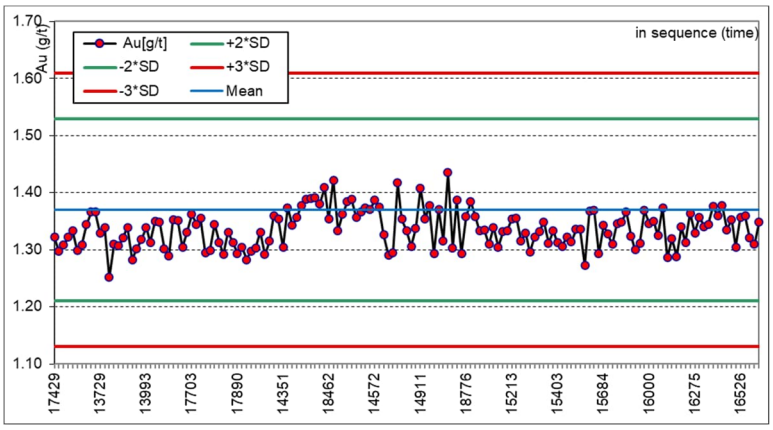


Figure 11-5: Certified Reference Material PM 1122, Gold Performance

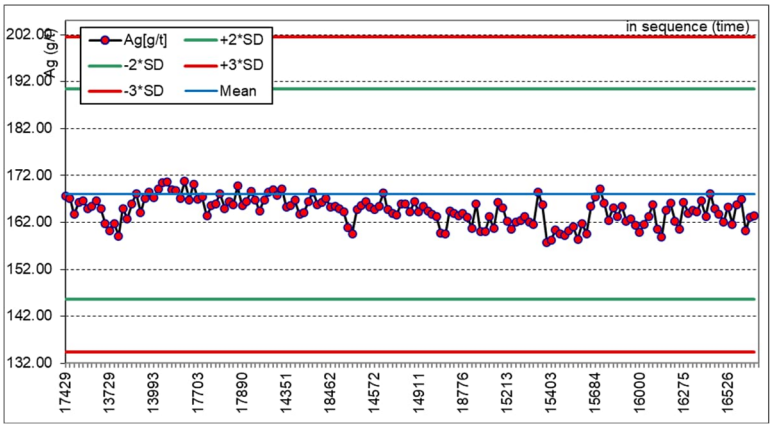


Figure 11-6: Certified Reference Material PM 1122, Silver Performance

Core duplicates were obtained from splitting half cores into two separate samples equivalent to 1/4 core with each sample bagged and labelled separately. Scattergrams for gold and silver duplicates have been presented in Figure 11-7 and Figure 11-8. Showing moderate to high variability. Core duplicates were observed to agree quite closely with original assays for gold and silver.

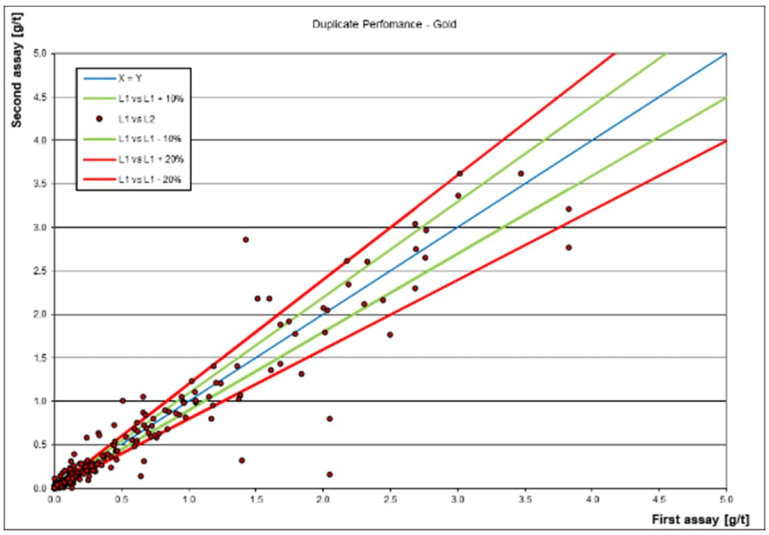


Figure 11-7: RMA Scattergram for Duplicate Performance of Gold

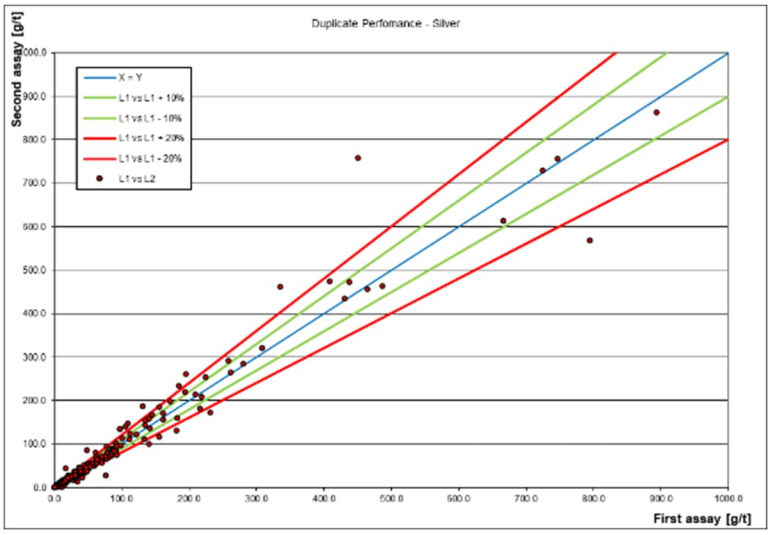


Figure 11-8: RMA Scattergram for Duplicate Performance of Silver

11.7.2 2021 to 2022 (First Semester)

AbraSilver decided to generate its own internal reference material. This was due to problems with the CRM supplier and the complexity of finding a reference material with oxidised matrix over the entire range of gold and silver.

Three types of internal refence material (IRM) were generated to comply with the grade range of mineralization. IRM was generated based on 50 kg of rejects returned by the SGS laboratory following an assay. The reference material was processed in a disk pulveriser to 85% passing a 75 µm (Tyler 200 mesh) screen. Then, the total of 50 kg was homogenized and divided into 500

individual sachets of 100-gr aliquots. A total of thirty samples were sent to certified laboratories to carry out round robin testing including ten samples to each of ALS Chemex, Alex Stewart Assayers and SGS. Following the receipt of assays, a statistical analysis was performed to determine the expected value and the standard deviation for future use.

The same protocol of insertion for the three types of control samples was continued, as adopted in 2017-2021. This included the insertion of blanks, standards (one of the three types), and core duplicates. IRM were inserted at a rate of one every 25 samples, blanks were inserted at a rate of approximately one every 25 samples, and core duplicates were taken approximately once every 25 samples. Using two IRM for each CRM.

The specifications of the standard are listed in Table 11-3.

Table 11-3: Certified Reference Materials

Certified and Internal Reference Material								
Parameter	STRT-04 (Certified)		ASDBL_Au-Ag_H01 (Internal)		ASDBL_Au-Ag_M01 (Internal)		ASDBL_Au-Ag_L01 (Internal)	
Element	Au	Ag	Au	Ag	Au	Ag	Au	Ag
Unit	[g/t]	[g/t]	[g/t]	[g/t]	[g/t]	[g/t]	[g/t]	[g/t]
Expected Value	0.861	26.8	3.092	358	0.904	109	0.436	38.7
Standard deviation	0.026	2.8	0.12	6	0.068	4	0.040	1.935

During the 2021 – 2022 program, quality control samples were routinely included in sample batches including Blanks, Standards and Duplicates. Details of the sample counts are provided in Table 11-4. Of the 14,633 samples submitted, QA/QC samples represented 16.51%, exceeding industry best practices recommendations of at least 10% total population.

Table 11-4: Summary of AbraSilver QA/QC Sample Counts

Sample Type	Count	Percentage	STRT-04	Core	Reject	Pulp
Number of samples	14,633	100%				
Original	12,948	89%				
Blank	959	7.2%				
CRM	219	2.2%	473			
ASDBL_Au-Ag_H01	76	1.2%				
ASDBL_Au-Ag_M01	83	1.3%				
ASDBL_Au-Ag_L01	98	1.4%				
Duplicate	706	5.5%		705	1	0
Validation	7	1%		705	1	0

Lower detection limits for ICP-AES analysis were 0.5 g/t Ag and 5.0 ppb Au. AbraSilver protocol for definition of a blank's failure is ten times the detection limit. All the gold blank samples passed while two silver blank samples exceeded the tolerance.

The gold and silver performance in blanks can be seen in Figure 11-9 and Figure 11-10.

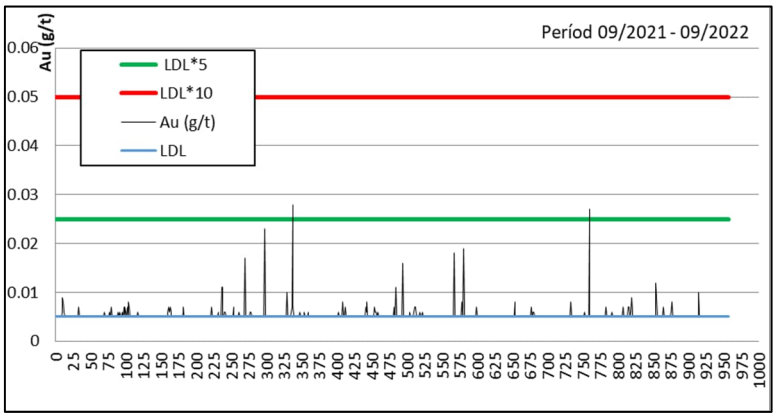


Figure 11-9: Gold Performance for Blank Samples

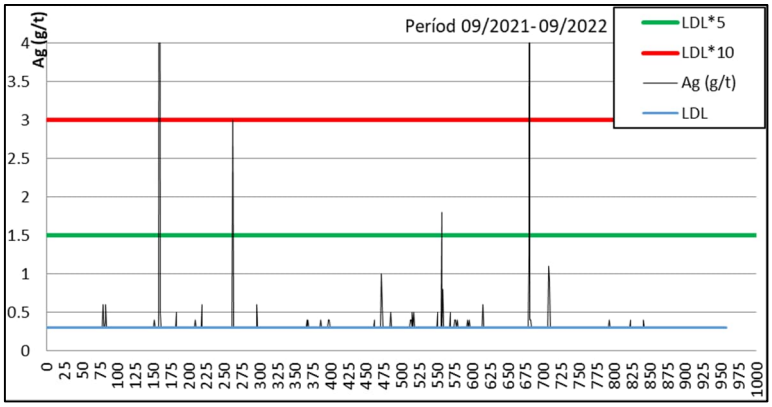


Figure 11-10: Silver Performance for Blank Samples

AbraSilver defines a reference material failure as differing from the recommended value of more than three times the standard deviation.

For CRM STRT-04, one result was outside this 5% error limit in Gold. No standards exceeded the limit in silver. The gold and silver performance for CRM STRT-04 can be seen in Figure 11-11 and Figure 11-12.

For the analysis of IRM H01, two standards returned a value outside the error limit in gold. Seven standards exceeded the lower limit in silver.

For the analysis of IRM M01, only one standard exceeded the limit for gold and none for silver. Finally for the analysis of IRM L01, one exceeded the limit value for gold and none for silver.

The gold and silver performance for IRM H01, M01 and L01 can be seen in Figure 11-13 to Figure 11-18.

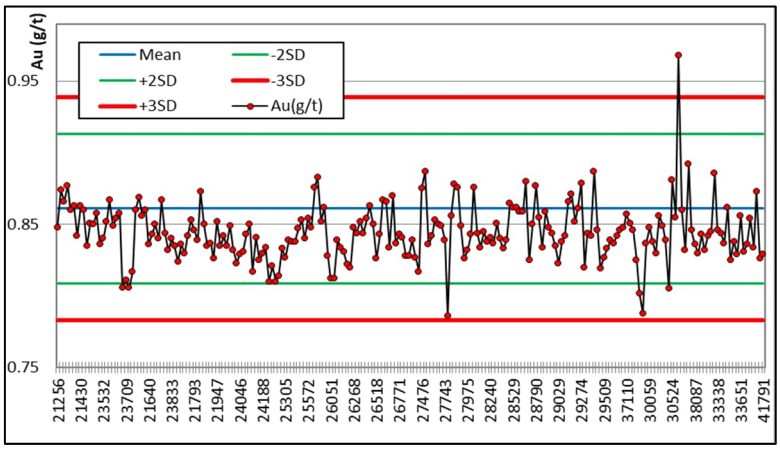


Figure 11-11: Certified Reference Material STRT-04, Gold Performance

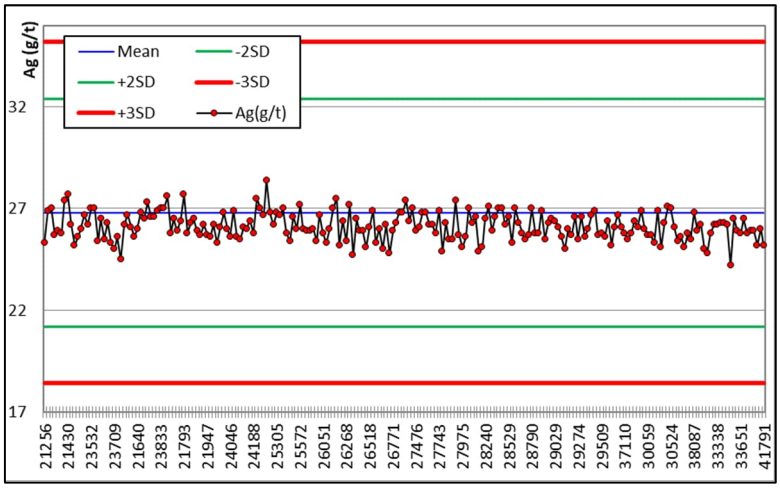


Figure 11-12: Certified Reference Material STRT-04, Silver Performance

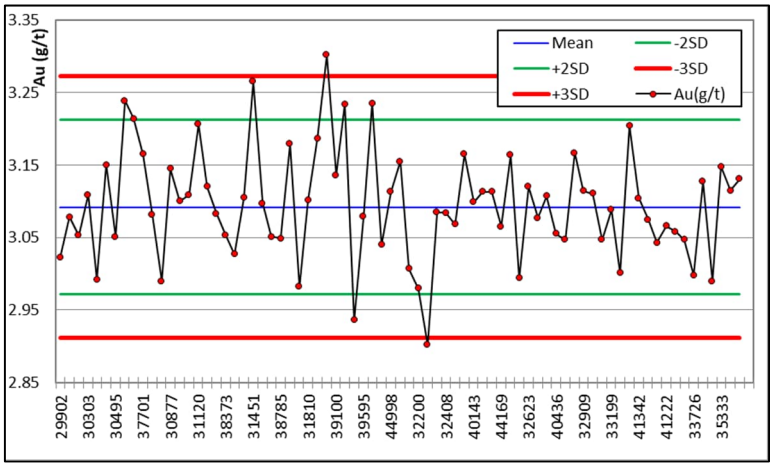


Figure 11-13: Internal Reference Material H01, Gold Performance

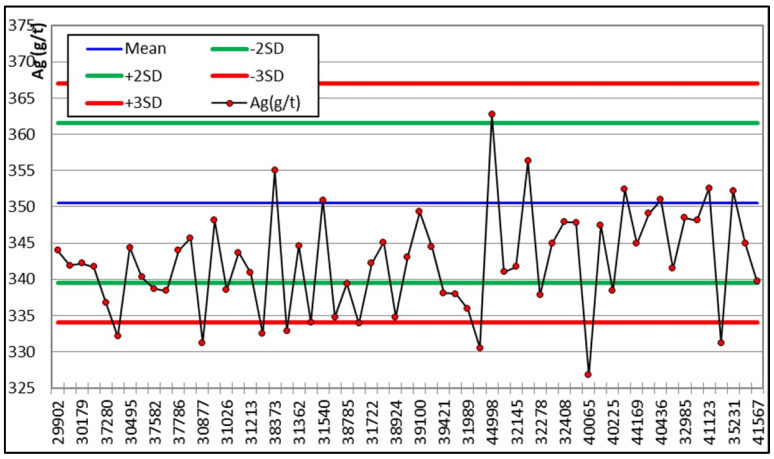


Figure 11-14: Internal Reference Material H01, Silver Performance

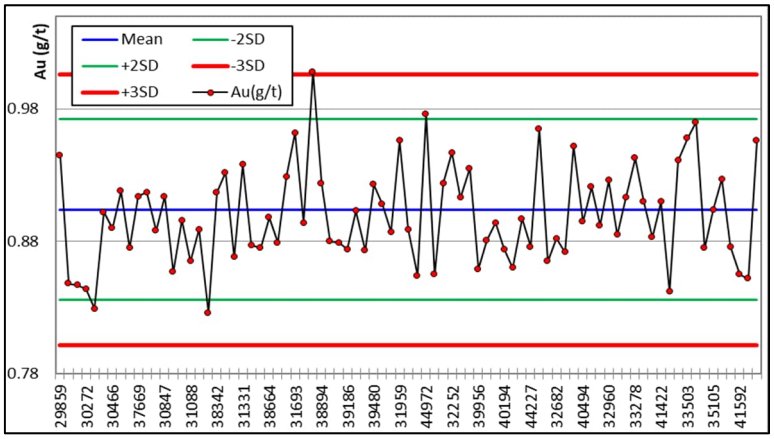


Figure 11-15: Internal Reference Material M01, Gold Performance

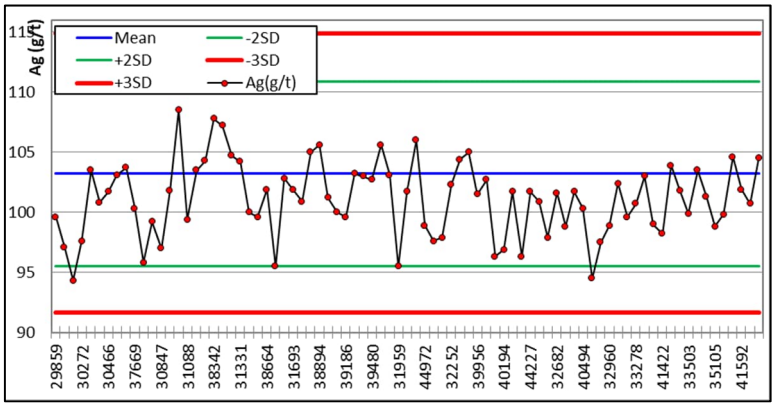


Figure 11-16: Internal Reference Material M01, Silver Performance

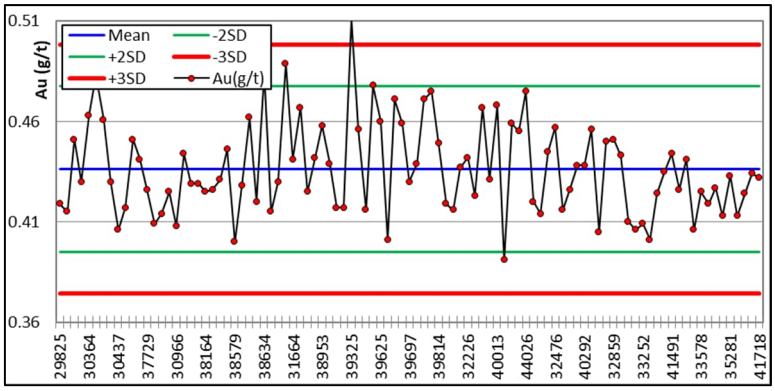


Figure 11-17: Internal Reference Material L01, Gold Performance

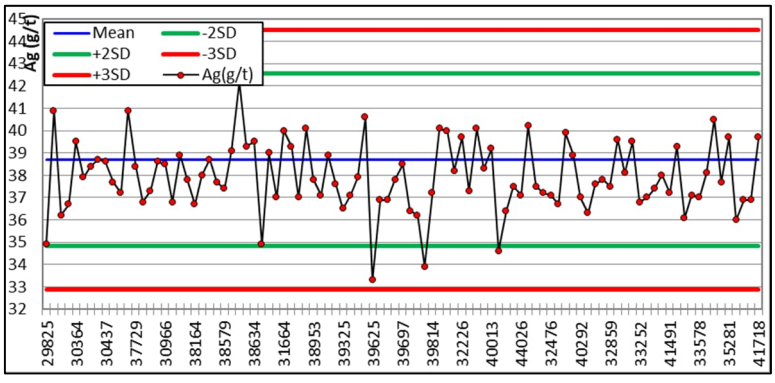


Figure 11-18: Internal Reference Material L01, Silver Performance

Core duplicates were obtained by dividing half the core into two separate 1/4 core equivalent samples, bagged and labelled separately. The results are shown in Figure 11-19 and Figure 11-20 with moderate to high variability.

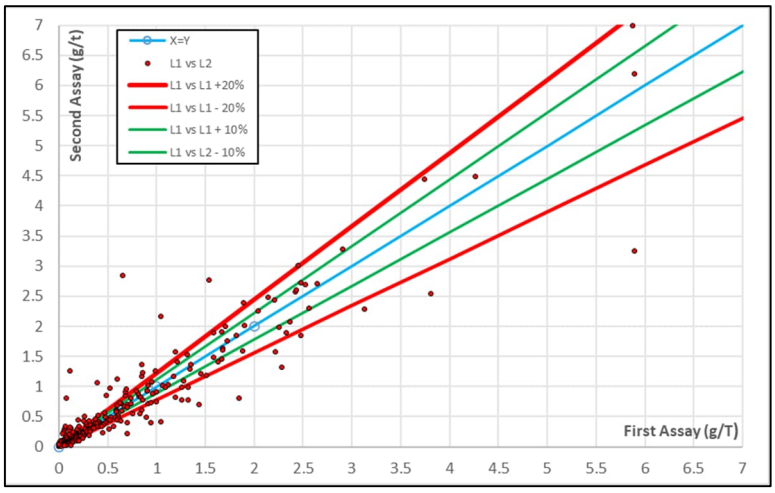


Figure 11-19: RMA Scattergram for Duplicate Performance of Gold

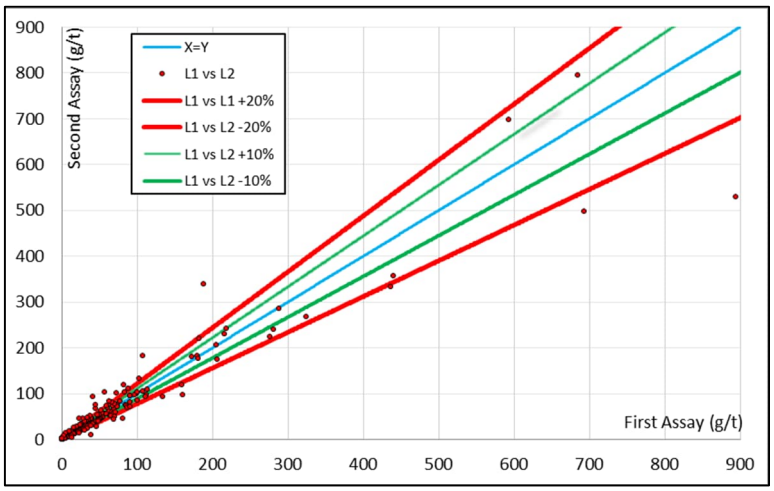


Figure 11-20: RMA Scattergram for Duplicate Performance of Silver

Based on the overall review and data analysis, the QP considers that the gold and silver accuracy during the 2017-2022 drilling campaigns is acceptable. Blank samples were assayed and most yielded values below the five times detection limit. There was no obvious evidence for gold or silver cross contamination during laboratory sample preparation. The RMA scattergram plots for gold and silver show good fit between the core duplicates and the original assays. A few outliers have however been observed due to high variability in the style of mineralization.

The author has concluded that the assay QA/QC protocols implemented by AbraSilver were consistent and no concerns were evident.

11.7.3 2022 (Second Semester) to 2023

Three types of internal refence material (IRM) were generated to cover the mineralisation grade range. IRM was generated based on 50 kg of rejects returned by assays at the SGS laboratory. The reference material was processed in a disk pulveriser to 85% passing a 75 µm (Tyler 200 mesh) screen. The total 50 kg was then homogenized and divided into 500 individual sachets of 100-gr aliquots. A total of thirty samples were sent to certified laboratories to carry out round robin testing. Ten samples were sent to each of ALS Chemex, Alex Stewart Assayers and SGS. Once assays were received, a statistical analysis was performed to determine the expected value and two-standard deviation.

The specifications of the internal reference material standards are listed in Table 11-5.

In addition, five certified reference material (CRM) standards were used. Compromising the range from high to low grade mineralization. The specifications of the standard are listed in Table 11-6.

The same protocol of insertion for the three types of control samples was continued, as applied in 2021-2022. Blanks, standards (one of the three types), and core duplicates were inserted into the sample stream. IRM and CRM were inserted at a rate of one every 25 samples, blanks at a rate of one every 25 samples, and core duplicates once every 25 samples. Using two IRM for each CRM.

Table 11-5: Internal Reference Materials Assay Specifications

Internal Reference Material						
Parameter	ASDBL_Au-Ag_H01		ASDBL_Au-Ag_M01		ASDBL_Au-Ag_L01	
Element	Au	Ag	Au	Ag	Au	Ag
Unit	[g/t]	[g/t]	[g/t]	[g/t]	[g/t]	[g/t]
Expected Value	3.092	358	0.9039	103.21	0.436	38.70
Standard deviation	0.12	4.58	0.034	3.849	0.040	1.935

Table 11-6: Certified Reference Materials Assay Specifications

Certified Reference Material										
Parameter	STRT-04		AuOx41		PLSUL59		AuOx-18		AuOx-33	
Element	Au	Ag	Au	Ag	Au	Ag	Au	Ag	Au	Ag
Unit	[g/t]	[g/t]	[g/t]	[g/t]	[g/t]	[g/t]	[g/t]	[g/t]	[g/t]	[g/t]
Expected Value	0.861	26.8	0.57	8.30	0.52	2 988	2.88	77.80	1.68	554.00
Standard deviation	0.026	2.8	0.04	0.44	0.05	44.71	0.20	1.93	0.10	16.65

During the 2022-2023 drilling campaign, sample batches included routine quality control samples. The detailed counts for each sample type have been included in Table 11-7. Of the 15,506 samples submitted, quality control samples represented 12.10%. Industry best practices recommend at least 10% of the total population.

Table 11-7: Summary of AbraSilver QA/QC Sample Counts

	Sample Type	Count	Percentage
	Number of samples	15,506	100%
	Original	13,607	87.75%
	Blank	803	5.18%
	Duplicate	539	3.48%
IRM	ASDBL_AU-AG_H01	99	0.64%
	ASDBL_AU-AG_M01	115	0.74%
	ASDBL_AU-AG_L01	137	0.88%
CRM	AuOx41	34	0.22%
	PLSUL59	29	0.19%
	AuOx-18	50	0.32%
	AuOx-33	53	0.34%
	STRT-04	40	0.26%

Laboratory detection limits for the ICP-AES analyses were 0.5 g/t Ag and 5.0 ppb Au. AbraSilver's procedure for defining the failure of a blank sample is ten times the detection limit. There were no failures for gold or silver values for blank samples.

The gold performance and silver performance in blanks can be seen in Figure 11-21 and Figure 11-22.

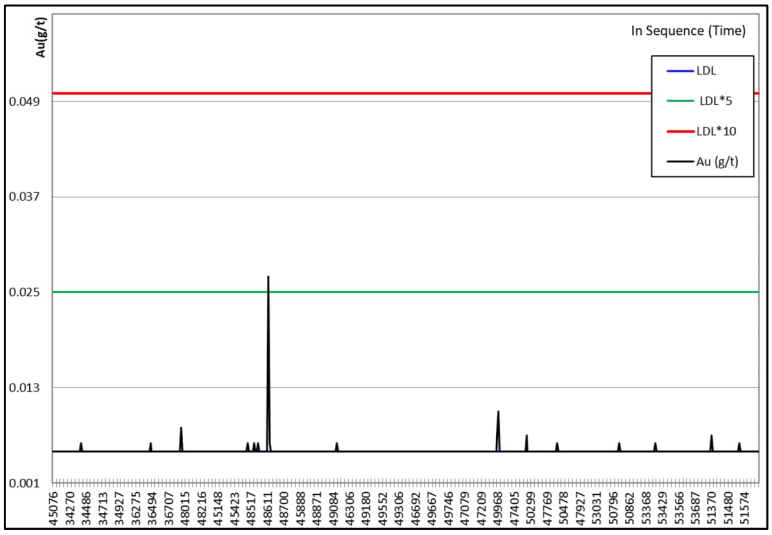


Figure 11-21: Internal Reference Material, Blank, Gold Performance

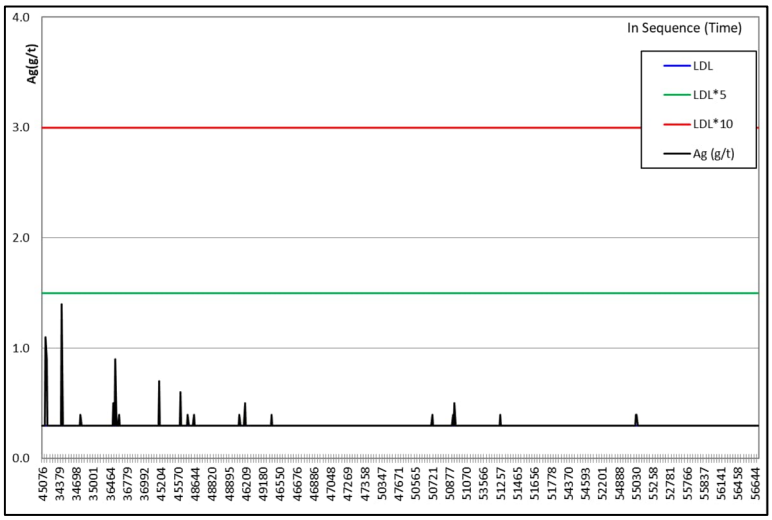


Figure 11-22: Internal Reference Material, Blank, Silver Performance

AbraSilver defines a reference material (IRM or CRM) failure as a value that differs from the recommended value by more than 5%. Approximately three times the standard deviation.

For the analysis of CRM STRT-04, no samples returned a value outside the error limit in gold or silver. The gold and silver performance for STRT-04 can be seen in Figure 11-23 and Figure 11-24.

For the analysis of CRM AuOx-41, no samples returned a value outside this 5% error limit in gold and silver. The gold and silver performance for AuOx-41 can be seen in Figure 11-25 and Figure 11-26.

For the analysis of CRM PLSUL59, no samples returned a value outside the error limit in gold and silver. The gold and silver performance for PLSUL59 can be seen in Figure 11-27 and Figure 11-28.

For the analysis of CRM AuOx-18, no samples returned a value outside this 5% error limit in gold and silver. The gold and silver performance for AuOx-18 can be seen in Figure 11-29 and Figure 11-30.

For the analysis of CRM AuOx-33, no samples returned a value outside this 5% error limit in gold and silver. The gold and silver performance for AuOx-33 can be seen in Figure 11-31 and Figure 11-32.

For the analysis of IRM ASDBL_H01, no samples returned a value outside this 5% error limit in gold and silver. The gold and silver performance for ASDBL_H01 can be seen in Figure 11-33 and Figure 11-34.

For the analysis of IRM ASDBL_M01, no samples returned a value outside this 5% error limit in gold and silver. The gold and silver performance for ASDBL_M01 can be seen in Figure 11-35 and Figure 11-36.

For the analysis of IRM ASDBL_L01, no samples returned a value outside this 5% error limit in gold and silver. The gold and silver performance for ASDBL_L01 can be seen in Figure 11-37 and Figure 11-38.

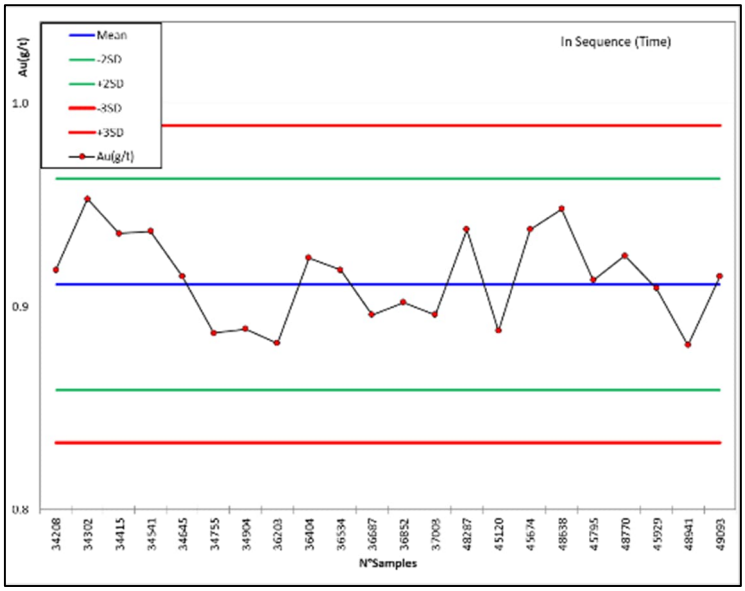


Figure 11-23: Certified Reference Material STRT-04, Gold Performance

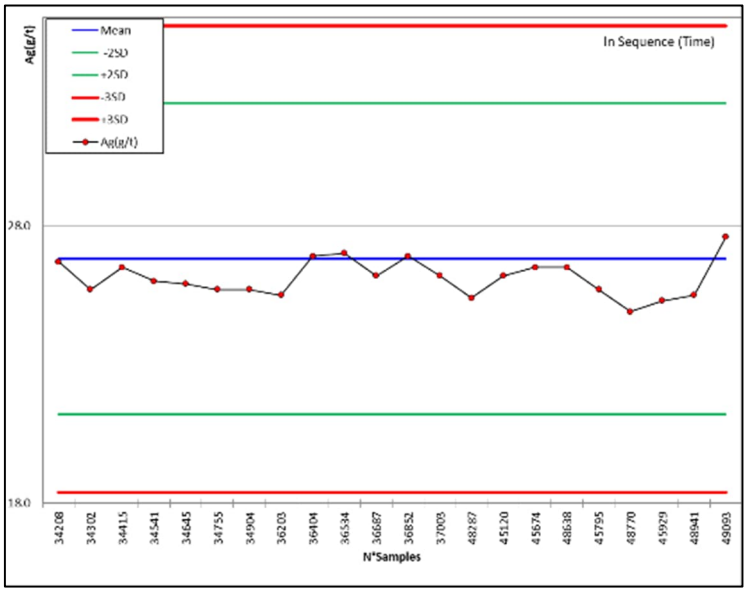


Figure 11-24: Certified Reference Material STRT-04, Silver Performance

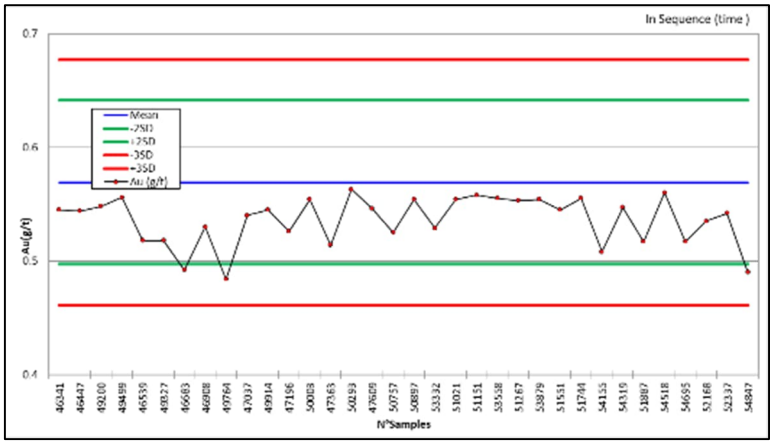


Figure 11-25: Certified Reference Material AuOx-41, Gold Performance.

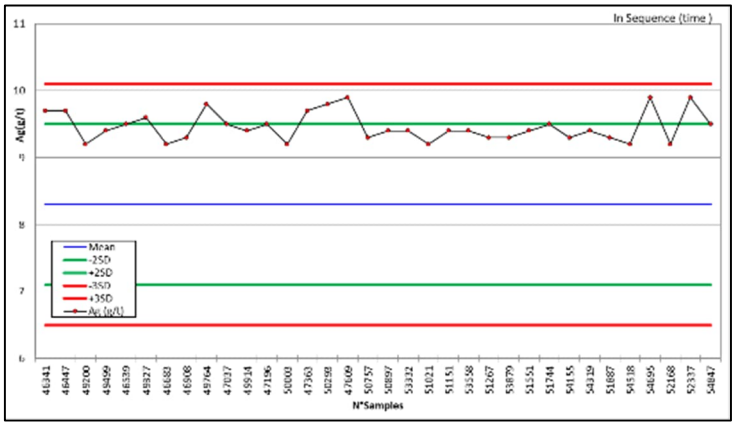


Figure 11-26: Certified Reference Material AuOx-41, Silver Performance

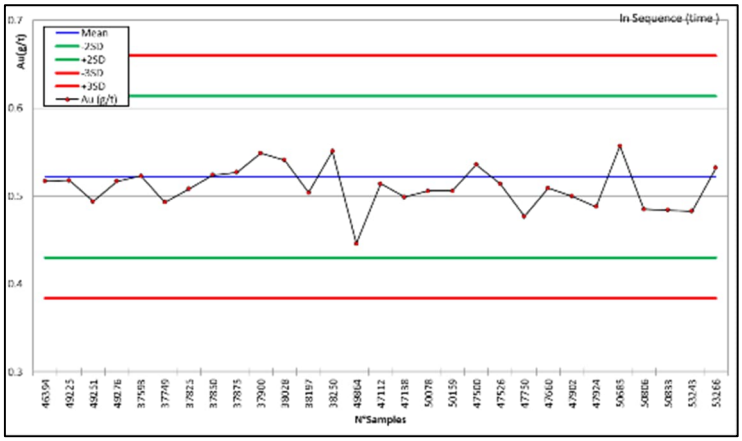


Figure 11-27: Certified Reference Material PLSUL59, Gold Performance

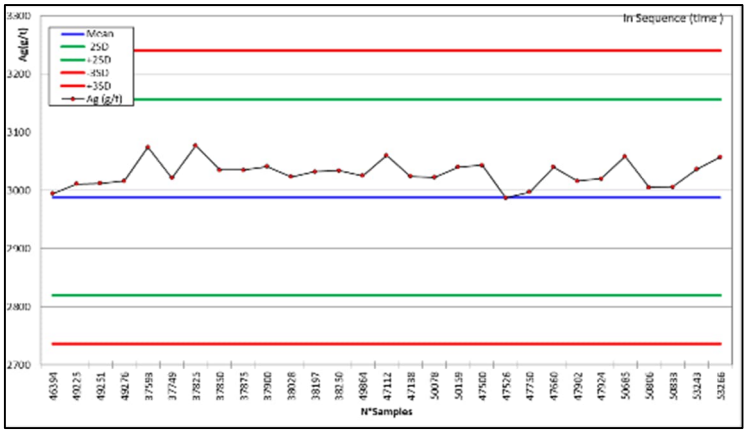


Figure 11-28: Certified Reference Material PLSUL59, Silver Performance

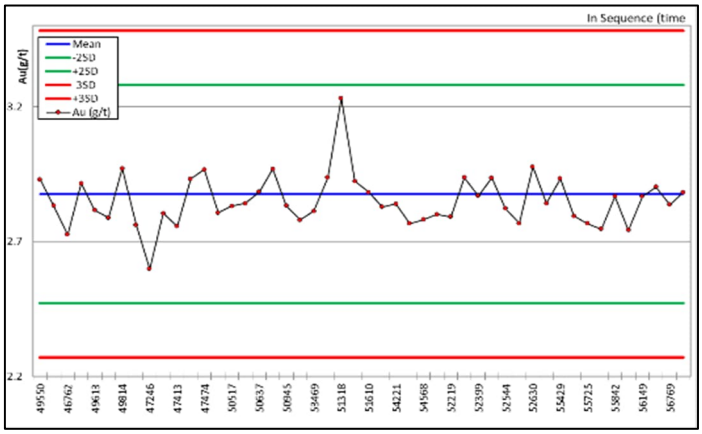


Figure 11-29: Certified Reference Material AuOx-18, Gold and Silver Performance

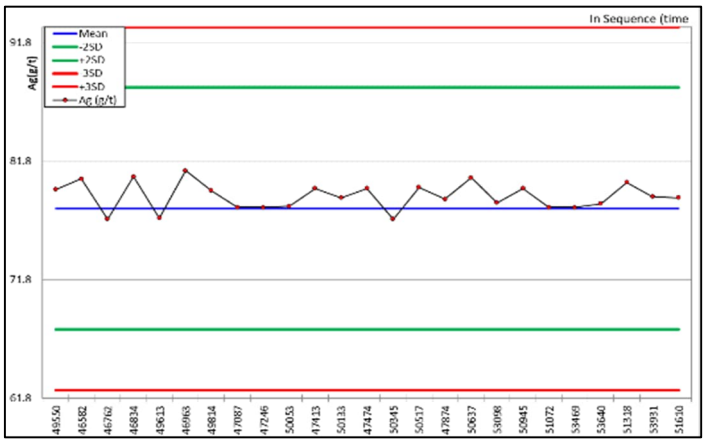


Figure 11-30: Certified Reference Material AuOx-18, Gold and Silver Performance

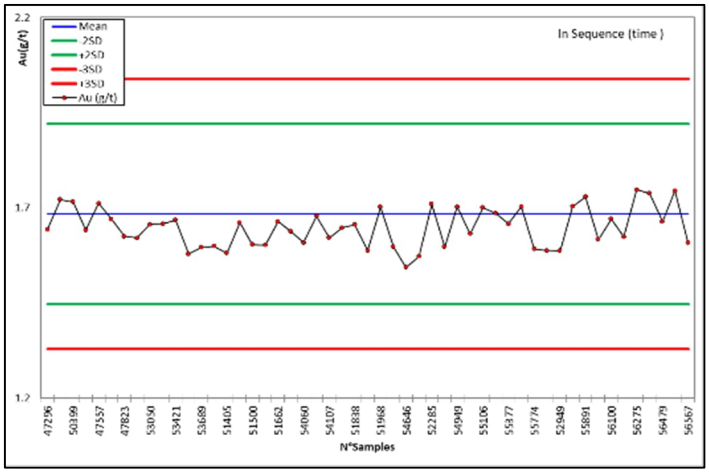


Figure 11-31: Certified Reference Material AuOx-33, Gold Performance

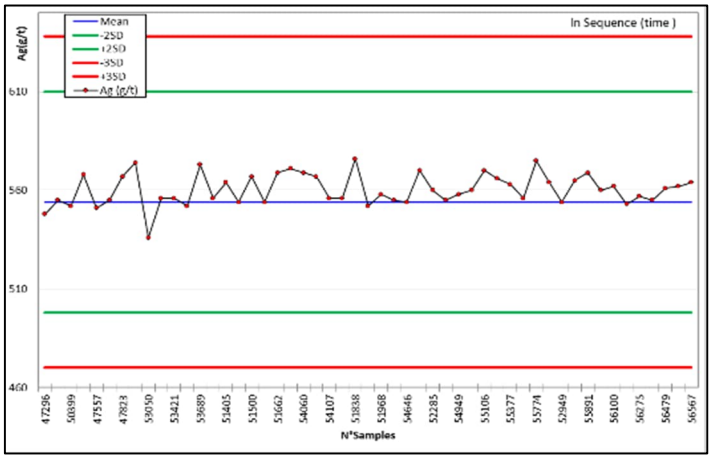


Figure 11-32: Certified Reference Material AuOx-33, Silver Performance

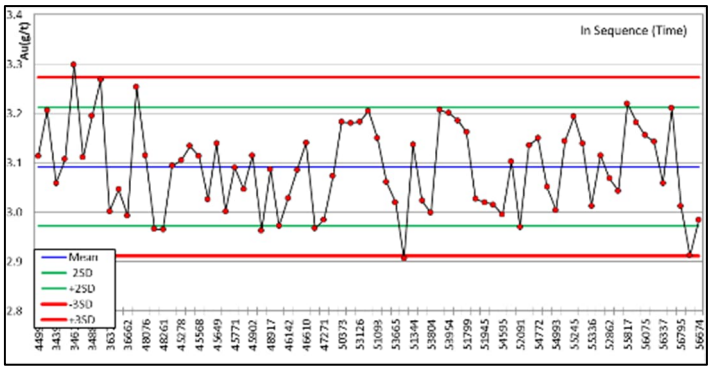


Figure 11-33: Internal Reference Material ASDBL_H01, Gold Performance

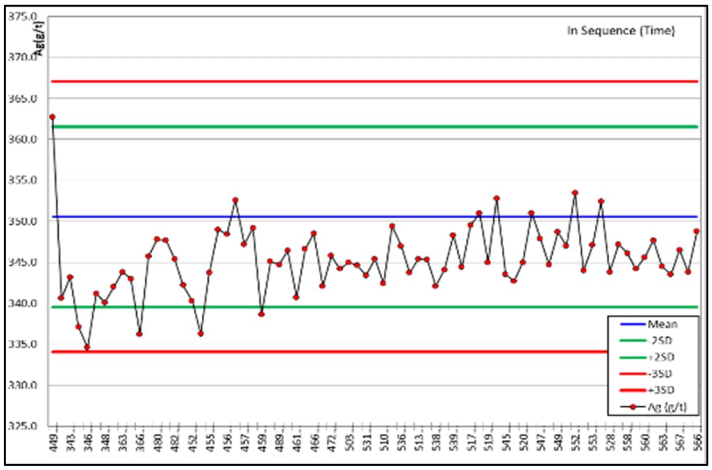


Figure 11-34: Internal Reference Material ASDBL_H01, Silver Performance

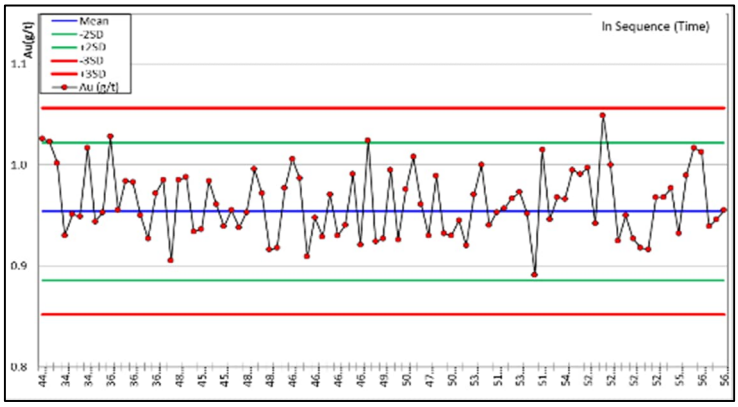


Figure 11-35: Internal Reference Material ASDBL_M01, Gold Performance

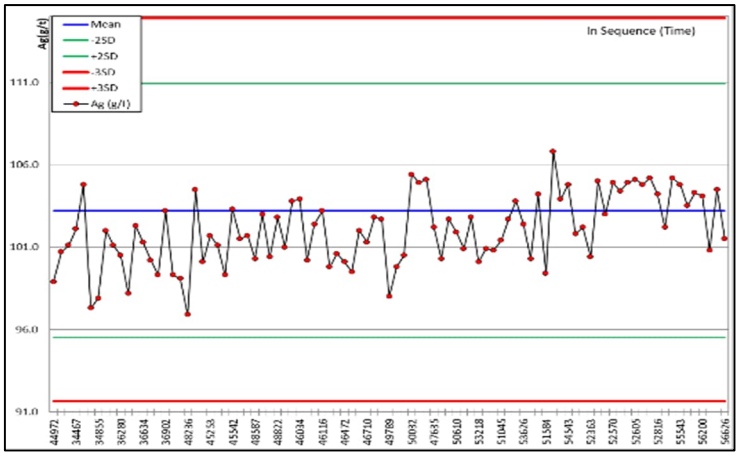


Figure 11-36: Internal Reference Material ASDBL_M01, Silver Performance

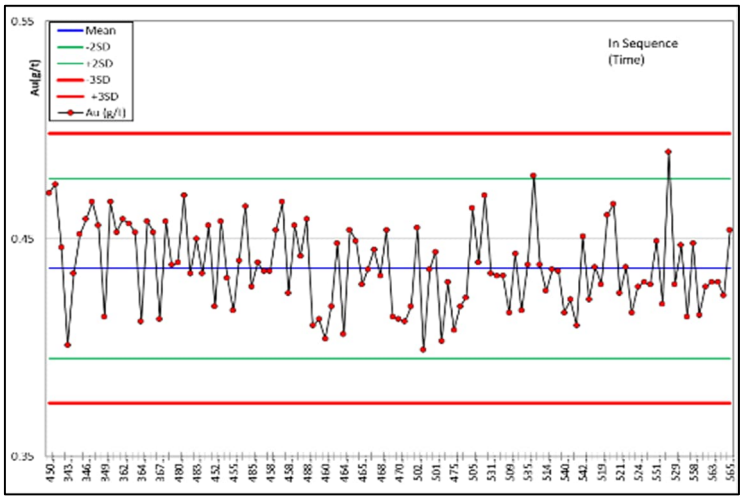


Figure 11-37: Internal Reference Material ASDBL_L01, Gold Performance.

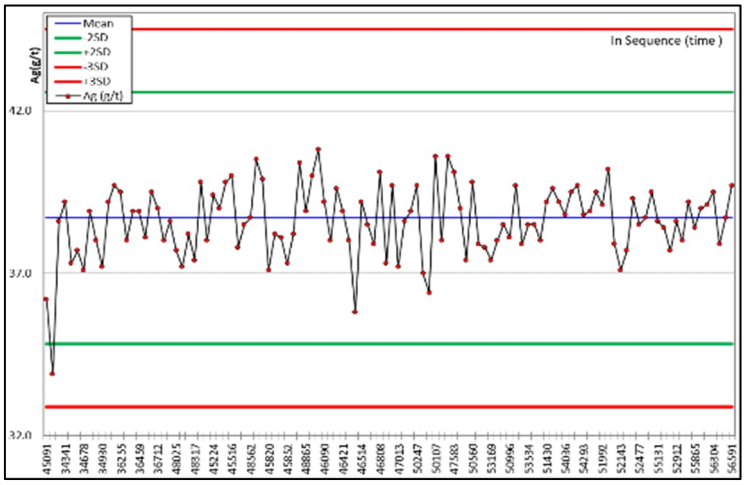


Figure 11-38: Internal Reference Material ASDBL_L01, Silver Performance.

For sample batches that included reference material failures, the laboratory was requested to re-assay samples. The corrected assay data was then loaded to the database.

Core duplicates were obtained by dividing half the core into two separate 1/4 core equivalent samples, bagged and labelled separately. Charts in Figure 11-39 and Figure 11-40 show the relevant RMA scattergrams and the Min-Max versus Hyperbolic Method scattergrams (Figure 11-41 and Figure 11-42). Indicating low to moderate variability.

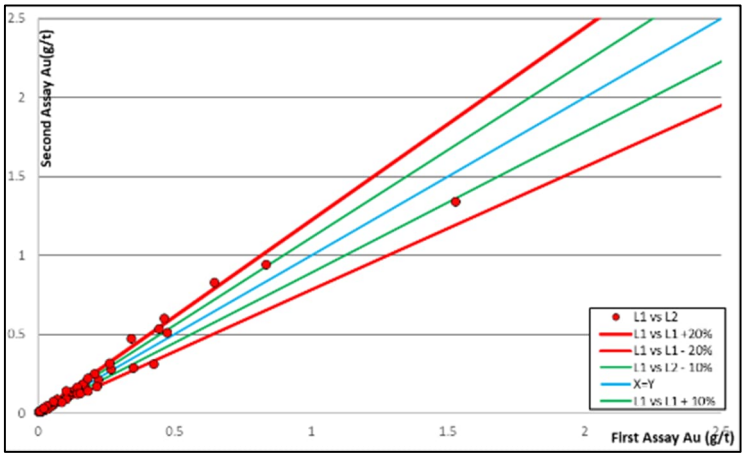


Figure 11-39: RMA Scattergram for Duplicate Performance of Gold

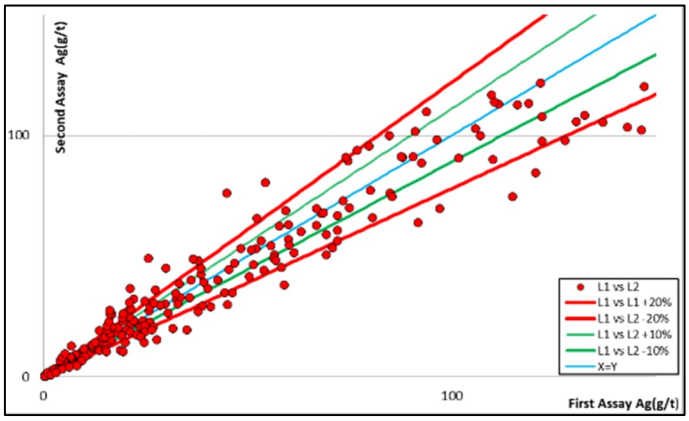


Figure 11-40: RMA Scattergram for Duplicate Performance of Silver

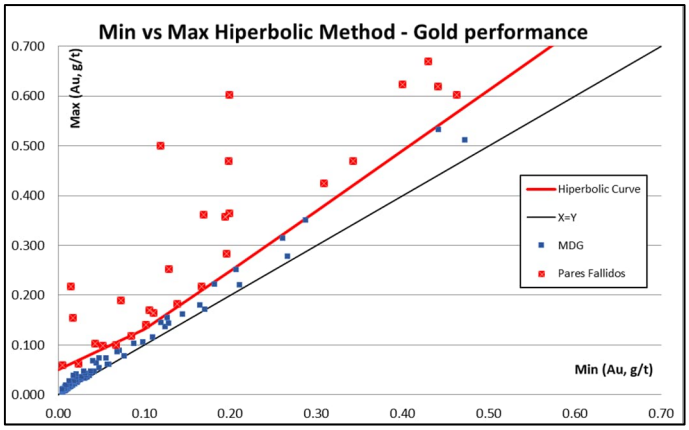


Figure 11-41: Min-Max versus Hyperbolic Method Scattergram for Duplicate Performance of Gold

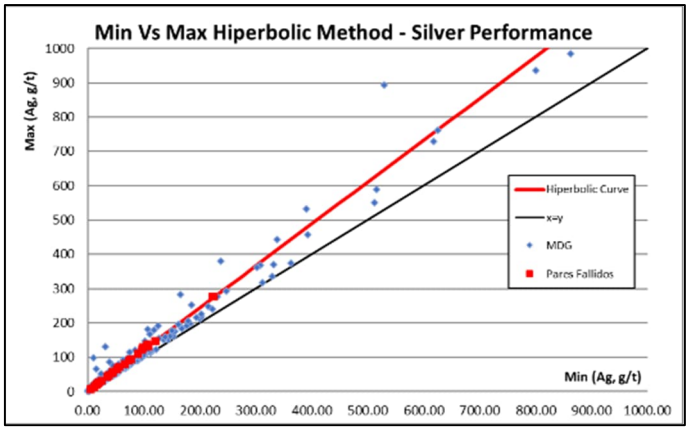


Figure 11-42: Min-Max versus Hyperbolic Method Scattergram for Duplicate Performance of Silver

11.7.4 Period 2024 to 2025. (Phase IV Drilling Campaign)

The same three types of internal refence material (IRM) used during Phase III Drilling Campaign were used. They generated internally to cover the mineralisation grade range. IRM was generated based on 50 kg of rejects returned by assays at the SGS laboratory. The reference material was processed in a disk pulveriser to 85% passing a 75 µm (Tyler 200 mesh) screen. The total 50 kg was then homogenized and divided into 500 individual sachets of 100-gr aliquots. A total of thirty samples were sent to certified laboratories to carry out round robin testing. Ten samples were sent to each of ALS Chemex, Alex Stewart Assayers and SGS. Once assays were received, a statistical analysis was performed to determine the expected value and two-standard deviation as mentioned in the previous sub section.

The specifications of the internal reference material standards are listed in Table 11-8.

In addition, eleven certified reference material (CRM) standards were used. Compromising the range from high to low grade mineralization. The specifications of the standard are listed in Table 11-9.

The same protocol of insertion for all types of control samples was continued at the Phase IV Drilling Campaign during 2024-2025, as applied in 2021-2022. Blanks, standards (one of the three types), and core duplicates were inserted into the sample stream. IRM and CRM were inserted at a rate of one every 25 samples, blanks at a rate of one every 25 samples, and core duplicates once every 25 samples. Using two IRM for each CRM approximately.

Table 11-8: Internal Reference Materials Assay Specifications

Internal Reference Material						
Parameter	ASDBL_Au-Ag_H01		ASDBL_Au-Ag_M01		ASDBL_Au-Ag_L01	
Element	Au	Ag	Au	Ag	Au	Ag
Unit	[g/t]	[g/t]	[g/t]	[g/t]	[g/t]	[g/t]
Expected Value	3.092	358	0.9039	103.21	0.436	38.70
Standard deviation	0.12	4.58	0.034	3.849	0.040	1.935

Table 11-9: Certified Reference Materials Assay Specifications

Certified Reference Material				
Parameter	Element	Unit	Expected Value	Two Standard deviation
AuOx-18	Au	[g/t]	2.876	0.1997
	Ag	[g/t]	77.80	1.9291
AuOx-33	Au	[g/t]	1.684	0.1180
	Ag	[g/t]	554.0	28.000
AuOx-52	Au	[g/t]	1.559	0.136

Certified Reference Material				
Parameter	Element	Unit	Expected Value	Two Standard deviation
	Ag	[g/t]	17.3	0.7
AuOx41	Au	[g/t]	0.569	0.0425
	Ag	[g/t]	8.300	0.4404
STRT-04	Au	[g/t]	0.861	0.0260
	Ag	[g/t]	26.80	2.8000
STRT-07	Au	[g/t]	1.085	0.0180
	Ag	[g/t]	41.50	1.8500
HDRT-02	Au	[g/t]	0.161	0.0120
	Ag	[g/t]	321.0	15.000
PLSUL59	Au	[g/t]	0.522	0.0492
	Ag	[g/t]	2 988	44.7080
OREAS 151C	Au	[g/t]	0.0802	0.0049
	Ag	[g/t]	----	----
OREAS 507-B	Au	[g/t]	0.175	0.0060
	Ag	[g/t]	----	----
OREAS 504-D	Au	[g/t]	1.460	0.0350
	Ag	[g/t]	2.690	0.114

During the 2024-2025 drilling campaign, sample batches included routine quality control samples. The detailed counts for each sample type have been included in Table 11-10. Of the 15,667 samples submitted, quality control samples represented 14.54%. Industry best practices recommend at least 10% of total population.

Table 11-10: Summary of AbraSilver QA/QC Sample Counts

	Sample Type	Count	Percentage
	Number of samples	15,667	100%
	Original	13,387	85.45%
	Blank	977	6.24%
	Duplicate	727	4.64%
	IRM	343	2.19%
	CRM	231	1.47%
IRM	ASDBL_Au-Ag_H01	55	0.35%
	ASDBL_Au-Ag_M01	148	0.94%
	ASDBL_Au-Ag_L01	140	0.89%
CRM	AuOx-18	97	0.62%
	AuOx-33	22	0.14%
	AuOx-52	11	0.07%
	AuOx41	0	0.00%
	STRT-04	2	0.01%
	STRT-07	7	0.04%
	HDRT-02	34	0.22%
	PLSUL59	44	0.28%
	OREAS 151C	6	0.04%
	OREAS 507-B	6	0.04%
	OREAS 504-D	2	0.01%

Laboratory detection limits for the ICP-AES analyses were 0.5 g/t Ag and 5.0 ppb Au. AbraSilver's procedure for defining the failure of a blank sample is ten times the detection limit. There were no failures for gold or silver values for blank samples.

The gold performance and silver performance in blanks can be seen in Figure 11-43 and Figure 11-44.

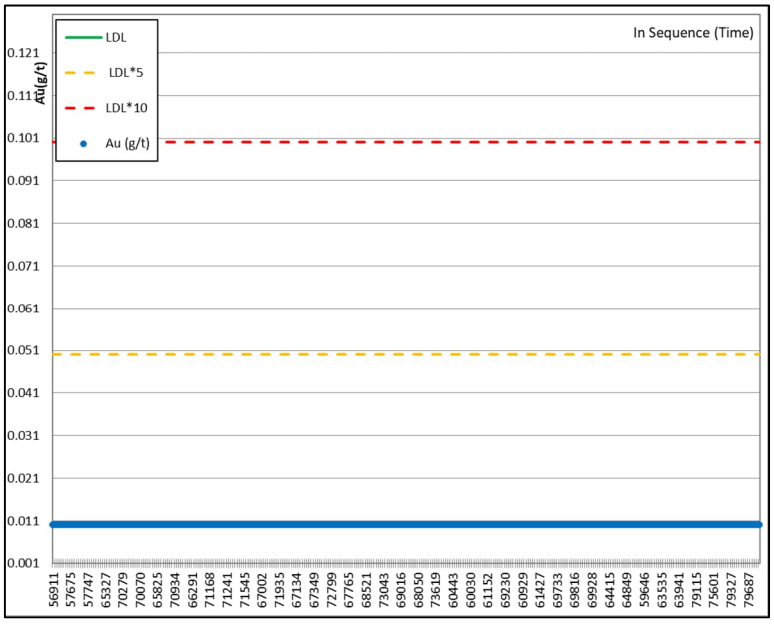


Figure 11-43: Internal Reference Material, Blank, Gold Performance

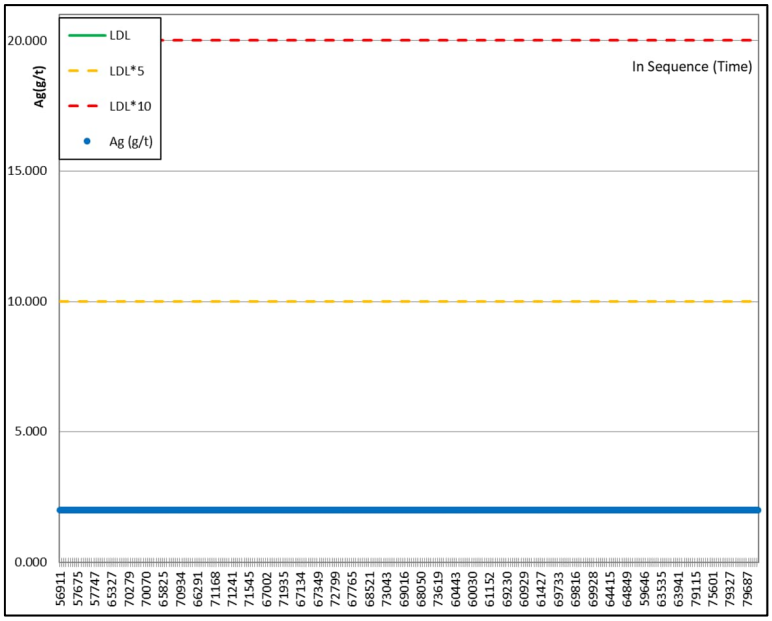


Figure 11-44: Internal Reference Material, Blank, Silver Performance

AbraSilver defines a reference material ("IRM or CRM") failure as a value that differs from the recommended value by more than 5%. Approximately three times the standard deviation.

For the analysis of CRM AuOx-18, no samples returned a value outside the error limit in gold or silver. The gold and silver performance for CRM AuOx-18 can be seen in Figure 11-45 and Figure 11-46.

For the analysis of CRM AuOx-33, no samples returned a value outside the error limit in gold or silver. The gold and silver performance for CRM AuOx-33 can be seen in Figure 11-47 and Figure 11-48.

For the analysis of CRM AuOx-52, no samples returned a value outside the error limit in gold or silver. The gold and silver performance for CRM AuOx-52 can be seen in Figure 11-49 and Figure 11-50.

For the analysis of CRM STRT-04, no samples returned a value outside the error limit in gold or silver. The gold and silver performance for CRM STRT-04 can be seen in Figure 11-51 and Figure 11-52.

For the analysis of CRM STRT-07, no samples returned a value outside the error limit in gold or silver. The gold and silver performance for CRM STRT-07 can be seen in Figure 11-53 and Figure 11-54.

For the analysis of CRM HDRT-02, no samples returned a value outside the error limit in gold or silver. The gold and silver performance for CRM HDRT-02 can be seen in Figure 11-55 and Figure 11-56.

For the analysis of CRM PLSUL-59, no samples returned a value outside the error limit in gold or silver. The gold and silver performance for CRM PLSUL-59 can be seen in Figure 11-57 and Figure 11-58.

For the analysis of CRM OREAS 151C, no samples returned a value outside the error limit in gold. The gold performance for CRM OREAS 151C can be seen in Figure 11-59.

For the analysis of CRM OREAS 507-B, no samples returned a value outside the error limit in gold. The gold performance for CRM OREAS 507-B can be seen in Figure 11-60.

For the analysis of CRM OREAS 504-D, no samples returned a value outside the error limit in gold or silver. The gold and silver performance for CRM OREAS 504-D can be seen in Figure 11-61 and Figure 11-62.

For the analysis of IRM ASDBL_H01, no samples returned a value outside this 5% error limit in gold and silver. The gold and silver performance for ASDBL_H01 can be seen in Figure 11-63 and Figure 11-64.

For the analysis of IRM ASDBL_M01, no samples returned a value outside this 5% error limit in gold and silver. The gold and silver performance for ASDBL_M01 can be seen in Figure 11-65 and Figure 11-66.

For the analysis of IRM ASDBL_L01, no samples returned a value outside this 5% error limit in gold and silver. The gold and silver performance for ASDBL_L01 can be seen in Figure 11-67 and Figure 11-68.

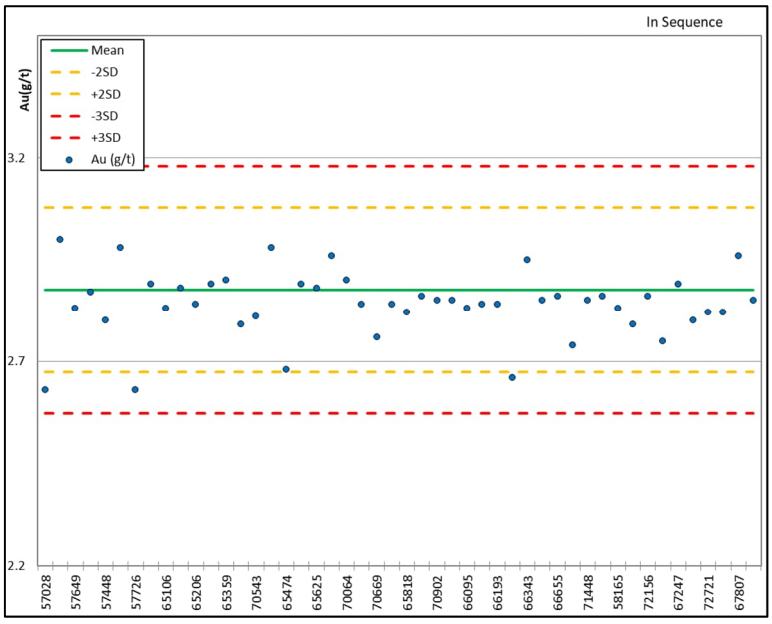


Figure 11-45: Certified Reference Material AuOx-18, Gold Performance

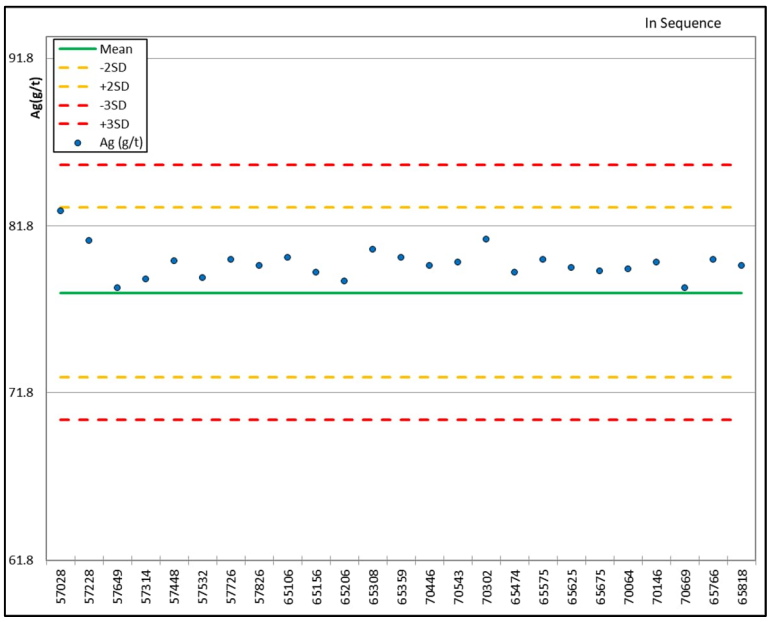


Figure 11-46: Certified Reference Material AuOx-18, Silver Performance.

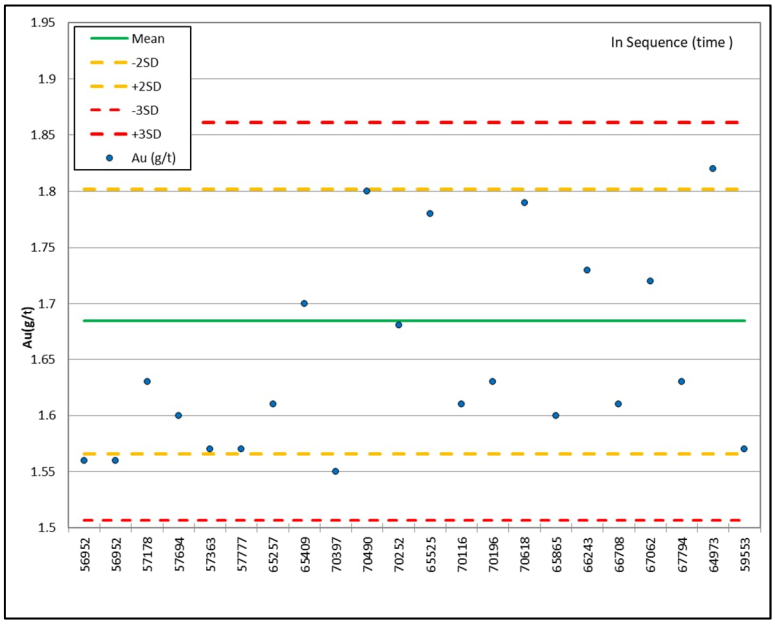


Figure 11-47: Certified Reference Material AuOx-33, Gold Performance

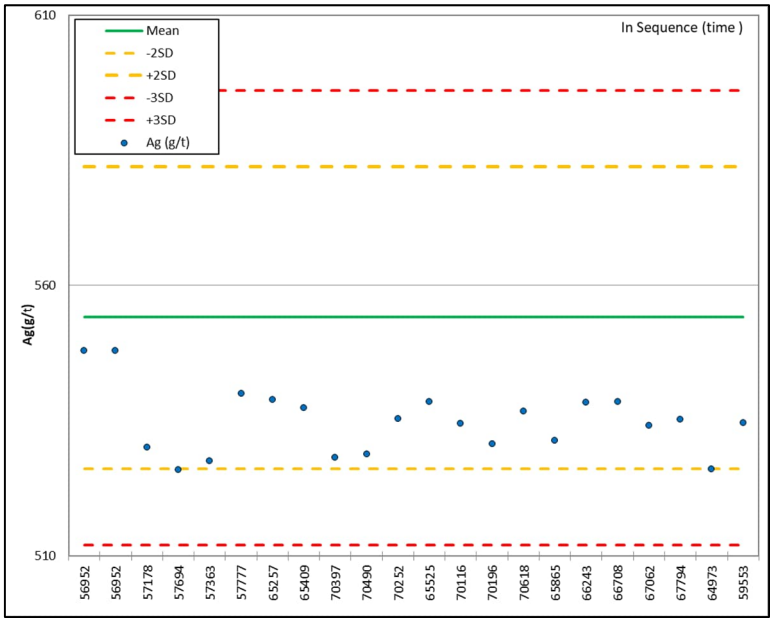


Figure 11-48: Certified Reference Material AuOx-33, Silver Performance

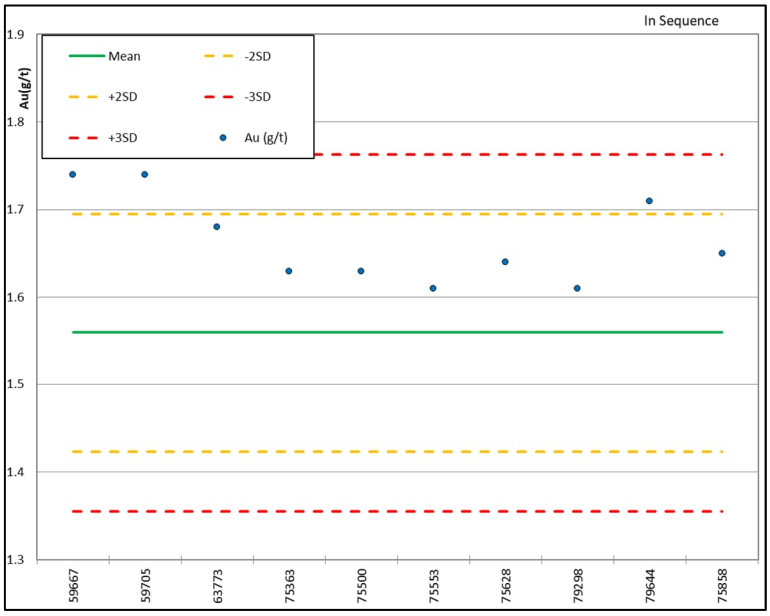


Figure 11-49: Certified Reference Material AuOx-52, Gold Performance



Figure 11-50: Certified Reference Material AuOx-52, Silver Performance

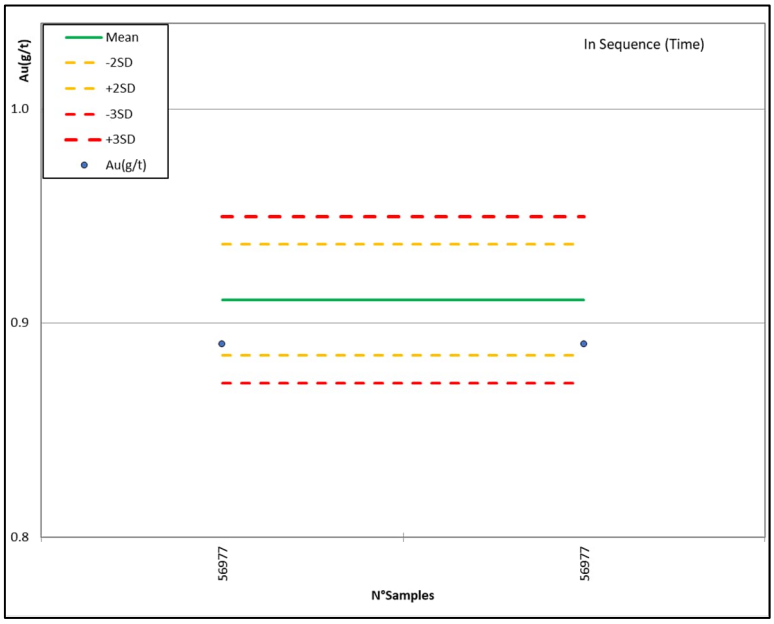


Figure 11-51: Certified Reference Material STRT-04, Gold Performance

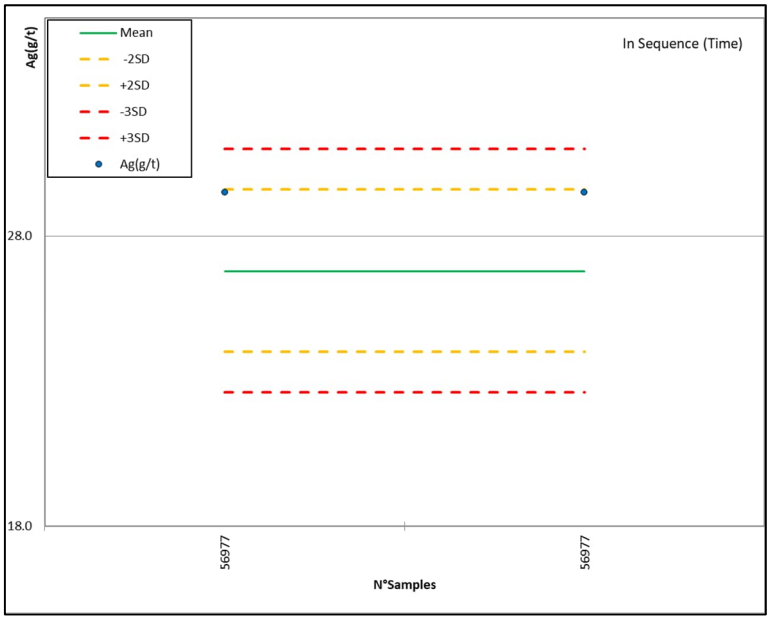


Figure 11-52: Certified Reference Material STRT-04, Silver Performance

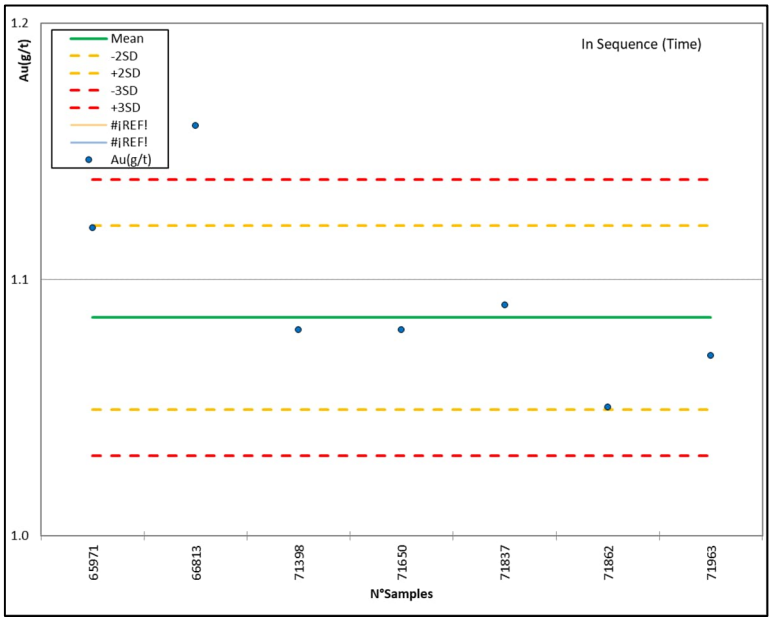


Figure 11-53: Certified Reference Material STRT-07, Gold Performance

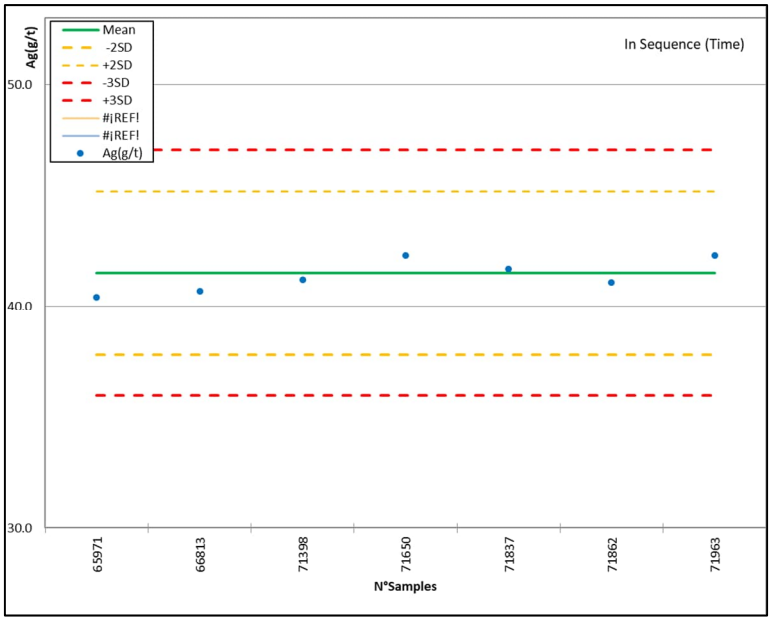


Figure 11-54: Certified Reference Material STRT-07, Silver Performance



Figure 11-55: Certified Reference Material HDRT-02, Gold Performance

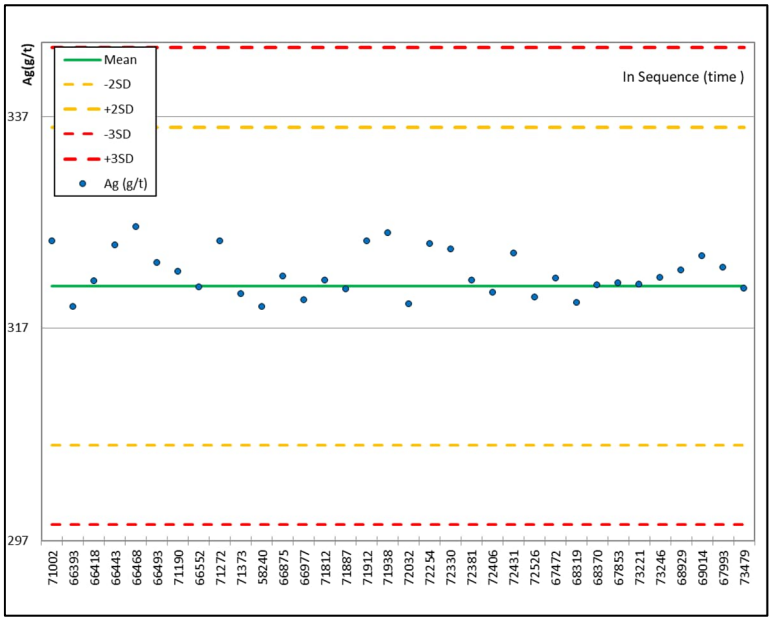


Figure 11-56: Certified Reference Material HDRT-02, Silver Performance

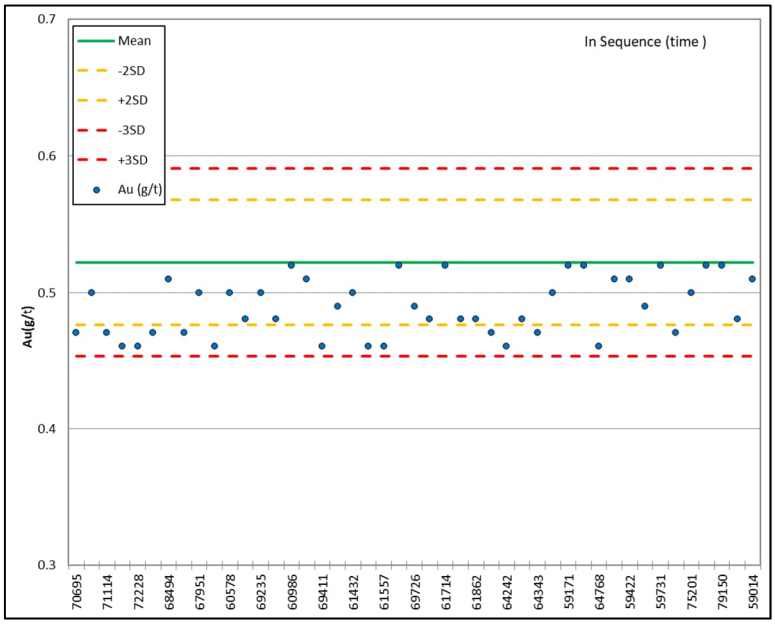


Figure 11-57: Certified Reference Material PLSUL59, Gold Performance

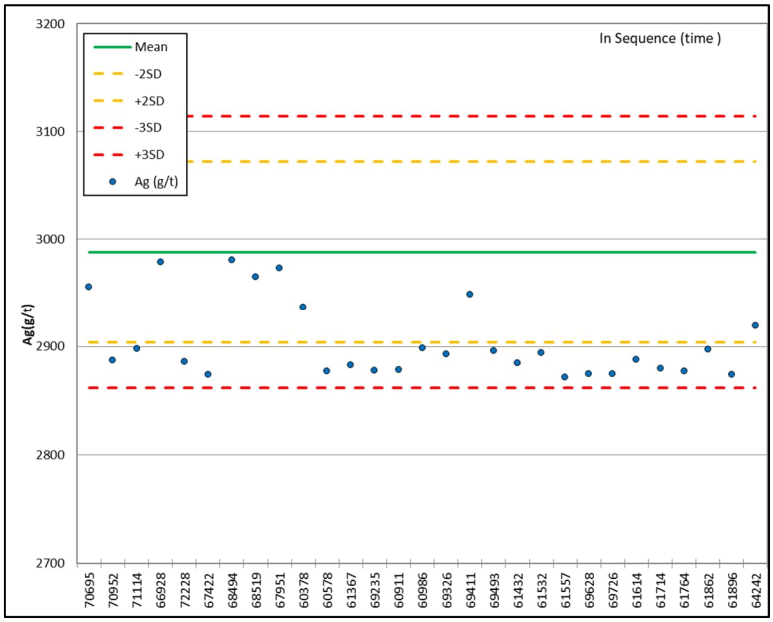


Figure 11-58: Certified Reference Material PLSUL59, Silver Performance

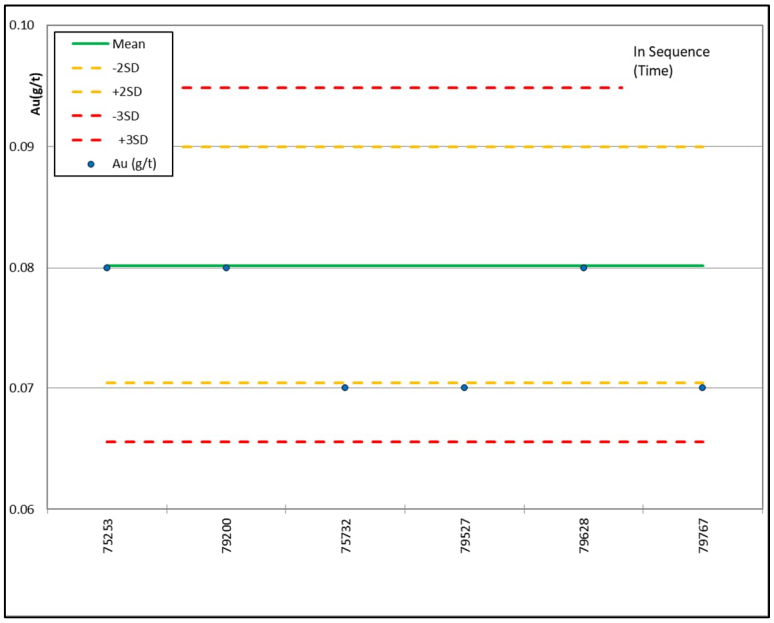


Figure 11-59: Certified Reference Material OREAS 151C, Gold Performance

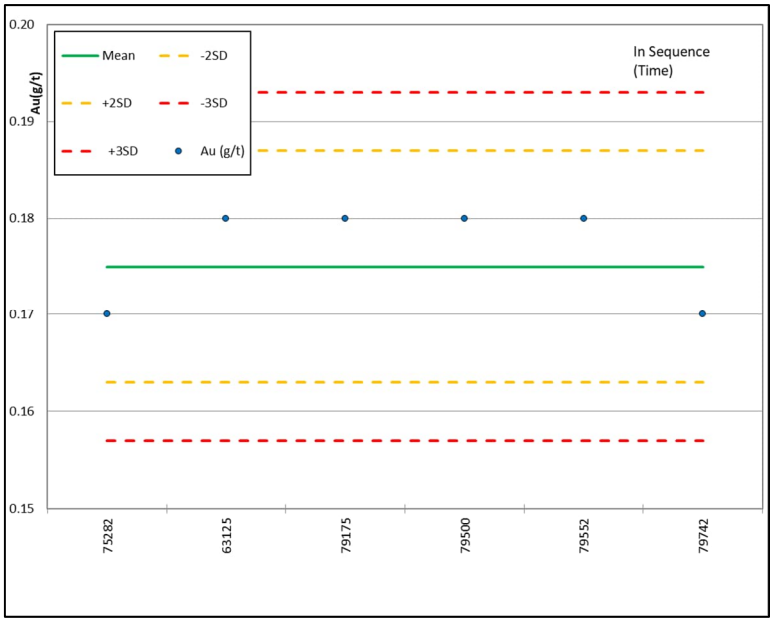


Figure 11-60: Certified Reference Material OREAS 507-B, Gold Performance

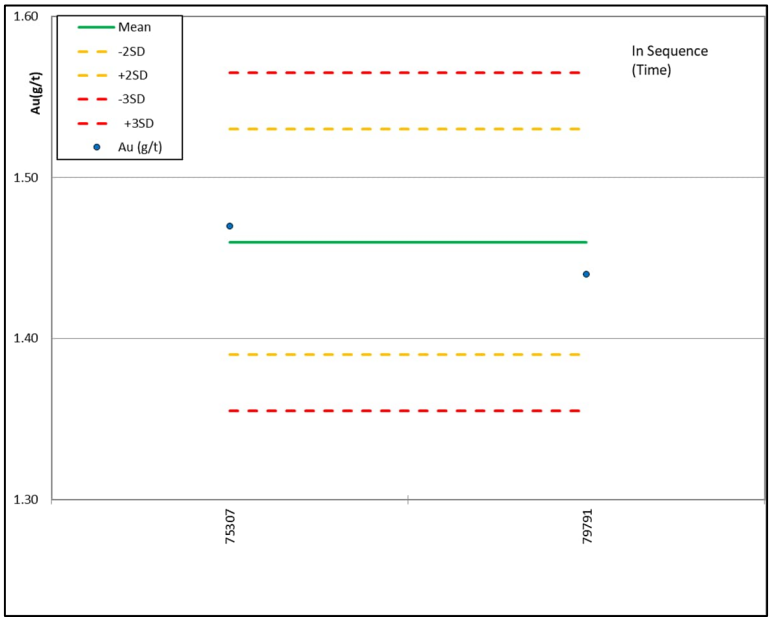


Figure 11-61: Certified Reference Material OREAS 504-D, Gold Performance

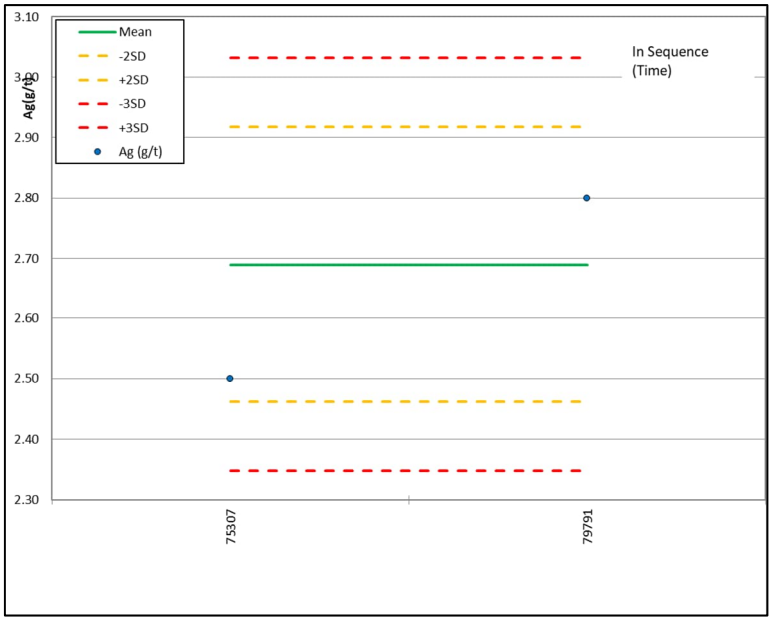


Figure 11-62: Certified Reference Material OREAS 504-D, Silver Performance

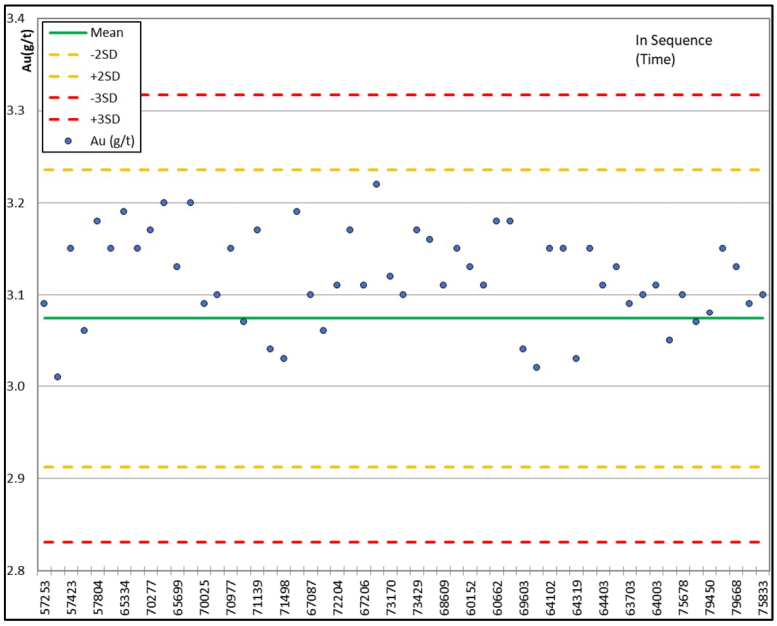


Figure 11-63: Internal Reference Material ASDBL_H01, Gold Performance

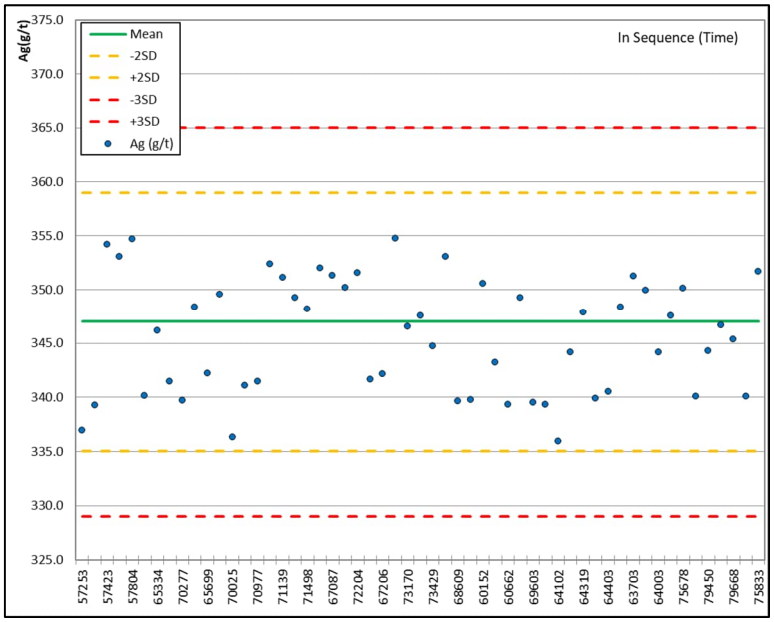


Figure 11-64: Internal Reference Material ASDBL_H01, Silver Performance

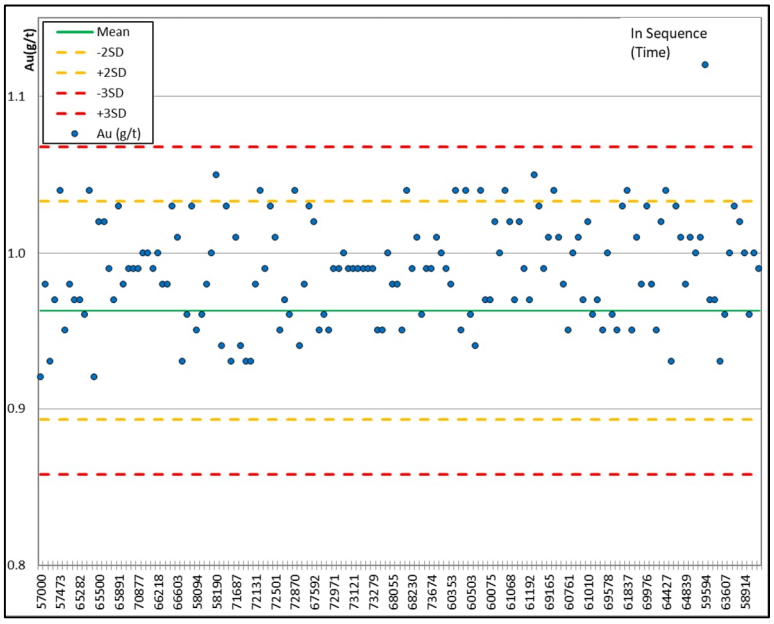


Figure 11-65: Internal Reference Material ASDBL_M01, Gold Performance

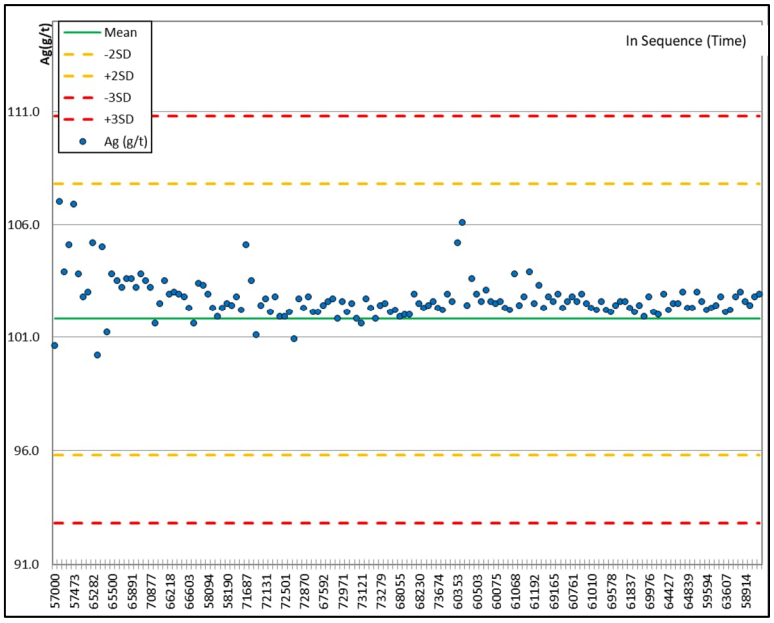


Figure 11-66: Internal Reference Material ASDBL_M01, Silver Performance

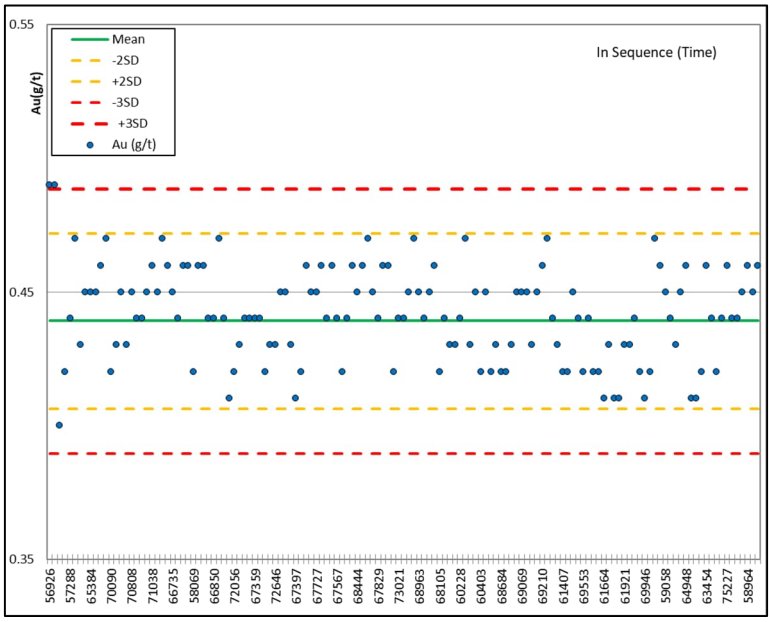


Figure 11-67: Internal Reference Material ASDBL_L01, Gold Performance

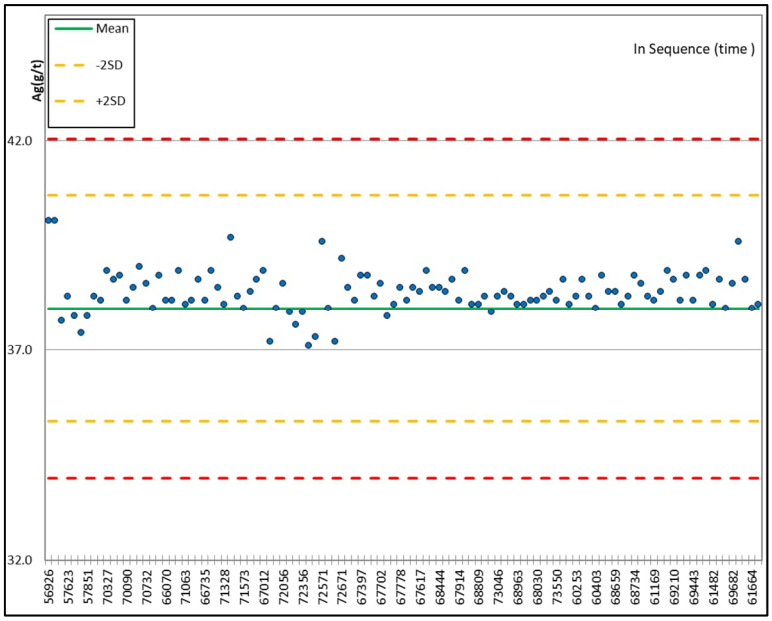


Figure 11-68: Internal Reference Material ASDBL_L01, Silver Performance

For sample batches that included reference material failures, the laboratory was requested to re-assay samples the entire batch. The corrected assay data was then loaded to the database.

Core duplicates were obtained by dividing half the core into two separate 1/4 core equivalent samples, bagged and labelled separately. Charts in Figure 11-69 and Figure 11-70 show the

relevant RMA scattergrams and the Min-Max versus Hyperbolic Method scattergrams (Figure 11-71 and Figure 11-72). Indicating low to moderate variability.

In addition, *Relative Differences scattergrams* were built. Relative difference scattergrams represent the cumulative percentage of samples, ordered by relative difference values versus the magnitude of the absolute relative difference. (Figure 11-73 and Figure 11-74).

In both cases, for gold and silver, was shown that approximately 90% of the data display negligible discrepancies while the upper 5–10% of the data exhibit significant deviations. These outliers may be associated with data sparsity, geological variability, or anomalous samples.

Also, *Analysis of Relative Difference versus Mean* for gold and silver were built to illustrate the relationship between differences (%) and the mean for each metal. The analysis indicates that relative differences are primarily driven by low-grade samples, where even small absolute discrepancies produce large relative variations. In contrast, medium to high-grade samples show tighter clustering around zero. (Figure 11-75 and Figure 11-76).

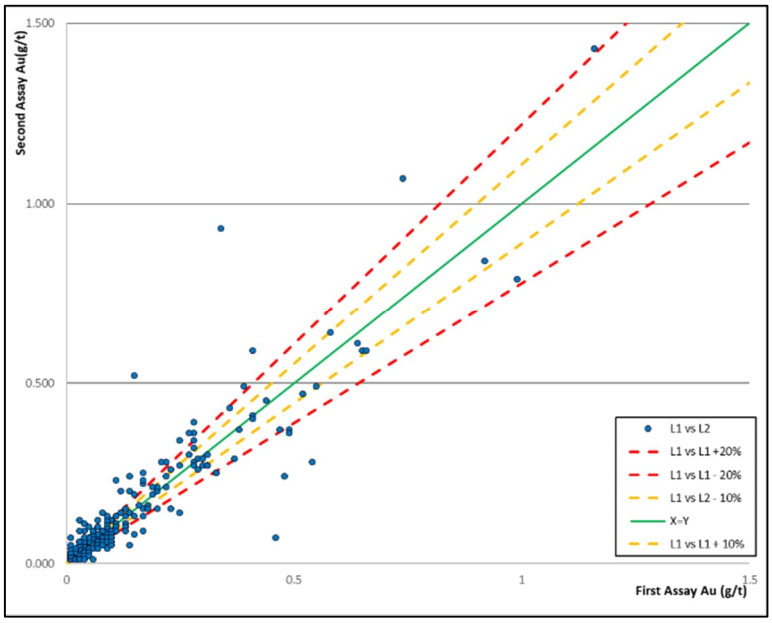


Figure 11-69: RMA Scattergram for Duplicate Performance of Gold

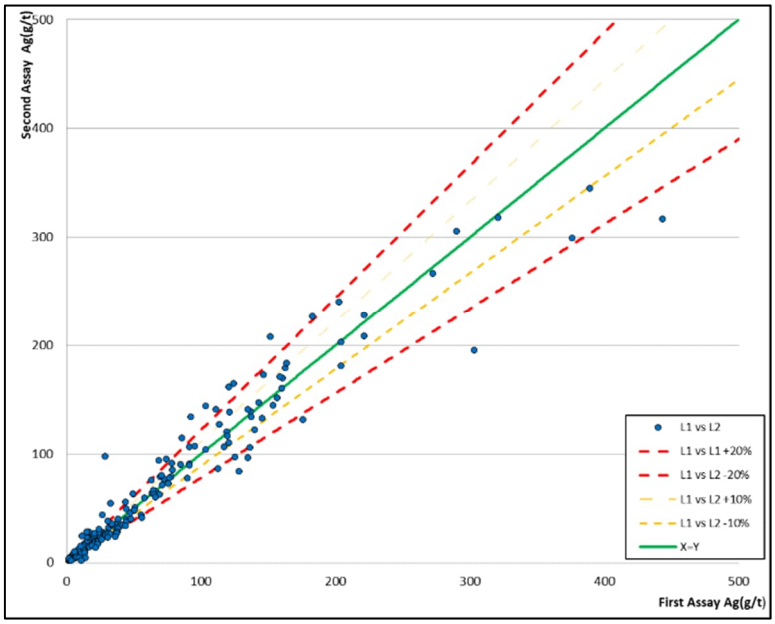


Figure 11-70: RMA Scattergram for Duplicate Performance of Silver

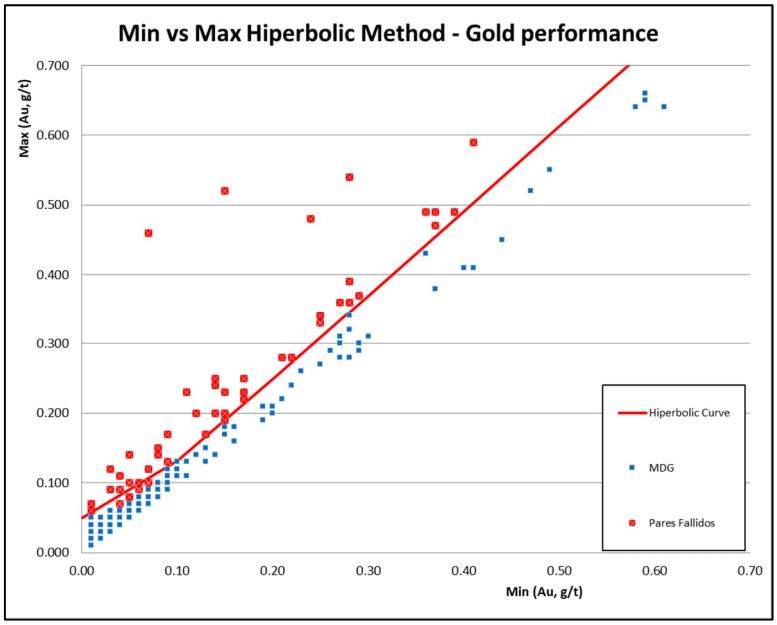


Figure 11-71: Min-Max versus Hyperbolic Method Scattergram for Duplicate Performance of Gold

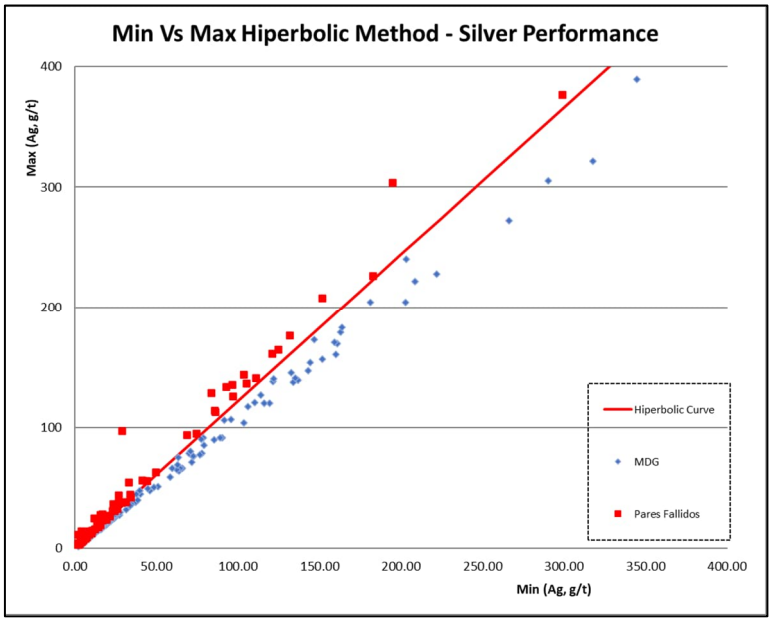


Figure 11-72: Min-Max versus Hyperbolic Method Scattergram for Duplicate Performance of Silver

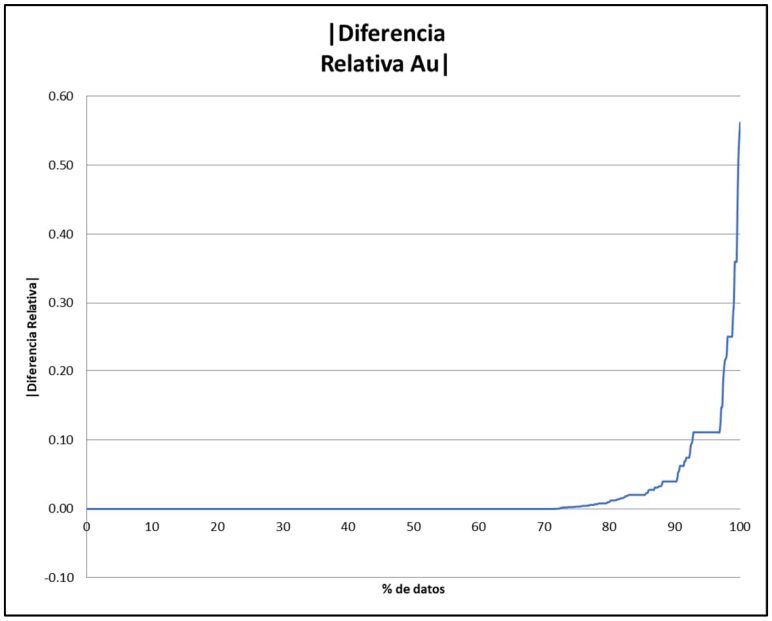


Figure 11-73: RD Scattergram for Duplicate Performance of Gold

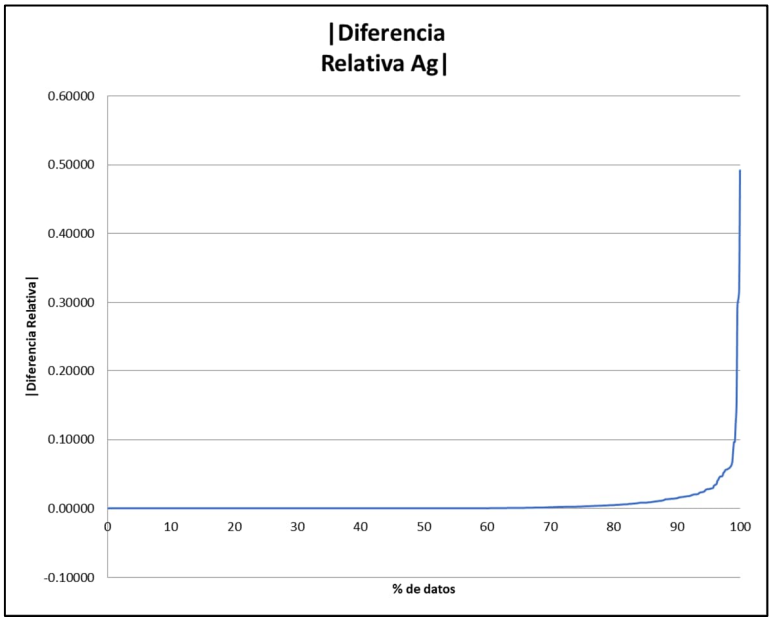


Figure 11-74: RD Scattergram for Duplicate Performance of Silver

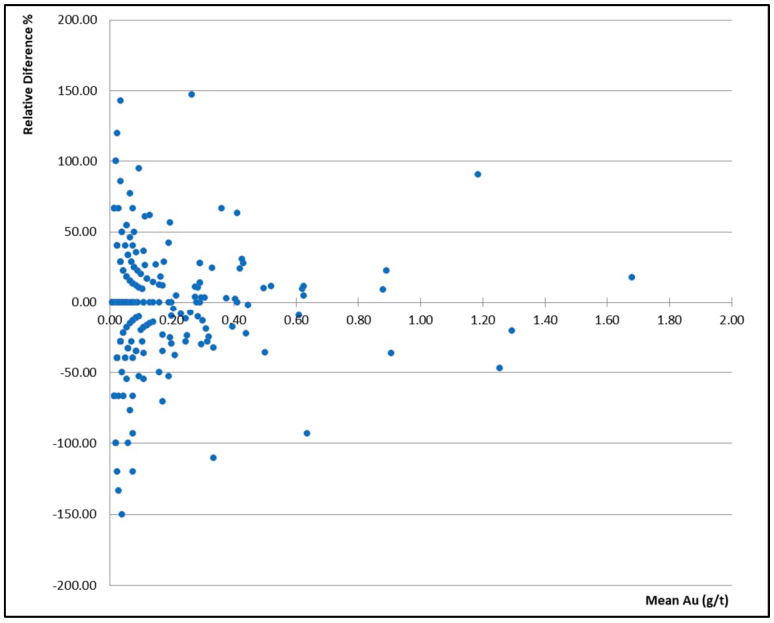


Figure 11-75: Analysis of Relative Difference versus Mean for Gold

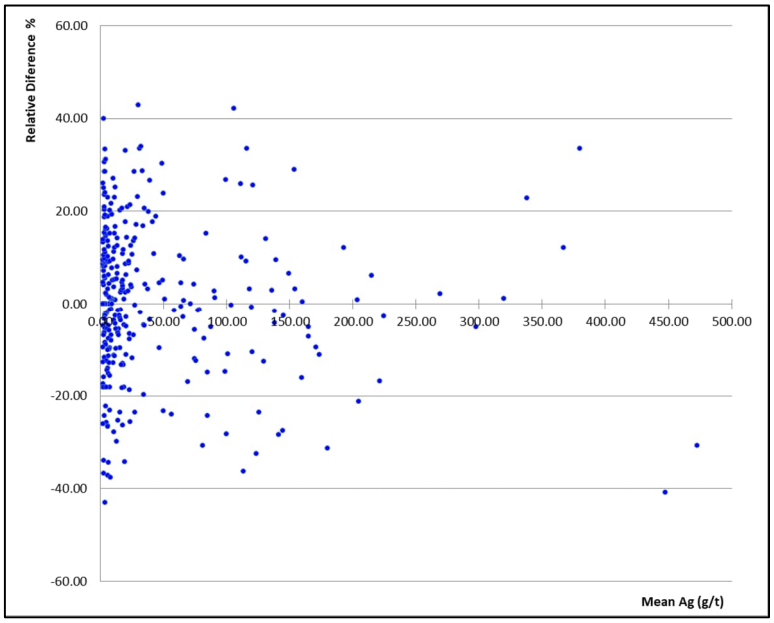


Figure 11-76: Analysis of Relative Difference versus Mean for Silver

11.7.4.1 Second Laboratory check

A second laboratory re-check was performed. Results from these samples corresponded with the general range of grades that had been reported during previous exploration.

Samples were selected in three intervals, based on the total population of samples from Phase IV Drilling Campaign. Core samples, reject samples and pulp samples were collected and sent to a secondary laboratory.

The intention of this re-sampling was to verify accuracy and precision of the principal laboratory used by AbraSilver and the sampling methodology of AbraSilver exploration staff.

All samples were sent to SGS Lab (“SGS”), with a preparation laboratory in San Juan and analytic laboratory in Lima, Peru, as a secondary laboratory. It is important to mention that Alex Stewart Assayers (“ASA”), with a preparation laboratory in Palpala, San Salvador de Jujuy and analytic laboratory in Mendoza, Argentina.

Once samples were assayed by SGS, three separated population were segregated cores, rejects and pulps. For each population, a set of statistical analysis was performed to validate the population and detect bias. RMA scatter plots were constructed for the studied elements. The RMA method offers an unbiased fit for two sets of pair values (original samples and check samples) that are considered independent from each other.

Figure 11-77: and Figure 11-78 shows RMA Scattergram for core duplicate performance of gold and silver. Figure 11-79 and Figure 11-80 shows RMA Scattergram for rejects duplicate

performance of gold and silver. Figure 11-81 and Figure 11-82 shows RMA Scattergram for pulps duplicate performance of gold and silver.

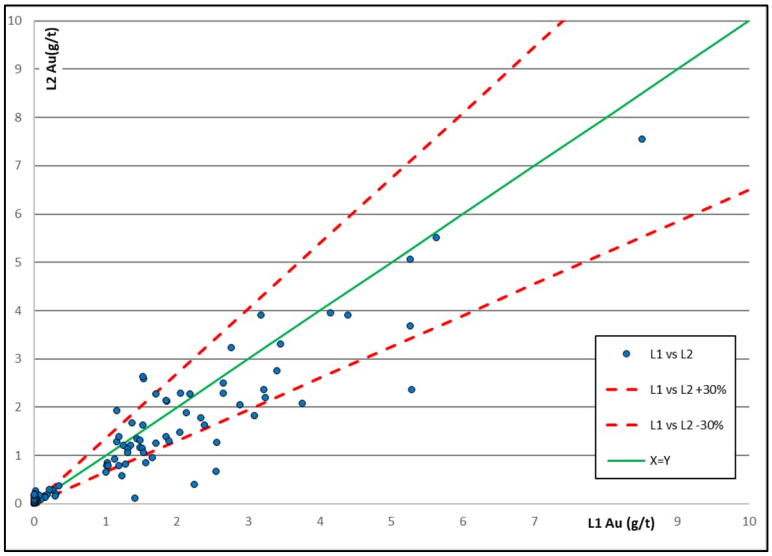


Figure 11-77: RMA Scattergram for Duplicate Performance of Gold (Cores)

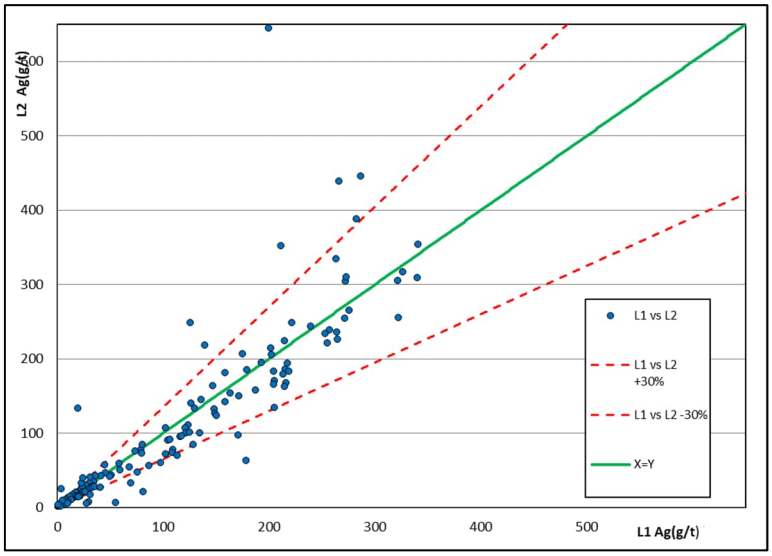


Figure 11-78: RMA Scattergram for Duplicate Performance of Silver (Cores)

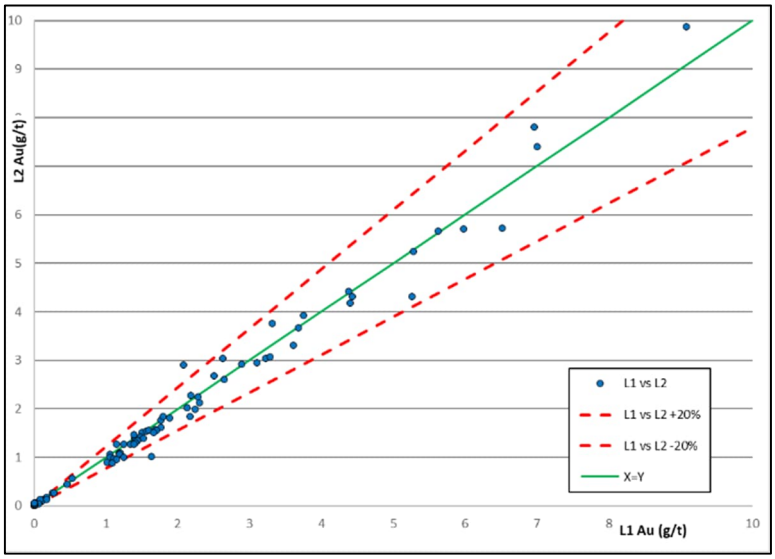


Figure 11-79: RMA Scattergram for Duplicate Performance of Gold (Rejects)

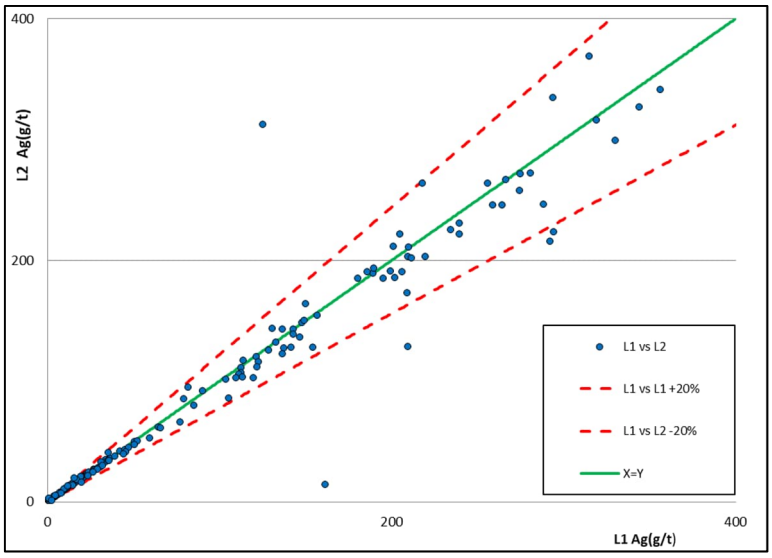


Figure 11-80: RMA Scattergram for Duplicate Performance of Silver (Rejects)

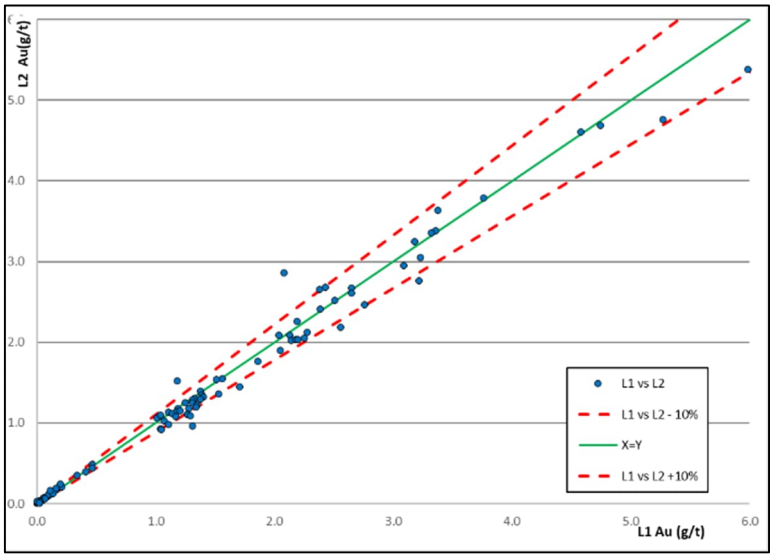


Figure 11-81: RMA Scattergram for Duplicate Performance of Gold (Pulps)

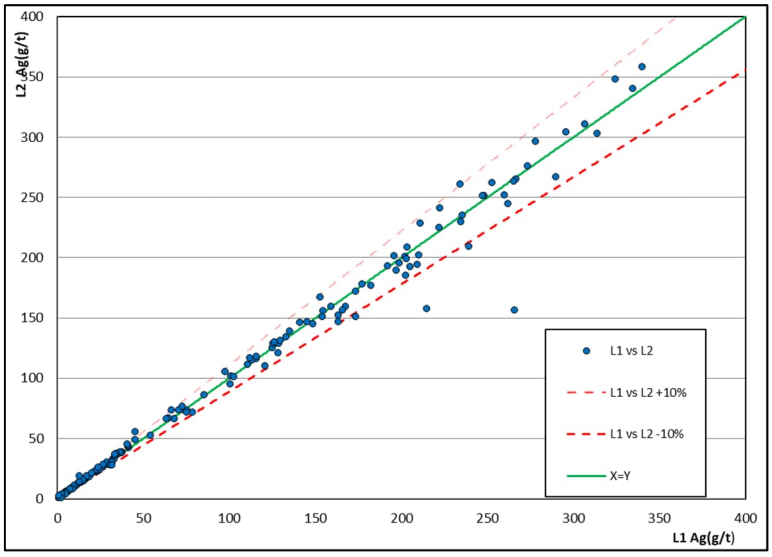


Figure 11-82: RMA Scattergram for Duplicate Performance of Silver (Pulps)

Relative (“RD”) versus Mobil Average (“MA”) plots were built also. In both cases, for gold and silver, was shown that approximately 90% of the data display negligible discrepancies while the upper 5–10% of the data exhibit significant deviations. These outliers may be associated with data sparsity, geological variability between two pieces of quarter core due to the nature of gold mineralization, or anomalous samples.

Figure 11-83 and Figure 11-84 shows Relative Differences Scattergram for core duplicate performance of gold and silver. Figure 11-85 and Figure 11-86 shows Relative Differences Scattergram for rejects duplicate performance of gold and silver. Figure 11-87

and Figure 11-88 shows Relative Differences Scattergram for pulps duplicate performance of gold and silver.

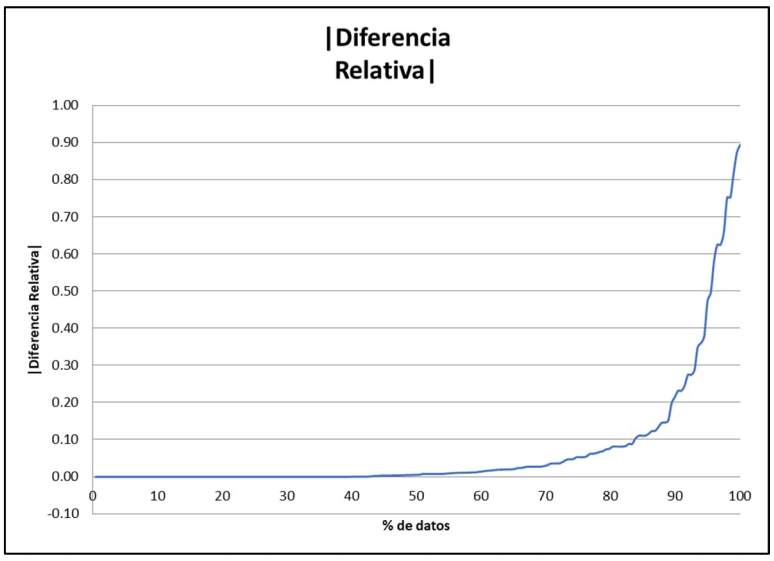


Figure 11-83: RD Scattergram for Duplicate Performance of Gold (Cores)

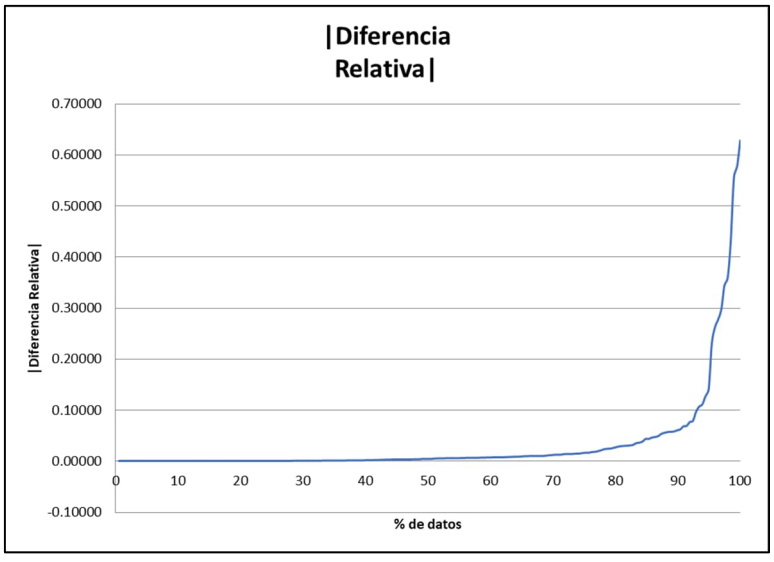


Figure 11-84: RD Scattergram for Duplicate Performance of Silver (Cores)

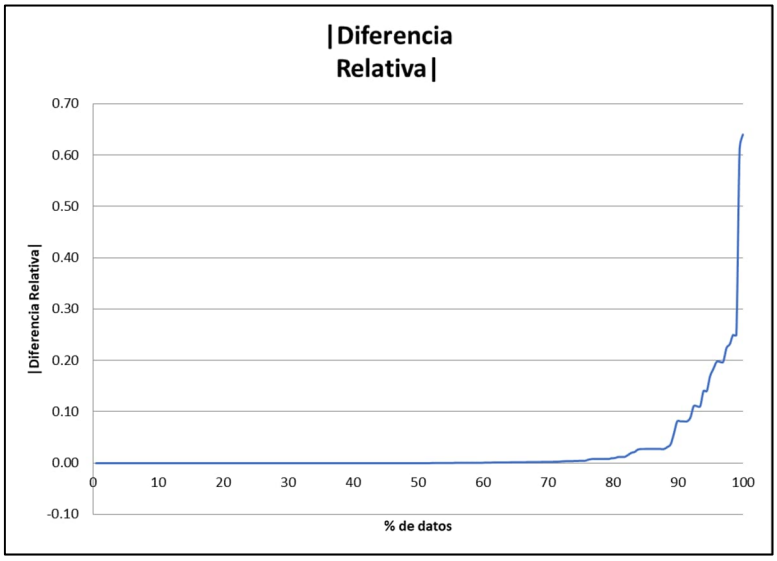


Figure 11-85: RD Scattergram for Duplicate Performance of Gold (Rejects)

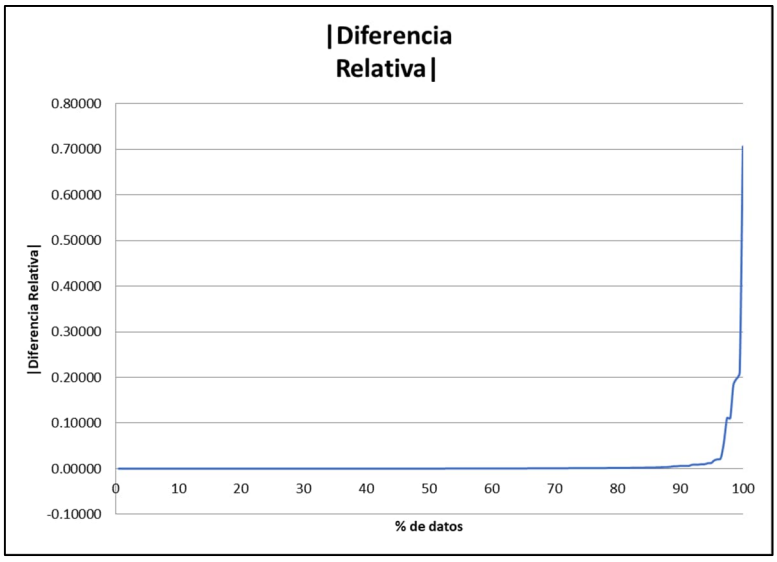


Figure 11-86: RD Scattergram for Duplicate Performance of Silver (Rejects)

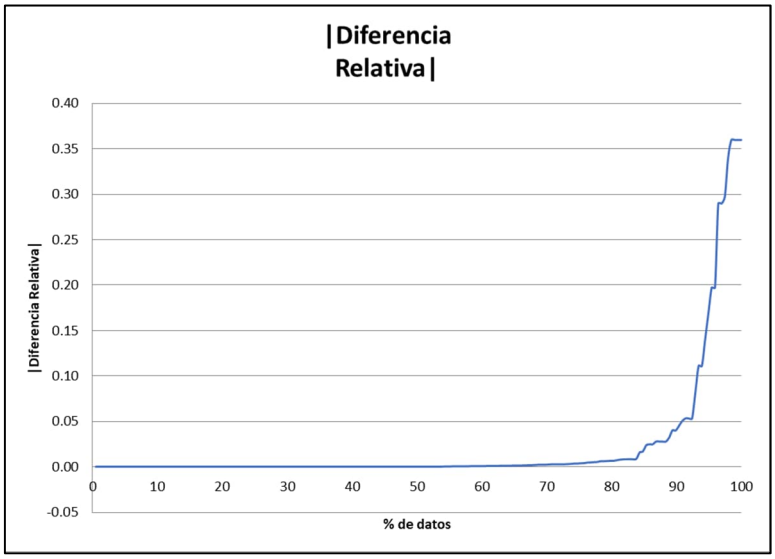


Figure 11-87: RD Scattergram for Duplicate Performance of Gold (Pulps)

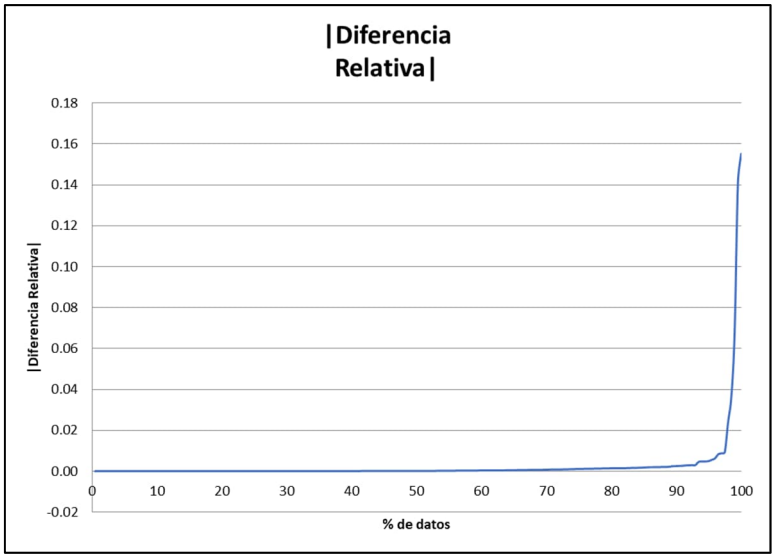


Figure 11-88: RD Scattergram for Duplicate Performance of Silver (Pulps)

11.7.5 Period 2025 (Phase V Drilling Campaign)

A comprehensive Quality Assurance and Quality Control (QA/QC) program was implemented throughout the 2025 Phase V drilling campaign to monitor the accuracy, precision and potential contamination associated with sampling and analytical procedures. The program was designed and supervised by AbraSilver's geological team and included the systematic insertion of certified reference materials (CRMs), coarse blank samples and quarter-core duplicates into the sample stream submitted to the primary analytical laboratory.

The QA/QC program evaluated a total of 9,597 drill core samples derived from 50 drill holes included in the Mineral Resource database. Quality control samples comprised 360 certified reference materials (approximately 3.8% of submitted samples), 612 blank samples (approximately 6.4%) and 453 quarter-core duplicates (approximately 4.7%), corresponding to insertion frequencies of approximately one CRM every 30 samples, one blank every 15 samples and one duplicate every 20 samples. This level of quality control exceeds industry-standard practices for advanced-stage precious metal exploration programs.

Multiple certified reference materials representing low-, medium- and high-grade gold and silver mineralization, as well as oxide- and sulphide-hosted matrices, were monitored during the campaign. The performance of these standards demonstrated that assay results remained consistently within acceptable control limits, with only minor deviations from certified values and no evidence of persistent analytical drift or systematic laboratory bias. Time-series control charts for the various CRMs indicate stable analytical performance throughout the duration of the campaign. See Table 11-11 and Figure 11-89 to Figure 11-120.

Table 11-11: Certified Reference Materials Assay Specifications

CRM	Matrix	Inserts (n)	Certified Au (g/t)	Mean Au (g/t)	Au Bias (%)	Certified Ag (g/t)	Mean Ag (g/t)	Ag Bias (%)	Assessment
ASDBH-01	Internal High Grade	47	3.074	3.08	0.3	347	345.26	-0.5	Pass
ASDBM-01	Internal Medium Grade	41	0.913	0.922	1	101.8	102.44	0.6	Pass
ASDBL-01	Internal Low Grade	18	0.439	0.44	1.1	38	38.54	1.4	Pass
PLSUL59	Sulphide	16	0.522	0.501	-4	2988	2931.53	-1.9	Pass
AuOx-18	Oxide	74	2.876	2.876	-1.4	77.8	78.23	0.6	Pass
AuOx-33	Oxide	20	1.684	1.684	-0.6	554	538.06	-2.9	Pass
AuOx-41	Oxide	12	0.569	0.56	-1.6	8.3	9.51	14.6	Ag affected by digestion
AuOx-52	Oxide	28	1.559	1.638	5.1	17.3	18.32	5.9	Method-related bias
OREAS 504-D	Oxide	11	1.46	1.454	-0.8	2.69	2.65	-1.7	Pass
OREAS 151-C	Oxide	28	0.0802	0.074	-8.3	N/A	N/A	N/A	Au monitoring only
OREAS 507-B	Oxide	20	0.175	0.174	-0.6	N/A	N/A	N/A	Au monitoring only

Blank samples inserted to monitor potential contamination returned results consistent with expected background concentrations, confirming the absence of significant sample carry-over or contamination during sample preparation and analytical procedures (Figure 11-115 and Figure 11-116).

Quarter-core duplicate analyses demonstrated acceptable levels of precision and reproducibility considering the inherent geological variability of epithermal precious metal mineralization (Figures Figure 11-117 to Figure 11-120).

Certain certified reference materials originally developed for three-acid digestion methods exhibited a positive analytical bias when analyzed using the four-acid digestion procedure adopted by AbraSilver for silver determinations. This behavior was anticipated based on the certification methodology of those materials and does not reflect analytical issues with the laboratory. These standards were subsequently discontinued or interpreted accordingly and did not affect the overall assessment of laboratory performance.

Based on the review of certified reference materials, blank samples and duplicate data, Mr. Peralta considers that the QA/QC program implemented during the 2025 Phase V drilling campaign demonstrates acceptable levels of accuracy, precision and contamination control. The assay results generated by the primary laboratory are therefore considered reliable and suitable for a Mineral Resource Estimate and all technical evaluations presented in this Technical Report.

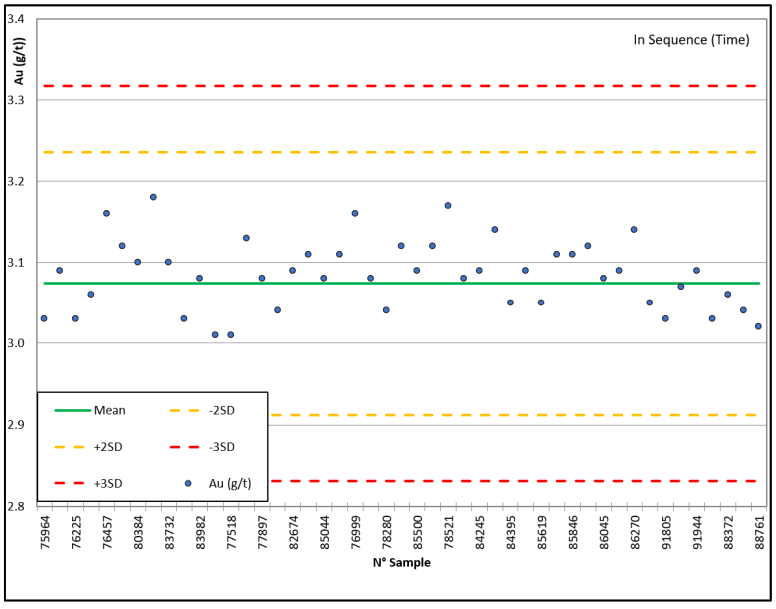


Figure 11-89: Internal Reference Material ASDBL_H01, Gold Performance

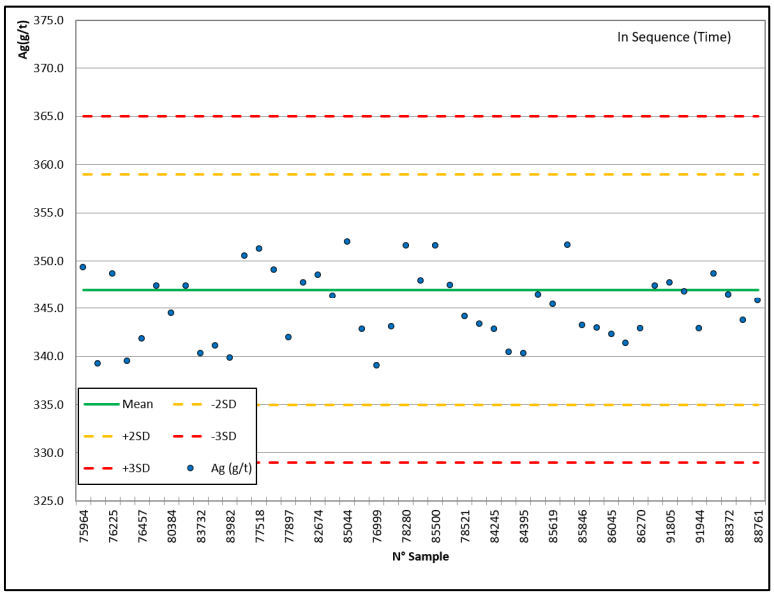


Figure 11-90: Internal Reference Material ASDBL_H01, Silver Performance

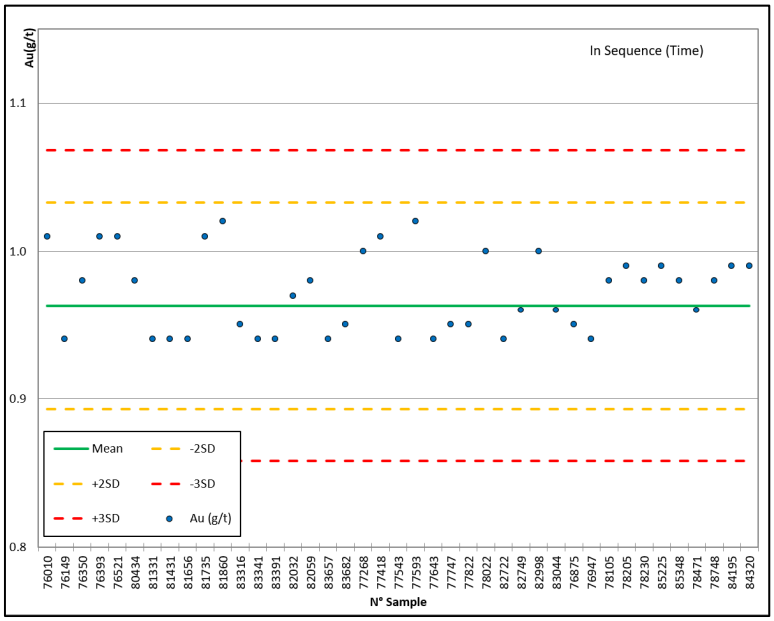


Figure 11-91: Internal Reference Material ASDBL_M01, Gold Performance

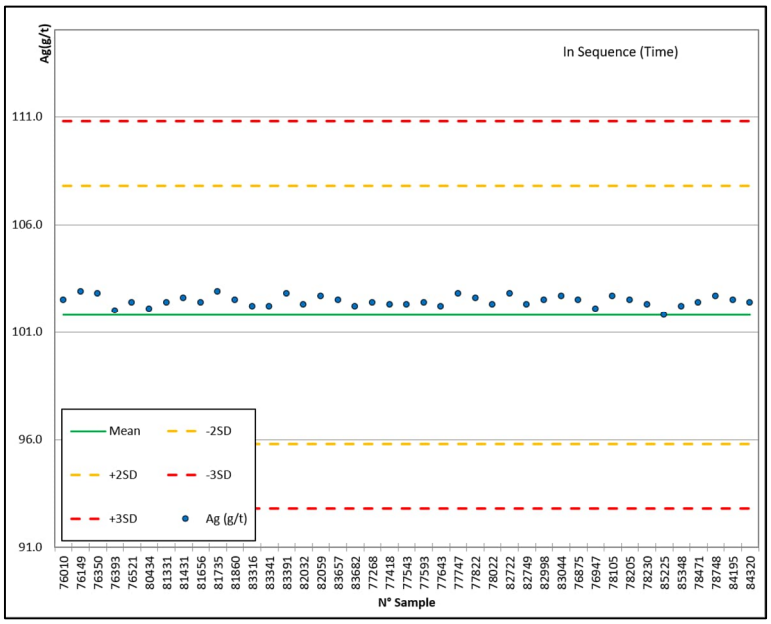


Figure 11-92: Internal Reference Material ASDBL_M01, Silver Performance

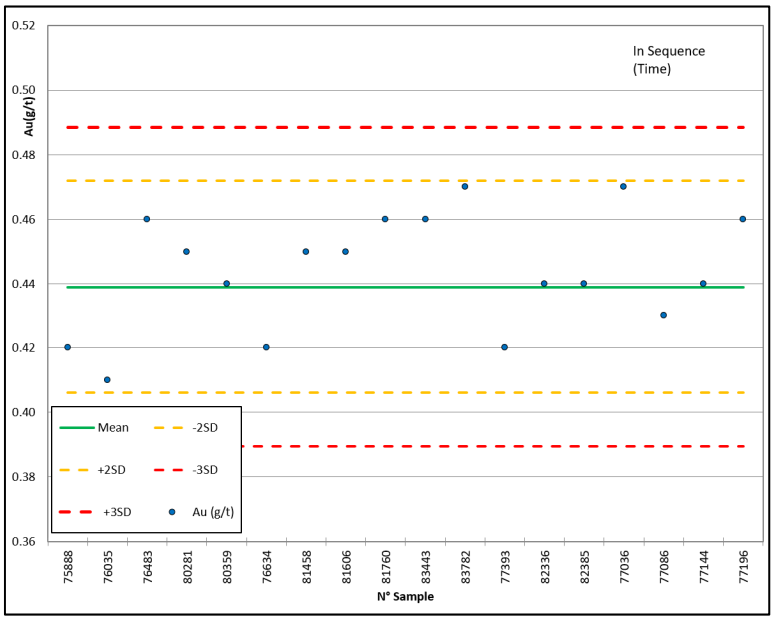


Figure 11-93: Internal Reference Material ASDBL_L01, Gold Performance

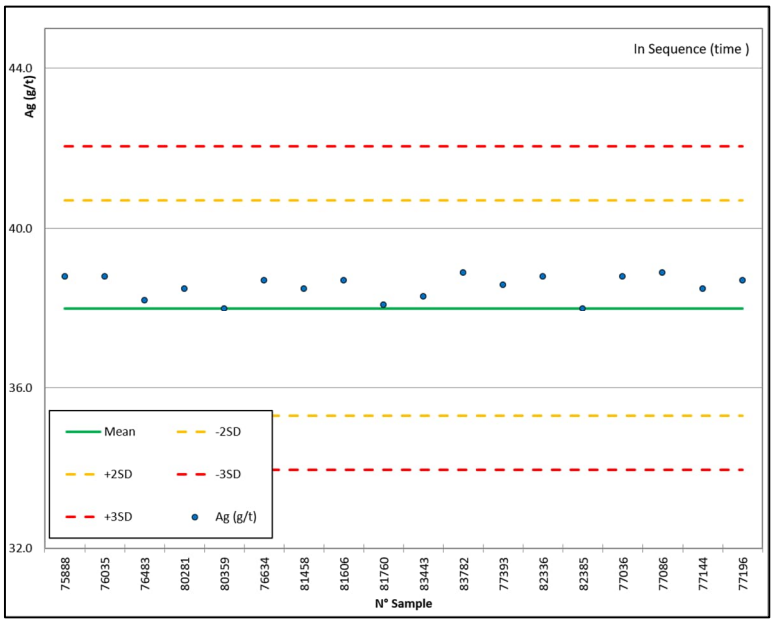


Figure 11-94: Internal Reference Material ASDBL_L01, Silver Performance

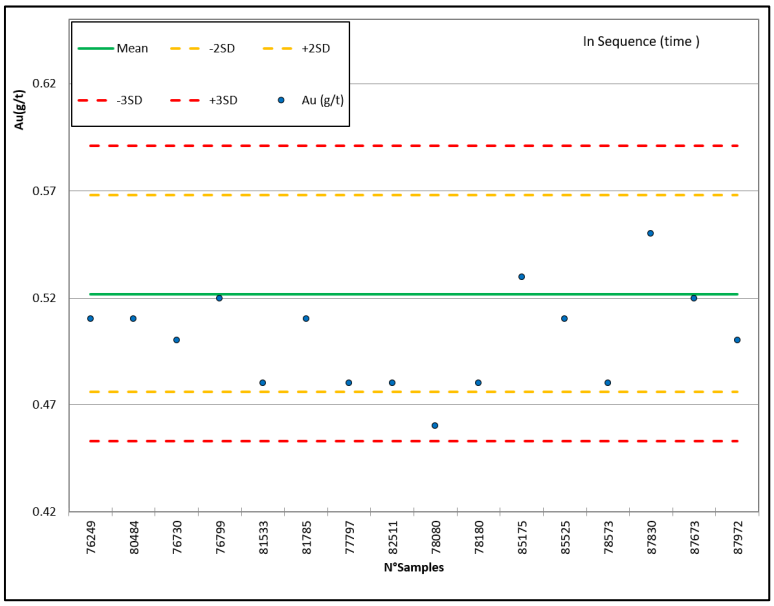


Figure 11-95: Certified Reference Material PLSUL59, Gold Performance

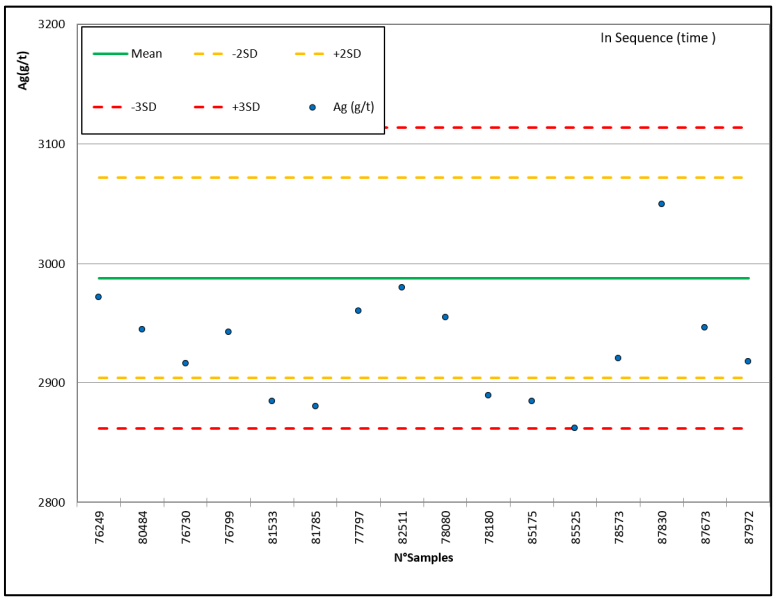


Figure 11-96: Certified Reference Material PLSUL59, Silver Performance

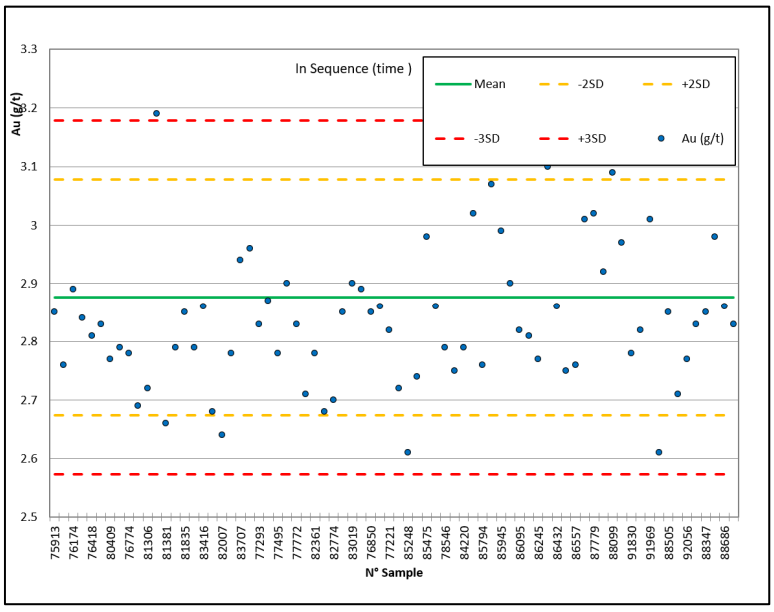


Figure 11-97: Certified Reference Material AuOx-18, Gold Performance

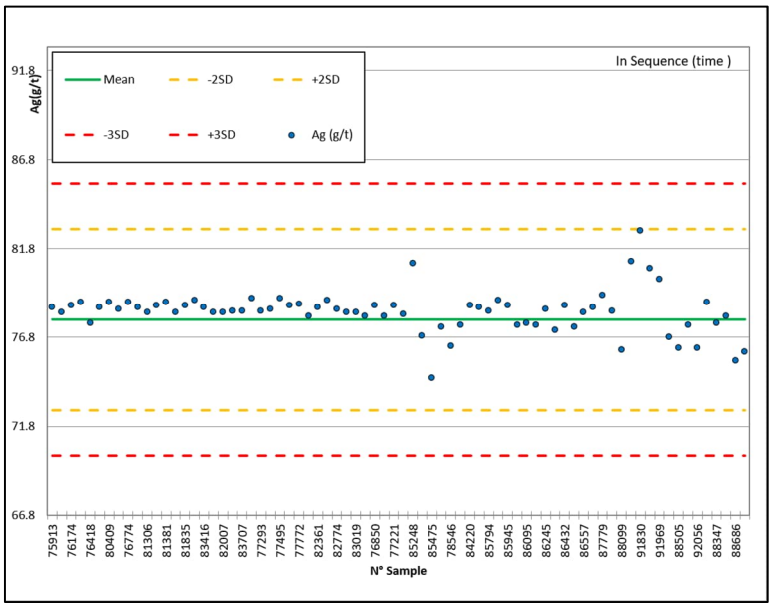


Figure 11-98: Certified Reference Material AuOx-18, Silver Performance

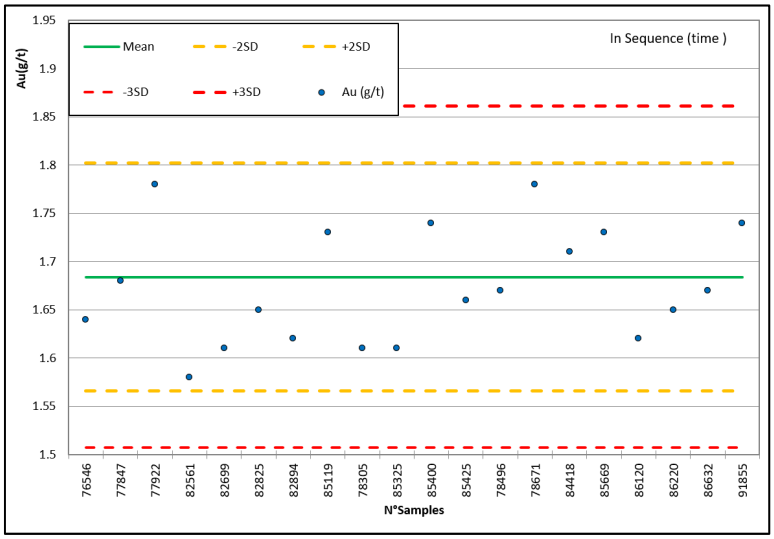


Figure 11-99: Certified Reference Material AuOx-33, Gold Performance

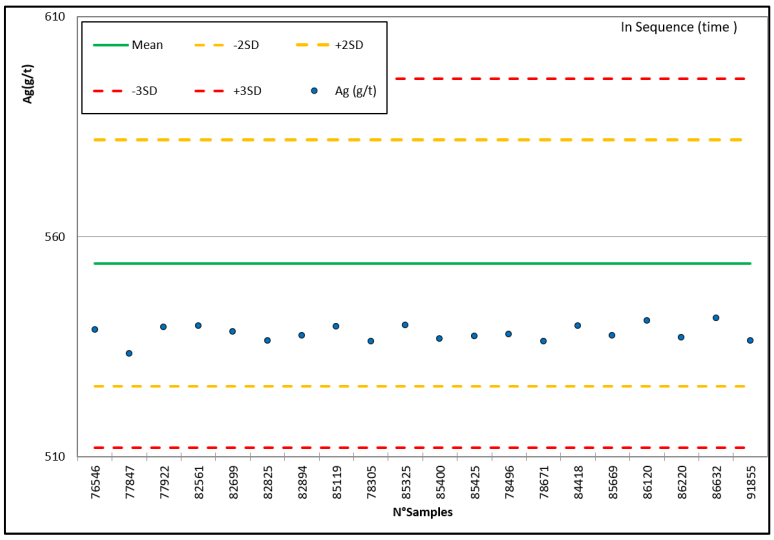


Figure 11-100: Certified Reference Material AuOx-33, Silver Performance

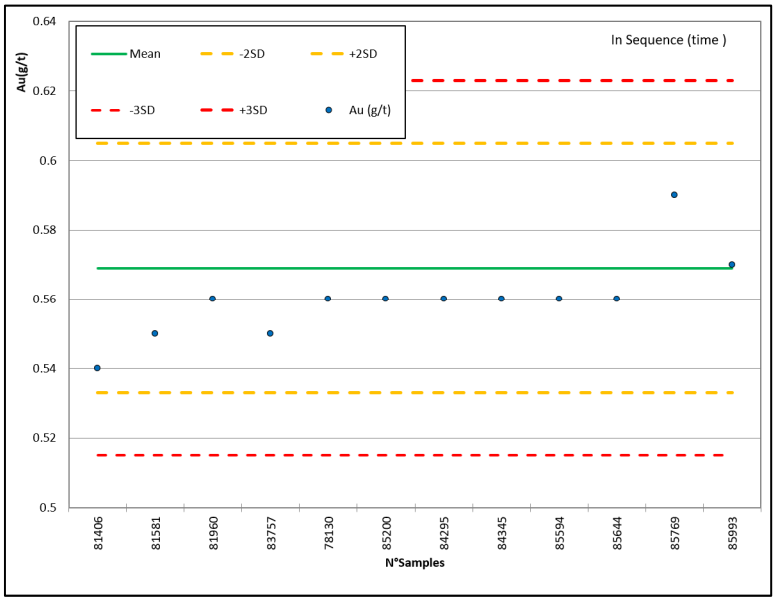


Figure 11-101: Certified Reference Material AuOx-41, Gold Performance

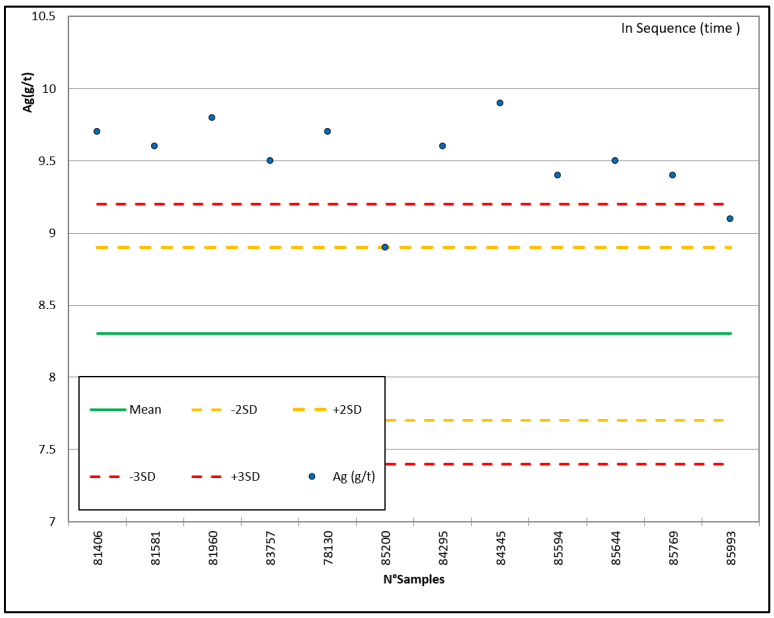


Figure 11-102: Certified Reference Material AuOx-41, Gold Performance

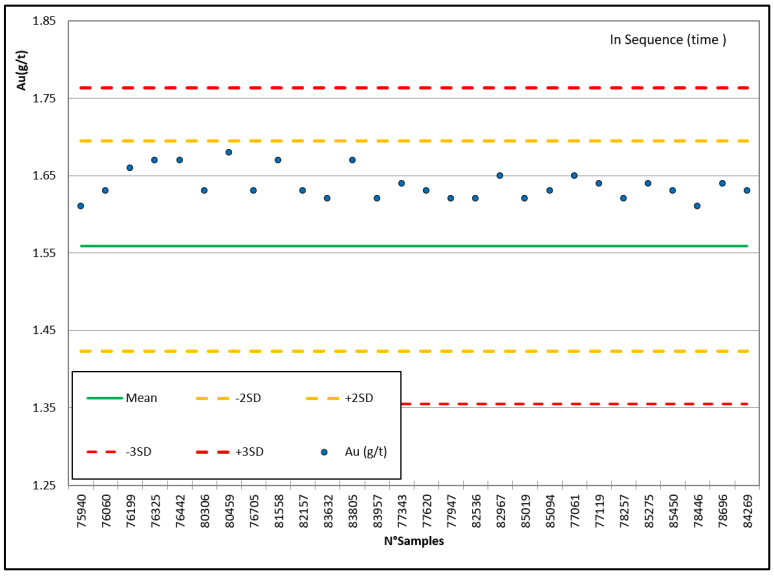


Figure 11-103: Certified Reference Material AuOx-52, Gold Performance

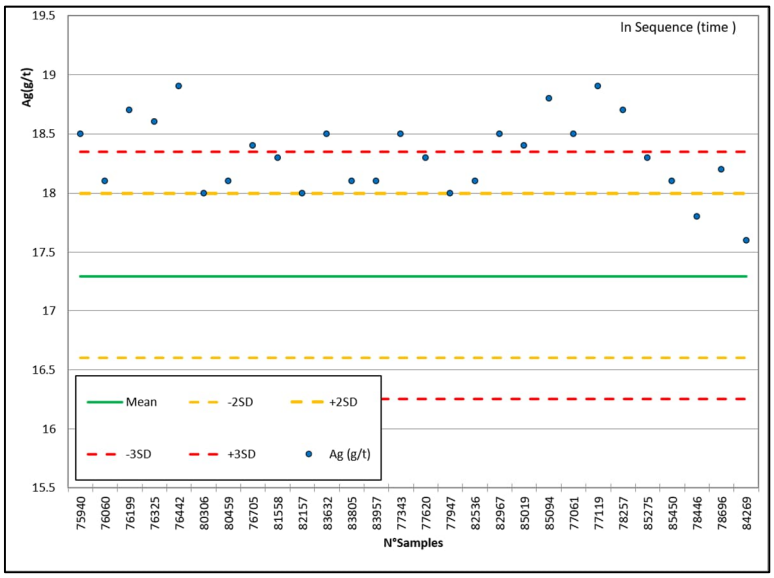


Figure 11-104: Figure 11-102: Certified Reference Material AuOx-52, Silver Performance

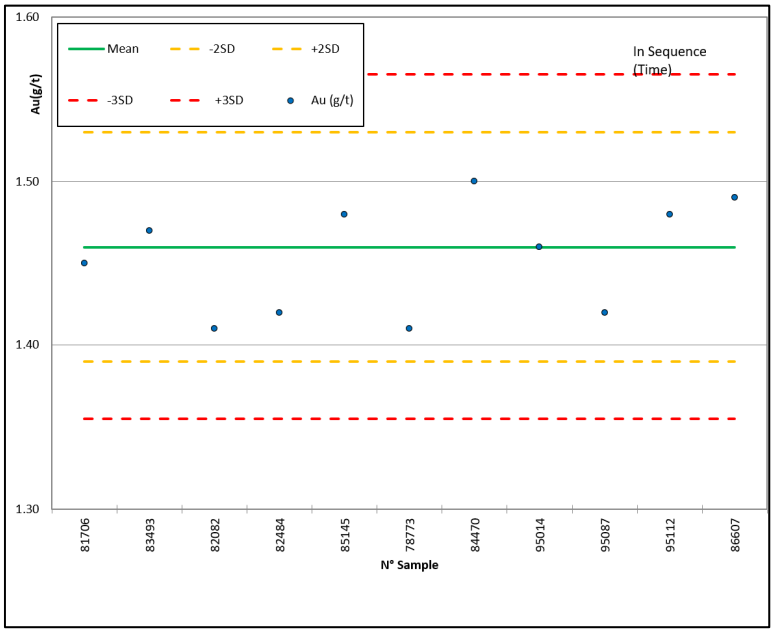


Figure 11-105: Certified Reference Material OREAS 504-D, Gold Performance

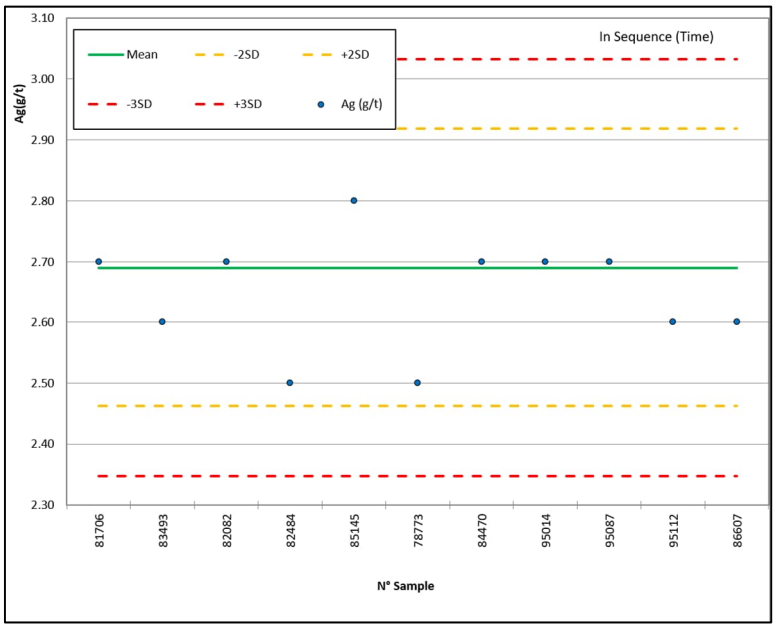


Figure 11-106: Certified Reference Material OREAS 504-D, Silver Performance

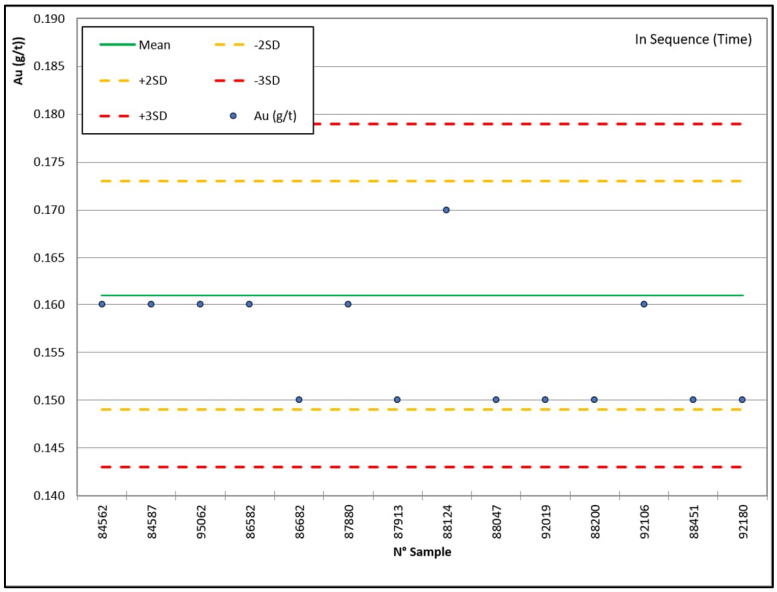


Figure 11-107: Certified Reference Material HDRT-02, Gold Performance

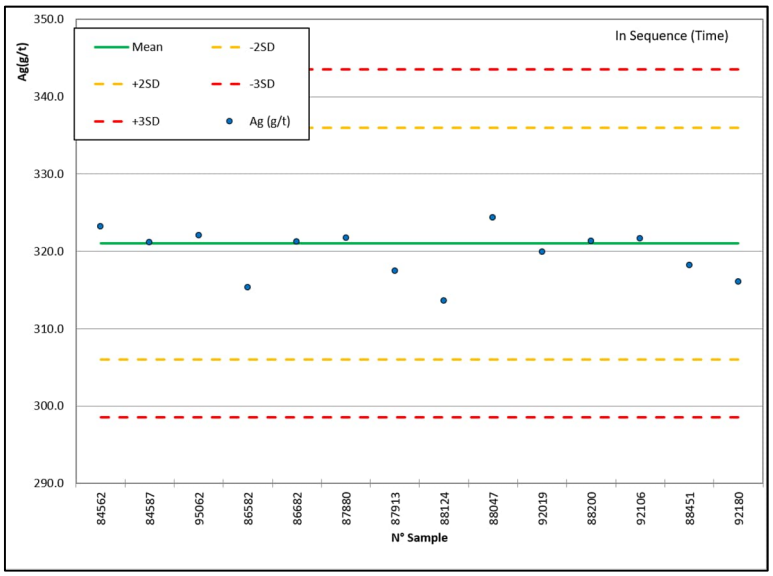


Figure 11-108: Certified Reference Material HDRT-02, Silver Performance

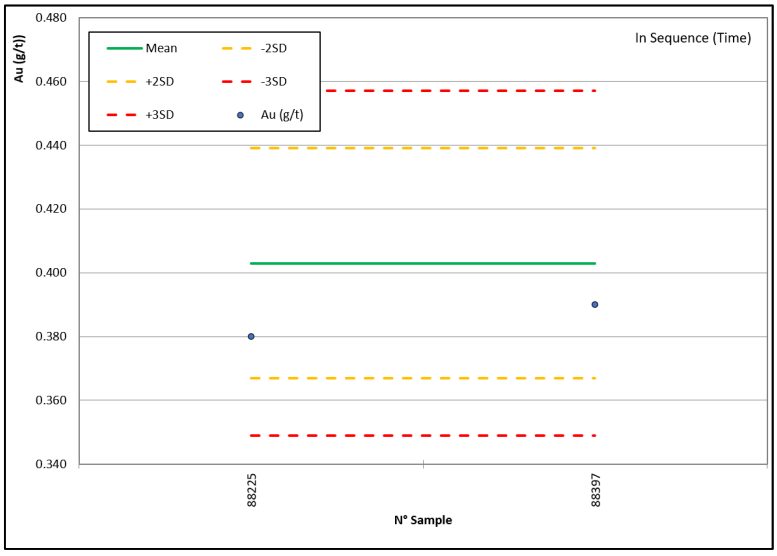


Figure 11-109: Certified Reference Material HDRT-03, Gold Performance

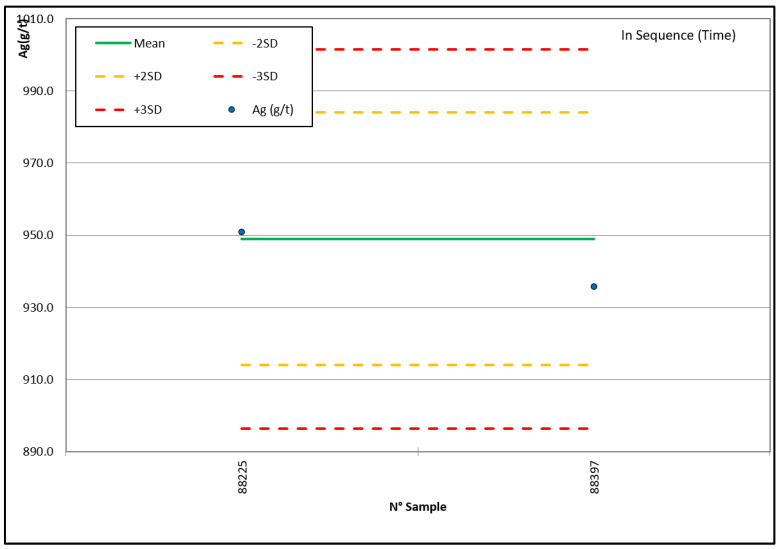


Figure 11-110: Certified Reference Material HDRT-03, Silver Performance

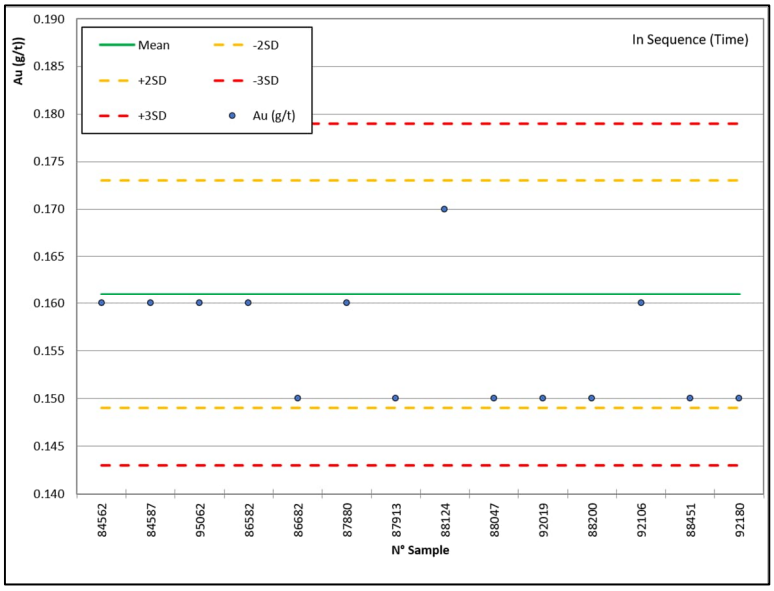


Figure 11-111: Certified Reference Material HDRT-02, Gold Performance

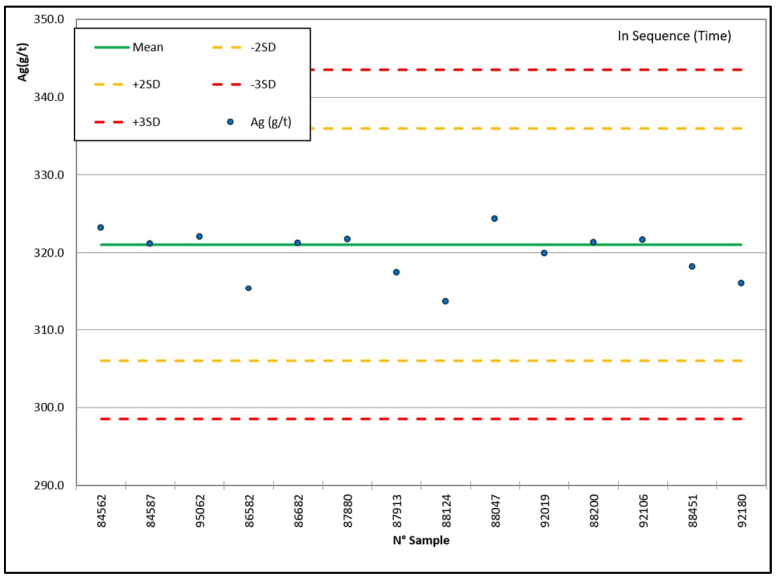


Figure 11-112: Certified Reference Material HDRT-02, Silver Performance

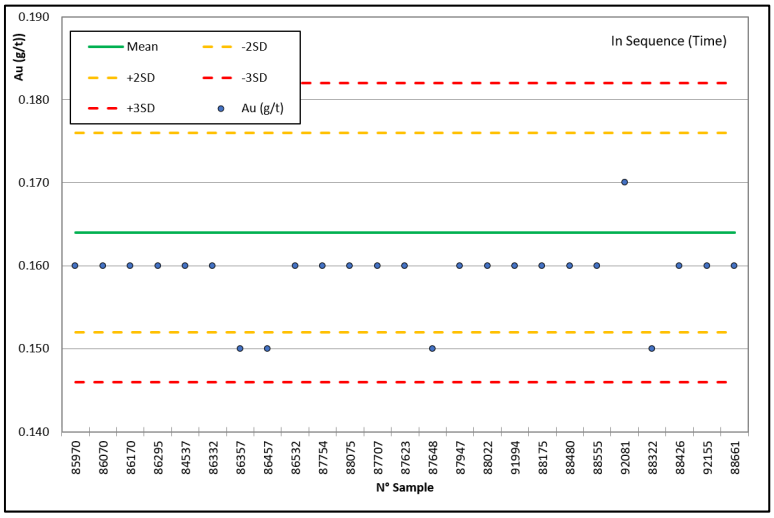


Figure 11-113: Certified Reference Material PLSUL-31, Gold Performance

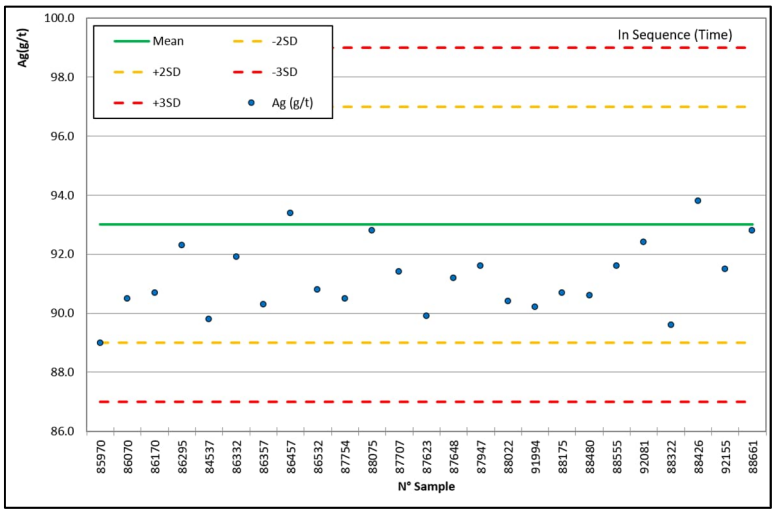


Figure 11-114: Certified Reference Material PLSUL-31, Silver Performance

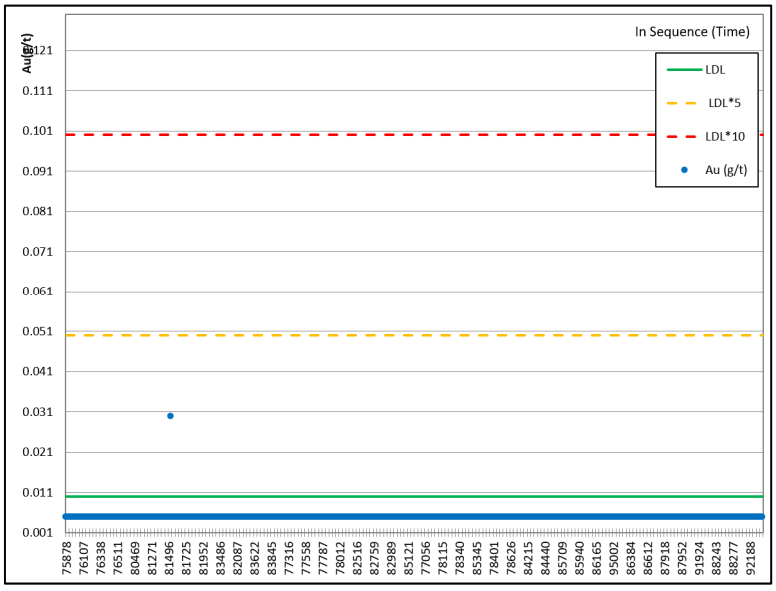


Figure 11-115: Internal Reference Material, Blank, Gold Performance

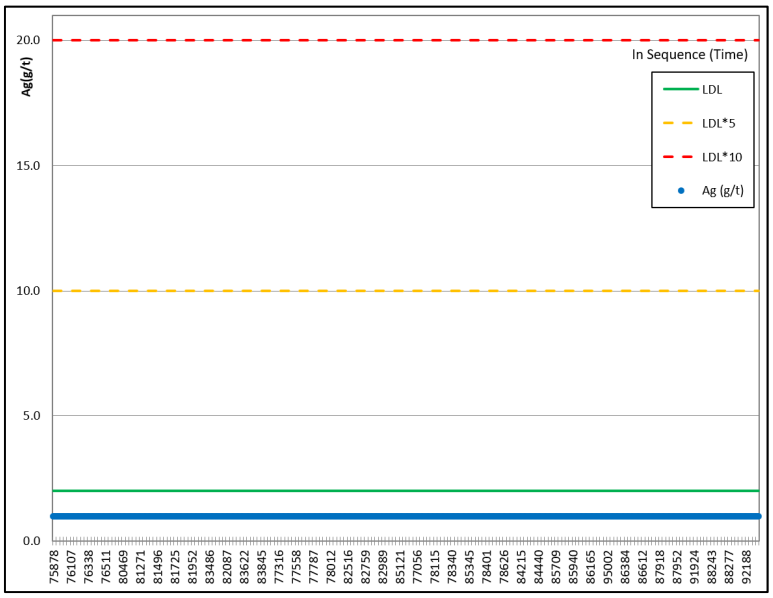


Figure 11-116: Internal Reference Material, Blank, Silver Performance

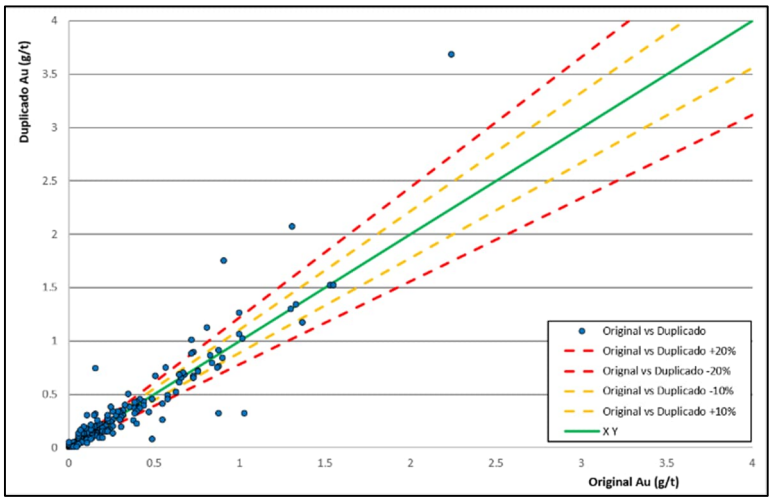


Figure 11-117: RMA Scattergram for Duplicate Performance of Gold

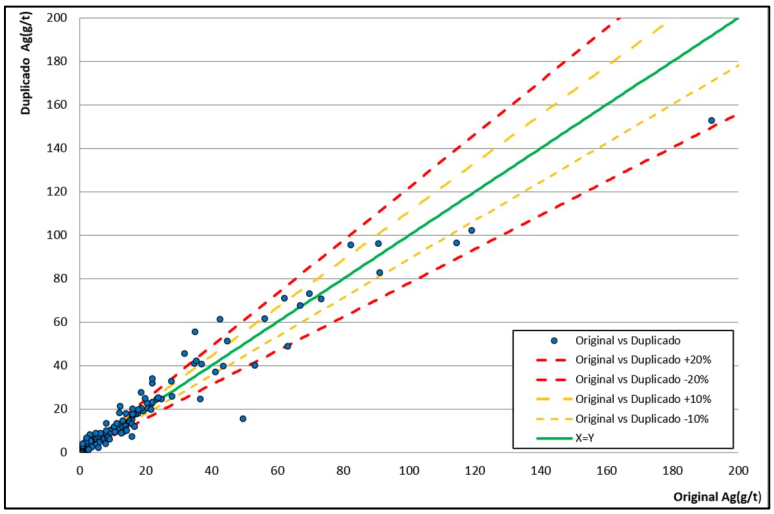


Figure 11-118: RMA Scattergram for Duplicate Performance of Silver

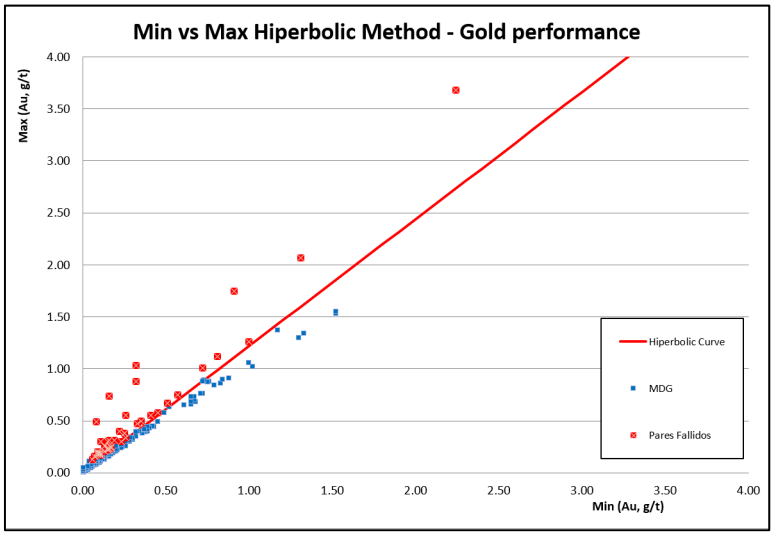


Figure 11-119: Min-Max versus Hyperbolic Method Scattergram for Duplicate Performance of Gold

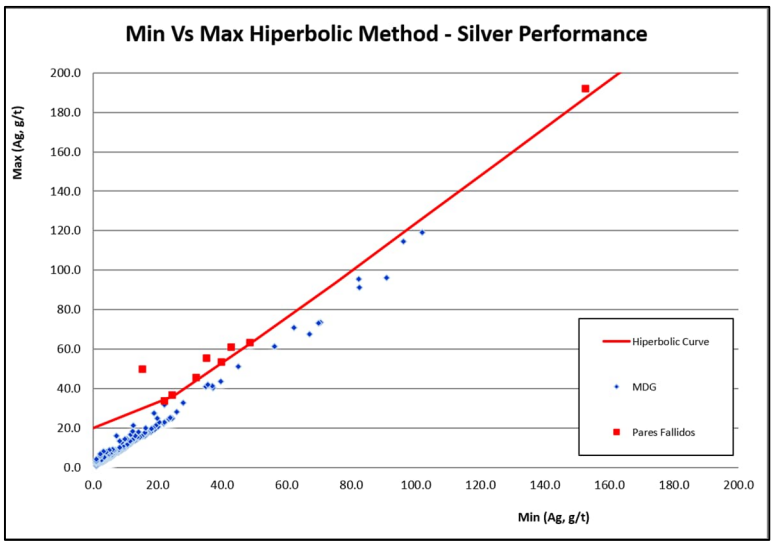


Figure 11-120: Min-Max versus Hyperbolic Method Scattergram for Duplicate Performance of Silver

11.7.5.1 Second Laboratory check for Phase V

As an independent verification of the primary analytical results, a comprehensive second laboratory check program was completed during the 2025 Phase V drilling campaign. The objective of this program was to assess the reproducibility of the analytical data and identify any potential analytical bias between laboratories.

The check assay program included three different types of duplicate material: coarse reject duplicates, pulp duplicates and quarter-core duplicates. These samples were selected from mineralized and barren intervals covering a broad range of gold and silver grades and were submitted to an independent external laboratory under separate sample batches while maintaining chain-of-custody procedures.

The comparison between the primary laboratory and the secondary laboratory included graphical and statistical evaluation of gold and silver assays using scatter plots, regression analysis, hyperbolic precision plots and relative error analysis. Separate evaluations were performed for coarse reject duplicates, pulp duplicates and core duplicates to assess variability introduced during sample preparation, subsampling and analytical determination. See Figure 11-121 and Figure 11-126.

The pulp duplicate dataset demonstrated an excellent level of agreement between laboratories for both gold and silver, reflecting the high analytical precision expected when identical prepared material is re-assayed

Coarse reject duplicates showed a slightly greater dispersion, consistent with the heterogeneity introduced during crushing and subsampling, but remained within acceptable limits for epithermal precious metal mineralization.

Quarter-core duplicate comparisons exhibited the highest variability of the three duplicate types, as expected due to the natural geological heterogeneity of mineralization and the nugget effect associated with gold and silver distribution.

Regression statistics indicate a strong correlation between laboratories with no evidence of significant systematic bias in either gold or silver analyses. Relative error plots and precision assessments confirm that analytical differences increase at lower grades, where results approach the limits of detection, but remain within industry expectations. At higher grades, the paired analyses display satisfactory reproducibility and no material discrepancies that would affect geological interpretation or Mineral Resource estimation.

Based on the review of duplicate core, coarse reject and pulp samples, Mr. Peralta considers that the inter-laboratory comparison demonstrates acceptable accuracy and precision for both gold and silver assays. The observed variability is consistent with that expected from the style of mineralization and the different stages of sample preparation represented by each duplicate type. Consequently, the analytical database generated during the Phase V drilling campaign is considered suitable for use in Mineral Resource estimation and supports the reliability of the assay data used in this Technical Report.

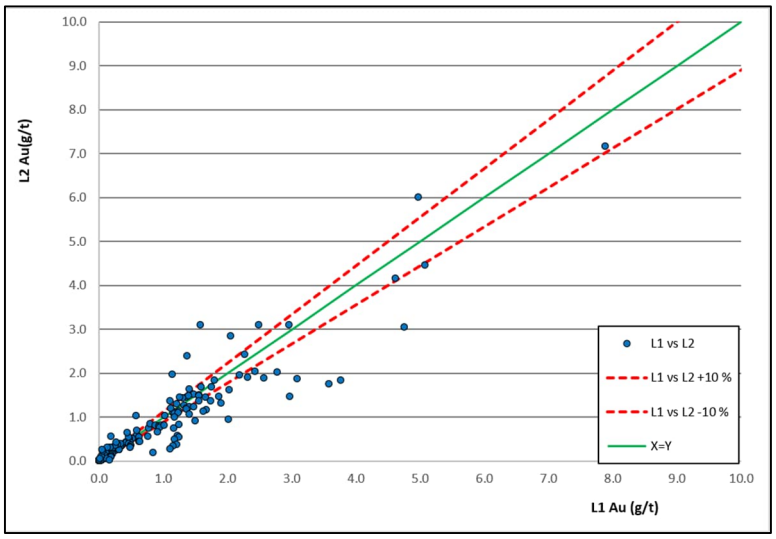


Figure 11-121: RMA Scattergram for Duplicate Performance of Gold (Cores)

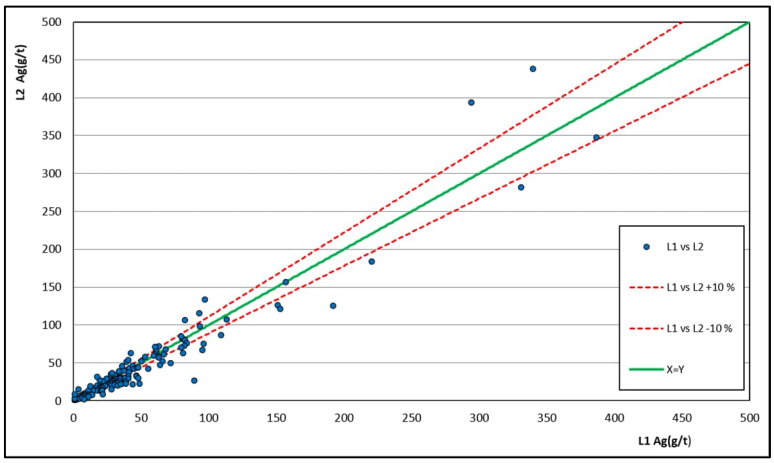


Figure 11-122: RMA Scattergram for Duplicate Performance of Silver (Cores)

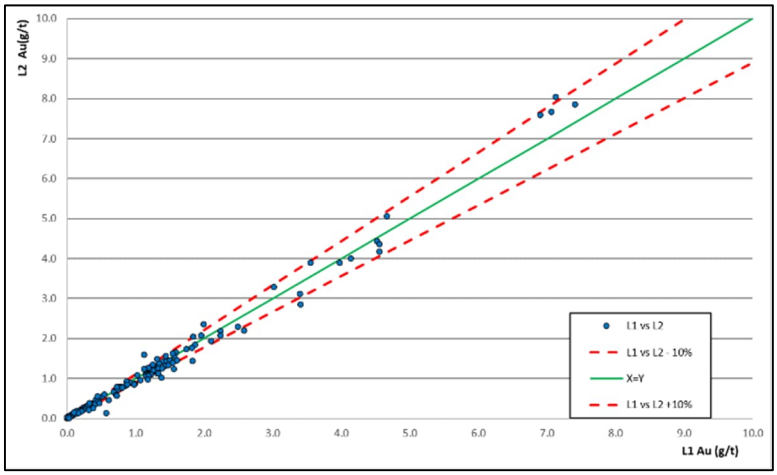


Figure 11-123: RMA Scattergram for Duplicate Performance of Gold (Rejects)

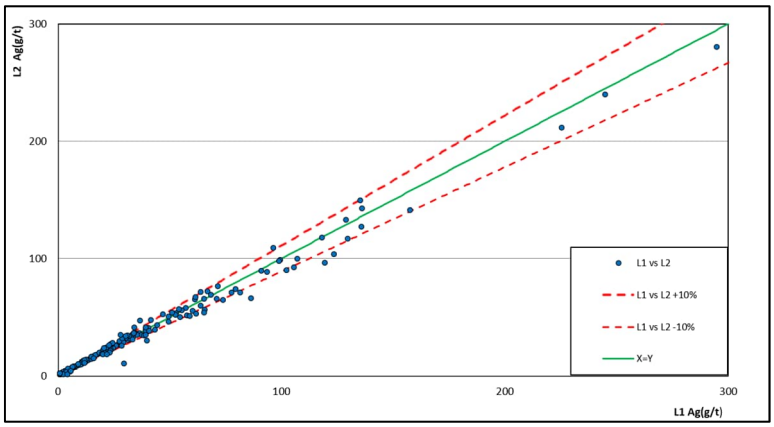


Figure 11-124: RMA Scattergram for Duplicate Performance of Silver (Rejects)

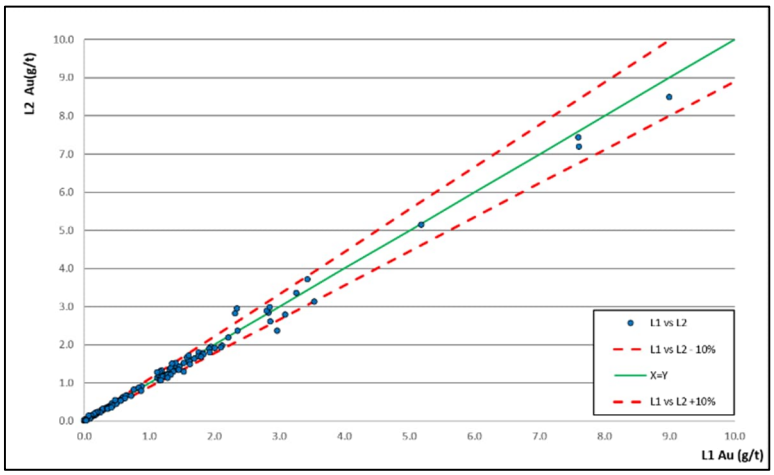


Figure 11-125: RMA Scattergram for Duplicate Performance of Gold (Pulps)

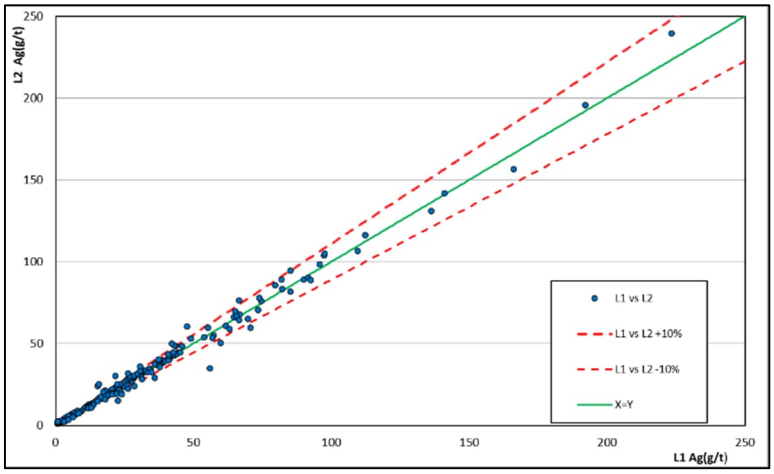


Figure 11-126: RMA Scattergram for Duplicate Performance of Silver (Pulps)

11.8 Discussion of QA/QC Assays AbraSilver

Based on this review and data analysis, Mr. Peralta concludes that the gold and silver accuracy was acceptable during the 2025 campaigns. Blank samples were assayed and most yielded values either below detection limits or below the five times detection limit. No obvious gold or silver cross contamination was identified during sample preparation. The RMA and Min-Max versus Hyperbolic Method scattergram plots for gold and silver show good fit between check assays and original assays. A few outliers have however been observed due to high variability in the style of mineralization.

The author has concluded that the assay QA/QC protocols implemented by AbraSilver were adequate for Mineral Resource estimation.

12. Data Verification

Mr. Peralta has reviewed the data compilation and has audited the drill hole database. The audit scope of work included reviewing:

- Collar locations.
- Downhole survey.
- Assays.
- Coincident samples.
- Twin holes.
- Bulk density.

The review also included checking 10% back to source data for collar location, survey, assay, density, and assay comparison analysis.

12.1 Collar Review

12.1.1 Collar Location

The review is based on 977 drillholes with a total depth of 193,131 meters. (see Table 12-1). The average drilling depth is 187 meters with a maximum of 1147 meters. There are 278 holes corresponding to reverse circulation (“RC”) with a total of 55,593 meters and 673 holes drilled with diamond (“DDH”) for a total of 134,299 meters.

Drilling campaigns expressed by year can be visualized in Table 12-1.

Table 12-1: Drill Campaign Summary by Year

Drilling Campaign	Type of Hole	Number of Holes	Meters Drilled	Average Meters Drilled	Min Metters Drilled	Max Metters Drilled
1987	RC	34	975	29	3	34
1990	RC	56	6 972	125	50	250
1993	DDH	5	1 002	200	146	254
1994	DDH	12	2 016	68	25	255
1996	RC	32	8 657	271	140	400
1997	RC	102	26 624	261	49	413
1997	DDH	19	4 558	240	31	380
1998	RC	24	7 547	314	220	70
1999	DDH	5	1 330	266	191	450
2003	RC	20	3 046	152	48	282
2003	DDH	6	397	66	46	76
2005	RC	10	1 772	177	101	252
2007	DDH	54	10 324	191	31	365
2007	Trenche	20	2 273	114	38	284

Drilling Campaign	Type of Hole	Number of Holes	Meters Drilled	Average Meters Drilled	Min Metters Drilled	Max Metters Drilled
2008	DDH	52	7 971	153	40	355
2012	DDH	19	1 679	88	41	126
2012	Trenche	6	569	95	47	145
2017	DDH	28	3 149	112	40	327
2019	DDH	2	844	422	38	464
2020	DDH	34	9 200	271	50	610
2021	DDH	69	16 136	233	50	451
2022	DDH	89	20 009	227	22	411
2023	DDH	82	14 683	179	30	245
2024	DDH	90	18 782	211	89	749
2025	DDH	107	22 617	199	101	1147
Subtotal	RC	278	55593	190	87	243
Subtotal	Trenche	26	2842	105	43	215
Subtotal	DDH	673	134299	204	62	439
Grand total		977	193131	187	67	352

12.1.2 Back to Source Checks

As the MRE was only executed for Oculto, JAC, Sombra, Fantasma and Laderas, coordinates consider only holes inside the block model area (described in Section 14). Identified by the “Area” field in the collar table, tagged within the Oculto, JAC, Sombra, Fantasma and Laderas zones. A perimeter surrounding these zones was used to cross check relevant holes.

This check included topographic survey details of 790 drillholes corresponding to 83% of total collars. The remaining 17% are located outside the Mineral Resource area.

The review confirmed that no drillholes reviewed presented differences between the original log and the database collar survey coordinates.

12.1.3 No Transcribed Coordinates

All drilling was presented with valid coordinates. No holes had absent data for collar location or final depth.

12.1.4 Maximum Depth versus Sampling and Logging Tables

The author carried out a review of the drilling data tables and did not find any discrepancy between maximum depth and the sampling or logging tables.

Table 12-2 shows the number of records per logging table while making the following observations:

- The drillholes have been selectively sampled, not all drilled intervals have been sampled continuously or until the end of the hole.
- Not all drillholes have a geology log until the end of hole depth.

- Some drillholes contain unlogged geology intervals.
- 94% of total meters drilled have been sampled.

Table 12-2: Number of Records per Logging Table

Tables	Number of Holes	Number of Records
Collars	791	791
Surveys	791	14,616
Assays	791	128,254
Lithology	791	8,316
Alteration	791	13,539
SG	111	8,624

12.1.5 Comparison of Collars and Topography Surface

A comparison of drillhole collar elevation with respect to the topographic surface was completed. Confirming that less than 1% of drillholes had a difference greater than 2 meters with respect to the topography.

Where the discrepancy was greater than 2m, it was recommended that the drillhole be projected onto topography and to adopt an Inferred classification. Only one historical hole presented such a difference.

12.2 Downhole Surveys

12.2.1 Downhole Surveys Station Analysis

As part of the data revision of the survey table, drillholes without downhole surveys have been excluded from the final database. The depth, dip and azimuth columns have been used for all selected drillholes. Details of survey records, listed by year, are shown in Table 12-3.

The author highlights the following from the review:

- All azimuth values are between 0 and 360.
- All dips are between -90 to -35 degrees.
- All azimuth values are between 0 and 360.
- All dips are between -90 to -35 degrees.

Drillholes with a single measurement and depth greater than 100 meters are not considered for a Measured categorization.

Table 12-3: Summary of Downhole Survey Station

Drilling Campaign	Number of Holes	Number of Records	Number of Holes			% Hole with Deviation Measurement
			1 Downhole Survey	2 Downhole Survey	>2 Downhole Survey	
1990	25	50	0	25	0	
1993	5	10	0	5	0	
1994	12	24	0	12	0	
1996	32	376	1	24	7	1
1997	113	1691	1	67	41	5
1998	24	1235	0	13	11	
1999	5	293	2	0	3	
2003	11	11	10	0	0	
2005	8	8	5	0	0	
2007	46	228	2	1	43	9
2008	49	185	2	13	33	11
2012	7	26	0	0	3	
2017	28	94	0	0	28	
2019	2	86	0	0	2	5
2020	34	274	0	0	34	
2021	61	904	0	0	61	
2022	81	2774	0	0	81	
2023	82	1815	0	0	82	
2024	80	1522	0	0	80	
2025	108	3725	0	0	108	
Grand Total	813	15331	23	160	617	31
Percentage			3%	20%	76%	

12.2.2 Assessing Any Corrections Applied

No global correction is suggested as most data in the downhole survey table is accurate and presents no meaningful deviation. The exception was second decimal place corrections.

It is suggested that holes with only one point station should not be used to categorize at a higher confidence level than Inferred. The exception being historical holes validated in a recent campaign, or where geological and alteration profile fits with recent models. In those cases, the confidence in hole trace is high.

As rounding issues are not considered material, no correction should be applied. However, it is suggested that original survey measurements are reloaded into the database.

12.3 Assays

12.3.1 Back to Source Data Checks

There has been changes in ownership and database systems over the last 35 years. Some of the historical data remains incomplete including flagging in the database, certified reference materials, blanks and duplicates. Recent project work over the last twenty years has however used exploration methodologies in line with industry best practice.

Back to source checks have been completed on the 2019 to 2025 drilling campaigns. For the remaining campaigns, gold and silver values have been verified using independent sampling of reject, pulps and cores from historical drillholes. (Refer to NI 43.101 Technical Report – Mining Plus, PEA 2022, Section 12 – Data Verification, Independent sampling check).

The author has checked approximately 0.72% of batch certificates back to source data comprising approximately 913 samples out of a total of 128,254.

The following conclusions are noted:

- The assay table includes 128,254 records of which 125,941 have gold values including 2,313 with zero value.
- The assay table includes 128,254 records of which 121,377 have silver values including 6,877 with zero value.
- Zero records have null values.
- No negative or non-numeric values were identified.
- The detection limit was replaced with half of the value, however, during the check back to source no data was detected below the detection limit.

The author has observed that there are no duplicate sample codes.

12.3.2 Overlapping Intervals

No overlapping samples were detected during the process of auditing the database.

No data entry errors were identified in the intervals.

12.3.3 Coincident Samples

No coincident samples were detected.

12.3.4 Kink Analysis

Kink analysis was performed over the 791 drillholes selected to be used in the MRE.

Kink analysis evaluates drillholes per year that have not passed the deviation analysis of survey points. This is when the change in azimuth is greater than 10 degrees, the change in dip limit is greater than 10 degrees or the angle of the drillhole is greater than 10 degrees.

A total of 32 drillhole survey point measurements has a deviation greater than 10 degrees. These 32 deviations represent less than 1% of the 15,341 points of observation.

After a detailed review and verification against the original certificates that did not pass the kink analysis, the conclusion is not to exclude any hole. The error in all cases was due to mistyping information or vertical holes with misinterpretations. All errors were corrected.

12.3.5 Comparison Analysis of Different Types of Data

In this report, no additional comparisons were conducted between data types as this work was previously completed (Refer to NI 43.101 Technical Report – Mining Plus, PEA 2022, Section 12 – Data Verification, Independent sampling check). The conclusions of that analysis are transcribed below.

- *"The comparison of RC vs DDH was performed within a limited area including the main mineralization. Results indicate that the sample results from RC drilling closely match those from diamond drilling and no bias is evident.*
- *The comparison of the 2008 and 2020 drilling campaigns is shown certain differences, mainly with the 2020 drilling campaign. This was attributed to the intercept of economic mineralization with significant values, causing mean, upper, and lower quartiles to be higher than the 2008 drilling campaign. It was thus concluded this was not evidence of bias."*

The QP concludes from the previous work that no bias is evident between RC and diamond drilling types.

12.3.6 Twinned Drill Holes

No analysis of twin holes has been made for this report as was done in a previous Technical Report NI 43.101 reported by Mining Plus, MP (2021). However, that study was not conclusive.

12.4 Site Visits

Mr. Peralta visited the Property from April 24th to May 03rd, from October 02nd to October 8th, 2023, from October 18th to October 22nd, from December 6th to December 11th, 2024, from April 8th to April 14th, from August 25th to August 29th, 2025, from April 13th to April 19th, 2026, for the purposes of this report.

He conducted a general site inspection, an inspection was made at and around the surface geology of the Oculto, JAC, Laderas, Sombra and Fantasma zones as discussed in Section 9. Several core samples were reviewed and compared them with logs, including drill collars, cores, logging facility, logging procedures, sampling methodologies and camp. Collar locations were confirmed by handheld GPS. Downhole survey station details were checked. Vertical cross sections and plan views with detailed geology, alteration and interpretation were discussed with AbraSilver geologists. Discussions included future exploration targets and near-term objectives.

In the author's opinion, the site was found to be as described in the Technical Reports, the facilities were well-maintained, and the core storage was orderly. (see Figure 12-1)



Source: AbraSilver, 2023.

Figure 12-1: Drill Core Storage Facility at Site

12.5 Discussion

In Mr. Peralta's opinion, the database is reasonably free from errors and suitable for use in the estimation of Mineral Resources.

13. Mineral Processing and Metallurgical Testing

13.1 Introduction

The Diablillos property, owned by AbraSilver, is a gold and silver deposit that has demonstrated good metal recoveries through laboratory-scale cyanidation testwork. Previous metallurgical testwork campaigns were carried out at several laboratories between 1996 and 2021 by the previous owners of the Project.

A more thorough metallurgical testwork program was completed by SGS Canada Inc., in Lakefield, ON (SGS Lakefield) in 2022 and 2023 for the PFS stage. Further testwork was conducted by ALS Metallurgy, in Kamloops, BC (ALS), and SGS Lakefield in 2025 for the DFS stage.

For the DFS, the testwork program focused on optimizing and validating the flowsheet established during the PFS, which consists of the following:

- Coarse primary grind (P_{80} 150 µm) to reduce the capital costs and energy demand
- Gravity concentration to recover silver/gold with intensive cyanidation
- Conventional leach with counter-current decantation (CCD)
- Merrill-Crowe for silver/gold recovery
- Cyanide destruction process.

The 2025 testwork program validated the PFS flowsheet, which will now be advanced into the DFS.

This chapter presents a brief overview of the historical testwork and provides more details on the latest testwork programs completed and upon which the design and recovery assumptions are resting.

13.2 Historical Testwork (1996 to 2023)

Table 13-1 presents the chronology and scope of the historical, pre-DFS stage, testwork conducted on the Diablillos Project.

Table 13-1: Summary of Historical Metallurgical Testwork

Year	Laboratory	Testwork Scope
1996	SGS Lakefield	Bottle Roll Cyanidation Tests
1998	SGS Lakefield	Bottle Roll Cyanidation Tests Column Leach Tests
2007	SGS Lakefield	Cyanidation Tests Bond Work Index (BWI) Mineralogy

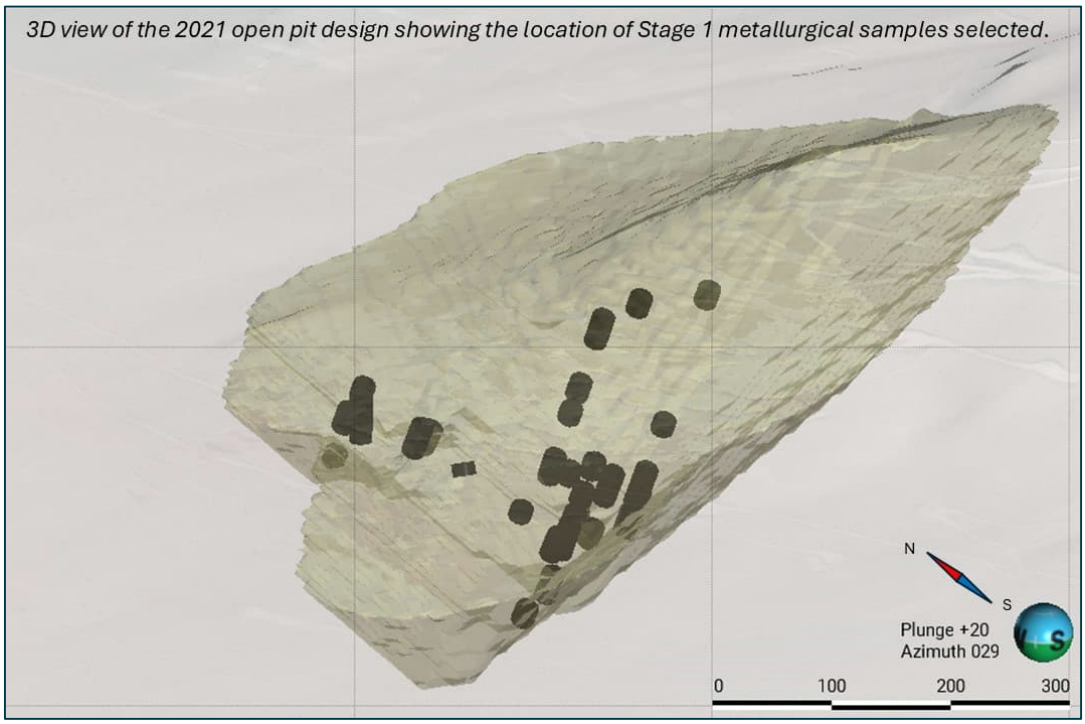
Year	Laboratory	Testwork Scope
2008	Process Research Associates	Sulphide Content BWI Bottle Roll Cyanidation Tests (Fine and Coarse Ore) Flotation
2009	Process Research Associates	48 hr Bottle Roll Cyanidation Column Leach Tests Leach Tests with Site Water
2019	ALS	Bottle Roll Cyanidation Tests Flotation
2021	ALS (Phase 1)	24 hr Cyanidation Tests Sedimentation Testing
2022/2023	SGS Lakefield An Investigation into the Diablillos Deposit, Project 18501-03 – Final Report (December 7, 2022) (Phase 2) An Investigation into the JAC and Fantasma Deposits, Project 18501-03 – Report #2 (May 12, 2023) (Phase 3)	Quantitative Evaluation of Materials by Scanning Electron Microscopy (QEMSCAN), Particle Mineral Analysis (PMA), Tescan Integrated Mineral Analyzer (TIMA) Semi-Autogenous Grinding (SAG) Mill Comminution (SMC), BWI, Abrasion Index (Ai) Extended Gravity Recoverable Gold (E-GRG) Cyanidation Tests (Fresh and gravity tails) Merrill-Crowe Cyanide Destruction with INCO/SO_2 Testing Sedimentation and Rheology Tests Acid Based Accounting (ABA), Net Acid Generating (NAG), Toxicity Characteristic Leaching Procedure (TCLP)
2022	Clean Mining	Alternative Leaching Reagent
2022	(In Peru)	Coarse Bottle Roll Cyanidation Column Tests

As the sample sourcing was confined to a smaller deposit footprint (or outside of the current pit limits) and that the cyanidation test conditions used for most work completed prior to the 2021 ALS test program are not relevant to the filtered database used to carry out the current analysis to determine the optimum design criteria for the plant, no discussion of these is provided herein. For further details about the historical testwork, refer to the PFS-stage NI 43-101 report (2025).

13.2.1 AbraSilver - Phase 1 (2021)

In 2021, AbraSilver sent ALS 71 samples, which were collected from 23 HQ diamond drill holes perforated between 2019 and 2020 in the Oculto deposit. This testwork program was labelled Phase 1 by AbraSilver.

The samples, tagged ME0001 to ME0071, were collected from single quartered drill core intercepts. These were confirmed to be within a resource-based pit shell envelope through plotting on plan views and sections of each intercept. Figure 13-1 is representative of such work, showing the intercepts crossing a particular horizon.



Source: AbraSilver, 2026

Figure 13-1: 3D View of the 2021 Open Pit Design Showing Phase 1 Metallurgical Sample Locations.

The mineralogy study indicated that there is a high level of silica present in the whole rock analysis, with an average content of 72% and a range of 40% to 94%. The geochemical inductively coupled plasma (ICP) indicated relatively high levels of arsenic, antimony, mercury, and lead in multiple samples.

QEMSCAN tests were conducted on all the samples and form the basis of the mineralogical knowledge generated from the testwork, with subsequent programs seeing more limited efforts. It was found that the major mineral component of most samples was quartz. The next abundant mineral is alunite, having a content of up to 45% in some intercepts. Overall, the material studied exhibits a large variability in the mineralogical constituents, texture, grain size, and silver-bearing minerals.

SAG mill comminution (SMC) hardness determinations were conducted on the 71 samples provided, with the Bond Ball Work Index (BBWi) involving only 16 samples. The SMC tests indicated an average Axb value of 61.7, which is considered to be medium-hard material in terms of this SAG milling parameter. The abrasion index (Ai) averaged 0.31. The BBWi of the samples tested averaged 14.8 kWh/t. The range of the above comminution data is quite wide, with Axb values varying between 28 and 127, the abrasion index (Ai) varying between 0.01 to 0.74, and the BBWi varying between 9.3 to 21.8 kWh/t.

The samples collected had an average head grade of 1.11 g/t of gold and 105 g/t of silver, which are contents somewhat higher than the current resource model averages. The content of copper is usually quite low, averaging around 0.01% once discounting 15 samples with high copper content that were excluded from the leach testwork program phase because they were deemed to be below the oxide layer and into the underlying sulphide zone.

Based on the observations from the testwork organized by Silver Standard in 2008 and 2009, the following test conditions were applied to 56 of the samples involved in hardness determinations:

- The grind sizes for bottle roll tests were targeted at P_{80} values of 100 µm and 150 µm, rather than 75 µm.
- The free sodium cyanide concentration was controlled at an initial 1 g/L for tests at the coarse P_{80} and 1.5 g/L for the fine P_{80} and then allowed to decay to 0.5 g/L after 6 hours in both cases.
- A leach time of 24 hours was employed.
- Oxygen was sparged into the leach bottles at the time intervals of 0, 1, 2, and 6 hours, as required to raise the dissolved oxygen (DO) level to 8-12 mg/L.

The leach test results indicated that there was no significant difference in metal recovery between grind sizes of 100 µm and 150 µm. Silver recovery appears to have more variation than gold, possibly due to the large variation in silver head grade of the individual samples. DO levels, with only sporadic injection of oxygen, were often measured at low levels at multiple solution sample extraction times, likely reducing the leaching efficiency achieved.

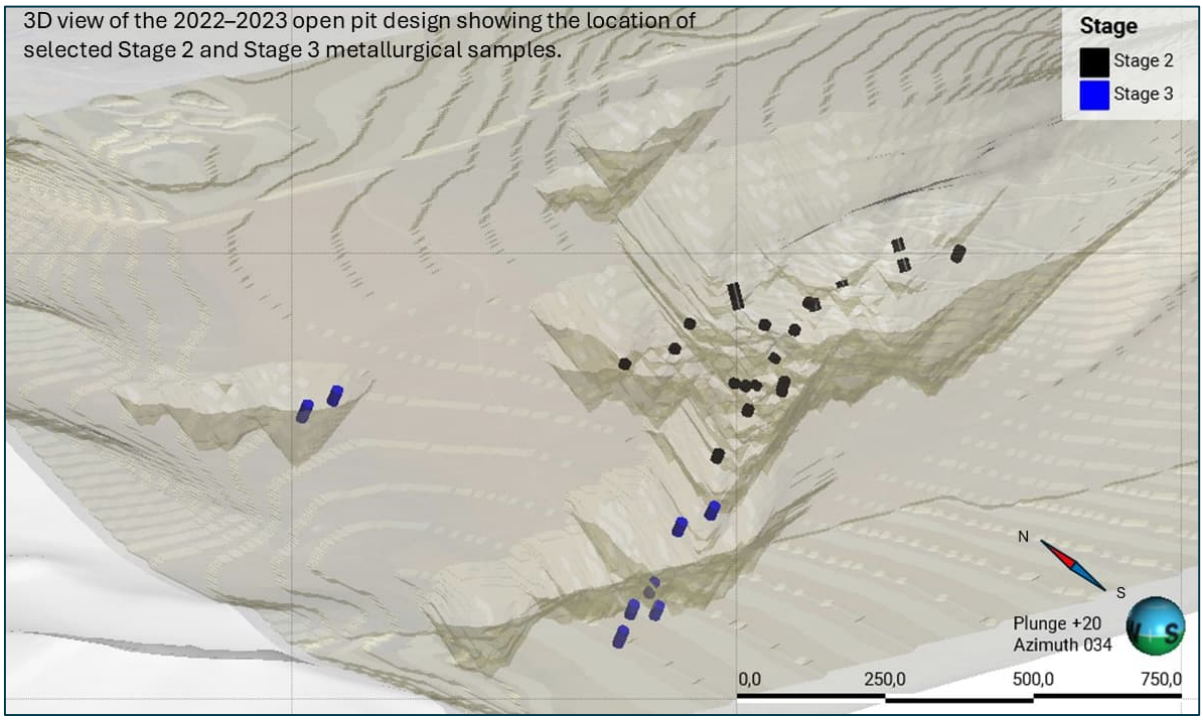
Diagnostic leach tests applied on the leach tails of two samples indicated that, after removing the residual portion leachable in cyanide, 60-70% of the unleached gold and silver in these tails may have been tied up in jarosite or plumbojarosite.

The sedimentation tests indicated that this material exhibited fast settling and produced a clear liquid overflow.

13.2.2 PFS Testwork, AbraSilver – Phase 2 and 3 (2022 – 2023)

In 2022, AbraSilver provided SGS Lakefield with 42 variability samples (tagged ME0072 to ME0114) and 16 composites prepared with portions of said samples from the Oculto deposit. These represented the Shallow Gold (SG), Silver Enrichment (SE), Deep Gold (DG, referred as Tesoro at times), and North East (NE) geometallurgical units (GU). In addition, a single "master composite" was received, blending individual composites in proportion representing the relative weight of each GU in the pit-confined resources. This testwork program was dubbed Phase 2 by AbraSilver.

The location of the variability samples was confirmed to correspond to the resources pit limits considered at the time, as illustrated by Figure 13-2.



Source: AbraSilver, 2026

Figure 13-2: Phase 2 -3 Variability Samples Location

In 2023, a further 12 samples from the newly discovered JAC deposit and two samples from the Fantasma deposit were submitted, as obtained from single intercepts within drill holes. These were used for the Phase 3 testwork program. As Fantasma remains excluded from the proposed mine plan at this time, no further discussion of this material is provided below.

QEMSCAN and PMA results on the 42 variability samples showed that most Oculto and JAC samples are predominantly composed of quartz, with notable amounts of alunite, jarosite, iron oxides, and minor free pyrite. TIMA analysis indicated that gold occurs mainly as native gold (82%), with the remainder as electrum (18%). The principal silver minerals identified were iodargyrite (42%) and chlorargyrite (32%), with the balance as acanthite or oxidized acanthite. Silver minerals are typically found within quartz/feldspar matrices and complex or liberated silver phases.

The 16 composite samples from Oculto were put through comminution testing (SMC, Ai and BBWi). Compositing material for hardness assessment has the drawback of potentially masking the actual variability though, within each of the GU represented. Such data should therefore not be used for design purpose and mixed with hardness determinations obtained on distinct samples when considering a statistical approach, such as the use of percentiles. This exercise at least demonstrated that the JAC material is significantly softer for SAG milling and less abrasive than Oculto ore, while still falling within a medium-hard range with respect to ball mill grinding.

A three-stage extended gravity recoverable gold (E-GRG) separation test was conducted on composite samples of both deposits, as well as composites from each geological unit within Oculto. The results for the Oculto composite indicated undiscounted E-GRG recoveries potential of 22.2% and 25.5% for gold and silver, respectively, while prospective gravity recovery from

the standard SGS Lakefield procedure called for 10.1% gold and 7.8% silver recoveries. The equivalent figures for JAC were, from the undiscounted E-GRG work, 24.9% gold and 34.5% silver, as compared to 17.3% and 9.1%, respectively, from the SGS Lakefield scheme.

Following comparative cyanide leach testing performed under different conditions with the "master composite", a set of optimum test conditions were retained and applied to leach tests involving the individual 42 variability samples. These conditions including a grind size of P_{80} 150 µm, a leach residence time of 48 hours at a pulp density of 45%, and an initial sodium cyanide concentration of 1.5 g/L, which was maintained as required for the first 12 hours and then allowed to decay.

These parameters yielded terminal gold recovery of 85% and 82%-84% for silver within the 48 hours of leach residence time tested, though the leach kinetics indicated that 36 hours of leaching time would be sufficient to provide maximum metals recovery.

Cyanide leach tests were also performed on nine of the gravity tailings, from the SGS Lakefield procedure, of Oculto and seven samples from JAC, indicating averages of 83% and 84% gold recovery and 80% and 85% silver recovery, respectively for each area. The JAC material generally requires greater cyanide and lime consumption due to higher silver and sulphide contents. Table 13-2 presents the reported gravity gold recoveries and subsequent 48 hour cyanidation recoveries of the gravity tails.

Table 13-2: SGS Lakefield Phase 2 – Recoveries from Gravity Concentration and Cyanidation of the Gravity Tails

Test	Shallow Gold	Silver Enrichment	Deep Gold	North East
Reported Gravity Recoveries				
Gold	16.2%	12.3%	8.6%	10.1%
Silver	16.4%	5.4%	3.3%,	5.2%
48 Hour Cyanidation Recoveries from Gravity Tails				
Gold	83.5%	86.6%	82.2%	88.1%
Silver	82.8%	54.3%	82.0%	80.2%

Solid/liquid separation and rheology tests were performed on leach slurry tails and detoxified slurry obtained from bulk cyanidation test products, indicating potential underflow target densities of 57% and 50% for Oculto and JAC, respectively. For the detoxified slurry, the critical solids density (CSD) was found to be 67% for Oculto and 55% for JAC.

Merrill-Crowe testwork confirmed that the ore is amenable to zinc dust precipitation. Based on the test results, a stoichiometric ratio of 3 for Oculto and 5 to 10 for JAC was recommended at a 3:1 CCD wash ratio. The leach and Merrill-Crowe recoveries were not affected by the type of water used in the tests, namely the brackish site water, synthetic site water created with aquarium salts and sulphuric acid mixed with deionized water, and tap water from the laboratory.

Cyanide destruction testing was performed on the Oculto composite leach tailings slurry obtained from the laboratory test program, with the goal to reduce the weak acid-dissociable

(WAD) cyanide level in the plant effluent to below 1 mg/L. The cyanide destruction method used in the test was the INCO SO_2/air process, using sodium metabisulfite as the sulphur dioxide source, copper sulphate as a copper ion source, and air to oxidize the weak acid-dissociable cyanide (CN_{wad}). The test indicated that less than 4.7 g SO_2/g CN_{WAD} and 1 hour of retention time will be required. Copper addition was not essential to achieve this outcome. The same cyanide destruction test procedure was applied to the JAC composite leach tails but a revised residual CN_{wad} target of <10 mg/L. The test indicated a requirement for about 4 g SO_2/g CN_{WAD} with 1 hour of retention time. Again, no copper was needed or added during the test.

Modified ABA tests, net acid generation tests, and TCLP analyses were conducted on the Oculto and JAC bulk leach and detoxified tailings, as well as on 24 leach tailings from Oculto and 14 of JAC, from samples involved in cyanidation variability tests. The results indicated that the tailings would be acid generating. However, to evaluate the potential for acid generation of leached tailings over a longer term at the mine site, more thorough testing is recommended. TCLP testing was also completed to determine the mobility of inorganic contaminants.

13.2.3 DFS Testwork (2025)

The DFS testwork program was initially structured as two phases: Phase 5A and Phase 5B. Phase 5A was to be completed concurrently at ALS and SGS Lakefield through the use of existing samples left from earlier testwork phases (Phase 1 for ALS and Phase 2 and 3 for SGS Lakefield). This phase focused on confirming the optimal leach conditions set during the PFS stage, revisiting parameters such as grind size, leach retention time, cyanide concentration, and DO levels. Merrill-Crowe and viscosity tests were also completed.

Phase 5B was carried out later at SGS Lakefield using newly selected samples shipped from the project site. This phase comprised a more extensive testwork program, which ended up being divided into a Phase 5B1 and Phase 5B2 for the sake of schedule and budget control.

The DFS process design, as developed by Worley, was largely based on testwork completed in 2025 under Phase 5A. As the Phase 5B testwork reports became available early in 2026, their outcomes were primarily used to complement the testwork databases relied upon for performing various data analysis and to strengthen the key process design assumptions, which provided additional confidence that the existing design is robust and technically sound. Worley recommended that the Phase 5B results be incorporated in the next project phase (Early Works) to further refine and optimize the current DFS design.

13.2.4 ALS Metallurgy – Phase 5A

13.2.4.1 Introduction

ALS conducted metallurgical testing on samples that were previously tested in 2022 under Phase 1. A total of 20 variability samples were retrieved from storage, with 12 simply renamed and tested as is while eight others, with low residual weights, were combined to form three composites along the GU lines.

The following tests were completed:

- Agitated whole ore cyanidation leach tests on the samples/composites at a nominal primary grind size of 150 µm and 200 µm K80, in duplicate, to assess the potential for cyanidation leach extraction of gold and silver.
- Agitated whole ore cyanidation leach tests at various sodium cyanide dosages and lead nitrate additions to assess their effect on gold and silver leach kinetics and terminal extractions.
- Agitated whole ore cyanidation leach tests over a range of slurry density.
- Viscosity tests on selected composites to assess the effect of pulp density and chemical additives on resulting slurry viscosity.

13.2.4.2 Metallurgical Testing

The metallurgical program involved agitated cyanidation leach tests on 15 composites to evaluate gold and silver recovery. Tests were performed under conditions simulating plant operations, using whole ore cyanidation at two primary grind sizes (150 µm and 200 µm K80), varying sodium cyanide concentrations, and three slurry densities.

13.2.4.3 Whole Ore Cyanidation – Primary Grind Series

Agitated cyanidation leach tests were performed on 15 composites at a target grind size of 150 µm K_{80} while actual sizings ranged from 130 µm to 174 µm K_{80} based on cyanidation tails sizing. Slurries were pre-aerated with oxygen for 16 hours, then leached for 24 hours at 1,500 mg/L sodium cyanide before residual cyanide was allowed to drift to 750 mg/L. The PFS leach slurry density of 45% w/w solids was used as the base case level and a pH of 10.5 was maintained in the slurry, which was kept under continuous agitation for 48 hours.

The results achieved indicate the following:

- Silver extraction: 45% to 92%, averaging 76%;
- Gold extraction: 47% to 95%, with the lowest values in low-grade SE composites;
- Sodium cyanide consumption: ~1.8 kg/t at 150 µm K_{80}; highest with high silver grades composites.

The agitated whole ore leaches were repeated for all composites at similar conditions but at a target primary grind size of 200 µm K_{80} while actual sizings ranged from 177 µm to 239 µm K_{80} based on cyanidation tails.

The results highlights are as follow:

- Silver extraction: 43% to 92%, averaging 75%;
- Gold extraction: 66% to 90% where, for composites with feed grades ≥0.5 g/t gold, the coarser grind reduced gold recovery by about 2%.
- Sodium cyanide consumption: ~1.7 kg/t at 200 µm P_{80}.

Gold and silver assays were performed on four size fractions of the leach residues from both set of tests completed at each grind size. For residues with higher gold content, the gold

distribution by size fraction generally followed the overall mass distribution. In contrast, silver showed a greater proportion of losses in the coarsest fraction (>150 µm) for some composites, suggesting that silver may be locked within larger particles.

13.2.4.4 Whole-Ore Cyanidation – Cyanide Dosage Series

Agitated whole ore leach tests were performed on six selected composites (three each from DG and SE) at a grind size of 150 µm K_{80} to evaluate the effects of varying sodium cyanide concentrations, cyanide addition during primary grinding, and lead nitrate addition. The samples were chosen either because of high head grades, to test the potential for reducing the cyanide dosage, or low overall Au/Ag grades, to see if an increase in cyanide addition would result in a higher recovery potential.

All tests included 16 hours of pre-aeration prior to leaching (the ALS Phase 1 standard), except for those where cyanide was introduced during grinding. High cyanide concentration tests targeted 2,500 mg/L in solution for 24 hours before allowing a drift to 1,500 mg/L, at 45% solids and a pH of 10.5. Low cyanide concentration tests were conducted at 750 mg/L, before being allowed to drift to 500 mg/L.

The results show that:

- The addition of lead nitrate did not enhance metal extraction.
- Increasing sodium cyanide concentration proved effective for one sample only, improving silver extraction by approximately 8%.
- Gold and silver extractions showed a significant decline in both gold and silver recovery for one sample at the lower cyanide concentration.
- For the single sample tested with cyanide and lime added during the primary grinding, negligible improvement of gold and silver extractions was seen.

Therefore, there seems to be no case for modifying the base case leaching conditions established from the PFS stage. Differential responses between tested samples, even of the same GU, are likely driven by distinct mineralogical characteristics between samples.

13.2.4.5 Whole-Ore Cyanidation – Pulp Density Effect

Viscosity testing was carried out on composites 25-SG-3 and 25-SG-5 at solids concentrations of 40%, 45%, and 50% using brackish water, so as to approach likely viscosity conditions with site water and compare the outcome with tests involving the addition of known slurry viscosity modifiers. The brackish water was prepared to simulate the mine site conditions based on AbraSilver's specifications. Aquarium-grade Red Sea salt was added to deionized water at a concentration of 1.45 g/L and the pH was adjusted to 5.5 using dilute sulphuric acid to match as closely as possible the site water content in chlorine, sodium, sulphates, and pH.

The viscosity measurements were performed under three conditions:

1. Without chemical additives
2. With sodium silicate at dosages of 250 g/t and 500 g/t
3. With carboxymethyl cellulose (CMC) at dosages of 250 g/t and 500 g/t.

The slurry without any additive also exhibited relatively low viscosity and shear stress, even at the higher density of 50% solids tested. The additives tested failed to provide a noticeable reduction in such readings. Based on these results, slurry densities of 45% and 50% solids were selected for comparative agitated cyanidation tests at a primary grind size of 150 µm K_{80} and without any chemical additives.

Slurries were pre-aerated for 16 hours, then leached at 1,500 mg/L of sodium cyanide for 24 hours, allowed to drift to 750 mg/L, and maintained at a pH of 10.5. Composite 25-SG-3, with head grades of 0.96 g/t Au and 33 g/t Ag, and composite 25-SG-5, with head grades of 0.34 g/t Au and 55 g/t Ag, were involved in the comparative tests. Table 13-3 summarizes the leach tests results.

Table 13-3: ALS Phase 5A - Leach Test Results at Variable Pulp Density

Test ID	Composite ID	Pulp % solids	Extraction (%)		Consumption (kg/t)	
			Au	Ag	NaCN	Lime
10	25-SG-3	45	90.5	54.4	1.8	1.6
57	25-SG-3	50	88.3	64.0	1.4	1.5
28	25-SG-5	45	88.4	77.5	2.0	2.2
58	25-SG-5	50	88.1	86.1	2.0	2.5

Per Table 13-3, gold recoveries varied slightly between the two slurry density conditions while silver recoveries were significantly higher. It is therefore possible to adopt a higher slurry density target or to allow such to drift higher than the PFS criteria of 45% solids under operating transient operating conditions.

13.2.5 SGS Lakefield – Phase 5A

13.2.5.1 Introduction

The program included 14 composites prepared from 24 individual samples previously tested under either AbraSilver's Phase 1 or Phase 2 testwork programs.

The 14 composites included three samples from the JAC deposit and 11 from the four mineral domains found within the Oculto deposit: Silver Enrichment (SE), Shallow Gold (SG), Deep Gold (DG), and North East (NE). Although some composites were made up of a single sample previously tested as such during Phase 2 or Phase 3, most combined the residual weights of multiple individual samples of the same mineral domain in order to have enough material to go through the intended testwork program with as many samples as possible.

13.2.5.2 Metallurgical Testing

Metallurgical testing followed a program scope fairly similar to that extended through the portion of the Phase 5A program, which was carried out simultaneously at ALS. It included head analyses and the effect of grind size, cyanide concentration, and lead nitrate on cyanidation recoveries. Additionally, Merrill-Crowe precipitation testing was completed.

Rheology measurements were performed on two high-clay samples to assess the impact of viscosity modifiers.

13.2.5.3 Effect of Grind Size on Cyanidation Results

Tests were performed at the then standard conditions, involving a P_{80} grind size of 150 µm, 40% w/w solids, and a pH of 10.5 to 11.0 using lime. Samples were pre-aerated for 4 hours with an air/oxygen mixture before cyanide addition. Sodium cyanide was maintained at 1.5 g/L for 24 hours, then at 0.75 g/L for the remainder of the 72 hour leach, with continuous aeration.

Ten composites then underwent cyanidation tests at a coarser P_{80} grind size of 200 µm, following the same cyanidation procedures as the initial tests. Gold leaching was slower at 200 µm, requiring the full 72 hours to maximize recovery, compared to approximately 36 hours at 150 µm. Additionally, residue gold grades were generally 0.04-0.10 g/t lower at the finer grind, representing a 1%-2% improvement in gold recovery. A similar improvement in silver leaching was observed at the finer P_{80}, with final residues typically containing 2-5 g/t less silver, which corresponds to a maximum 5% increase in silver recovery.

13.2.5.4 Cyanide Concentration Testing

Four composites (25-SE-1, 25-DG-1, 25-NE-1, and 25-JAC-2) were selected for cyanidation testing at a higher cyanide concentration, while four others (25-DG-2, 25-DG-4, 25-NE-2, and 25-JAC-3) were tested at a lower concentration. The high concentration tests were conducted at 2.5 g/L of initial cyanide concentration, held for 24 hours and then maintained at 1.5 g/L for the remainder of the 48 hour leach duration.

The four tests conducted under higher cyanide concentration conditions, when compared to the corresponding grind size tests on the same samples, indicated that increased cyanide dosage had minimal impact on the overall results. The most notable difference was in cyanide consumption, which ranged from 0.81 kg/t to 1.67 kg/t for the tests with higher cyanide levels and 0.63 kg/t to 1.05 kg/t for those with the lower levels.

The lower cyanide concentration tests used an initial 0.75 g/L of sodium cyanide, maintained for 24 hours, then allowed to decay and maintained at 0.50 g/L for the remainder of the leach. The four tests under these conditions, when compared to the corresponding grind size tests, showed a slight negative impact on recoveries, with gold's averaging ~1.5% lower and silver's ~2.8% lower. Average final tailings grades were 0.07 g/t higher for gold and 3.3 g/t higher for silver in the lower cyanide tests. Cyanide consumption ranged from 0.37 kg/t to 0.49 kg/t in these tests, as compared to 0.54 kg/t to 1.16 kg/t in the higher cyanide tests.

When combined with the similar test program completed at ALS under the concurrent Phase 5A, the SGS Lakefield work reaches similar conclusions in terms of the limited benefits of straying too much from an initial cyanide addition of 1.5 g/L. The point through the test when the residual level may be allowed to drift lower naturally and at what minimum level it should be held not to be detrimental to the leach kinetics of the metals may be further assessed, with the current 0.75 g/L level held at the 24 hour mark deemed conservative and

possibly leading to higher cyanide consumption than otherwise achievable, but better left as an optimization aspect for the operations to test.

13.2.5.5 Lead Nitrate Cyanidation Testing

Three composites (25-SE-2, 25-DG-3, and 25-SG-2) were selected for cyanidation testing with a 300 g/t lead nitrate ($Pb(NO_3)_2$) dosage. These tests followed the same procedure as the cyanide concentration tests, except that 300 g/t lead nitrate was added prior to the standard sodium cyanide addition (1.50 g/L for the first 24 hours, then allowed to decay and maintained at 0.75 g/L for the remainder of the 48 hour leach).

The tests with lead nitrate, when compared to corresponding grind size tests, showed no positive impact from lead nitrate addition. Gold extractions were slightly lower (by ~1% on average), while silver extractions were minimally affected. Other reagent consumptions were not significantly influenced by lead nitrate addition.

As per the work done under ALS Phase 5A, the indications from the additional tests realized by SGS Lakefield are that lead nitrate has a very limited impact on the resulting extraction of either gold or silver.

13.2.5.6 Cyanide Addition to Grinding Stage Testing

One composite (25-SG-1) was selected for cyanidation testing with cyanide addition during grinding. The objective was to assess whether cyanide addition during grinding could improve silver recovery.

In this test, the sample was pulped in the grinding mill at 50% w/w solids with pH-adjusted water and a NaCN dosage of 1.5 g/L. After milling, the pulp was transferred to a bottle, adjusted to 45% w/w solids, and the NaCN was re-adjusted to 1.5 g/L. The remainder of the test followed the same procedures as the lead nitrate tests.

The results were similar to the equivalent grind evaluation test, with low silver extractions in both cases, at 30% for no cyanide to grind and 46% with it. The direct gold head grade for 25-SG-1 was only 0.03 g/t, making the reported gold recoveries insignificant. Adding cyanide to the grinding stage substantially increased its consumption, from 0.43 kg/t for the base case to 1.72 kg/t.

As per a similar testing completed under the ALS Phase 5A test program, there is no incentive to move forward with initial cyanide addition, from the beginning of the leach circuit and into the grinding environment.

13.2.5.7 Merrill-Crowe Testing

The Merrill-Crowe process requires complete clarification and de-aeration of the solution prior to zinc dust addition. All operations were conducted under an inert nitrogen atmosphere. For each test, the solution was de-aerated by sparging with nitrogen for 30 minutes in a semi-sealed vessel, followed by the addition of lead nitrate and then zinc dust. Nitrogen flow was maintained for an additional 30 minutes to ensure thorough mixing and prevent oxygen re-absorption. Upon completion, the contents were filtered under oxygen free conditions.

Merrill-Crowe tests were conducted on pregnant leach solution (PLS) blends of multiple leach tests with both NE and JAC samples using zinc dust additions at 1.5, 3, and 5 times the stoichiometric requirement, and with lead nitrate usually added at one-quarter of the zinc dust mass. For tests on diluted PLS, the dilution was made with pH adjusted water at a various water-to-PLS ratio, to simulate the eventual dilution brought in a CCD circuit, as required to minimize losses of metal in the leach solution. These conditions might be considered a worst case scenario in terms of dissolved metals concentration, as an actual CCD circuit would leave more of these metals, now left with the pulp fraction at the laboratory-scale, into the stream entering the Merrill-Crowe circuit instead. Lower metal concentrations are expected to require higher stoichiometric ratios of zinc in order to achieve equivalent cementation efficiency.

Two tests were conducted on each diluted sample. For JAC, the first test (MC-2E) mirrored MC-1E conditions, while the second test (MC-2H) repeated MC-2E, but with a total feed cyanide level adjusted at 262 mg/L, from a minimum targeted concentration of 250 mg/L.

Tests on threefold diluted solutions showed significantly lower recoveries (45 to 65%), as can be expected from reduced initial gold and silver concentrations. Table 13-4 presents results for the NE PLS sample while Table 13-5 covers those for the JAC.

With the undiluted NE PLS sample, gold and silver precipitation efficiencies improved with increasing zinc dust addition, from approximately 70% at a 1.5:1 stoichiometric ratio to 90 to 95% at a 5:1 ratio, indicating that higher zinc additions are required to achieve near-complete (~100%) recovery. Dilution of the PLS in a 3:1 ratio lowered the efficiencies seen. The resulting fourfold dilution of the total cyanide (CN) content, through the addition of water, was thought to be the cause of such an outcome. This led to test MC-1H, where fresh CN was added to reach a total CN level of about half what was seen in the undiluted PLS. Although the efficiencies were pushed upwards, the resulting 54% for gold and 66.5% for silver are still insufficient to foresee a decrease in the zinc dust addition requirements below the 5:1 ratio on diluted solution.

With the JAC PLS sample, very high efficiencies were achieved for gold, even at the 1.5:1 ratio or from the lowest 3:1 ratio tested with diluted PLS. With undiluted solution, cementation efficiency was near perfect for silver as well, starting from a 3:1 zinc dust addition ratio. Once diluted, the silver efficiency decreased somewhat and a marginal increase in the total CN did not help to bring it back near the 100% level seen on the undiluted solution.

These mixed results indicate that further investigation is required to establish conditions for achieving near complete metal recoveries.

Table 13-4: SGS Lakefield Phase 5A - Merrill-Crowe Testing of a PLS Blend from NE Samples

Test ID	Feed Dilution Water:PLS Ratio	Zn Stoich. Addition Ratio	Zn:Lead Nitrate Addition Ratio	Solution Assays			Recovery	
				Au (mg/L)	Ag (mg/L)	CN_T (mg/L)	Au (%)	Ag (%)
Feed	-	-	-	2.00	19.1	533	-	-
MC-1A		1.5	4:1	0.65	5.6	624	67.5	70.7
MC-1B		3	4:1	0.28	1.7	599	86.0	91.4
MC-1C		5	4:1	0.20	1.0	661	90.0	94.9
MC-1F		3	4:1	0.70	7.2	591	65.0	62.6
MC-1G		5	4:1	0.20	2.2	591	90.0	88.6
Diluted Feed	3:1	-	-	0.50	4.8	133	-	-
MC-1E		3	4:1	0.28	2.1	155	44.0	56.6
Diluted Feed	3:1	-	-	0.50	4.8	275	-	-
MC-1H		5	4:1	0.23	1.6	265	54.0	66.5

Table 13-5: SGS Lakefield Phase 5A - Merrill-Crowe Testing of a PLS Blend from JAC Samples

Test ID	Feed Dilution Water:PLS Ratio	Zn Stoich. Addition Ratio	Zn:Lead Nitrate addition ratio	Solution Assays			Recovery	
				Au (mg/L)	Ag (mg/L)	CN_T (mg/L)	Au (%)	Ag (%)
Feed	-	-	-	0.29	94.2	587	-	-
MC-2A		1.5	4:1	<0.01	2.6	624	100.0	97.2
MC-2B		3	4:1	<0.01	0.3	686	100.0	99.7
MC-2C		5	4:1	<0.01	0.2	661	100.0	99.8
MC-2F		3	10:1	<0.05	3.3	657	100.0	96.5
MC-2G		2	4:1	<0.05	11.1	624	100.0	88.2
Diluted Feed	3:1	-	-	0.07	23.6	147	-	-
MC-2E		3	4:1	<0.05	1.0	160	100.0	95.7
Diluted Feed	3:1	-	-	0.07	23.6	262	-	-
MC-2H		3	4:1	<0.05	2.5	245	100.0	89.3

13.2.5.7.1 High-Clay Samples Rheology Testing

Two composite samples (25-SG-1 and 25-JAC-3), both containing higher clay content, were submitted for viscosity optimization testing at various pulp densities. Using the optimum P_{80} grind size of approximately 150 µm, slurry viscosity was evaluated on “as-is” ground material and with the addition of viscosity modifiers, including sodium silicate, cellulose-based Depramin C, and Calgon. All samples were pulped to the target density of 45% solids using synthetic process water, to emulate the probable quality and viscosity of the site’s fresh water.

Rheology measurements indicated that sample 25-SG-1 settled rapidly even at 50% w/w solids; therefore, tests planned at lower slurry densities were not performed. Only the unsheared portion of the 50% solids test was valid and reported. Sample 25-JAC-3 remained stable across all tested solids contents, with shear rates ranging from 0 to 300 s^{-1} and a constant shear rate of 200 s^{-1}, as specified. Both samples exhibited Bingham plastic behavior with low yield stress and viscosity. Additions of viscosity modifiers had no measurable impact on the already low viscosities.

As with the ALS Phase 5A work, the results indicated that a slurry density of 50% solids could be sustained within the leach circuit without an undue increase to the viscosity or agitator power requirements.

13.2.5.8 Phase 5A Conclusions and Recommendations

The conclusions and recommendations from this testwork program are as follows:

- The standard agitated leach test conditions would be as follows:
  - Target grind size of P80 of 150 µm
  - Pulp density of 45% w/w solids, but with leeway to allow an increment towards 50%
  - pH maintained between 10.5 and 11.0 using lime
  - Leach retention time of 36 hours
  - Sodium cyanide (NaCN) concentration initially set at 1.5 g/L for 24 hours, then allowed to decay and maintained at 0.75 g/L for the remainder of the leach
  - Pre-aeration for 4 hours using air/oxygen sparging until the DO reaches 10 to 12 ppm. As the intent of this pre-aeration stage is to reintroduce DO in the slurry, after the period spent in a laboratory-scale enclosed grinding mill where DO depletion can occur from reactions with the slurry minerals, and thus simulate DO inflow as with open-ended industrial grinding mills and subsequent retention within a pre-leach thickener before entering the leach circuit proper, the duration of this lab-scale pre-aeration stage and the level of DO to be met are subject to further revision.
  - Air/oxygen sparging during leach to a target DO of 10 to 12 ppm. Lower levels are to be tested for reduction of the eventual associated oxygen cost, which will be supplied instead of air due to the low atmospheric pressure at the Project’s site altitude.
- Average gold extractions under semi-optimized conditions ranged from 76% to 92% while silver extractions ranged from 48% to 89%, with silver seemingly more sensitive to a combination of mode of occurrence and texture of the silver-bearing material.
- No beneficial effect on metal recoveries was observed when lead nitrate was added to the leach circuit or from cyanide addition to the grinding mill instead of leach.

- Additional cyanidation tests are recommended to refine operating conditions and maximize gold and silver extraction, especially for samples that performed poorly under the established standard conditions.
- Preliminary Merrill-Crowe testing indicated that gold and silver can be recovered efficiently from cyanide leach solutions. However, in some cases, high zinc dosages may be necessary (>5x the stoichiometric requirement). This is particularly true after PLS dilution because it simulates the wash water addition during an industrial scale CCD process. Future testwork should focus on leach solutions that perform poorly under standard Merrill-Crowe conditions to determine the optimum parameters for near-complete (~100%) gold and silver recovery and identify any interfering elements present in the PLS.
- Rheology testing showed that thickener underflows exhibited a Bingham flow behavior and were generally thixotropic, characterized by low yield stress and low viscosities. None of the tested viscosity modifiers had any effect on the already low viscosities. Settling rates were observed to be rapid.

13.2.6 SGS Lakefield – Phase 5B1 Test Program

The following sections summarize the Phase 5B1 testwork program and results. For full details, refer to the Diablillos Deposit report prepared by SGS Lakefield, dated 20 March 2026.

The objectives of this program were to evaluate the response of fresh Diablillos samples, as received from AbraSilver, for the purposes of expanding the database of hardness and cyanidation test results under optimum conditions through the inclusion of more samples from the most significant mineral domains in the deposits. The lithology and alteration associated with samples from the same mineral domain were also to be assessed as potential markers of the cyanidation response variability observed earlier. In this regard, care was taken to limit the testing to one alteration per sample material to be included within each single drill hole intercept, in addition to providing a good spatial distribution and head grade variability.

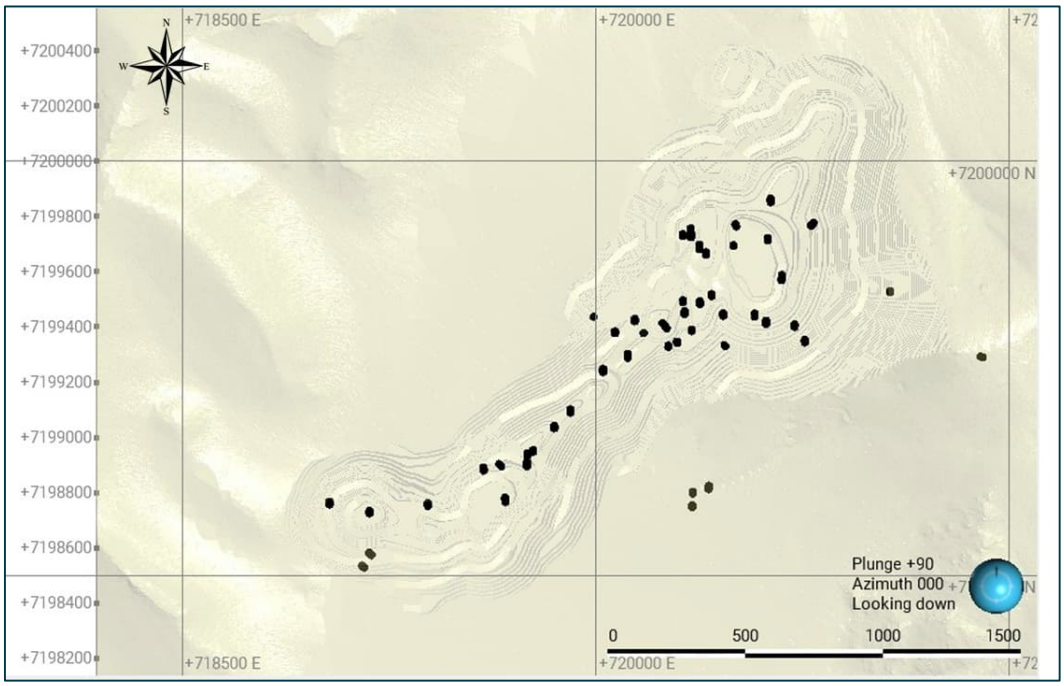
The metallurgical testwork program included the following:

- Sample characterization of a Year 1-5 composite, including mineralogical assessment using QEMSCAN and TIMA analysis
- Whole ore cyanidation
- Extended gravity recoverable gold and associated silver recovery
- Cyanidation of gravity tail products from SGS Lakefield's gravity concentration procedure
- Merrill–Crowe cementation tests
- Oxygen uptake evaluation
- Cyanide destruction testing
- Rheology of leach and detoxified tails.

13.2.6.1 Sample Receipt and Preparation

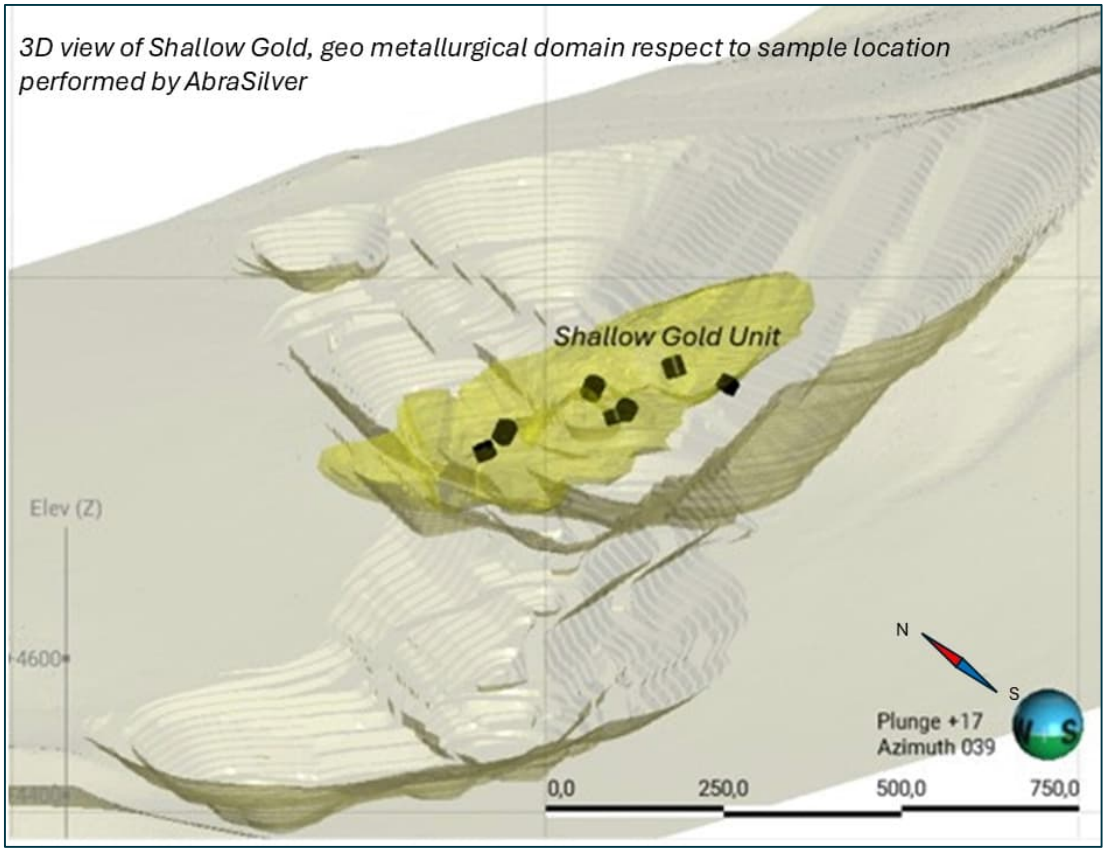
In June and July 2025, four shipments of samples from AbraSilver's Diablillos deposit were received at SGS Lakefield for the test program. These included 388 individual quarter-core intervals. From the material received, SGS Lakefield prepared 52 single drill core intercept samples, 10 E-GRG composites, and seven metallurgical (MET) composites.

Figure 13-13 illustrates the relative position of each of the 52 samples involved in the Phase 5B programs, with the three Sombra samples shown outside of the PFS based pit limits on the south side of the Oculto deposit. Figure 13-4 to Figure 13-8, in turn, present the spatial distribution from a 3D model of the deposits.



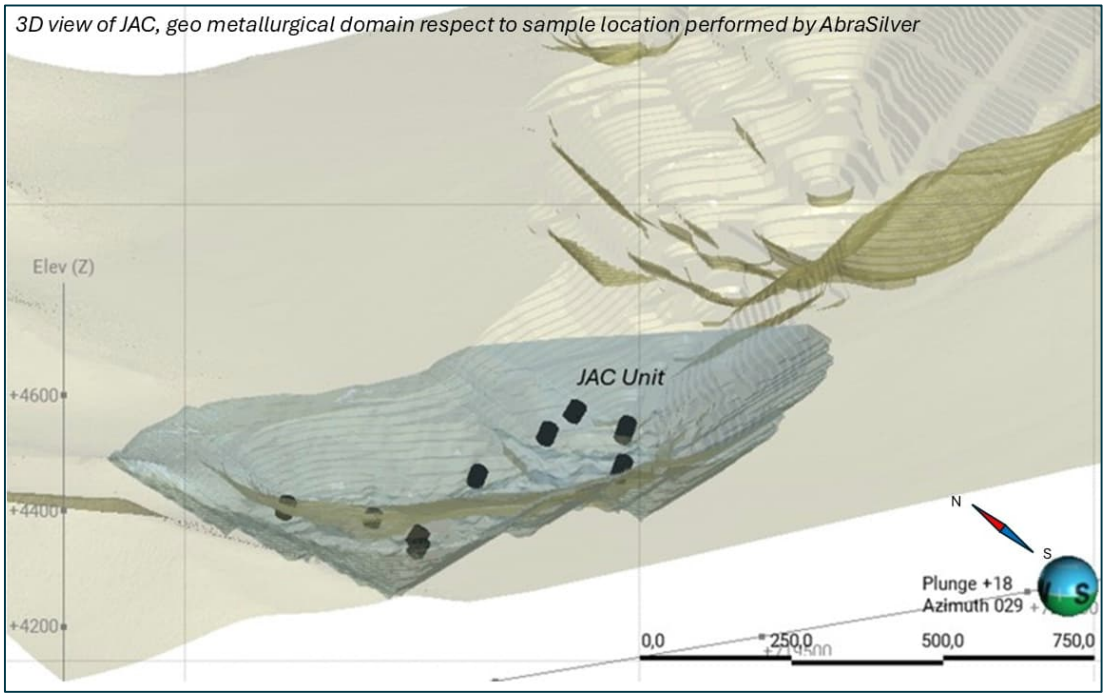
Source: AbraSilver, 2026

Figure 13-3: Relative Location of Phase 5B Drill Hole Intercepts within Oculto and JAC Deposits



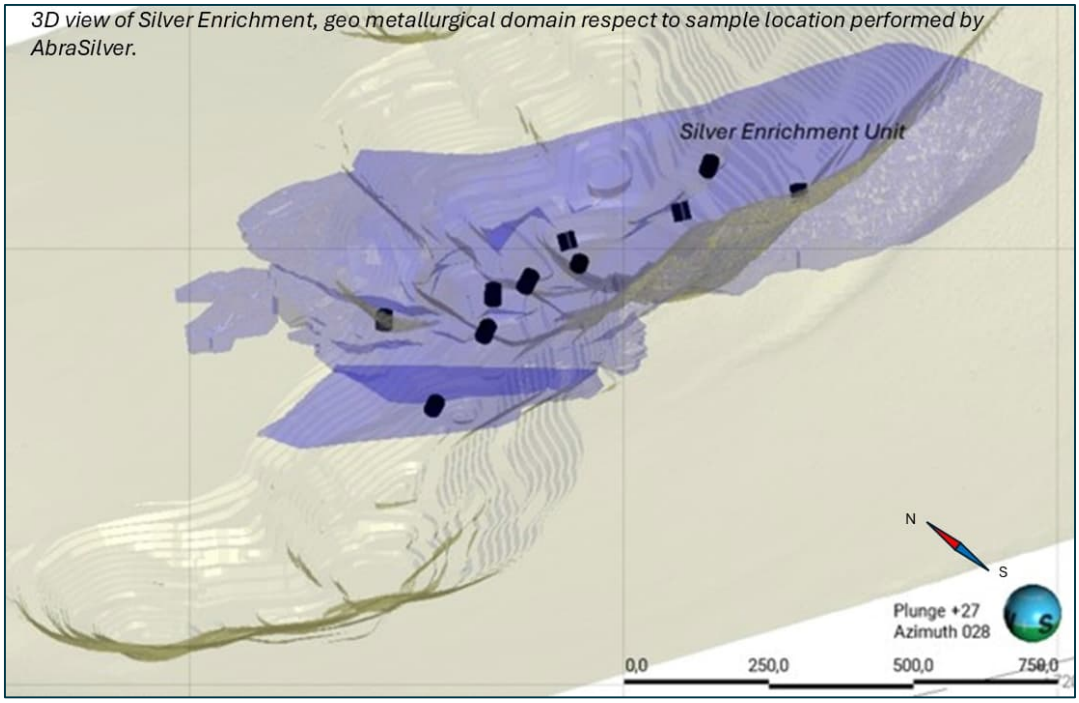
Source: AbraSilver, 2026

Figure 13-4: Phase 5B - SG Sample Locations within Mineral Domain (AbraSilver)



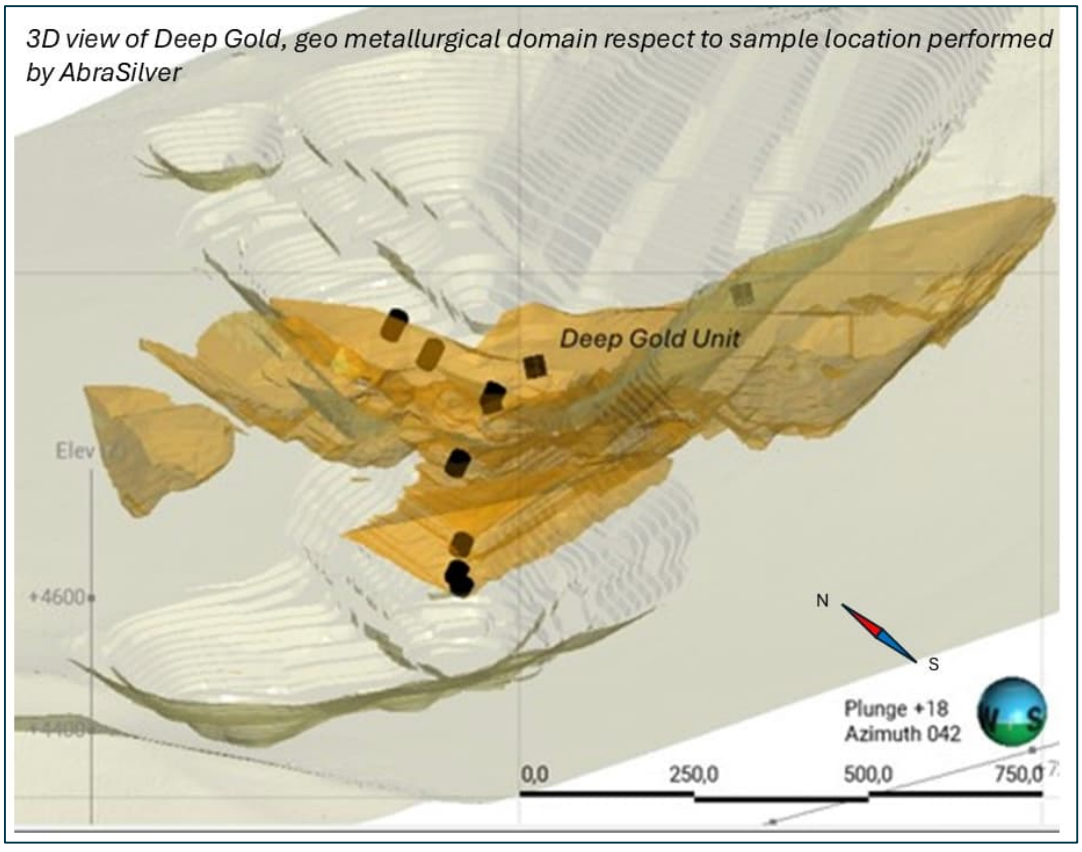
Source: AbraSilver, 2026

Figure 13-5: Phase 5B - JAC Sample Locations within Mineral Domain (AbraSilver)



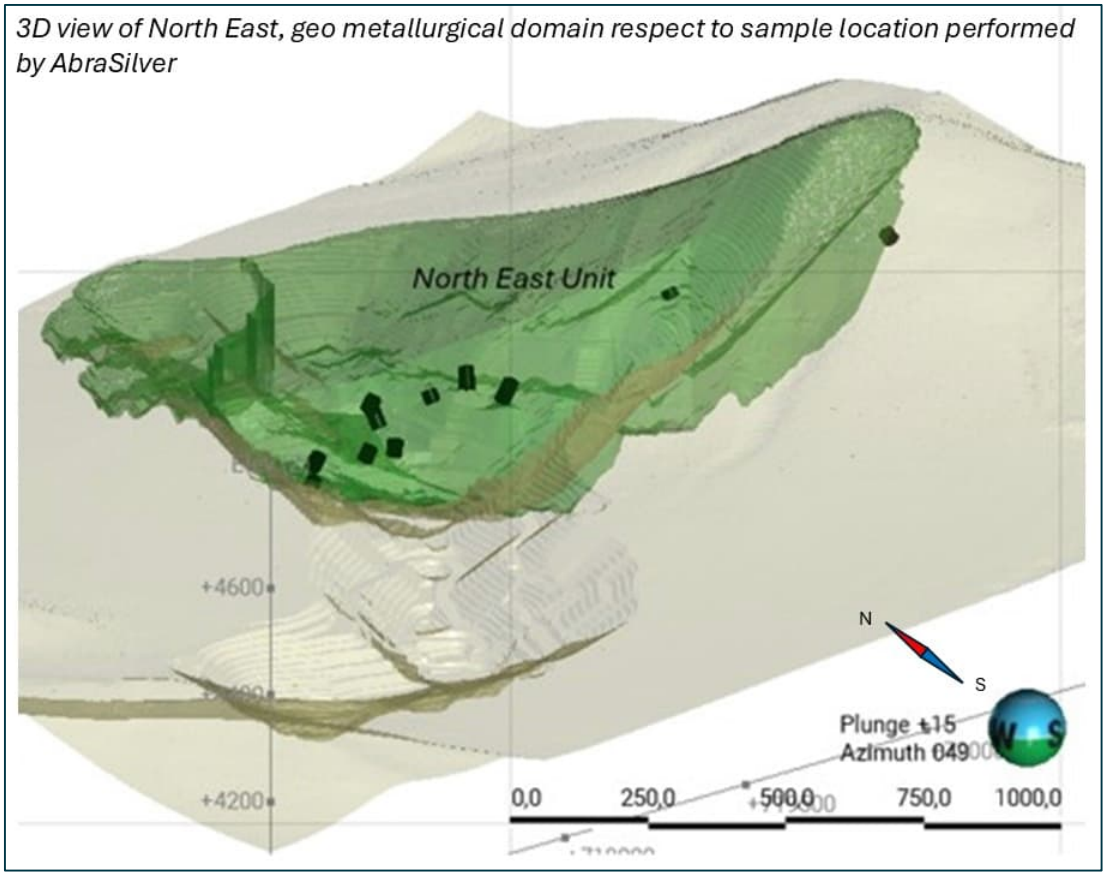
Source: AbraSilver, 2026

Figure 13-6: Phase 5B – SE Sample Locations within Mineral Domain



Source: AbraSilver, 2026

Figure 13-7: Phase 5B - DG Sample Locations within Mineral Domain



Source: AbraSilver, 2026

Figure 13-8: Phase 5B – NE Sample Locations within Mineral Domain

During composite preparation, the variability samples were stage crushed to generate material for 14 SMC tests and 45 Bond ball mill grindability tests. Due to the eventual testwork scope resulting from involving all these samples at once, the overall foreseen program was divided into two stages (Phase 5B1 and Phase 5B2). Samples were thus deployed during either one or the other stage, except for three variability samples from the Sombra (SOM) mineral domain, which is not included in the DFS mine plan and will therefore not be discussed further.

A Year (YR) 1–5 Composite was also prepared by blending material from 16 drill hole samples collected across the five mineral domains and was meant to emulate the first five years of plant feed blend, as expected at that time. Per such indications, the YR 1-5 composite held 19.7% of JAC, 10.4% of SG, 40.5% of SE, 20.4% of DG, and 8.9% of NE as individual weight fractions.

Nine additional composites were prepared for the intended E-GRG test program. These composites were produced by blending selected individual drill interval samples from each mineral domain while also incorporating a single alteration into each resulting E-GRG composite.

Finally, six MET composites were prepared. These MET composites were produced along mineral domain lines and contained 30-40 kg of material each to cover specific additional metallurgical testwork following, at times, gravity separation ahead of cyanidation. Therefore,

the testing applied to these tails may also have included Merrill–Crowe processing, cyanide destruction (CND), solid–liquid separation and rheology testing.

13.2.6.2 Comminution Test Program

Fourteen composites were submitted for SMC testing, while 26 of the 45 samples prepared were submitted for Bond Ball Mill grindability (BWi) testing.

13.2.6.3 SAG Mill Comminution (SMC) Test

SMC tests were conducted on 10 Oculto samples from the NE mineral domain and the YR 1-5 composite provided. All the samples were initially picked from the NE mineral domain because it was known to be the main contributor to the resources tonnage but had limited historical data available to qualify it.

The A×b parameter values ranged from 39.2 to 68.7 for the NE samples and was 72.3 for the YR 1-5 composite. When compared with the JK Tech database, these values correspond to hardness percentiles ranging from 12 to 64, indicating that the material is classified as soft to moderately hard.

13.2.6.4 Bond Ball Mill Grindability Test

The Bond ball mill grindability work index (BBWi) determinations were also carried out with a closing screen of 65 Mesh, versus the historical data generated with a 100 Mesh screen, as the typical product from such selection was nearer the adopted 150 µm grind target, at ~160 µm. The earlier work yielded product at a finer ~120 µm, thus possibly overestimating the grinding power.

The results for the 25 Phase 5B1 samples tested and the YR 1-5 composite ranged from 11.8 kWh/t to 27.3 kWh/t.

13.2.6.5 Mineralogy

Thirty-five drill core interval composites and three MET composites were submitted for head characterization analysis, completed by whole-rock-analysis (WRA) chemistry.

In addition, the YR 1-5 composite and some selected leach residues were involved in additional mineralogical investigations.

13.2.6.5.1 QEMSCAN Mineralogy

The YR 1–5 Composite was submitted for automated mineralogical analysis using QEMSCAN technology. The results indicate that the sample is composed predominantly of quartz and sulphates, with trace amounts of pyrite. Mineral abundance classifications identified quartz as a major component (>30%), alunite in moderate concentrations (10-30%), and iron oxides and jarosite in minor amounts (2-10%). Various additional minerals were present only in trace quantities (<2%).

13.2.6.5.2 TIMA-X Mineralogy (Alternative Automated Mineralogical Analysis)

The YR 1–5 Composite and four cyanide leach residues with low recoveries were submitted for automated mineralogy analysis using TIMA technology.

For the YR 1–5 Composite, both gold and silver bearing minerals were evaluated for mineral associations, liberation characteristics, and grain abundance. Native gold was the only gold bearing phase identified, with all four observed grains measuring less than 3 µm in size. The primary silver minerals detected were chlorargyrite and iodargyrite, with several additional silver phases present in minor to trace amounts. Silver mineral liberation was generally high, measured at 92.1%, 87.4%, 77.5%, and 97.3% for the +150 µm, –150/+75 µm, –75/+20 µm, and –20 µm size fractions, respectively.

Selected leach residues, from tests CN-1, CN-6, CN-14 and CN-33, were submitted for silver specific TIMA analysis to shed light on the source of poor leach recovery. Each sample was screened into the +150 µm, –150/+75 µm, –75/+20 µm, and –20 µm size fractions, and six polished sections from each fraction were prepared for silver scanning and accompanying assay.

The detection limits and reported silver assay results varied considerably. All size fractions from CN-1 and the two coarse fractions from CN-6 were reported as below the 50 g/t detection limit, whereas most other fractions returned values between 10 g/t and 26 g/t. Silver bearing particle statistics were considered reasonable for the CN-1 fractions and the two coarse fractions from CN-33. In contrast, particle statistics for CN-6 and CN-14 were poor, with no silver minerals detected in the –75/+20 µm and –20 µm fractions for either sample. Only a single silver grain was identified in the CN-14 sample. These outcomes limit the validity of the exercise.

The silver deportment and liberation data indicated that CN-1 contained a mixture of silver minerals that were not recovered, with liberation values of 1% in the +150 µm fraction, 46% in the –150/+75 µm fraction, 0% in the –75/+20 µm fraction, and 82% in the –20 µm fraction. In the CN-6 residue, iodargyrite was the only silver mineral detected, occurring exclusively as locked particles. The single silver grain identified in the CN-14 residue was acanthite, which was also found locked within silicate minerals. For CN-33, iodargyrite was the primary silver mineral observed, again present as locked particles.

13.2.6.5.3 Extended Gravity Recoverable Gold (E-GRG)

The composites prepared for E-GRG testwork were meant to cover specific alteration types within each of the mineral domain. As such, composites bearing the most prevalent argillic (ARG) alteration were targeted by this Phase 5B1 testwork program, leaving others with the secondary alterations for Phase 5B2.

13.2.6.5.4 SE-ARG Composite E-GRG

E-GRG testing on the SE-ARG Composite determined overall E-GRG recoveries of 19.0% for gold and 19.5% for silver. The calculated head grades from the E-GRG test - 0.09 g/t gold and 19.4 g/t silver - aligned closely with the screen fraction analysis results of 0.10 g/t gold and 19.7 g/t silver.

Across the three sequential gravity stages, the gold and silver distributions were as follows:

- Stage 1 (P_{80} 578 µm): 9.0% gold and 14.4% silver
- Stage 2 (P_{80} 187 µm): 6.6% gold and 4.4% silver
- Stage 3 (P_{80} 77 µm): 3.4% gold and 0.6% silver

The screen fraction analysis (SFA) for the head sample and the E-GRG test results are provided in Table 13-6 and Table 13-7. Per Table 13-6, nearly half of the gold is quite fine, reporting in the -25 µm size fraction. Both gold and silver do show some ~25-30% in the +35 Mesh range but these values align with the mass distribution and are not indicative of a specific size concentration.

Table 13-6: SE-ARG Composite Head SFA

Particle Size		Mass (%)	Distribution, %	
Mesh	µm		Au	Ag
20	850	0.6	1.1	0.6
28	600	15.4	11.1	12.7
35	425	15.9	13.1	12.9
48	300	11.7	6.0	9.1
65	212	8.5	5.2	9.2
100	150	6.1	2.5	6.3
150	106	4.8	3.0	7.0
200	75	4.2	1.7	5.1
270	53	3.5	1.5	6.9
400	38	3.8	2.7	7.8
500	25	3.3	4.1	6.2
-500	-25	22.2	48.0	16.1
Total		100.0	100.0	100.0

Table 13-7: SE-ARG Composite E-GRG Results

Grind Size P_{80} (µm)	Product ID	Mass		Gold		Silver	
		g	%	Assay (g/t)	Dist'n (%)	Assay (g/t)	Dist'n (%)
P_{80} 578 µm	Stage 1 Conc Sampled Tails	66.5 292	0.33 1.46	2.40 0.13	9.0 2.2	842 16.2	14.4 1.2
P_{80} 187 µm	Stage 2 Conc Sampled Tails	68.4 244	0.34 1.22	1.72 0.08	6.6 1.0	252 15.4	4.4 1.0
P_{80} 77 µm	Stage 3 Conc Final Tails	55.5 19,273	0.28 96.4	1.09 0.07	3.4 77.7	40.9 15.8	0.6 78.3
Totals (Head) Knelson Conc		20,000 190	100.0 0.95	0.09 1.77	100.0 19.0	19.4 396	100.0 19.5

13.2.6.5.5 DG-ARG Composite E-GRG

E-GRG testing on the DG-ARG Composite determined overall E-GRG recoveries of 16.7% for gold and 4.1% for silver. The calculated head grades from the E-GRG test of 0.38 g/t gold and 19.1 g/t silver showed close agreement with the screen fraction analysis results of 0.37 g/t gold and 20.1 g/t.

The three sequential gravity concentration stages produced the following gold and silver distributions:

- Stage 1 (P_{80} 564 µm): 7.2% gold and 1.9% silver
- Stage 2 (P_{80} 201 µm): 5.0% gold and 1.7% silver
- Stage 3 (P_{80} 80 µm): 4.5% gold and 0.5% silver

The screen fraction analysis for the head sample and E-GRG test results are presented in Table 13-8 and Table 13-9. The behaviour of the metal units was fairly similar to what was seen in Table 13-6 for the SE-ARG sample, with nearly half the gold reporting to the pan fraction and the rest of the distribution of gold and silver units espousing that of the weight.

Table 13-8: DG-ARG Composite Head SFA

Particle Size		Mass	Distribution (%)	
Mesh	µm	(%)	Au	Ag
+20	850	1.0	1.2	2.0
28	600	16.0	13.4	16.7
35	425	14.3	11.2	14.2
48	300	12.1	9.4	11.9
65	212	10.8	7.3	11.0
100	150	9.0	4.8	8.3
150	106	7.3	3.6	6.2
200	75	6.4	3.1	5.7
270	53	5.2	2.8	4.3
400	38	4.1	3.2	3.8
500	25	2.8	2.4	3.0
-500	-25	11.0	37.5	13.0
Total		100.0	100.0	100.0

Table 13-9: DG-ARG Composite E-GRG Results

Grind Size P_{80} (µm)	Product ID	Mass		Gold		Silver	
		g	%	Assay (g/t)	Dist'n (%)	Assay (g/t)	Dist'n (%)
P_{80} 564 µm	Stage 1 Conc	73.6	0.37	7.45	7.2	97.0	1.9
	Sampled Tails	254	1.27	0.37	1.2	17.9	1.2

Grind Size P_{80} (µm)	Product ID	Mass		Gold		Silver	
		g	%	Assay (g/t)	Dist'n (%)	Assay (g/t)	Dist'n (%)
P_{80} 201 µm	Stage 2 Conc Sampled Tails	81.6 269	0.41 1.35	4.59 0.33	5.0 1.2	80.3 18.6	1.7 1.3
P_{80} 80 µm	Stage 3 Conc Final Tails	54.2 19,267	0.27 96.3	6.26 0.32	4.5 80.9	33.4 18.5	0.5 93.4
Totals (Head) Knelson Conc		20,000 209	100.0 1.05	0.38 6.03	100.0 16.7	19.1 74.0	100.0 4.1

13.2.6.5.6 NE-ARG Composite E-GRG

E-GRG testing on the NE-ARG Composite determined overall E-GRG recoveries of 31.6% for gold and 6.0% for silver. The calculated head grades from the E-GRG test of 0.72 g/t gold and 19.8 g/t silver correlated closely with the screen fraction analysis results of 0.66 g/t gold and 20.0 g/t silver.

The three sequential gravity concentration stages produced the following gold and silver distributions:

- Stage 1 (P_{80} 577 µm): 7.4% gold and 2.3% silver
- Stage 2 (P_{80} 238 µm): 15.8% gold and 3.1% silver
- Stage 3 (P_{80} 74 µm): 8.3% gold and 0.6% silver

The screen fraction analysis for the head sample and the E-GRG test results are summarized in Table 13-10 and Table 13-11, respectively.

Table 13-10: NE-ARG Composite Head SFA

Particle Size		Mass (%)	Distribution (%)	
Mesh	µm		Au	Ag
+20	850	2.6	5.8	6.4
28	600	14.5	14.9	16.7
35	425	16.6	14.3	15.7
48	300	13.8	8.7	11.6
65	212	10.6	6.4	7.4
100	150	8.1	4.7	5.5
150	106	6.1	3.5	4.4
200	75	5.1	2.6	4.0
270	53	3.9	2.3	3.1
400	38	3.1	2.7	2.7
500	25	2.4	3.9	2.4
-500	-25	13.3	30.1	20.0
Total		100.0	100.0	100.0

Table 13-11: NE-ARG Composite E-GRG Results

Grind Size P_{80} (µm)	Product ID	Mass		Gold		Silver	
		g	%	Assay (g/t)	Dist'n (%)	Assay (g/t)	Dist'n (%)
P_{80} 577 µm	Stage 1 Conc Sampled Tails	70.7 287	0.35 1.43	15.2 0.67	7.4 1.3	131 18.3	2.3 1.3
P_{80} 238 µm	Stage 2 Conc Sampled Tails	69.2 262	0.35 1.31	33.0 0.56	15.8 1.0	176 18.5	3.1 1.2
P_{80} 74 µm	Stage 3 Conc Final Tails	56.2 19,256	0.28 96.3	21.3 0.50	8.3 66.1	42.8 18.8	0.6 91.4
Totals (Head) Knelson Conc		20,000 196	100.0 0.98	0.72 23.2	100.0 31.6	19.8 121	100.0 6.0

13.2.6.6 Cyanidation Testwork

A single 1 kg bottle roll cyanidation test was performed on twenty-six samples and on three MET composites to evaluate gold and silver extraction, as well as lime and cyanide consumption. An additional test was conducted on the YR 1–5 composite.

All cyanidation tests were carried out at a target grind size of P_{80} of 150 µm and a pulp density of 45 % solids (w/w) using tap water. The pH was maintained between 10.5 and 11.0 using lime added as calcium hydroxide. The bottles were placed on rolling agitators and air was sparged for 4 hours to pre aerate the pulp, targeting a DO concentration of 6-8 mg/L. Following pre aeration, sodium cyanide (NaCN) was added to reach 1.5 g/L, which was maintained for 24 hours before being allowed to naturally decay to 0.75 g/L. This concentration was then maintained for the remainder of the 48 hour leach period. During leaching, an air/oxygen mixture was continuously sparged to maintain DO levels between 8 to 12 mg/L.

The final gold extractions for the thirty cyanidation tests ranged from 58% to 98% after 48 hours of leaching, with 26 of the 30 tests achieving gold extractions of 80% or higher. Corresponding tailings gold grades ranged from below the detection limit of 0.01 g/t to 0.43 g/t. Silver extractions ranged from 26% to 91%, with 18 tests returning silver extractions of 70% or greater. Good agreement was observed between the calculated and direct head grades for both gold and silver.

Sodium cyanide consumption ranged from 0.46 kg/t to 4.70 kg/t, with 22 of the 30 tests reporting consumption below 1.0 kg/t. The high value is the result of a high copper content in JAC MET composite (CN-33), at 0.175%Cu, which is much higher than the indicated 200 g/t used for delimiting the oxide and sulphide boundary of the deposit. Sample 04_MET_F5_NE also surpassed this limit, with 712 g/t Cu. Lime consumption ranged from 0.37 kg/t to 2.83 kg/t, with 28 tests reporting consumption below 2.0 kg/t. Detailed summaries for each cyanidation test are provided in Table 13-12 and Table 13-13.

Table 13-12: Phase 5B1 - Gold Extraction by Cyanidation

Test ID	Sample ID	CN Residue P_{80} (µm)	% Au Extraction (Hours)							Residue Au (g/t)	Au Head Grade (g/t)	
			2	4	8	12	24	36	48		Calc	Direct
CN-1	01-MET-F5-JAC	149	79	80	81	83	84	86	87	<0.01	<0.08	<0.01
CN-2	02-MET-F5-JAC	158	79	80	82	83	84	86	87	<0.01	<0.08	<0.01
CN-3	05-MET-F5-JAC	147	52	53	54	55	56	57	58	<0.05	<0.12	0.09
CN-33	06-MET-F5-JAC	166	49	70	71	73	64	73	84	<0.02	0.13	0.08
CN-4	08-MET-F5-JAC	150	77	83	85	86	87	89	90	<0.01	<0.10	<0.01
CN-5	10-MET-F5-JAC	144	43	59	90	81	78	83	81	0.06	0.32	0.33
CN-6	02-MET-F5-SE	153	67	76	64	89	90	88	89	0.05	0.47	0.38
CN-7	03-MET-F5-SE	159	56	57	81	82	84	85	86	0.02	0.11	0.10
CN-8	08-MET-F5-SE	147	74	75	77	78	79	81	82	0.02	<0.08	0.02
CN-9	01-MET-F5-DG	189	68	72	80	81	86	84	86	0.05	0.35	0.36
CN-19	03-MET-F5-DG	167	74	81	82	82	82	83	83	0.14	0.82	0.64
CN-11	04-MET-F5-DG	158	67	72	74	78	79	80	81	0.42	2.16	1.92
CN-12	06-MET-F5-DG	167	25	43	80	85	87	86	87	0.13	0.99	0.90
CN-13	07-MET-F5-DG	158	48	54	55	61	62	63	59	0.10	0.23	0.14
CN-14	08-MET-F5-DG	160	77	94	100	100	100	98	98	0.02	1.04	1.26
CN-15	09-MET-F5-DG	171	29	44	58	66	81	82	82	0.43	2.38	2.19
CN-16	10-MET-F5-DG	168	79	91	96	95	93	95	94	0.03	0.40	0.37
CN-17	11-MET-F5-DG	169	70	77	80	81	80	82	82	0.13	0.72	0.60
CN-18	02-MET-F5-NE	157	74	81	83	84	86	81	89	0.03	0.22	0.18
CN-10	03-MET-F5-NE	134	56	75	81	82	83	84	83	0.32	1.85	1.89
CN-20	04-MET-F5-NE	148	69	70	71	72	73	74	75	0.03	<0.12	0.08
CN-21	06-MET-F5-NE	144	55	65	62	59	60	66	62	0.11	0.29	0.26
CN-22	08-MET-	157	79	79	81	78	77	81	83	0.06	0.32	0.30

Test ID	Sample ID	CN Residue P_{80} (µm)	% Au Extraction (Hours)							Residue Au (g/t)	Au Head Grade (g/t)	
			2	4	8	12	24	36	48		Calc	Direct
	F5-NE											
CN-23	09-MET-F5-NE	137	38	61	75	78	79	79	81	0.28	1.49	1.38
CN-24	10-MET-F5-NE	142	71	80	82	82	81	84	84	0.25	1.58	1.38
CN-25	11-MET-F5-NE	152	70	87	94	96	92	95	95	0.05	0.92	0.86
CN-27	JAC MET Comp	129	69	69	70	72	73	88	89	0.01	0.09	0.08
CN-28	Oculto MET Comp	161	68	77	81	82	83	85	85	0.17	1.11	0.95
CN-29	YR 1-5 MET Comp	148	70	84	87	89	86	90	89	0.06	0.54	0.51
CN-26	YR 1-5 Comp	147	70	87	93	90	91	91	90	0.06	0.55	0.46

Table 13-13: Phase 5B1 - Silver Extraction by Cyanidation

Test ID	Sample ID	CN Residue P_{80} (µm)	% Ag Extraction (Hours)							Residue (Ag g/t)	Ag Head Grade (g/t)	
			2	4	8	12	24	36	48		Calc	Direct
CN-1	01-MET-F5-JAC	149	41	45	49	53	56	58	61	64.8	165	144
CN-2	02-MET-F5-JAC	158	84	85	84	85	87	85	84	6.0	38.1	41.0
CN-3	05-MET-F5-JAC	147	61	64	71	72	69	70	71	11.0	37.6	36.2
CN-33	06-MET-F5-JAC	166	59	64	64	65	69	70	71	28.0	95.0	83.5
CN-4	08-MET-F5-JAC	150	58	59	60	61	63	64	64	22.2	61.9	56.3
CN-5	10-MET-F5-JAC	144	69	71	69	73	75	78	77	78.8	347	344
CN-6	02-MET-F5-SE	153	55	57	59	60	61	61	61	20.2	51.8	52.0
CN-7	03-MET-F5-SE	159	49	52	57	61	63	64	66	16.8	48.6	45.0
CN-8	08-MET-F5-SE	147	52	52	54	53	54	54	52	19.6	41.0	40.1
CN-9	01-MET-F5-DG	189	77	76	78	79	80	80	81	11.3	59.8	59.0
CN-19	03-MET-F5-DG	167	50	51	53	53	54	55	55	7.0	15.4	15.0
CN-11	04-MET-F5-DG	158	71	74	74	70	78	81	79	23.4	113	107

Test ID	Sample ID	CN Residue P_{80} (µm)	% Ag Extraction (Hours)							Residue (Ag g/t)	Ag Head Grade (g/t)	
			2	4	8	12	24	36	48		Calc	Direct
CN-12	06-MET-F5-DG	167	76	77	77	79	80	79	80	7.0	35.0	33.7
CN-13	07-MET-F5-DG	158	19	21	23	25	26	26	26	20.5	27.8	22.2
CN-14	08-MET-F5-DG	160	44	46	46	47	47	46	47	17.0	31.7	34.0
CN-15	09-MET-F5-DG	171	58	64	72	77	82	84	85	13.6	90.3	86.2
CN-16	10-MET-F5-DG	168	72	74	77	77	80	79	80	27.9	137	142
CN-17	11-MET-F5-DG	169	74	77	81	80	82	81	81	8.0	41.4	39.5
CN-18	02-MET-F5-NE	157	80	80	81	80	78	81	81	24.7	129	131
CN-10	03-MET-F5-NE	134	56	60	64	66	67	68	68	8.0	24.7	24.3
CN-20	04-MET-F5-NE	148	49	55	63	66	68	71	73	8.0	29.6	28.1
CN-21	06-MET-F5-NE	144	57	58	59	59	58	60	61	<2	<5.1	<10
CN-22	08-MET-F5-NE	157	39	40	40	40	40	41	41	<2	<3.4	<10
CN-23	09-MET-F5-NE	137	63	68	72	74	74	74	74	5.0	19.4	19.7
CN-24	10-MET-F5-NE	142	71	72	73	73	73	73	74	9.0	34.6	31.5
CN-25	11-MET-F5-NE	152	50	53	54	54	55	55	55	10.2	22.8	21.3
CN-27	JAC MET Comp	129	1	4	38	60	79	84	87	34.5	274	259
CN-28	Oculto MET Comp	161	87	90	91	91	92	92	91	12.1	136	134
CN-29	YR 1-5 MET Comp	148*	63	66	67	67	68	69	70	25.3	82.9	81.9
CN-26	YR 1-5 Comp	147	64	68	69	69	69	70	70	24.7	81.5	81.0

13.2.6.7 Cyanidation on Gravity Separation Testwork Products

The gravity response of the YR 1–5, JAC MET, and Oculto MET composites was evaluated using approximately 30 kg of sample for each test. This work was conducted along with the SGS Lakefield standard procedure where the material is first ground to the P80 targeted as leach circuit feed. The material is then passed into a lab-scale Knelson concentrator, the concentrate of which is then further upgraded onto a Mozley vibrating table to reduce the residual concentrate weight to a target of 0.6% of the total Knelson feed weight.

Table 13-14 summarizes the Knelson gravity separation test results.

Table 13-14: Phase 5B1 – SGS Lakefield Gravity Concentration Procedure Applied to MET Composites

Test ID	Sample ID	Tailings P_{80} (µm)	Concentrate			Recovery		Head Grade			
								Au, g/t		Ag, g/t	
			wt. (%)	Au* (g/t)	Ag* (g/t)	Au (%)	Ag (%)	Calc	Direct	Calc	Direct
G-1	YR 1-5 Met Comp	157	0.62	13.6	1,749	16.2	13.7	0.52	0.51	79.4	81.9
G-2	JAC Met Comp	118	0.68	1.63	6,084	12.3	16.4	0.09	0.08	253	258
G-3	Oculto Met Comp	142	0.62	47.7	7,901	27.0	34.3	1.09	0.95	142	134

Notes:

*Gold and silver concentrate grades are derived from the subsequent intensive cyanidation calculated head grades.

13.2.6.7.1 Gravity Concentrate Intensive Cyanidation

An intensive cyanidation test was conducted on the gravity concentrates produced from the YR 1–5, JAC MET, and Oculto MET gravity separation tests.

Each concentrate was pulped to 25% solids using tap water and the pH was maintained between 10.5 and 11.0 with lime added as calcium hydroxide (CaO). The bottles were placed on rolling agitators and pre-aerated for 4 hours using an air/oxygen sparge to achieve a dissolved oxygen level of 8-12 ppm. Aeration was maintained throughout the 72 hour leach period, during which sodium cyanide (NaCN) was added at 5 g/L and held at this concentration for the duration of the test.

The final gold extraction for all three gravity concentrates was 98%, while silver extractions ranged from 97% to 100% after 72 hours.

Sodium cyanide consumption during the intensive cyanidation tests ranged from 7.52 kg/t to 37.4 kg/t, while lime (CaO) consumption ranged from 0.32 kg/t to 1.64 kg/t. The high consumption reading, tied to the JAC Met composite, is deemed the result of copper bearing material having reported to the gravity concentrate and consuming cyanide: the PLS assay of this test returned values of 2,780 mg/L Cu whereas those of the two other composites showed 13 and 97 mg/L Cu.

13.2.6.7.2 Gravity Tails Bulk Cyanidation

Based on the conditions established during the cyanidation testwork, bulk cyanidation leaching of the gravity tailings streams was conducted on the three MET composite samples to generate sufficient leach solutions and residues for subsequent Merrill–Crowe, solid/liquid separation, rheology, and cyanide destruction testing.

After pulping to 45% solids the ground charges, two 1 kg (dry basis) charges were removed from each composite and submitted for oxygen uptake testing. Aquarium salts were then added to the remaining pulp to achieve a concentration of 1.32 g/L chloride and simulate the

level expected from the slightly brackish site water. Following conditioning, the pulp underwent agitated cyanide leach under the usual optimum conditions set earlier.

The bulk cyanidation tails pulp was then divided into two equal portions. One portion was forwarded for solid/liquid separation testing. The second portion was diluted 2:1 with synthetic process water and 0.5 g/L NaCN. The slurry was then decanted back to the original test density of 45% solids to simulate the dilution occurring within the CCD circuit through the addition of wash water. The decanted solution was submitted for Merrill–Crowe testing, while the remaining pulp was forwarded for cyanide destruction testing.

The bulk leach cyanidation tests (BL-1 to BL-3) conducted on the gravity tailings from the MET composite samples achieved gold extractions ranging from 75% to 84% and silver extractions ranging from 66% to 89%. As previously stated, the results from the JAC MET composite are tainted by the inclusion of material containing much higher levels of copper than expected when the mining operations are restricted to the oxide layer of the deposit.

13.2.6.7.3 Gravity and Cyanidation Overall Recovery

The overall gold recoveries obtained through gravity separation, as performed along the standard SGS Lakefield procedure, followed by cyanidation of both the gravity concentrate and tailings ranged from 78% to 86%, while silver recoveries ranged from 70% to 92%. These gold and silver recoveries were generally consistent with those achieved through whole ore cyanidation.

The one exception was the JAC MET composite, where gold recovery by gravity plus cyanidation was 78%, as compared with 89% from whole ore cyanidation. This discrepancy is likely an anomaly, as the combined circuit (gravity plus cyanidation of all products) should theoretically achieve recoveries equal to or better than whole-ore cyanidation. The anomaly is most likely related to the very low gold head grade of the JAC MET composite (0.08 g/t) and inherent analytical limitations at such low concentrations.

Table 13-15 summarizes the overall recovery results for gravity and gravity-cyanidation.

Table 13-15: Phase 5B1 - Overall Gold and Silver Recoveries from Combined Gravity-Cyanidation Testwork (SGS Lakefield Gravity Concentration Procedure)

MET Composite ID	Grind P_{80} Size (µm)	Metal Distribution					Gravity Concentrate			Gravity Tailing			Overall Recovery		Whole Ore		
		CN Test ID	Au		Ag		CN Test ID	Extraction		CN Test ID	Extraction				CN Test ID	Extraction	
			Conc (%)	Tail (%)	Conc (%)	Tail (%)		Au (%)	Ag (%)		Au (%)	Ag (%)	Au (%)	Ag (%)		Au (%)	Ag (%)
YR 1-5	157	G-1	16.2	83.8	13.7	86.3	CN-30	98	97	BL-1	84	66	86	70	CN-29	89	70
JAC	118	G-2	12.3	87.7	16.4	83.6	CN-31	98	99	BL-2	75	85	78	87	CN-27	89	87
Oculto	142	G-3	27.0	73.0	34.3	65.7	CN-32	98	100	BL-3	77	89	82	92	CN-28	85	91

Merrill-Crowe Testwork

A portion of each bulk leach tailing sample, as generated from the bulk cyanidation leaching of the gravity tailing products from the three MET composites, was subjected to a simulated CCD step, which consisted of diluting the tails from the cyanidation tests with synthetic site water that was laced with cyanide at a 0.5 g/L concentration at variable ratios and then removing the supernatant to leave a slurry at 45% solids, which then proceeded to cyanide destruction testwork. The solution removed then represented the equivalent of the CCD circuit thickener overflow, or the pregnant leach solution to be presented to the Merrill-Crowe circuit. This dilution simulates the addition of wash water within the industrial CCD circuit, which is used to displace the dissolved metals away from the thickened solids before they proceed to cyanide destruction. A real CCD circuit would have left more of the dissolved metals in the overflow, somewhat increasing the actual concentration of these presented to the Merrill-Crowe circuit.

Gold and silver were precipitated using zinc dust from these simulated CCD overflow solutions

The Merrill–Crowe process requires complete clarification and deaeration prior to zinc dust addition. All operations were carried out under an inert nitrogen atmosphere. For each test, the solution was first de-aerated by sparging nitrogen for 30 minutes in a semi-sealed vessel. The required amount of lead nitrate was then added, followed by the specified addition of zinc dust. Nitrogen flow was maintained for an additional 30 minutes to ensure thorough mixing of the zinc dust and to prevent oxygen reabsorption.

Initially, zinc and lead nitrate additions for all initial tests were consistent, with five times the stoichiometric zinc dust requirement and lead nitrate at one quarter of the zinc mass (tests MC-1A to MC-1C, MC-2A to MC-2C, MC-3A to MC-3C). Subsequent tests were carried out to minimize the zinc dust addition while maintaining a high cementation efficiency. Therefore, changes to the dilution ratio and/or stoichiometric zinc dust addition rates and ratio of lead nitrate to zinc dust were introduced.

Across all tests performed on the three composite samples, gold recovery was consistently 100% under every test condition. Even though the gold grades in solution were low, ranging from below the detection limit of 0.05 mg/L up to 0.16 mg/L, the Merrill–Crowe process still effectively precipitated all dissolved gold.

For the YR 1-5 MET composite (MC-1A to MC-1G), silver extraction ranged from 90.1% to 99.3%. For the JAC MET composite (MC-2A to MC-2G), silver extraction ranged from 96.8% to 100%. For the Oculto MET composite (MC-3A to MC-3G), silver extraction ranged from 88.3% to 98.3%.

The best overall performance for all three samples was observed under the following conditions:

- 2:1 CCD-simulated dilution ratio
- Five times stoichiometric zinc dust addition
- Lead nitrate addition equal to one-quarter of the zinc mass.

These optimal conditions correspond to tests MC-1A, MC-2A, and MC-3A, which minimize the solution losses in the CCD circuit. A detailed summary of the results is provided in Table 13-16. At the preferred 3:1 dilution ratio, the silver cementation efficiency with the 5 times stoichiometric zinc dust ratio drops by only 1.3% for the YR 1-5 MET, 0.4% for the JAC MET and 3.8% for the Oculto MET.

Table 13-16: Phase 5B1 - Merrill-Crowe Test Results

Test ID	Synthetic Process Water to PLS Ratio	Zn Stoichiometric Addition Times	Zn:Pb(NO_3)$_2$ Addition Ratio	Feed Solution		Barren Solution		Extraction	
				Au (mg/L)	Ag (mg/L)	Au (mg/L)	Ag (mg/L)	Au (%)	Ag (%)
MC-1A	2:1	5	4:1	0.10	22.1	<0.05	0.15	100	99.3
MC-1B	3:1	5	4:1	0.07	12.1	<0.05	0.24	100	98.0
MC-1C	5:1	5	4:1	<0.05	6.04	<0.05	0.60	100	90.1
MC-1D	2:1	3	4:1	0.10	12.3	<0.05	0.55	100	95.5
MC-1E	3:1	3	4:1	0.07	9.17	<0.05	0.83	100	90.9
MC-1F	2:1	3	2:1	0.10	12.3	<0.05	0.48	100	96.1
MC-1G	3:1	3	2:1	0.07	9.04	<0.05	0.67	100	92.6
MC-2A	2:1	5	4:1	<0.05	48.3	<0.05	<0.03	100	100
MC-2B	3:1	5	4:1	<0.05	36.1	<0.05	0.13	100	99.6
MC-2C	5:1	5	4:1	<0.05	24.4	<0.05	0.32	100	98.7
MC-2D	2:1	3	4:1	<0.05	48.3	<0.05	0.37	100	99.2
MC-2E	3:1	3	4:1	<0.05	37.2	<0.05	0.81	100	97.8
MC-2F	2:1	3	8:1	<0.05	53.3	<0.05	0.48	100	99.1
MC-2G	3:1	3	8:1	<0.05	38.9	<0.05	1.25	100	96.8
MC-3A	2:1	5	4:1	0.16	21.5	<0.05	0.36	100	98.3
MC-3B	3:1	5	4:1	0.12	16.4	<0.05	0.97	100	94.1
MC-3C	5:1	5	4:1	0.07	10.9	<0.05	1.47	100	86.5
MC-3D	2:1	3	4:1	0.16	21.5	<0.05	1.37	100	93.6
MC-3E	3:1	3	4:1	0.12	16.5	<0.05	1.90	100	88.5
MC-3F	2:1	3	2:1	0.16	22.5	<0.05	1.08	100	95.2
MC-3G	3:1	3	2:1	0.12	16.6	<0.05	1.95	100	88.3

Note: Values less than detection limit (<) are assumed to be zero in the % extraction calculation

13.2.6.8 Oxygen Uptake Testwork

Two oxygen uptake tests were conducted on the gravity tailing material from each of the three MET composites (tests OU-1 to OU-6). For each composite, one test was completed using air, targeting a DO concentration of approximately 6 mg/L to 8 mg/L, while the second test was completed using oxygen, targeting a DO concentration of 10 mg/L to 12 mg/L. Two additional oxygen uptake tests were also performed: one on the JAC E-GRG composite (OU-8) and one on the Oculto MET composite (OU-7), both using oxygen at a DO target of 10 mg/L to 12 mg/L.

The test conditions were designed to simulate cyanide leach conditions. Oxygen consumption was measured over a 24 hour period and oxygen uptake rates (mg/L/min) were calculated at 1, 2, 3, 4, 5, 6, and 24 hours. Throughout the tests, cyanide and lime concentrations were maintained at target levels.

Although the 24 hour-mark measurements for tests OU-1 and OU-5 showed an increase in oxygen uptake rate, the overall results demonstrated that oxygen demand generally decreased over the duration of the tests. This trend is reflected in the downward slopes of the linear trendlines for each test. After 24 hours, oxygen uptake rates across all tests ranged from 0.001 mg/L/min to 0.176 mg/L/min. The much higher oxygen demand, as demonstrated by the JAC MET, is attributed to its high copper content, making this sample unrepresentative of the mineral domain.

13.2.6.9 Cyanide Destruction (CND) Testwork

Cyanide destruction testwork was carried out on the bulk leach tailings from the YR 1-5 MET composite, the JAC MET composite, and the Oculto MET composite. The primary focus of the CND test program was on the decanted diluted leach tailings, representing simulated CCD tailings, to replicate a leach/CCD/CND process flowsheet.

The objective of the cyanide destruction process was then to produce a treated discharge pulp with a CN_{WAD} concentration of <10 mg/L in the solution phase. Optimization efforts focused mainly on adjusting the sodium metabisulphite (SMBS) dosage, along with evaluating the effects of retention time and copper addition on overall process performance. All CND tests were conducted at 45% solids (w/w) and ambient temperature in a 1.0 L reactor.

Following initial batch testing, an extended continuous cyanide destruction test was conducted to generate sufficient treated material for solid–liquid separation and rheology evaluations. After achieving an initial residual CN_{WAD} concentration of <10 mg/L, additional testwork was performed to optimize the process conditions, targeting a more stringent CN_{WAD} concentration of <1 mg/L for both the YR 1-5 MET composite and the JAC MET composite.

Barren leach solutions from tests with YR 1-5 MET (BL-1, BL-1B), JAC MET (BL-2, BL-2B), and Oculto MET (BL-3) were analyzed for cyanide speciation and subjected to an ICP scan. These results are summarized in Table 13-17.

Table 13-17: Phase 5B1 - CND Feed Solution Chemical Analysis

Element (mg/L)	CND Solution				
	BL-1	BL-1B	BL-2	BL-2B	BL-3
CN_T	449	406	899	674	462
CN_{WAD}	412	393	824	624	437
CNS	<2	<2	120	130	6
CNO	12	23	20	43	13
Ag	20.7	2.74	50.9	34.4	22.5
Al	0.5	<0.2	<0.2	<0.2	4.6
As	<3	<3	<3	<3	<3
Ba	0.047	0.046	0.034	0.055	0.04
Be	<0.002	<0.002	<0.002	<0.002	<0.002
Bi	<1	<1	<1	<1	<1
Ca	71.9	69.6	37	73.3	69
Cd	<0.09	<0.09	<0.09	<0.09	<0.09
Co	<0.3	<0.3	<0.3	<0.3	<0.3
Cr	<0.1	<0.1	<0.1	<0.1	<0.1
Cu	99.1	1.39	443	275	11.3
Fe	0.8	0.19	0.9	0.98	7.7
K	7	5	9	9	21
Li	<2	<2	<2	<2	<2
Mg	0.3	0.22	0.13	0.28	0.69
Mn	<0.04	<0.04	<0.04	<0.04	0.15
Mo	<0.6	<0.6	<0.6	<0.6	<0.6
Na	982	987	1320	1280	951
Ni	<0.6	<0.6	<0.6	<0.6	<0.6
P	<5	<5	<5	<5	<5
Pb	<2	<2	<2	<2	<2
Sb	<1	<1	<1	<1	<1
Se	<3	<3	<3	<3	<3
Sn	<2	<2	<2	<2	<2
Sr	0.12	0.1	0.076	0.099	0.17
Ti	<0.02	<0.02	<0.02	<0.02	<0.02
Tl	<3	<3	<3	<3	<3
V	<0.2	<0.2	<0.2	<0.2	<0.2
Y	<0.02	<0.02	<0.02	<0.02	<0.02
Zn	0.9	<0.7	3.7	2.7	<0.7

13.2.6.9.1 CND Testwork on YR 1-5 MET Composite

The YR 1-5 MET composite gravity tails cyanidation residues were put through the SO_2/air cyanide destruction steps. The tails were first diluted at a 2:1 ratio and repulped to 45% solids, to simulate washing within the CCD circuit.

The early optimization tests did not include any copper addition and evaluated only the sulphur dioxide requirement. It was subsequently determined that effective cyanide destruction required elevated copper levels, approximately 20 g/t, to adequately catalyze the SO_2/air reaction.

The SO_2/air process proved successful during the third continuous CND test. The initial reagent conditions included approximately 5.5 g SO_2 per g CN_{WAD}, 20 g/t added copper, and a 60 minute retention time. The copper addition was increased to 50 g/t, the SO_2 dosage and retention time remained similar to CND-1-3 but there was no improvement in cyanide destruction efficiency relative to the 20 g/t copper addition.

The following three tests focused on maintaining the copper addition at 25 g/t while optimizing the SO_2 dosage and retention time to consistently achieve a residual CN_{WAD} of <10 mg/L. Reduction of the SO_2 addition was not successful in achieving a residual CN_{WAD} of <10 mg/L.

The reactor size was then increased from 1 L to 2 L to improve agitation and oxygen dispersion, with the DO increasing from 0.2 mg/L to 4.2 mg/L. Further cyanide destruction optimization was carried out on the BL□1B pulp, obtained from bulk leaching the YR 1-5 MET composite, with a revised target of <1 mg/L CN_{WAD}. One batch test and two continuous tests were completed on the YR 1□5 MET composite bulk leach tailings. Both continuous tests successfully achieved the target of <1 mg/L CN_{WAD}. The resulting barren solution contained 0.56 mg/L CN_{WAD}, 0.78 mg/L CN_T, <0.1 mg/L Cu, and <0.1 mg/L Fe.

Upon completion of the program, it was determined that the following operating conditions will achieve a discharge CN_{WAD} concentration of <1 mg/L for the YR 1-5 CCD simulated composite:

- 45% solids (w/w)
- 6.0 g equivalent SO_2 per gram CN_{WAD} (achieving <1 mg/L)
- 35 mg/L copper addition at a pH of 8.5 – lime added as needed (value can be optimized and adjusted at site and additions will likely be less than the values used in the testwork).
- 60 minute retention time.

13.2.6.9.2 CND Testwork on JAC MET Composite

The JAC MET composite gravity tails cyanidation rejects (BL-2) underwent cyanide destruction testing under the test series 2 (CND 2-X), following an approach that was similar to that applied to the YR 1-5 composite tested in Series 1. Two testwork phases were completed on the cyanidation residues from the YR 1-5 MET composite, wherein both cases were diluted with cyanide solution in a 2:1 ratio and then repulped to 45% solids. The first phase was performed on the cyanidation residues of the gravity tails (BL-2) and sought a residual CN_{wad} content of <10 mg/L. The second phase was performed on residues from bulk cyanidation of JAC MET material (BL-2B) and then sought a lower residual content of <1 mg/L.

The SO_2/air process needed approximately 6.5 g SO_2 per g CN_{WAD} and 30 g/t of copper to achieve the <10 mg/L CN_{wad} target, along with a 60 minute retention time. For the more stringent target of <1 mg/L, about 7.2 g SO_2 per g CN_{WAD} and 40 g/t of copper were needed, with an extension of the residence time to 90 minutes.

13.2.6.9.3 CND Testwork on Oculto MET Composite

The Oculto MET composite also underwent cyanide destruction optimization under CND Series 3. It was only tested with the objective of achieving a residual CN_{WAD} of <10 mg/L.

Early destruction tests were performed in a 1 L reactor, after which the system was upscaled to 2 L for the final two optimization tests. During the initial tests, the pulp became viscous, which limited effective air dispersion and agitation. As a result, DO dropped to 0.7 mg/L in test CND 3-2, negatively affecting the efficiency of the cyanide destruction. Increasing the reactor volume to 2 L improved mixing and gas dispersion, which raised the DO levels to 3-6 mg/L in the final tests.

Overall, the results indicate that cyanide in the Oculto MET composite is more easily reduced to the target level than in the previous two composites, requiring lower reagent dosages to achieve <10 mg/L CN_{WAD}. The optimized CND 3 conditions included a 60 minute retention time, 4.7 g SO_2 per g CN_{WAD}, and 10 g/t of added copper.

The final solution analyses from the successful destruction tests (CND 1-3, CND 1B-2, CND 2-3, CND 2B-4, and CND 3-3) are summarized in Table 13-18.

Table 13-18: Phase 5B1 - Overall CND Testwork Discharge Solution Analysis

Element (mg/L)	CND Product Analysis				
	CND 1-3	CND 1B-2	CND 2-3	CND 2B-4	CND 3-3
CN_{wad} target	< 10	< 1	< 10	< 1	< 10
CN_T	3.9	0.78	2.58	0.81	23.6
CN_{WAD}	3.41	0.56	1.87	0.5	6.89
CNS	3	<2	180	100	7
CNO	660	770	1,300	1,100	740
Hardness as $CaCO_3$	1520	1,470	1,770	2,280	1,400
Ag	0.513	0.73	0.0529	0.27	0.305
Al	0.015	0.03	0.002	<0.01	0.39
As	0.245	0.254	0.0312	0.124	0.085
Ba	0.188	0.251	0.194	<0.0008	0.0791
Be	<0.000007	<0.00007	<0.000007	<0.00007	<0.00007
B	0.2	0.17	0.116	0.13	0.47
Bi	0.00049	0.0011	0.00005	0.0011	0.0004
Ca	604	583	701	907	555
Cd	0.000024	<0.00003	0.000175	0.0001	<0.00003
Co	0.0274	0.0122	0.107	0.172	0.0381
Cr	0.0113	0.0137	0.0178	0.0194	0.0079
Cu	0.282	0.09	13.2	0.92	0.11
Fe	0.187	0.21	<0.007	0.1	8.53
K	14.7	11.4	21.3	21.4	35.2
Li	0.0115	0.009	0.0117	0.01	0.02
Mg	3.7	3.11	5.04	4.1	3.24

Element (mg/L)	CND Product Analysis				
	CND 1-3	CND 1B-2	CND 2-3	CND 2B-4	CND 3-3
Mn	0.0109	0.0043	0.126	0.0194	0.0167
Mo	0.0322	0.013	0.0566	0.063	0.044
Na	1600	1690	2790	2690	1650
Ni	0.0874	0.031	0.0138	0.013	0.187
P	0.15	0.12	0.02	<0.03	<0.03
Pb	0.00519	0.0042	0.00176	0.0021	0.0027
Sb	0.0565	0.046	0.372	0.286	0.069
Se	0.0185	0.006	0.321	0.461	0.0376
Si	45.3	54.6	19.5	31.7	1.7
Sn	0.00014	<0.0006	0.0001	<0.0006	0.0009
Sr	0.784	0.689	0.758	0.767	0.621
Ti	0.0009	0.004	0.0001	<0.001	<0.001
Tl	0.0012	0.00102	0.00513	0.0109	0.00109
U	0.00193	0.00118	0.00672	0.00173	0.00597
V	0.00387	0.0053	0.00061	0.0028	0.0007
W	0.0177	0.0121	0.0744	0.084	0.027
Y	<0.00002	<0.0002	<0.00002	<0.0002	<0.0002
Zn	0.004	<0.02	0.004	<0.02	<0.02

13.2.6.10 Rheology Characterization

Slurry rheology evaluations were conducted on three leach tailings samples and three detox tailings samples. All six samples were received as thickened products generated by Metso as part of their settling and dynamic thickening testwork program (see Section 13.2.7).

Rheology testing was performed on each underflow sample across a range of solids contents that were selected based on the slurry's stability and flowability. All samples exhibited negligible inter-particle interactions, as indicated by their α-values being very close to 1.00.

13.2.6.10.1 Rheological Behaviour of Cyanidation Tails

The critical solids density (CSD) of the JAC MET composite leach tails was approximately 56% w/w solids. At this density, the material exhibited a yield stress of 28 Pa under unsheared conditions and 5 Pa under sheared conditions (measured after three minutes of constant shearing).

Thixotropic behaviour was observed at slurry densities ≥55.6% w/w solids. At densities ≥58.9% w/w solids, the sample displayed plug flow behaviour during unsheared measurements.

The CSD of the Oculto MET composite leach tails was approximately 67% w/w solids, determined over a tested solids range of 60.5 to 73.5 w/w. At this density, the slurry exhibited a yield stress of 12 Pa under unsheared conditions. Sheared rheology data at ≤66.0% w/w solids were excluded due to solids settling during measurement, rendering those results invalid.

Rheopectic behaviour was observed at slurry densities ≥70.4% w/w solids. Rheopexy is a "flow-adverse" response in which a material's resistance to flow increases under constant shear.

The CSD of the YR 1-5 MET composite leach tails was approximately 68% w/w solids, as determined over a tested solids range of 60.3 to 71.5% w/w. At this density, the slurry exhibited a yield stress of 22 Pa under unsheared conditions. All sheared rheology data were excluded due to the rapid settling behaviour of the slurry, which rendered those measurements invalid.

Testing at 72.3% w/w solids resulted in a torque overload condition, indicating that the applied load exceeded the rheometer's maximum torque capacity. Rheopectic behaviour was observed at slurry densities ≥68.6% w/w solids. The CSD of the JAC MET composite detoxified tails was approximately 56.5% w/w solids. At this density, the slurry exhibited a yield stress of 27 Pa under unsheared conditions and 13 Pa under sheared conditions. Sheared data collected at 45.4% w/w solids were deemed invalid due to solids settling during the measurement.

Thixotropic behaviour was observed at slurry densities ≥55.7% w/w solids, indicating a reduction in flow resistance under sustained shear.

13.2.6.10.2 Rheological Behaviour of Detoxified Tails

The CSD of the Oculto MET composite detox tails was approximately 71% w/w solids. At this density, the slurry exhibited a yield stress of 33 Pa under unsheared conditions and 6 Pa under sheared conditions. Sheared data at ≤65.2% w/w solids were excluded due to solids settling during measurement, which rendered the results invalid.

A test conducted at 75.5% w/w solids resulted in a torque overload condition, indicating that the rheometer's maximum torque capacity was exceeded. Thixotropic behaviour was observed at slurry densities ≥69.9% w/w solids, reflecting a decrease in flow resistance under constant shear.

The CSD of the YR 1-5 MET composite detox tails underflow was approximately 69% w/w solids, as determined over a tested solids range of 60.2 to 72.0% w/w. At this density, the slurry exhibited a yield stress of 24 Pa under unsheared conditions and 3 Pa under sheared conditions. Sheared data at ≤66.7% w/w solids were excluded due to solids settling during measurement.

Rheopectic behaviour was observed at slurry densities ≥70.1% w/w solids, indicating an increase in flow resistance under sustained shear.

13.2.6.11 Phase 5B1 Testwork Program Conclusions and Recommendations

The conclusions and recommendations from the Phase 5B1 testwork were as follows:

- The gold and silver grades of the composites used in the test program varied widely, ranging from <0.01 g/t to 2.19 g/t gold and <10 g/t to 344 g/t silver.
- Comminution testing showed that the composites ranged from soft to moderately hard in the SMC test (relative to the JK Tech database) and from moderately soft to very hard in the BBWI test, with work indices between 11.8 kWh/t and 27.3 kWh/t.
- TIMA analysis of the YR 1-5 Composite identified native gold as the only gold bearing phase, while chlorargyrite and iodargyrite were the dominant silver minerals, with minor acanthite. Silver minerals liberation was high, ranging from 78% to 97%.
- All silver minerals identified are cyanide-soluble. TIMA analysis of cyanide leach residues indicated that most remaining silver was poorly liberated and likely locked within cyanide-insoluble host minerals.
- Gravity separation testing for three composites, YR 1-5 MET, JAC MET, and Oculto MET, achieved gold recoveries of 12% to 27% and silver recoveries of 14% to 34% using the SGS Lakefield standard testing procedure, not the E-GRG protocol.
- Preliminary cyanidation testing at a P_{80} of ~150 µm produced gold extractions of 58% to 98% and silver extractions of 26% to 91% after 48 hours. Lower recoveries correlated with lower head grades. Notably, 26 of 30 composites achieved ≥80% gold extraction and 18 of 30 achieved ≥70% silver extraction. Sodium cyanide consumption ranged from 0.46 kg/t to 4.70 kg/t and lime consumption ranged from 0.37 kg/t to 2.83 kg/t. Elevated cyanide consumption in some samples was associated with cyanide-soluble copper minerals.
- Cyanidation of gravity products (for YR 1-5 MET, JAC MET, and Oculto MET) showed that intensive cyanide leaching of gravity concentrates yielded 98% gold extraction for all three composites and 97% to 100% silver extraction. Bulk cyanide leaching of the gravity tailings achieved 75% to 84% gold extraction and 66% to 89% silver extraction. Combined gravity and cyanidation recoveries ranged from 78% to 86% for gold and 70% to 92% for silver. As these recoveries were comparable to those from whole-ore cyanidation, no benefit was identified from incorporating a gravity separation circuit into the flowsheet.
- Merrill-Crowe testing on simulated CCD overflow solutions achieved 100% gold recovery in all tests, while silver recoveries ranged from 88% to 100%. Optimal performance was obtained using a 2:1 dilution of the leach solution with CCD solution, five times the stoichiometric zinc dust addition based on combined Au + Ag concentrations, and a lead nitrate addition equal to one quarter of the zinc dosage.
- Cyanide destruction using the SO_2/air process successfully achieved the more stringent target residual cyanide concentration indicated of <1 mg/L and optimal operating conditions were established for all three of the composite samples. Better agitation and higher DO levels may reduce the reagent consumption and/or improve the destruction kinetics.
- Slurry rheology testing showed that the CSD of the three composites ranged from 56% to 71% w/w solids, with unsheared yield stress values between 12 Pa and 33 Pa.
    - Thixotropic behaviour was observed with the JAC leach tailings, JAC detox tailings, and Oculto detox tailings, indicating decreasing flow resistance under sustained shear.
    - Rheopectic behaviour was observed in the Oculto leach tailings, YR 1–5 leach tailings, and YR 1–5 detox tailings, indicating increasing flow resistance under sustained shear.

13.2.7 Metso Settling and Dynamic Thickening Testwork

The purpose of the program was to refine the thickener sizing criteria through testing of the cyanide leach tailings from the YR 1–5, Oculto, and JAC composites, together with their respective post-cyanide destruction tailings.

The specific objectives of the testwork were to determine the following:

- Optimum flocculant type and dosage
- Thickener unit area requirement determination versus thickener feed density and underflow density target
- Achievable overflow clarity
- Underflow rheology, expressed as yield stress.

The results are summarized in this section.

13.2.7.1 Sample Characterization and Preparation

SGS Lakefield prepared and supplied three cyanide leach tailings samples and three post‑cyanide destruction tailings samples from their leach testwork program. As for other portions of SGS Lakefield's work, synthetic water was formulated with aquarium salts to closely match the expected site water chemistry (relative to Cl, SO_4, Na, and pH). This synthetic water was used for slurry dilution during thickening testwork.

All samples were received as slurries. Prior to flocculant screening and dynamic thickening tests, the cyanide leach samples pH was adjusted with lime to 10.5–11. All testing was conducted at ambient temperature.

13.2.7.2 Flocculant Screening

Six SNF flocculants were evaluated during the screening testwork. Of these, SNF 905 VHM and SNF 913 SH both achieved clear overflow performance at dosages of 10 g/t or greater. SNF 913 SH demonstrated the best overall performance, producing larger, well-formed floc agglomerates at lower dosages and showing no evidence of floc fragility. Based on these results, SNF 913 SH was selected for all subsequent dynamic thickening testwork.

13.2.7.3 Dynamic Thickening Tests

Multiple dynamic thickening test runs were completed on the Year 1–5, Oculto, and JAC composites cyanide leach tailings, as well as their respective post-cyanide destruction tailings. The testwork was carried out across a range of solids loading rates and flocculant dosages to assess their effects on underflow density and overflow clarity. The resulting data are meant to support thickener sizing and establish appropriate design criteria for processing these tailings streams.

13.2.7.3.1 Year 1-5 Composite Leach Tailings

Dynamic thickening tests on the Year 1-5 composite leach tailings showed that an underflow density of ~60% solids was achieved across all solids loading rates tested (0.5-1.0 t/m²/hr) at a flocculant dosage of 15 g/t. The optimum feed slurry density was found to be 18% w/w solids. Additional dosage optimization at a fixed loading rate of 1.0 t/m²/hr indicated underflow solids between 54.6% and 65.9% achieved with optimal performance at 12 g/t. Overflow clarity remained consistently below 100 ppm for all test conditions, indicating suitability for process water reuse.

13.2.7.3.2 Oculto Composite Leach Tailings

Initial testing at a flocculant dosage of 15 g/t across slurry loading rates of 0.5-1.0 t/m²/hr consistently achieved an underflow density of 60% solids. Optimum feed slurry density was established as 18% w/w solids. Further optimization at a fixed loading rate of 1.0 t/m²/hr with flocculant dosages between 5 and 15 g/t produced stable underflow solids of approximately 69.5%, with the highest value at 15 g/t. Overflow clarity remained below 100 ppm for all tests, which confirms its suitability for process water reuse.

13.2.7.3.3 JAC Composite Leach Tailings

Dynamic thickening tests on the JAC composite leach tailings were conducted at an initial flocculant dosage of 60 g/t across slurry loading rates of 0.5-0.9 t/m²/hr, achieving an underflow density of 50% to 60% solids under all conditions tested. As for the other composites tested, the optimum feed slurry density was measured as 18% w/w solids. Further optimization at a fixed loading rate of 0.9 t/m²·h showed that reducing the flocculant dosage to 50 g/t increased underflow solids from 49.7% to 55.3%. However, this reduction resulted in overflow suspended solids exceeding 100 ppm. At the 60 g/t dosage, overflow clarity consistently remained below 100 ppm, indicating suitability for process water reuse.

13.2.7.3.4 Year 1-5 Composite Cyanide Destruction Tails

Dynamic thickening tests on the Year 1-5 composite cyanide destruction tailings were conducted at an initial flocculant dosage of 20 g/t across slurry loading rates of 0.5-0.9 t/m²/hr. Under all conditions tested, an underflow density of 60% solids was achieved. 18% w/w solids was still the optimum feed density. Further optimization at a fixed loading rate of 0.5 t/m²/hr with flocculant dosages between 20 g/t and 30 g/t resulted in underflow solids ranging from 67.0% to 68.7%, with the highest underflow density achieved at 20 g/t. Overflow clarity remained consistently below 100 ppm for all test runs, indicating suitability for process water reuse.

13.2.7.3.5 Oculto Composite Cyanide Destruction Tails

Dynamic thickening tests on the Oculto composite cyanide destruction tailings were conducted at an initial flocculant dosage of 20 g/t across slurry loading rates of 0.5-0.9 t/m²/hr. Under all conditions tested, an underflow density of 60% solids was achieved. The optimum feed density was 14.8% w/w solids. Further optimization at a fixed loading rate of 0.9 t/m²·h with flocculant dosages between 10 g/t and 20 g/t resulted in underflow solids ranging from 66.5%

to 73.5%, with the highest underflow density achieved at 10 g/t. Overflow clarity remained consistently below 100 ppm for all test runs, indicating suitability for process water reuse.

13.2.7.3.6 JAC Composite Cyanide Destruction Tails

Dynamic thickening tests on the JAC composite cyanide destruction tailings were conducted at initial flocculant dosages of up to 60 g/t and slurry loading rates of 0.5-0.7 t/m²·h. At a fixed loading rate of 0.5 t/m²·h, reducing flocculant dosage from 60 g/t to 40 g/t maintained acceptable overflow quality, while further reduction to 25 g/t resulted in deteriorated overflow clarity despite achieving underflow density of 50-60% solids. The optimum feed density was established as 15.9% w/w solids. Additional testing at a higher loading rate of 0.7 t/m²·h with flocculant dosages between 25 and 60 g/t produced underflow solids ranging from 50.4% to 53.0%, with the highest underflow density achieved at 60 g/t.

Overflow clarity met the target of less than 100 ppm at flocculant dosages of 40 g/t and 60 g/t, although some ultra fines were observed. At a reduced dosage of 25 g/t, the overflow suspended solids increased significantly to 139 ppm and 591 ppm at loading rates of 0.5 t/m²·h and 0.7 t/m²·h, respectively, indicating that higher flocculant dosages are required to maintain acceptable overflow quality.

13.2.7.3.7 Metso Dynamic Thickening Testwork Conclusions

Thickening testwork conducted for the Project on Year 1–5, Oculto, and JAC composites cyanide leach tailings, together with their post-cyanide destruction slurry counterparts, demonstrated that all materials can be successfully thickened. In all cases, an underflow density of approximately 50% to 60% solids was met or exceeded.

Testing of the Year 1–5 and Oculto composite leach tailings and their post-cyanide destruction products produced overflow suspended solids concentrations of less than 100 ppm, indicating that the overflow water would be suitable for reuse as process water.

For the JAC composite leach tailings and its post-cyanide destruction counterpart, overflow suspended solids concentrations exceeded 100 ppm when flocculant dosages were below 60 g/t. The Year 1-5 composite contained 20% of JAC material in its composition.

Thickener sizing based on non-JAC material could likely sustain a feed rate of 1.0 t/m$^{2/}$hr and still achieve underflow density in the 55-60% solids range with ~15 g/t of flocculant. JAC, with its higher clay content, would on the other hand require no more than 0.7 t/m$^{2/}$hr and be limited to no more than 55% solids. The higher flocculant dosage of ~60 g/t required to maintain a clear overflow with the JAC material may reduce the actual underflow density achievable when blends of Oculto and JAC material are to be processed to the 50-55% solids range.

The above indications would dictate to limit the proportion of JAC material fed to the plant at any time in order to ensure the most optimum operation of the CCD circuit, pre-leach thickener, and final tails thickener. Under such a scenario, a design solids loading of 0.8 t/m^2/hr could be adopted and with concurrent setting of the underflow target at 50-55% solids and ~30 g/t as flocculant addition rate.

13.2.8 SGS Lakefield – Phase 5B2 Test Program

A summary of the Phase 5B2 testwork program and results are presented in the following sections. For full details, refer to the report by SGS Lakefield, dated 29 April 2026.

13.2.8.1 Sample Receipt and Preparation

In January 2026, 11 boxes of individually packaged drill core interval samples were received in a single shipment at SGS Lakefield. Six boxes contained material in the extension area of the North East (NE) domain, with samples to be used for compositing. In addition, material previously submitted at the laboratory at the onset of the Phase 5B1 program but left untested was now covered by the scope of Phase 5B2. In total, 27 samples were tested individually and used to create four E-GRG composites and three MET composites.

Four composites were prepared for E-GRG testing by blending selected charges derived from the individual samples. The underlying intercepts were selected in order to create mineral domain composites, with the other main alteration encountered in each of them, besides the argillic material tested under Phase 5B1.

In addition, three MET composites were also tested. Two were created by blending selected charges from the individual samples of either the Oculto or JAC deposits, while the third was produced by combining equal portions of the first two MET composites, yielding a 50:50 blend of Oculto and JAC materials. Each MET composite was later involved in bulk cyanidation, CND, and rheology testing.

13.2.8.2 Phase 5B2 Samples Head Characterization

The 27 samples and three MET composites were submitted for head characterization and whole-rock analysis. Across the 30 composites analyzed, gold grades ranged from below the detection limit (0.01 g/t) to 4.99 g/t, while silver grades ranged from below the detection limit (10 g/t) to 1,380 g/t. Total sulphur values ranged from 0.57% to 11.3%, while arsenic ranged from below the detection limit (0.003%) to 0.26%.

It is worth noting that some of the samples provided, typically those with high total sulphur content, contained elevated amounts of copper, indicating that they were taken below the supergene-to-hypogene transition point. Of note, in this regard, are 09_MET_F5_JAC, with 0.52%Cu, and 13_MET_F5_NE, with 0.28%Cu. The latter thus contaminated the E-GRG composite NE-EXT, which assayed 0.1%Cu.

13.2.8.3 Phase 5B2 Metallurgical Testing

13.2.8.3.1 Extended Gravity Recoverable Gold (E-GRG) Testwork

Four composites were prepared for E-GRG testing: DG-VSIL, NE-SIL, SE-VSIL, and NE-EXT. The first three are covering the second most prevalent alteration in each of the mineral domain: either silicified (SIL) or vuggy silica (VSIL). The NE composite contained intercepts with the recently added pit extension (EXT) within this mineral domain.

E-GRG testing of the DG-VSIL composite determined final gravity recoverable values of 29.3% for gold and 22.5% for silver, with calculated head grades of 2.57 g/t gold and 72.9 g/t silver. Gravity concentration across three sequential stages produced gold and silver distributions as follows:

- Stage 1 (P_{80} 547 µm): 14.1% gold and 15.7% silver
- Stage 2 (P_{80} 190 µm): 7.8% gold and 5.3% silver
- Stage 3 (P_{80} 80 µm): 7.3% gold and 1.6% silver.

E-GRG testing of the NE-SIL composite yielded final gravity recoverable values of 27.2% for gold and 15.5% for silver, with calculated head grades of 0.75 g/t gold and 19.9 g/t silver. Gravity concentration across three sequential stages produced gold and silver distributions of:

- Stage 1 (P_{80} 557 µm): 11.7% gold and 6.8% silver
- Stage 2 (P_{80} 193 µm): 8.6% gold and 5.8% silver
- Stage 3 (P_{80} 80 µm): 6.9% gold and 2.8% silver.

E-GRG testing of the SE-VSIL composite produced final gravity recoverable values of 18.1% for gold and 54.3% for silver, with calculated head grades of 0.57 g/t gold and 507 g/t silver. The three-stage gravity process yielded gold and silver distributions of:

- Stage 1 (P_{80} 547 µm): 5.8% gold and 17.2% silver
- Stage 2 (P_{80} 190 µm): 7.5% gold and 27.0% silver
- Stage 3 (P_{80} 80 µm): 4.8% gold and 10.0% silver.

E-GRG testing of the NE-EXT composite yielded final gravity recoverable values of 16.1% for gold and 1.4% for silver, with calculated head grades of 0.64 g/t gold and 10.0 g/t silver. Across three sequential gravity stages, the gold and silver distributions were:

- Stage 1 (P_{80} 547 µm): 4.3% gold and 0.4% silver
- Stage 2 (P_{80} 190 µm): 5.7% gold and 0.5% silver
- Stage 3 (P_{80} 80 µm): 6.2% gold and 0.5% silver.

Table 13-19 to Table 13-22 summarize the E-GRG results for each of these composites.

Table 13-19: Phase 5B2 - DG-VSIL Composite E-GRG Results

Grind Size P_{80} (µm)	Product ID	Mass		Gold		Silver	
		g	%	Assay (g/t)	Dist'n (%)	Assay (g/t)	Dist'n (%)
P_{80} 547 µm	Stage 1 Conc Sampled Tails	76.1 278	0.38 1.39	95.44 2.22	14.1 1.2	2,965 57.7	15.7 1.1
P_{80} 190 µm	Stage 2 Conc Sampled Tails	78.1 318	0.39 1.59	51.46 2.18	7.8 1.3	984 58.6	5.3 1.3
P_{80} 80 µm	Stage 3 Conc Final Tails	55.7 19,194	0.28 96.0	67.36 1.82	7.3 68.2	416.6 57.1	1.6 75.1
	Totals (Head)	20,000	100.0	2.57	100.0	72.9	100.0
	Knelson Conc	210	1.05	71.63	29.3	1,558	22.5

Table 13-20: Phase 5B2 - NE-SIL Composite E-GRG Results

Grind Size P_{80} (µm)	Product ID	Mass		Gold		Silver	
		g	%	Assay (g/t)	Dist'n (%)	Assay (g/t)	Dist'n (%)
P_{80} 557 µm	Stage 1 Conc Sampled Tails	75.5 263	0.38 1.32	23.35 0.64	11.7 1.1	358 19.6	6.8 1.3
P_{80} 193 µm	Stage 2 Conc Sampled Tails	89.4 334	0.45 1.67	14.54 0.63	8.6 1.4	261 18.3	5.8 1.5
P_{80} 80 µm	Stage 3 Conc Final Tails	69.4 19,169	0.35 95.8	14.90 0.55	6.9 70.3	162.2 17.0	2.8 81.7
	Totals (Head)	20,000	100.0	0.75	100.0	19.9	100.0
	Knelson Conc	234	1.17	17.49	27.2	263	15.5

Table 13-21: Phase 5B2 - SE-VSIL Composite E-GRG Results

Grind Size P_{80} (µm)	Product ID	Mass		Gold		Silver	
		g	%	Assay (g/t)	Dist'n (%)	Assay (g/t)	Dist'n (%)
P_{80} 569 µm	Stage 1 Conc Sampled Tails	71.6 270	0.36 1.35	9.24 0.56	5.8 1.3	24,399 392.2	17.2 1.0
P_{80} 214 µm	Stage 2 Conc Sampled Tails	87.7 317	0.44 1.58	9.66 0.50	7.5 1.4	31,237 276.5	27.0 0.9
P_{80} 101 µm	Stage 3 Conc Final Tails	71.4 19,183	0.36 95.9	7.65 0.47	4.8 79.1	14,172 231.5	10.0 43.8
	Totals (Head)	20,000	100.0	0.57	100.0	507	100.0
	Knelson Conc	231	1.15	8.91	18.1	23,833	54.3

Table 13-22: Phase 5B2 - NE-EXT Composite E-GRG Results

Grind Size P_{80} (µm)	Product ID	Mass		Gold		Silver	
		g	%	Assay (g/t)	Dist'n (%)	Assay (g/t)	Dist'n (%)
P_{80} 568 µm	Stage 1 Conc Sampled Tails	83.6 239	0.42 1.22	6.46 0.68	4.3 1.3	10.0 10.0	0.4 1.2
P_{80} 204 µm	Stage 2 Conc Sampled Tails	104.4 264	0.53 1.34	6.91 0.66	5.7 1.4	10.0 10.0	0.5 1.3
P_{80} 80 µm	Stage 3 Conc Final Tails	92.2 18,894	0.47 96.0	8.53 0.55	6.2 81.2	10.2 10.0	0.5 96.0
	Totals (Head)	19,678	100.0	0.64	100.0	10.0	100.0
	Knelson Conc	280	1.42	7.31	16.1	10.1	1.4

13.2.8.3.2 Preliminary Cyanidation Testing

Twenty two samples underwent single 1 kg bottle roll cyanidation tests to evaluate gold and silver extraction, as well as lime and cyanide consumption. The purpose of this work was to expand the database of cyanidation results and assess further the variability of the response within each of the mineral domains.

The samples were tested along the standard conditions adopted during the Phase 5B1 testwork. Namely, the samples were ground to a target P_{80} of ~150 µm and leached at 45% solids using tap water, with pH maintained at 10.5-11.0 using lime. After four hours of air pre-aeration with air, sodium cyanide was added at 1.5 g/L for 24 hours, then allowed to decay to and maintained at 0.75 g/L for the remainder of the leach. Dissolved oxygen was controlled at 8-12 mg/L throughout leaching using air/oxygen sparging.

Final gold extraction from the 22 cyanidation tests ranged from 42% to 93% after 36 hours, with 16 samples achieving gold recoveries of 80% or higher. Three samples (CN-58 to CN-60) were leached for 48 hours, yielding gold extractions between 60% and 91%. Corresponding tailings gold grades ranged from below 0.01 g/t to 0.74 g/t. Silver extraction after 36 hours varied widely, from 8% to 96%, with only four samples exceeding 70% recovery. The three 48 hour tests produced silver extractions of 8% to 24%.

Sodium cyanide consumption ranged from 0.44 kg/t to 9.35 kg/t, the high end seen with samples of high copper content, with most samples below 1.10 kg/t, while lime consumption ranged from 0.39 kg/t to 3.59 kg/t, with the majority below 2.0 kg/t. Detailed results are presented in Table 13-23 and Table 13-24.

Table 13-23: Phase 5B2 - Gold Extraction by Cyanidation

Test ID	Sample ID	CN Residue P_{80} (µm)	% Au Extraction (Hours)						CN Residue Au (g/t)	Head Grade Au Calc (g/t)	Head Grade Au Direct (g/t)
			2	4	8	12	24	36			
CN-34	03_MET_F5_JAC	146	82	82	84	85	86	88	<0.01	<0.08	<0.01
CN-35	04_MET_F5_JAC	147	80	81	83	84	85	87	<0.01	<0.08	0.02
CN-36	07_MET_F5_JAC	154	80	82	84	85	86	87	<0.01	<0.08	0.01
CN-37	09_MET_F5_JAC	148	22	22	31	37	37	42	0.17	0.28	0.27
CN-38	01_MET_F5_SH	134	76	77	79	78	80	79	0.26	1.21	1.03
CN-39	02_MET_F5_SH	125	71	73	74	75	76	78	0.03	0.11	0.06
CN-40	03_MET_F5_SH	145	84	89	90	89	91	92	0.27	3.31	2.54
CN-41	04_MET_F5_SH	122	68	71	76	77	80	84	0.19	1.17	1.11
CN-41R	04_MET_F5_SH	143	71	75	78	80	83	85	0.19	1.22	1.11
CN-42	06_MET_F5_SH	156	69	82	85	85	85	86	0.22	1.53	1.40
CN-43	07_MET_F5_SH	121	85	90	88	87	88	89	0.05	0.46	0.41
CN-43R	07_MET_F5_SH	157	65	68	79	80	77	79	0.12	0.56	0.41
CN-44	01_MET_F5_SE	147	61	81	83	93	83	93	0.01	0.13	0.04
CN-45	04_MET_F5_SE	155	41	61	75	79	84	87	0.08	0.58	0.53
CN-46	05_MET_F5_SE	143	70	79	81	82	83	83	0.20	1.21	1.01
CN-47	06_MET_F5_SE	131	80	80	83	85	86	87	0.01	0.08	0.02
CN-48	10_MET_F5_SE	166	81	87	89	87	88	89	0.03	0.27	0.23
CN-49	02_MET_F5_DG	150	53	71	73	72	74	75	0.15	0.57	0.55
CN-50	05_MET_F5_DG	136	52	77	85	85	86	86	0.74	5.28	4.99
CN-51	05_MET_F5_NE	134	69	79	82	80	84	85	0.05	0.30	0.41
CN-52	07_MET_F5_NE	150	35	58	89	91	92	92	0.14	1.70	1.63
CN-58	12_MET_F5_NE	155	54	54	64	61	58	55	0.12	0.30	0.27

Test ID	Sample ID	CN Residue P_{80} (µm)	% Au Extraction (Hours)						CN Residue Au (g/t)	Head Grade Au Calc (g/t)	Head Grade Au Direct (g/t)
			2	4	8	12	24	36			
CN-59	13_MET_F5_NE	140	26	43	55	61	66	69	0.43	1.50	1.46
CN-60	14_MET_F5_NE	152	77	91	93	92	91	91	0.04	0.44	0.45

Table 13-24: Phase 5B2 - Silver Extraction by Cyanidation

Test ID	Sample ID	CN Residue P_{80} (µm)	% Ag Extraction (Hours)						CN Residue Ag (g/t)	Head Grade Ag Calc (g/t)	Head Grade Ag Direct (g/t)
			2	4	8	12	24	36			
CN-34	03_MET_F5_JAC	146	12	12	12	12	12	13	26.0	29.7	20.4
CN-35	04_MET_F5_JAC	147	68	69	71	71	73	72	<10	36.0	36.7
CN-36	07_MET_F5_JAC	154	51	52	55	57	57	58	43.9	104	96.9
CN-37	09_MET_F5_JAC	148	0.2	0.2	0.7	2	6	14	546	632	619
CN-38	01_MET_F5_SH	134	14	14	15	15	15	16	62.3	73.9	70.0
CN-39	02_MET_F5_SH	125	21	22	22	22	22	22	<10	12.8	<10
CN-40	03_MET_F5_SH	145	46	46	46	46	46	46	11.3	20.9	12.5
CN-41	04_MET_F5_SH	122	85	89	90	89	89	91	12.3	144	145
CN-41R	04_MET_F5_SH	143	83	81	86	85	87	88	17.1	147	145
CN-42	06_MET_F5_SH	156	8	9	9	9	10	10	<10	11.1	<10
CN-43	07_MET_F5_SH	121	13	14	14	14	14	15	<10	11.7	<10
CN-43R	07_MET_F5_SH	157	10	11	11	11	12	12	<10	<11	<10
CN-44	01_MET_F5_SE	147	77	84	85	84	84	86	10.0	70.1	69.6
CN-45	04_MET_F5_SE	155	72	89	92	89	93	96	55.6	1403	1380
CN-46	05_MET_F5_SE	143	24	28	31	31	32	34	14.6	22.0	20.7

Test ID	Sample ID	CN Residue P_{80} (µm)	% Ag Extraction (Hours)						CN Residue Ag (g/t)	Head Grade Ag Calc (g/t)	Head Grade Ag Direct (g/t)
			2	4	8	12	24	36			
CN-47	06_MET_F5_SE	131	45	46	46	46	45	47	<10	18.8	18.6
CN-48	10_MET_F5_SE	166	16	19	20	21	22	22	<10	12.9	<10
CN-49	02_MET_F5_DG	150	60	62	63	63	64	64	24.8	68.6	64.7
CN-50	05_MET_F5_DG	136	60	63	64	65	66	69	24.3	76.9	66.4
CN-51	05_MET_F5_NE	134	16	17	17	17	18	18	<10	12.2	<10
CN-52	07_MET_F5_NE	150	43	53	61	64	66	67	17.0	51.9	49.2
CN-58	12_MET_F5_NE	155	18	20	22	23	24	24	<10	<13	<10
CN-59	13_MET_F5_NE	140	1	2	3	4	6	8	<10	<11	<10
CN-60	14_MET_F5_NE	152	24	23	23	23	24	24	<10	<13	<10

13.2.8.3.3 Cyanidation Testwork at Finer Grind

Five additional cyanidation tests were conducted at a finer grind size, targeting a P_{80} of 80 µm, to evaluate the effect of increased liberation on gold and silver extraction on samples that provided poor extractions at the standard P_{80} of 150 µm. All other test conditions remained the same as those used in the initial cyanidation tests.

Detailed extraction efficiency results are provided in Table 13-25 and Table 13-26, where the 36 hour data is also presented in brackets the equivalent extraction seen from the same sample under the initial coarser grind size tested during Phase 5B1.

Although the results at a finer grind provided an improvement relative to those at the coarser target, silver dissolution remained relatively muted in all cases and only two samples performed much better for gold, albeit usually from a higher starting point (exception of 06_MET_F5_JAC, which saw an improvement from 73% to a reported 100% - or 86% if taking the average between the 24 and 48 hour readings).

Table 13-25: Phase 5B2 - Finer Grind Size Cyanidation: Gold Extraction Summary

Test ID	Sample ID	CN Residue P_{80} (µm)	% Au Extraction (Hours)							CN Residue Au (g/t)	Head Grade Au Calc (g/t)	Head Grade Au Direct (g/t)
			2	4	8	12	24	36	48			
CN-53	01_MET_F5_JAC	90	78	80	81	82	84	85 [86]	88	<0.01	<0.08	<0.01
CN-54	06_MET_F5_JAC	86	77	79	96	97	84	100 [73]	88	<0.01	<0.08	0.08
CN-55	02_MET_F5_SE	73	84	89	88	89	88	90 [88]	89	0.05	0.47	0.38
CN-56	03_MET_F5_SE	83	69	71	83	85	87	100 [85]	91	0.01	0.11	0.10
CN-57	08_MET_F5_DG	79	85	94	94	94	94	95 [98]	93	0.08	1.12	1.26

Table 13-26: Phase 5B2 - Finer Grind Size Cyanidation: Silver Extraction Summary

Test ID	Sample ID	CN Residue P_{80} (µm)	% Ag Extraction (Hours)							CN Residue Ag (g/t)	Head Grade Ag Calc (g/t)	Head Grade Ag Direct (g/t)
			2	4	8	12	24	36	48			
CN-53	01_MET_F5_JAC	90	46	51	56	59	62	64 [58]	64	60.4	166	144
CN-54	06_MET_F5_JAC	86	71	74	76	78	78	77 [70]	80	52.8	91.3	83.5
CN-55	02_MET_F5_SE	73	58	61	61	63	63	64 [61]	63	18.9	51.3	52.0
CN-56	03_MET_F5_SE	83	50	55	60	64	68	69 [64]	71	13.8	47.0	45.0
CN-57	08_MET_F5_DG	79	48	49	50	50	51	52 [46]	50	17.6	35.4	34.0

13.2.8.3.4 Bulk Cyanidation Testing

Bulk whole-ore cyanidation leaching was conducted on three MET composite samples to produce sufficient leach tailings for subsequent cyanide destruction and rheology testing. Leaching was performed at the same conditions as the preliminary cyanidation tests except that the fresh water added was treated with lime, aquarium salts - to simulate expected site water chloride levels - and 0.5 g/L of cyanide.

Bulk cyanidation tests (BL-4 to BL-6) on the JAC #2, Oculto #2, and 50:50 JAC/Oculto MET composite samples achieved gold extractions of 76% to 87% and silver extractions of 74% to 92%.

13.2.8.3.5 Counter-Current Decantation Simulation

Each bulk leach tailing from the three MET composite cyanidation tests underwent a simulated CCD step to produce underflow material for cyanide destruction testing. Following completion of bulk leaching, the tailings were diluted 2:1 with synthetic process water with pH adjusted to 10.5–11.0 using lime and 0.5 g/L of sodium cyanide. The diluted pulp was mixed, allowed to settle, and then decanted to restore a pulp density of 45% solids. The decanted solution represented the simulated CCD overflow and was sampled for analysis, while the remaining slurry (simulated CCD underflow) was advanced to cyanide destruction testing.

Table 13-27 lists the assay results for the simulated CCD solutions which, as the overflow of this circuit, would feed the Merrill-Crowe circuit (albeit not tested in Phase 5B2) and, as part of the underflow slurry, would report to the CCD and cyanide destruction circuits.

Table 13-27: Phase 5B2 - CCD Simulated Solution Chemical Assays

Element	Units	BL-4 [JAC]	BL-5 [Oculto]	BL-6 [50:50]
CN_T	mg/L	408	441	408
CN_{WAD}	mg/L	391	424	416
CN_F	mg/L	419	409	429
CNS	mg/L	<2	8.5	2.7
CNO	mg/L	<2.5	<2.5	<2.5
Hg	mg/L	0.19	0.11	0.15
Al	mg/L	<0.2	3.2	0.3
As	mg/L	<3	<3	<3
Ba	mg/L	0.043	0.150	0.059
Be	mg/L	<0.002	<0.002	<0.002
Bi	mg/L	<1	<1	<1
Ca	mg/L	89.4	55.3	80.7
Cd	mg/L	<0.09	<0.09	<0.09
Co	mg/L	<0.3	<0.3	<0.3
Cr	mg/L	<0.1	<0.1	<0.1
Cu	mg/L	2.8	11	6.8
Fe	mg/L	0.6	10.7	3.7

Element	Units	BL-4 [JAC]	BL-5 [Oculto]	BL-6 [50:50]
K	mg/L	12	27	13
Li	mg/L	<2	<2	<2
Mg	mg/L	0.35	0.81	0.51
Mn	mg/L	0.5	0.75	1.28
Mo	mg/L	<0.6	<0.6	<0.6
Na	mg/L	881	981	943
Ni	mg/L	<0.6	<0.6	<0.6
P	mg/L	<5	<5	<5
Pb	mg/L	<2	<2	<2
Sb	mg/L	<1	<1	<1
Se	mg/L	<3	<3	<3
Sn	mg/L	<2	<2	<2
Sr	mg/L	0.092	0.12	0.089
Ti	mg/L	<0.02	<0.02	<0.02
Tl	mg/L	<3	<3	<3
V	mg/L	<0.2	<0.2	<0.2
Y	mg/L	<0.02	<0.02	<0.02
Zn	mg/L	<0.7	<0.7	<0.7

13.2.8.3.6 Cyanide Destruction Testing

CND testing was conducted using the SO_2/air process on the simulated CCD underflow slurry from the JAC #2, Oculto #2, and JAC/Oculto MET composite samples. The objective was to achieve a solution-phase CN_{WAD} concentration of less than 1 mg/L. After completing preliminary batch tests, extended continuous CND testing was performed under varying operating conditions until the target CN_{WAD} levels were consistently met. For the 50:50 JAC/Oculto composite, the treated pulp from optimized conditions was subsequently advanced to rheology testing.

An initial batch test on the JAC #2 MET composite tails from BL-4 nearly met the target, achieving 1.28 mg/L CN_{WAD} with additions of 7.13 g SO_2/g CN_{WAD} and 50 mg/L copper at a 90 minute retention time.

Subsequent continuous tests varied SO_2 dosage, copper addition, retention time, and reactor configuration but did not consistently achieve the <1 mg/L CN_{WAD} target, with final CN_{WAD} values ranging from 1.70 to 6.58 mg/L. Improvements were observed when retention time and reactor number were increased, but results remained above the target.

Successful cyanide destruction was ultimately demonstrated in a continuous test with increased copper addition (260 mg/L), 8.96 g SO_2/g CN_{WAD}, a 90 minute retention time, pH 8.5, and a single 2 L reactor. This test achieved CN_{WAD} concentrations of 1.05 mg/L by titration and 0.55 mg/L by direct assay. The Oculto #2 MET composite (BL-5) feed solution characterization indicated elevated cyanide levels (441 mg/L CN_T and 424 mg/L CN_{WAD}) with moderate copper and iron concentrations. Preliminary batch testing was successful, achieving a CN_{WAD} concentration of 1.20 mg/L using 5.07 g SO_2/g CN_{WAD}, 70 mg/L copper, and a 60 minute retention time.

However, subsequent continuous testing under variable SO_2 dosage, copper addition, retention time, and reactor configuration, was unable to consistently achieve the target CN_{WAD} level of <1 mg/L. The most effective continuous configuration was provided when employing a two-stage reactor system (total retention time of ~182 minutes), increased SO_2 and copper additions (8.64 g SO_2/g CN_{WAD} and 510 mg/L Cu), and pH control at 8.5. This test reduced CN_{WAD} to approximately 1.25-1.29 mg/L.

Overall, the testing demonstrated that additional copper was required to catalyze cyanide destruction, but excessive copper increased SO_2 demand due to stable copper-cyanide complexes, resulting in higher reagent consumption. Copper additions evaluated during the program ranged from 80 mg/L to 510 mg/L.

The test program determined that a discharge CN_{WAD} concentration of less than 1 mg/L for the simulated CCD underflow of the Oculto #2 composite can be achieved under the following operating conditions:

- 45% solids (w/w)
- 9.0 g equivalent SO_2/g CN_{WAD}
- 510 mg/L copper addition
- pH 8.5
- 180 minutes retention time.

A third composite, the JAC/Oculto composite (a 50/50 blend of JAC #2 and Oculto #2 composites), underwent cyanide destruction optimization. Preliminary batch testing was successful, achieving a CN_{WAD} concentration of 1.27 mg/L using 4.59 g SO_2/g CN_{WAD}, 50 mg/L copper, and a 60 minute retention time.

Subsequent continuous testing generally achieved the target residual CN_{WAD} of <1 mg/L by adjusting SO_2 dosage, copper addition, and retention time, with final CN_{WAD} values ranging from 0.37 mg/L to 1.06 mg/L. The most effective result achieved a CN_{WAD} of 0.49 mg/L with a copper addition of 240 mg/L.

Overall, the SO_2/air process was demonstrated to be effective for the JAC/Oculto composite, with optimized conditions identified as 7.72 g SO_2/g CN_{WAD}, 240 mg/L copper, pH 8.5 and 60 minute retention time, producing CN_{WAD} concentration of 0.37 mg/L by direct assay.

13.2.8.3.7 Rheology Testing

A slurry rheology test was conducted on a post-cyanide destruction tailings sample from the JAC/Oculto MET composite (CND 6). The sample was received as a naturally thickened slurry without flocculant and was tested as received. Rheological behavior was evaluated over a range of solids contents based on stability and flowability. The intent was to probe the behaviour of this specific 50:50 JAC:Oculto split as it was touted as the upper limit of the JAC content that should be tolerated as plant feed, which would be managed on the mining side through stockpiles. This establishes the basis for sizing of the plant's thickeners.

The CND 6 tailings had an initial solids concentration of approximately 65% (w/w), exhibiting a yield stress of 22 Pa under unsheared conditions and 12 Pa after sustained shearing. Measurements at or below 59.6% solids were deemed unreliable due to solids settling during testing, and testing at 45% solids was not conducted because of slurry instability and rapid settling.

At solids contents of 64.7% (w/w) and above, the sample displayed rheopectic (flow-adverse) behavior, where resistance to flow increased under constant shear. Table 13-28 summarizes the rheology results.

Table 13-28: Phase 5B2 - Rheological Characteristics of JAC/Oculto MET Composite Detoxified Leach Tails

Test Code	Solids % w/w	Unsheared Sample			Constant Shearing	Sheared Sample Shear			Observations
		Shear Rate $\dot{g}$	Yield Stress $\tau_{y\beta}$	Plastic Viscosity η_P	Viscosity η_C	Shear Rate $\dot{g}$	Yield Stress $\tau_{y\beta}$	Plastic Viscosity η_P	
		range, 1/s	Pa	mPa.s	mPa.s	range, 1/s	Pa	mPa.s	
CSD = ~65% solids, corresponding to ~22 Pa unsheared and 12 Pa sheared yield stress.									
T53	71.4	Torque Overload							
T49	69.6	400-600	127	144	424-467	100-300	48	300	Rheopectic
T50	67.5	400-600	74	41	210-251	100-300	25	172	Rheopectic
T51	64.7	200-400	19	49	97-123	100-300	12	89	Rheopectic
T52	61.6	200-400	8.6	31	54-71	100-300	6.2	54	None
T48	59.6	200-400	5.0	24	39-54	100-300	4.0	44	Minor settling
T47	54.3	200-400	1.9	14	22-39	Invalid due to fast settling			Moderate settling
T46	49.3	200-400	0.7	9.7	14-47				Moderate settling

Notes:

The CND 6 Detox Tailings samples did not contain flocculant.

The values are based on data produced by the unsheared and sheared slurry sample.

Variable shearing was produced in the 0 to 600 s^{-1} range, increasing and decreasing (up and down curves). Constant shearing was produced by subjecting the slurry sample to a constant rotation at 300 1/s for 180 seconds. Bingham Plastic parameters: yield stress ($\tau_{g\beta}$) and plastic viscosity (η_P) values, for the specified $\dot{g}$ range.

$\dot{g}$ - Shear rate range at which the rheological parameters were calculated.

13.2.8.3.8 Phase 5B2 Conclusions

Whole-ore cyanidation testing at a P_{80} of 150 µm expanded the database of tests performed at the retained optimum conditions. Large variability in the results was still observed, with the tests in Phase 5B2 yielding gold extractions between 42% and 93% after 36 hours, while silver extractions ranged from 8% to 96%. Lower recoveries were generally associated with lower head grades. Sixteen of the 22 composites achieved gold recoveries of 80% or greater, whereas only four achieved silver recoveries above 70%. This outcome for silver is mostly pegged on the more variable modes of occurrence and texture than seen for gold.

Finer grind cyanidation testing at a P_{80} of 80 µm resulted in consistently high gold recoveries of 88% to 91% and improved silver recoveries of 50% to 80% after 48 hours.

Cyanide destruction tests performed on bulk cyanidation tails, using the SO_2/air process, were successful in achieving a target residual CN_{WAD} concentration of <1 mg/L and optimal operating conditions were defined for the three composites tested. Improved mixing conditions and higher DO levels were shown to be potential avenues for improving reagent consumptions and/or reaction kinetics.

Slurry rheology testing on the JAC/Oculto post-cyanide destruction tailings indicated a critical solids density (CSD) of approximately 65% (w/w), with an unsheared yield stress of about 22 Pa. The tailings exhibited rheopectic behavior at slurry density above the CSD, indicating increased resistance to flow under sustained shear. Blending down the content of JAC fed to the plant and reporting to tailings, and likely within the CCD circuit, to no more than 50% may therefore circumvent the more restrictive operating conditions around thickening duties that were seen when pure JAC samples were submitted to rheology and dynamic thickening testwork earlier.

13.3 Analysis of Testwork Databases

The valid data generated from the various testwork phases has been analyzed in order to provide indications as to the relevant design criteria for plant design. Where applicable, data validity was confirmed on the basis of applying filters meant to remove data points with feed grades out of the range expected as plant feed material, from samples out of the proposed pit limits, or obtained without having applied many of the optimum leach feed test conditions retained for plant design.

Excluding samples with gold or silver feed grades at multiple times higher than the expected reserves average grades, with a copper content likely associating them to the hypogene zone of the deposit, or that did not meet a silver equivalent threshold grade, used as an equivalent to a cut-off grade, was performed by applying the following filters:

- Gold feed grade >3.0 g/t
- Silver feed grade >600 g/t
- Silver equivalent feed grade <40 g/t
- Copper content > 200 g/t

13.3.1 Optimum Grind Size

Figure 13-9 and Figure 13-10 plot the 24 hour leach recoveries achieved at different grind size versus the gold and silver head grades. The same filters described under Section 13.3 were applied to the database. The 24 hour mark was retained as it is the longest leach time applied to most of the tests realized at the finer grind size obtained in Phase 1; therefore, it was the only leach time that facilitated a direct comparison over an extended range of P_{80}.

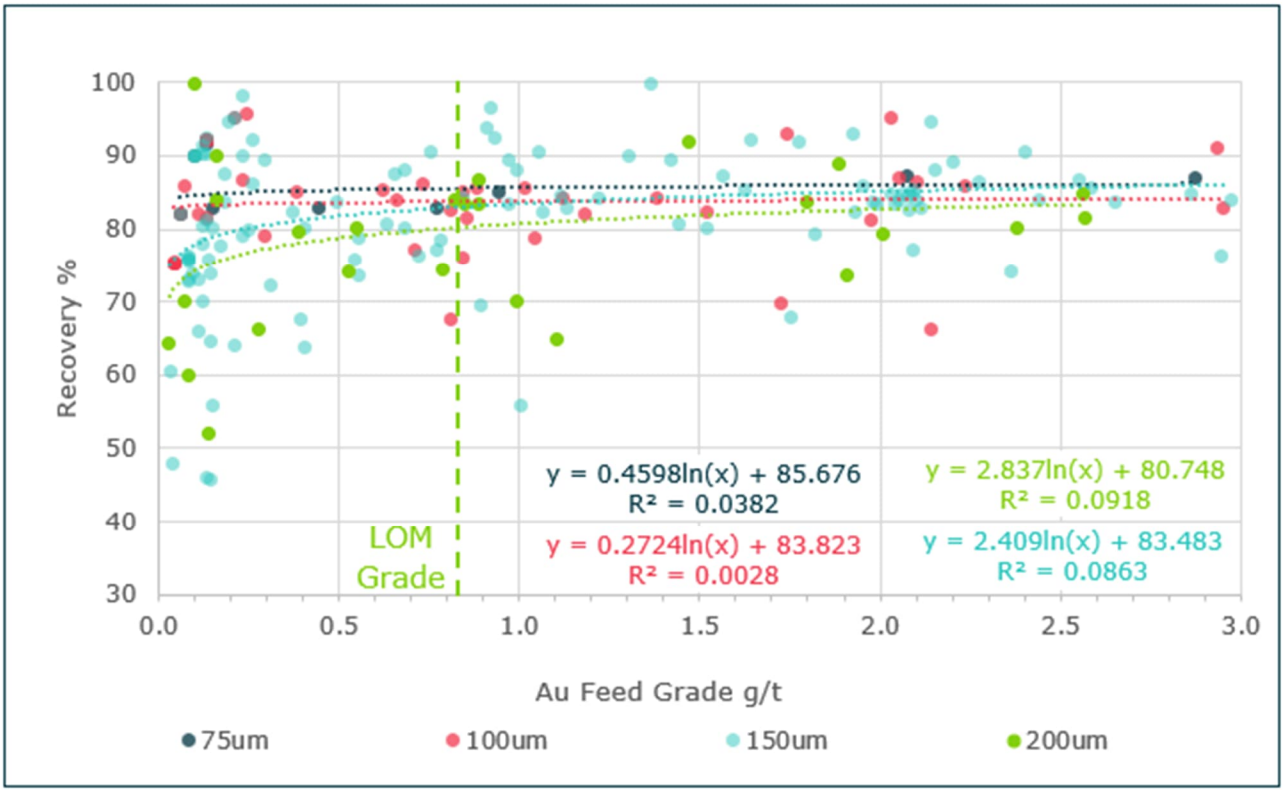


Figure 13-9: Gold Leach Recovery versus Gold Feed Grade at Variable P_{80}– 24- hours Leach

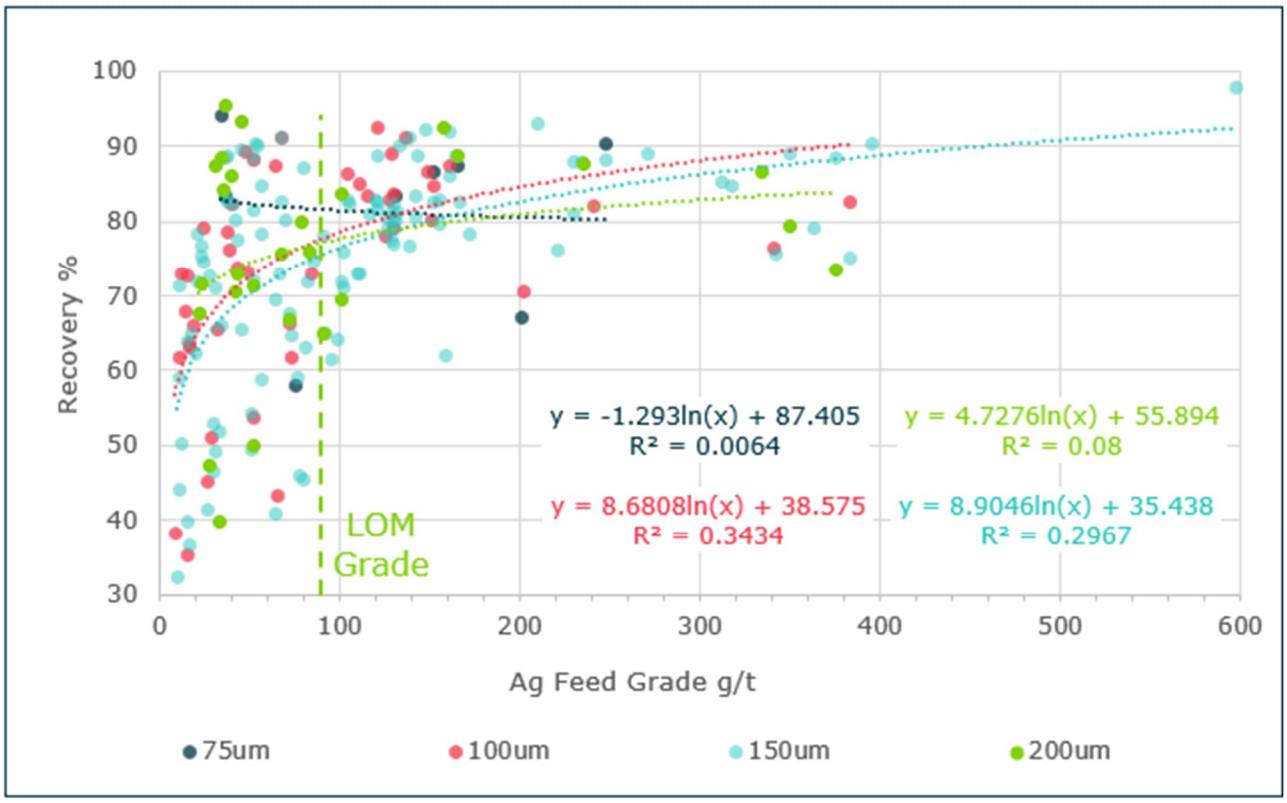


Figure 13-10: Silver Leach Recovery versus Silver Feed Grade at Various P_{80} – 24 hours Leach

Although such analysis bears the caveat of comparing data generated under different leach conditions besides the noted change of P_{80} (starting and held cyanide concentrations, DO levels), the figures illustrate that grinding to 100 µm P_{80} does not provide a significant improvement in leach recovery, especially for gold, when compared to 150 µm. However, increasing the grind size to 200 µm substantially reduces gold and silver recoveries.

This specific comparison is further strengthened by the analysis of the paired data sets generated during Phase 5A, at both 150 and 200 µm. When the outcome of both Phase 5A programs (ALS and SGS Lakefield) are analyzed together, the sets of paired tests evaluating the effect of the primary grind size and leach time duration on the eventual leach recovery of gold and silver indicate unweighted recovery differentials, as outlined in Table 13-29.

Table 13-29: Combined Phase 5A Analysis of Grind Size and Leach Retention Time Effect on Gold and Silver Extraction by Cyanidation

Mineral Domain	Number of Paired Tests	Average P_{80} (µm)	% Au Extraction (Hours)			% Ag Extraction (Hours)		
			24	36	48	24	36	48
Deep Gold	8	157	84.0	84.9	85.7	81.7	83.1	84.3

Mineral Domain	Number of Paired Tests	Average P_{80} (µm)	% Au Extraction (Hours)			% Ag Extraction (Hours)		
			24	36	48	24	36	48
		204	81.1	81.8	83.6	80.5	81.4	83.0
		differential	2.9	3.1	2.1	1.2	1.7	1.3
JAC	2	149	77.0	78.5	80.0	79.3	79.9	80.1
		198	82.0	77.0	79.0	78.2	77.9	78.7
		Differential	-5.0	1.5	1.0	1.1	2.0	1.5
North East	5	143	84.9	86.5	86.8	84.4	85.6	85.8
		202	82.7	82.8	83.3	81.5	81.5	82.0
		Differential	2.2	3.8	3.5	2.8	4.1	3.8
Shallow Gold	3	164	80.2	81.2	86.4	76.2	76.5	78.1
		205	68.4	73.0	83.5	66.5	70.5	74.6
		Differential	11.8	8.2	2.9	9.7	6.0	3.5
Silver Enrichment	5	145	77.9	78.8	75.3	75.4	76.4	76.6
		196	74.1	83.4	85.3	74.2	74.8	76.8
		Differential	3.8	-4.5	-10.0	1.2	1.6	-0.2
Overall (weighted[1])	23	150	83.3	84.5	84.7	80.5	81.6	82.2
		200	80.2	81.8	83.5	78.5	79.0	80.4
		Differential	3.0	2.7	1.2	2.0	2.6	1.8

Note:

[1]Weighted values are considering the ounces distribution between mineral domains as per the PFS data.

Per Table 13-29, the weighted recovery benefits for providing the finer P_{80} of 150 µm is 2.6% for gold and 2.7% for silver, in absolute terms. Relative to the leach circuit retention time provided, there is a diminishing return in extending the size of the circuit above 36 hours. While going from 24 to 36 hours would deliver a recovery increment of 1.2% for gold and 1.1% for silver, pushing from 36 hours to 48 hours would only increase these values by a further 0.2% and 0.6%, respectively. It is noted that, in the case of some mineral domains, such conclusions rely on a limited number of paired tests.

Nevertheless, the current indications are that, without providing more tankage volume, the operators could take advantage of the limited viscosity increment seen with a higher pulp density than the 45% solids used as a base case scenario and select a higher value in the plant to deliver the incremental retention time at no extra capital cost.

The grade characteristics of each mineral domains identified and tested individually are important for interpreting the differential recovery data generated under different test conditions in a proper light: a drop of 10% in gold recovery for the SG domain may seem quite relevant but is considered differently once the proportion of ounces to be supplied from it, based on a prospective mine plan, is known to be limited to 5.3% of the project total.

Along the same lines, relying on mathematical averages to interpret results within a given domain is also a potential source of bias as the feed grade ranges and mineralogy between tested samples can be quite variable.

13.3.2 Grindability Data Analysis

The most relevant data was provided by Phase 1, Phase 3, and Phase 5B1. Phase 1 provided 30 sets of SMC data (Axb, ta, SAG circuit specific energy (SCSE), and drop weight index (Dwi)) and 31 BBWi. Phase 3, covering only JAC, provided a further nine values of SCSE, DWi and BBWi. Phase 5B1 added 25 BBWi values.

Table 13-30 provides a statistical analysis of the available SAG milling-related test data, by GU, as well as indicating weighted average values for Oculto and the whole LOM tonnage base, as available from a February 2026 prospective mine plan.

Table 13-30: Average and Percentiles of Filtered SMC Grindability Data

GU	LOM Tonnage (%)	Data Count	SCSE (kWh/t)		Dwi (kWh/m^3)	
			Average	P_{80}	Average	P_{80}
SG	4.0	14	8.43	9.24	4.27	5.37
SE	17.4	18	8.34	9.00	4.15	5.00
DG	20.2	21	8.37	9.08	4.41	5.49
NE	42.6	12	8.88	9.21	4.94	5.62
Oculto	84.3	66	8.62	9.14	4.62	5.45
JAC	15.7	9	7.26	7.97	1.89	2.47
Diablillos	100.0	75	8.41	8.95	4.19	4.98

Some of the SMC testwork realized during Phase 1 involved composite samples (15 of 75 samples), integrating multiple drill intercepts in the material tested. Such an approach tends to mask the variability that may exist within each GU, albeit likely providing an average aligned with the eventual value if enough determinations are available. Using such data to size grinding mills by relying on some percentile value of the database (e.g. P75 or P80) may be fraught because of this hidden variability, and issue which may be further compounded if the number of data is limited.

Table 13-31 presents the data relevant to ball milling. Because theclosing screen in use was changed between the earlier determinations and the ones from Phase 5B1 to reflect a product P_{80} more aligned with the now settled grind target of 150 µm, the two data sets may not be totally compatible. It is noted as well that the “estimated” BBWi values, extended by ALS during Phase 1 to nearly all of the samples on the basis of correlations between grind calibration time and the actual BBWi of some samples, are not considered in Table 13-31 as it not obtained following the standard Bond procedures.

Table 13-31: Average and Percentiles of Filtered BBWi Grindability Data

GU	LOM Tonnage	# BBWi data		BBWi, all data (kWh/t)		BBWi @ ~120 µm (kWh/t)		BBWi @ ~160 µm (kWh/t)
	%	@~120 µm	@160 µm	Average	P80	Average	P80	Average
SG	4.0	1	2	18.53	20.44	21.80	Na	16.90
SE	17.4	2	5	15.65	17.33	14.35	15.10	16.17
DG	20.2	11	9	15.73	17.04	11.69	16.10	14.48
NE	42.6	7	7	16.85	19.14	16.22	16.94	17.24
Oculto	84.3	21	23	16.41	18.32	15.01	16.58	16.34
JAC	15.7	9	0	14.86	16.60	14.86	16.60	N/A
Diablillos	100.0	30	23	16.17	18.05	14.99	16.59	N/A

The number of individual BBWi measurements per GU is limited in most cases to a total that does not lend much credibility to the statistics presented in Table 13-31, especially for a percentile-based analysis.

Potential correlations linking any of the hardness parameters with some head grade components available in the block model were attempted but none provided a fit sufficient to justify such for predictive purposes.

13.3.3 Leach Recovery Prediction Analysis

All available leach data from the historical and Phase 5A testwork described in Section 13.2.3 are plotted in Figure 13-11 and Figure 13-12. The graphs show the leach recoveries versus the gold and silver head grade, respectively, at different leach retention times.

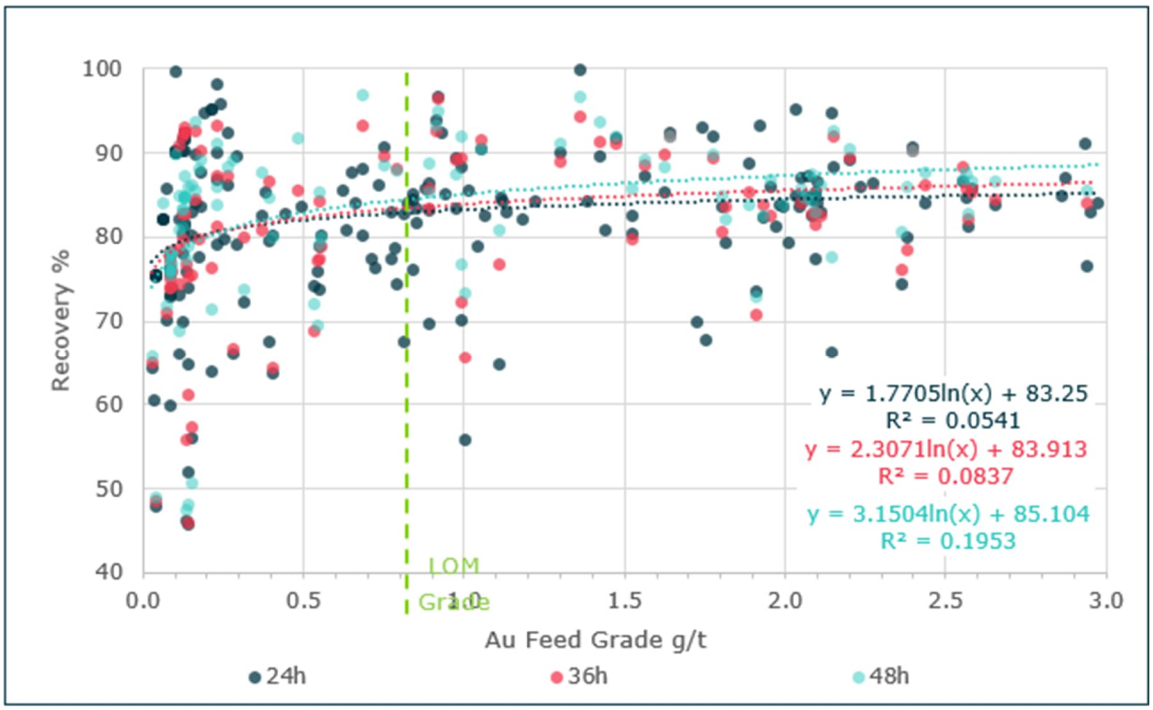


Figure 13-11: Gold Leach Recovery versus Gold Feed Grade at Variable Leach Duration – Filtered Data

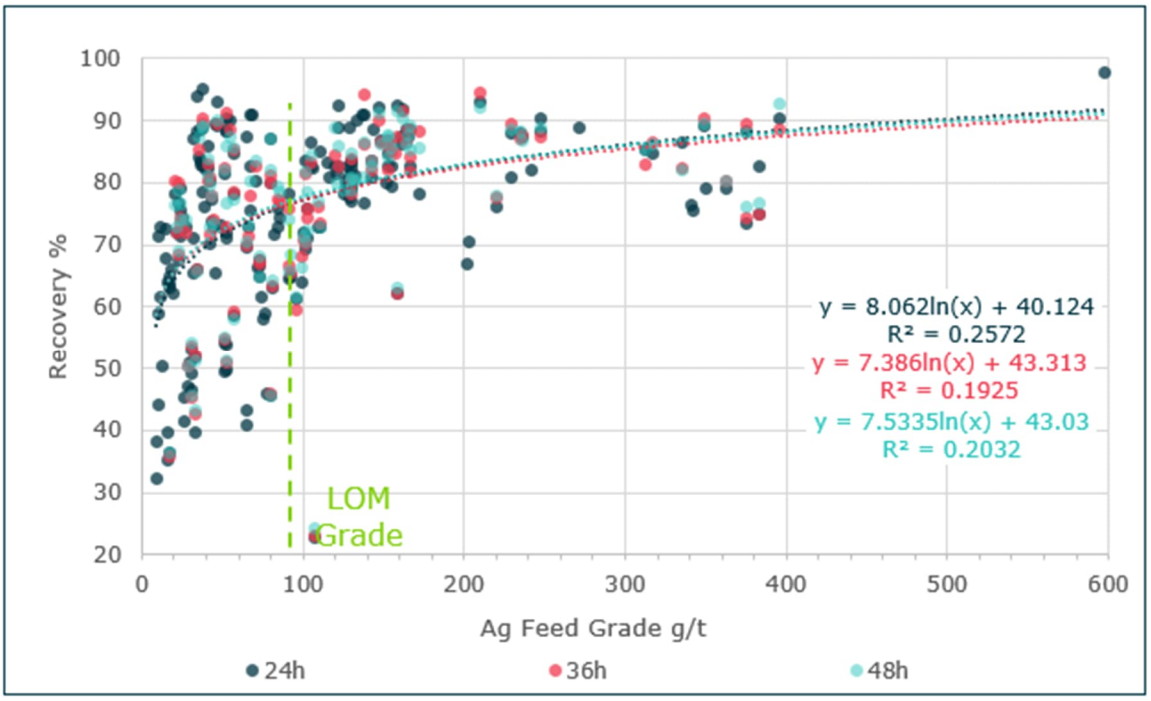


Figure 13-12: Silver Leach Recovery versus Silver Feed Grade at Variable Leach Duration – Filtered Data

Figure 13-11: shows a slight increase in gold recovery when retention time is extended from 36 to 48 hours near the LOM grade. However, Figure 13-12 shows that silver recoveries remain virtually unchanged across the three retention times at similar grades. Considering the uncertainty in the recovery curves, it was concluded that the optimal retention time remains 36 hours. Extending retention to 48 hours offers no significant improvement in recovery and is therefore not economically justified.

Figure 13-13 and Figure 13-14 plot the leach recoveries versus the gold and silver head grades, respectively, but only for the tests performed at the design conditions for retention time and grind size, namely 36 hour retention time and P_{80} grind size of ~150 µm. The same filters defined above were applied to the database for excluding deemed outliers.

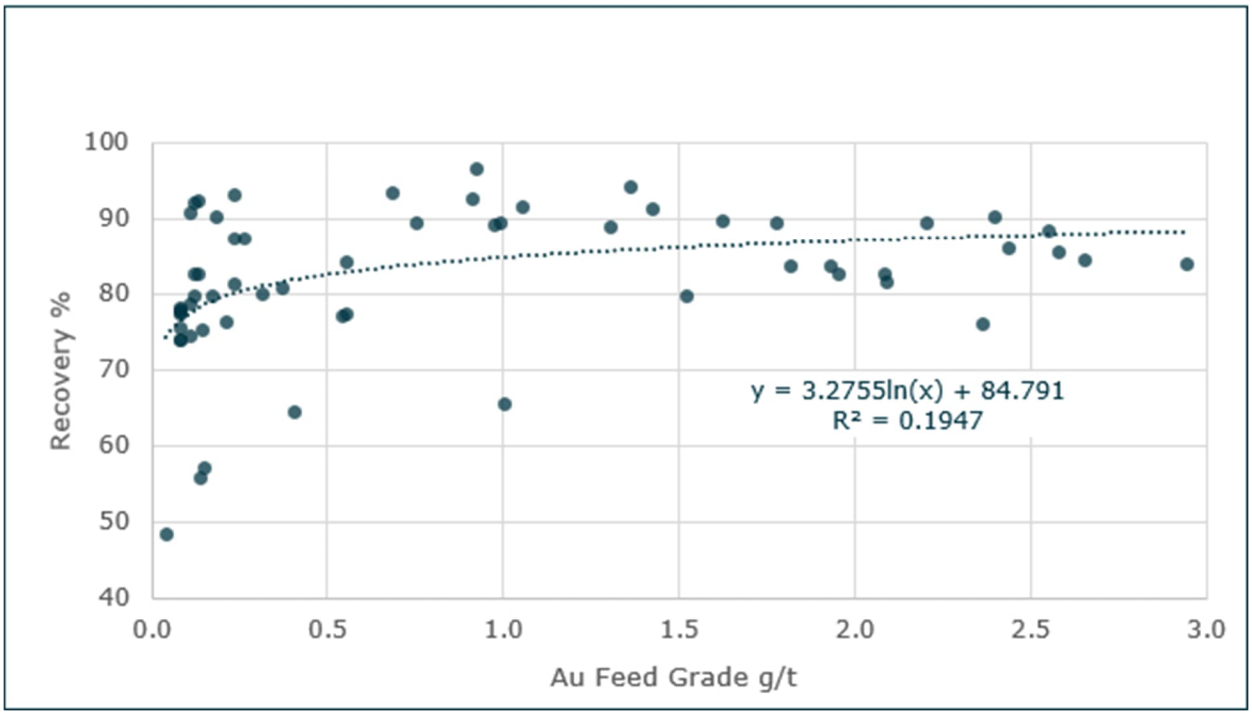


Figure 13-13: Gold Leach Recovery versus Gold Feed Grade – at Design Conditions and Filtered Data

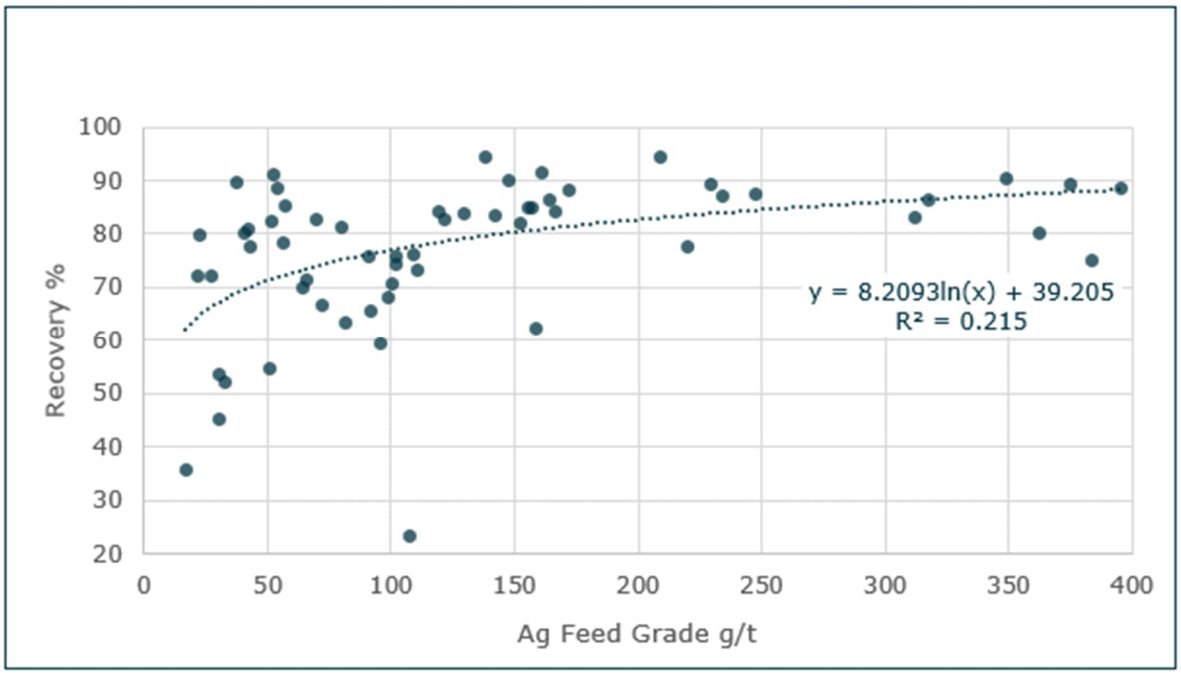


Figure 13-14: Silver Leach Recovery versus Feed Grade – at Design Conditions and Filtered Data

As there is no obvious correlation linking feed grade to recovery expectation, as per Figure 13-13 and Figure 13-14, the filtered leach recovery data at design conditions were analyzed after grouping the data points by the individual GU that each represents. The resulting scatter plots are presented in Table 13-15 and Figure 13-16.

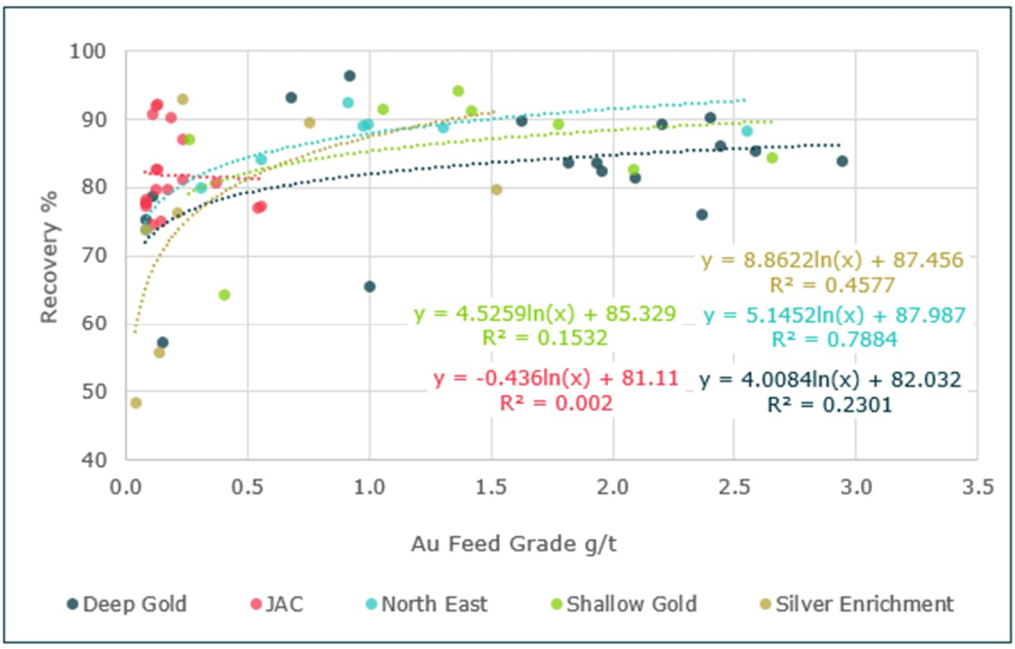


Figure 13-15: Gold Leach Recovery versus Gold Feed Grade per GU - Filtered Data

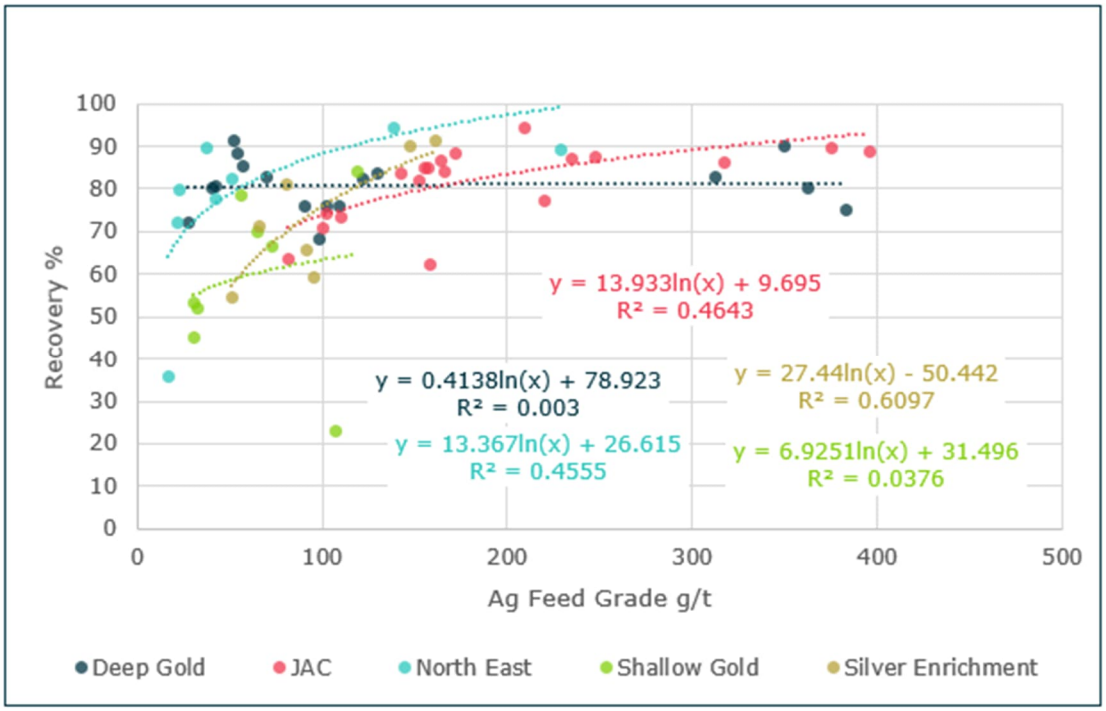


Figure 13-16: Silver Leach Recovery versus Silver Feed Grade per GU - Filtered Data

Recovery equations are typically derived using the type of relationship identified in Table 13-15 to Figure 13-16 to support project's economics. However, statistical analysis of the relevant datasets indicates an absence of significant correlation (low R-Squares) between the feed grade and metallurgical recovery, with the possible exceptions of gold recovery for the NE GU and silver recovery with SE GU.

Given the lack of predictive reliability, it is recommended that fixed recovery values be adopted for the DFS. Because of the wide range of feed grades tested, assigning a direct mathematical average that is derived from testwork results of tests completed at conditions similar to those adopted from Phase 5B1 is not deemed appropriate. Instead, it is proposed that the Project considers a weighted average against the metal units represented by each of the relevant tests.

This approach is assuming that there is no synergistic, nor detrimental effect in processing a blend of individual ore blocks, as represented by each of the samples tested – and no indication to the contrary has been unearthed by the testwork programs. It further assumes that each of these samples within a given GU is carrying the same representativity within said GU. This aspect may be contested as a wide range of metallurgical responses, with similar feed grades and a single GU, have been registered – mostly affecting silver recovery. Through the data analysis, it was found likely that unquantified and variable mineralogical factors, other than those defined by the classification within a given GU, or by alteration, lithology or mineralization types, are driving the metallurgical response. Because of the lack of specific markers available within the block model, the weighting along metal units remains the most appropriate approach at this time to help modulate the expectations of metallurgical response.

As an illustration of such an assertion, feeding the plant with material from a sample (assimilated to an ore block) of a given GU that carries a head grade of 50 g/t of silver and provides a test leach recovery of 50% and simultaneously another, as a blend, from the same GU but with 150 g/t of silver that yields a test recovery of 80% should not be deemed as projecting an overall recovery of 65% ([50+80]/2) for the contained silver units, as would be derived on a mathematical basis. Instead, it should be deemed to be 72.5% ([50%*50 + 80%*150]/200), as obtained on a silver units weighing.

In a similar fashion, the eventual calculation of the overall metal recoveries for the regrouped GUs forming Oculto should be based on the individual contribution of each GU towards the total metal units provided.

Table 13-32 summarizes the leach recoveries per GU from the filtered test data from Phase 2 to Phase 5B2, on the basis of the metal unit weighting principle defined above. The Phase 1 data is specifically not applicable as the leach duration of 24 hours used then is shorter than the adopted 36 hour retention time. This leaves a database containing 88 valid data points.

Table 13-32: Proposed Leach Recoveries per GU on a Metal Unit Weighted Approach Basis

Mineral Domain	Number of Data Points	Au Recovery (%)	Ag Recovery (%)
Shallow Gold	12	85.9	60.4
Silver Enrichment	13	83.3	73.8
Deep Gold	26	84.1	79.5
North East	16	86.0	82.2
JAC	21	80.2	80.5

The filtered data typically show a bias towards higher median tested head grades than those announced by the mine plan, albeit with the silver content in the SE samples being an exception with a much lower value than the LOM average, as illustrated in Table 13-33.

Table 13-33: Comparison of Filtered Cyanidation Database Median Head Grades versus LOM Head Grades

Mineral Domain	Testwork Data		LOM Plan	
	Au	Ag	Au	Ag
Shallow Gold	1.29	60.1	0.87	53.9
Silver Enrichment	0.21	65.8	0.45	119.0
Deep Gold	1.33	129.0	0.90	87.9
North East	0.92	23.8	0.84	30.3
JAC	0.11	158.0	0.06	135.5

Although no definite correlation was found between recovery expectations and feed grades, it is recommended to introduce some samples in an expanded database that would allow the revised median values to approach further the indicated LOM averages. This could also be used incorporate more samples of the selected mineral domain carrying the argillic alteration as, although it is found to be predominant for all domains but especially so for DG, NE and SE, it is under represented in the database, with a higher weighing towards the vuggy silica alteration.

13.3.4 Gravity Concentration Recovery Analysis

The gravity recovery expectations are derived from the E-GRG testwork completed through Phase 5B. The data generated using an approach where all the material is ground to the final leach feed P_{80}, presented to a centrifugal device and the concentrate of which is then tabled to reduce the residual weight to about 0.6% of the total feed weight (the SGS standard procedure) is not deemed to provide a testing environment aligned with an actual industrial gravity concentration circuit, as demonstrated by the later development of the E-GRG methodology.

The E-GRG data generated facilitates segregation of the recovery expectations, not only per GU but also while considering the expected split of resource tonnage and metal units per the main alteration types present in each GU.

The interpretation of the raw E-GRG data into industrial-scale projection relies on a few assumptions laid out in a paper by Dr. A. Laplante, the developer of the E-GRG procedure. It is mostly stating that:

- Centrifugal concentrator, or centrifuge, used to recover gravity-recoverable gold were found to be quite efficient down to a fine grain size of the liberated precious metals (PM)
- Coarse PM grains will recirculate multiple times into the centrifuge, before reporting to the downstream circuit, because of their high specific gravity (S.G.) and the poor separation efficiency with such material of the cyclones found in typical close-circuited ball milling circuit. This allows for a high probability of recovery by the centrifuge
- Fine PM grains have a higher probability of reporting to the cyclone overflow and thus to not be presented in the underflow through the centrifuge, despite the benefit provided by the high S.G. Their probability of capture into a gravity concentrate will be more closely associated with the so-called “intensity” of the gravity circuit, defined as the proportion of the overall cyclone underflow stream that will be regularly presented to the gravity circuit proper.
- The E-GRG procedure, as deployed at laboratory scale, was found to be indicative of the optimum recovery that may be expected from the coarse fraction of a given ore. Because of the typical limitations encountered in an industrial environment, multiple benchmarking exercises indicated that a well operated gravity circuit with coarse PM may be expected to achieve no more than 70% of the E-GRG indicated recovery potential.

From the above, the actual E-GRG data was interpreted in such a way as to assign a recovery efficiency of 70% to coarse PM particles and an efficiency aligned with the intended “intensity” of the industrial circuit to the fine PM particles. The transition between coarse and fine in this case was taken conservatively as 53 µm.

Table 13-34 presents the E-GRG test results analysis and the breakdown of the overall E-GRG contributed by the +53 and -53 µm material. For assigning a recovery expectation to the fines fraction, a gravity circuit intensity (GCI) of 50% and 25% were calculated.

Table 13-34: Proposed Gravity Concentration Recoveries per GU - Weighted per Main Alterations

Sample	E-GRG Recoveries (%)						Scaled-Up Overall Rec (%)			
	Overall		>53 µm		<53 µm		@ 50% GCI		@ 25% GCI	
	Au	Ag	Au	Ag	Au	Ag	Au	Ag	Au	Ag
JAC-ARG	24.9	34.5	7.2	30.3	17.7	4.2	13.9	23.3	9.5	22.3
SE-ARG	19.0	19.5	3.7	12.9	15.4	6.6	10.3	12.3	6.4	10.7
SE-VSIL	18.1	54.3	10.2	51.0	7.9	3.3	11.1	37.4	9.1	36.5
DG-ARG	16.7	4.1	2.4	2.8	14.3	1.3	8.8	2.6	5.2	2.3
DG-VSIL	29.3	22.5	10.3	18.8	19.0	3.7	16.7	15.0	12.0	14.1
NE-ARG	31.6	6.0	4.2	3.0	27.4	3.0	16.6	3.6	9.8	2.9
NE-SIL	27.2	15.5	3.8	13.8	23.4	1.7	14.4	10.5	8.5	10.1

Based on the differential recovery expectations between a gravity circuit allowing an intensity of 50% versus 25%, the higher value was selected by AbraSilver to proceed with the inclusion of such circuit in the design criteria. These differentials are mostly driven by the amount of gold and silver expected to report to the -53 µm size fraction and the large difference in recovery expectation that halving the gravity circuit intensity would then imply.

Based on a preliminary mine plan, Table 13-35 provides a view of what an estimate of the tonnage and metal units distribution between the different alterations looks like for each GU. The distribution of tonnage proportions per alteration were obtained from the PFS mine plan.

Table 13-35: Tonnage and Metal Units Distribution Between Mineral Domain and Alterations

GU	% of Prospective LOM			Alteration	% of GU Tonnage
	Resource Tonnage	Au Units	Ag Units		
Deep Gold	20.2			ARG	51.1
		27.4	23.7	SIL	12.0
				VSIL	36.8
North East	42.6			ARG	50.7
		54.0	17.2	SIL	33.1
				VSIL	16.1
Shallow Gold	4.0			ARG	34.9
		5.3	2.9	SIL	28.9

GU	% of Prospective LOM			Alteration	% of GU Tonnage
	Resource Tonnage	Au Units	Ag Units		
				VSIL	36.2
Silver Enrichment	17.4			ARG	52.1
		11.9	27.7	SIL	8.5
				VSIL	39.3
JAC	15.7	1.4	28.4	ARG	100

Based on Table 13-35, the E-GRG composites tested would be representative of 77% of the GU/alteration combinations found in the deposit. As for the gold and silver units, if it is assumed that each alteration within a given GU holds the overall metal units of said GU in equal proportions as to each of the alteration tonnage ratio, the coverage provided by the E-GRG testwork would be 68% for the gold units and 92% for silver units. The actual grades per GU/alteration combinations could be used to further refine the assignment of an overall gravity recovery projection for each GU, if and when available.

Combining the data from Table 13-33 and Table 13-34 forms the basis for assigning a gravity recovery for each GU, as the LOM does not carry the details of the breakdown of tonnages and head grades by alterations. The following assumptions were made for this exercise:

- The head grades of each alteration are the same as the one for the related GU (e.g. no in situ enrichment between GU)
- The gravity recovery of the untested alteration within a given GU will be attributed the weighted average (by tonnage proportion) of the other alterations for which data is available.
- As no E-GRG data was generated for SG, representing only 5.3% of the LOM gold and 2.9% of the LOM silver, the overall gravity recovery was calculated by using the lowest recovery per alteration type from the other GUs tested and applying the tonnage distribution for SG as shown in Table 13-34.
- As the high E-GRG result for silver recovery with the SE-VSIL composite (37.4%) is deemed to be a potential artefact of the elevated silver grade of the sample processed (507 g/t), the value was cut back to the same level as achieved for gold with the SE-VSIL composite (11.1%).

Integrating these considerations with the results presented in Table 13-34 and Table 13-35 allows for the derivation of the overall gravity concentration recovery projections for each GU, as provided in Table 13-36.

Table 13-36: Projected Metal Recovery by Gravity Concentration – Pre-Intensive Leaching

Deposit	Au Recovery (%)	Ag Recovery (%)
Shallow Gold	11.2	9.4
Silver Enrichment	10.6	11.8

Deposit	Au Recovery (%)	Ag Recovery (%)
Deep Gold	12.1	7.8
North East	16.4	7.1
JAC	13.9	23.3

The recovery data presented in Table 13-36 has to be further derated for consideration of the efficiency of the downstream circuits that will be handling the centrifuge concentrate, before the product reaches a doré format. The PFS design criteria included an intensive leach reactor (ILR) for dissolving the gold and silver into a high-grade PLS. The DFS design proposes to retain this step but replace the intended mixing of the ILR PLS with the Merrill-Crowe feed with a dedicated electrowinning cell, plating the precious metals onto wool cathodes as a sludge, to be treated by mercury retorting before being smelted in an induction furnace.

Although some tests were carried out by SGS Lakefield on a “proxy” gravity concentrate, as produced by the SGS standard gravity concentration procedure, with a resulting dissolution rate of 98% for gold within 72 hours and 100% for silver within 48 hours, it is expected that an industrial unit would benefit from the use of higher DO levels than the 8-12 mg/L used in the SGS test. As well, the addition of a leach accelerant – either hydrogen peroxide or a proprietary formulation from ILR vendors – can be expected to reduce the required retention time into the typical ~16 hours for a batch to be complete while yielding similarly high metal dissolution. Values of 99% for gold and 97% for silver ILR recoveries are proposed for the design criteria. Any metals not dissolved through the ILR are deemed refractory and will not provide further recovery potential once fed to the subsequent agitated leach circuit. The net recovery for electrowinning is deemed to be 100% as the step recovery is likely to approach 99% and the balance not recovered will remain in the agitated leach circuit and will be recirculated in the barren solution, where losses to CCD are deemed to be ≤0.1% for gold and ≤0.2% for silver.

13.3.5 Overall Recovery Projections

Table 13-37 presents the overall process plant recoveries for each GU. These values are calculated based on the leach recoveries presented in Table 13-32 and considering the contribution of the gravity circuit pursuant to Table 13-34. Reductions for the ILR and CCD circuit efficiencies are included.

Table 13-37: Process Plant Overall Recoveries

Deposit	Au Recovery (%)	Ag Recovery (%)
Shallow Gold	87.7	64.1
Silver Enrichment	85.5	76.4
Deep Gold	85.8	80.6
North East	88.2	82.1
JAC	82.9	83.9

13.4 Conclusions and Recommendations

An extensive amount of testwork was performed on the Diablillos deposit during 2021 to 2023, from Phase 2 and Phase 3, to prepare the PFS. The samples were collected from different mineral domains of the deposit and the testwork shows that the upper oxide layer of the orebody is amenable to cyanidation while care had to be taken to not reach into the underlying sulphide zone, where the elevated amount of copper would not conform to the design criteria developed based on the testwork herein. For the DFS, more testwork under Phase 5A and Phase 5B, with new and old samples, was performed to confirm and further optimize the PFS flowsheet.

The mineralogy analysis on Oculto and JAC samples revealed that the primary components are quartz, alunite, jarosite, and iron oxides. Pyrite occurs mainly as free, liberated, or associated with hard silicates and complex forms. Gold is present as native gold or electrum, while silver appears preferentially as iodargyrite and chlorargyrite, although acanthite and “oxidized” silver minerals are also reported. Sulphide content in the oxide layer is low and the total and organic carbon were below the detection limits, suggesting that the material is unlikely to exhibit preg-robbing behaviour during processing.

Based on the comminution tests, the majority of the material is of friable to moderate competency and of medium to high hardness. This means that the ore breaks readily under impact to generate a mid-size product in the SAG mill. When this friable to moderate competency is combined with high ore hardness, which is typical of most Diablillos geological samples, the ball mill can become the circuit bottleneck.

Gravity tests on Oculto and JAC materials confirmed the presence of gravity-recoverable gold and silver. The SGS Lakefield approach of realizing gravity concentration testing, followed by cyanidation of the gravity tails, did not provide a firm basis for expecting incremental recovery versus a whole-ore leach approach. Nevertheless, gravity concentration was applied in the procedure on 100% of the material ground to the target grind because agitated leach does not reflect the real environment of an industrial-scale circuit. AbraSilver conducted an internal analysis and directed that gravity concentration and associated equipment be included in the DFS flowsheet.

Due to the high silver to gold ratio in the orebody, Merrill-Crowe remains the most economical process option for the Project. Tests conducted in the PFS stage confirmed that Merrill-Crowe was very efficient in recovering precious metals from the PLS.

New cyanide destruction testwork results were available with the Phase 5B2 testwork. They indicated that residual WAD cyanide concentrations of less than 1 g/t may be achievable under strictly controlled conditions; however, this outcome was dependent on reagent addition rates and retention time considered for the test conditions. Confirmation of performance at a DFS design scale and operating conditions will require additional verification should the CN_{WAD} limit of 1 mg/L be maintained. The results indicated that better mixing and, likely, higher DO levels (instead of the fixed air flowrate used in the test procedure) could improve kinetics and reagent consumption.

The use of oxygen to replace air in the industrial leach circuit should also benefit the CND circuit efficiency.

Based on the historical testwork results, the following process conditions are considered for the DFS:

- Grind size of P_{80} 150 µm
- Incorporation of a gravity concentration circuit, along with intensive cyanidation and electrowinning
- 36 hour agitated leach retention time at 45% w/w solids, with limited efficiency penalty for increasing towards 50% solids under higher plant feed throughput
- Six stages of CCD with a wash ratio of three at 50% w/w solids, with 55% solids feasible under low proportion of JAC content in the plant feed
- One hour of retention time for cyanide destruction
- Tailings thickener underflow target of 55% w/w solids, with the possibility to reach 60% solids with a low proportion of JAC content in the plant feed
- Reagent consumptions:
  - 1.3 (Oculto) and 1.2 (JAC) kg/t of sodium cyanide for leach
  - 1.1 (Oculto) and 2.9 (JAC) kg/t of lime for leach
  - 300% stoichiometric addition of zinc to gold, silver, and mercury at a 2:1 dilution ratio. Further dilution to 3:1 may require incremental zinc.
  - 4.7 (Oculto) and 4.1 (JAC) g SO_2/g CN_{WAD} for cyanide destruction to achieve less than 10 mg/L of CN_{WAD}.

During the PFS phase, SGS Lakefield determined that leach and Merrill-Crowe precious metal recoveries showed no significant variation when using tap water, site water from Pond 7, or synthetic water formulated to replicate Pond 7 water. Based on this observation, most DFS testwork was conducted with tap water except for the bulk cyanidation tests, which utilized synthetic water for replicating possible viscosity effects in the CCD and CND circuits

Although metallurgical testwork indicates that the use of site water does not materially impact metal recovery performance, process water quality is expected to deteriorate relative to site water due to continuous recirculation within the plant and the ingress of suspended solids and high levels of ions from the tailings storage facility (TSF) reclaim water.

The following activities are recommended to support the next phase of the Project:

- Enlarge the BBWI database with additional hardness determinations that cover most mineral domains (less the marginal SG, tonnage-wise) as the current extent thereof is insufficient to apply a percentile approach to the design criteria.
- Enlarge the cyanidation database by including more samples that meet the grade filters used for the data analysis above and carrying the argillic alteration in samples at or below the average LOM head grades from the DG, SE, and NE mineral domains.

- Perform additional oxygen-uptake testwork to confirm the eventual consumption rate.
- Introduce Eh and DO as control variables while carrying CND optimization testwork instead of using fixed reagent addition rates.
- Confirm the quality of the TSF reclaim water and evaluate the impact on metal recoveries.

14. Mineral Resource Estimate

14.1 Summary

Mr. Luis Rodrigo Peralta, FAusIMM, CP (Geo), Principal Resource Geologist and independent external consultant from INSA Consultora for AbraSilver Resource Corp. (AbraSilver), is responsible for the Mineral Resource Estimate ("MRE") for the Oculto, JAC, Fantasma, Sombra and Laderas zones of the Diablillos Project. This MRE incorporates revised metal prices and metallurgical recoveries for the Diablillos Project, a new geological and alteration model for the Oculto, JAC, Fantasma, Sombra and Laderas zones and a doble process mineral resource for the tank and heap leaching material.

All holes not drilled by AbraSilver are referred to as historical drillholes. AbraSilver completed diamond drill holes between 2017 to 2026. This totalled 790 drill holes and 170,368m. The cut-off date for the diamond drill holes is December 31st, 2025.

The MRE has been based on a subset of drilling data as detailed in Section 10. Drill holes have been excluded that were outside of the Diablillos block model limit and drill holes without assay results have been excluded from the MRE. The subset of drilling data included 790 drill holes (both diamond and reverse circulation) of which 365 were historical drillholes and 425 were AbraSilver drillholes, this totalled 92,647 meters of drilling.

Verification of drill data was summarised in Section 12 and the Mr. Peralta was satisfied that drill data was collected in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Mineral Exploration Best Practice Guidelines (CIM, 2018) and Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (CIM, 2019). It was deemed suitable for use in a Mineral Resource estimate.

Diablillos is a high sulphidation epithermal silver-gold deposit with strong supergene overprinting. The principal controls to alteration and mineralization are predominantly structural with a mixed influence imparted by lithology. The combination of structural and alteration control has generated a steeply dipping and shallowly dipping control considered in the Mineral Resource estimate. The estimation domains were defined using a combination of alteration domains and lithology domains. A subset of mixed domains was defined for each lithology and alteration type.

Based on the drill hole database a new 3D Model of lithology and alteration domains was created. The block model was generated in MS MinePlan mining software. A statistical study of the gold and silver behaviour for each domain was used to support block model grade interpolation. Gold and silver grades were estimated using Ordinary Kriging ("OK") and potential bias was reviewed using an Inverse Distance method ("ID2"). Drill hole intervals have been composited to a length of 1 meter, which is the most common sample length.

Grade capping has been applied to composited grade intervals on a case-by-case basis in each domain.

A specific gravity (SG) model has been built and applied to the block model, based on measurements from 8,624 core samples. Bulk density was assigned to the block model as averages for each lithology and alteration domain, including oxidation / sulphides zone subsets. The average bulk density is 1.82 t/m3 for cover material, 2.32 t/m3 for mineralized material and 2.23 t/m3 for waste material.

Mr. Peralta (“QP”) has performed a visual comparison of block model sections against drill traces; a review of comparison statistics; and undertaken check estimates. He is satisfied that the MRE is consistent with the CIM best practice guidelines (CIM, 2019).

The MRE for the Diablillos deposit, with an effective date of May 6th, 2026, has been constrained by an optimized open pit shell. It is reported at an approximate equivalent cut-off grade of 39 g/t AgEq for tank leach process and 15 g/t AgEq for heap leach process. The MRE has been categorized in accordance with the CIM Definition Standards (CIM, 2014) and comprises Measured, Indicated and Inferred Mineral Resources as summarized in Table 14-1.

To define the cut-off grade, a methodology was used based on a Net Value per Block (“NVB”) calculation. The NVB was based on "Benefits = Income - Cost" being positive, where,

> *Income = [(Au Selling Price (USD/oz) - Au Selling Cost (USD/oz)) x (Au grade (g/t)/31.1035)) x Au Recovery (%)] + [(Ag Selling Price (USD/oz) - Ag Selling Cost (USD/oz)) x (Ag grade (g/t)/31.1035)) x Ag Recovery (%)]*
>
> *Cost = Mining Cost (USD/t) + Process Cost (USD/t) + Transport Cost (USD/t) + G&A Cost (USD/t) + [Royalty Cost (%) x Revenue].*

The NVB method assumed a total mine operating cost of USD 32.58/t for tank leach process which resulted in an average equivalent cut-off grade of approximately 39 g/t AgEq. For heap leach process an average equivalent cut-off grade of approximately 15 g/t AgEq was used.

Table 14-1: Total Diablillos Mineral Resource Summary (Tank & Heap Leach) – As of April 30, 2026.

Zone	Zone	Category	Tonnes (000 t)	Ag (g/t)	Au (g/t)	AgEq (g/t)	Contained Ag (000 Oz Ag)	Contained Au (000 Oz Au)	Contained AgEq (000 Oz Ag)
Tank Leaching	Oxides	Measured	41042	100	0.68	159	131 668	896	209 281
		Indicated	60978	41	0.58	92	81 060	1 143	180 078
		Measured & Indicated	102021	65	0.62	119	212 728	2 039	389 359
		Inferred	14400	25	0.57	74	11 468	262	34 187
Heap Leaching	Oxides	Measured	25469	13	0.09	19	10 997	76	15 425
		Indicated	104491	7	0.13	15	24 328	428	49 342
		Measured & Indicated	129960	8	0.12	16	35 325	503	64 767
		Inferred	34947	6	0.14	14	6 939	158	16 153
Total	Oxides	Measured	66512	67	0.45	105	142 665	971	224 706
		Indicated	165469	20	0.30	43	105 388	1 570	229 420
		Measured & Indicated	231981	33	0.34	61	248 053	2 542	454 127
		Inferred	49347	12	0.26	32	18 406	420	50 340

Notes for Mineral Resource Estimate:

1. Mineral Resources are not Mineral Reserves and have not demonstrated economic viability.
2. The formula for calculating AgEq is as follows: Silver Eq Oz = Silver Oz + Gold Oz x (Gold Price/Silver Price) x (Gold Recovery/Silver Recovery).
3. The Mineral Resource model was populated using Ordinary Kriging grade estimation within a three-dimensional block model and mineralized zones defined by wireframed solids, which are a combination of lithology and alteration domains. The 1m composite grades were capped where appropriate.
4. The Mineral Resource is reported inside a conceptual open pit shell derived using US$ 34.50/oz Ag price, US $3,200/oz Au price, 86.6% process recovery for Au, and 80.9% process recovery for Ag, for the tank leaching and 74.3% process recovery for Au, and 46.8% process recovery for Ag, for the secondary heap leaching.

5. Open pit optimization was constrained using a dual-process approach, with tank leaching as the primary process (total opex of US$32.58/t) and heap leaching as the secondary process (total opex of US$7.00/t).
6. The MRE has been categorized in accordance with the CIM Definition Standards (CIM, 2014).
7. A Net Value per block [NVB] calculation was used to constrain the Mineral Resource, determine the "Benefits = Revenue-Cost", where, Revenue = [(Au Selling Price (US$/oz) - Au Selling Cost (USD/Oz)) x (Au grade (g/t)/31.1035)) x Au Recovery (%)] + [(Ag Selling Price (US$/oz) - Ag Selling Cost (USD/Oz)) x (Ag grade (g/t)/31.1035)) x Ag Recovery (%)] and Cost = Mining Cost (US$/t) + Process Cost (US$/t) + Transport Cost (US$/t) + G&A Cost (US$/t) + [Royalty Cost (%) x Income]
8. The Mineral Resource is sub-horizontal with sub-vertical feeders and has a reasonable prospect for eventual economic extraction by open pit methods.
9. In-situ bulk densities were assigned to each model domain, according to samples averages for each lithology domain, separated by alteration zones and subset by oxidation.
10. All tonnages reported are dry metric tonnes and ounces of gold and silver contained are troy ounces.
11. Mining recovery and dilution factors have not been applied to the Mineral Resource estimates.
12. The Mineral Resource was estimated by Luis Rodrigo Peralta, B.Sc., FAusIMM CP (Geo), an INSA Consultora Managing Principal Geologist, and an Independent Qualified Person under NI 43-101.
13. Mr. Peralta is not aware of any environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues that could materially affect the potential development of the Mineral Resource.
14. All figures are rounded to reflect the relative accuracy of the estimates. Minor discrepancies may occur due to rounding to appropriate significant figures.

14.2 Drilling Data

The MRE has been based on a subset of the drill hole database reported in Section 10 of the Technical Report. The subset included 790 drill holes, with 596 diamond drill holes ("DDH") for 123,013m and 194 reverse circulation drill holes ("RC") for 47,355m. In total 170,368 meters of drilling for the Oculto, JAC, Fantasma, Laderas, and Sombra zones.

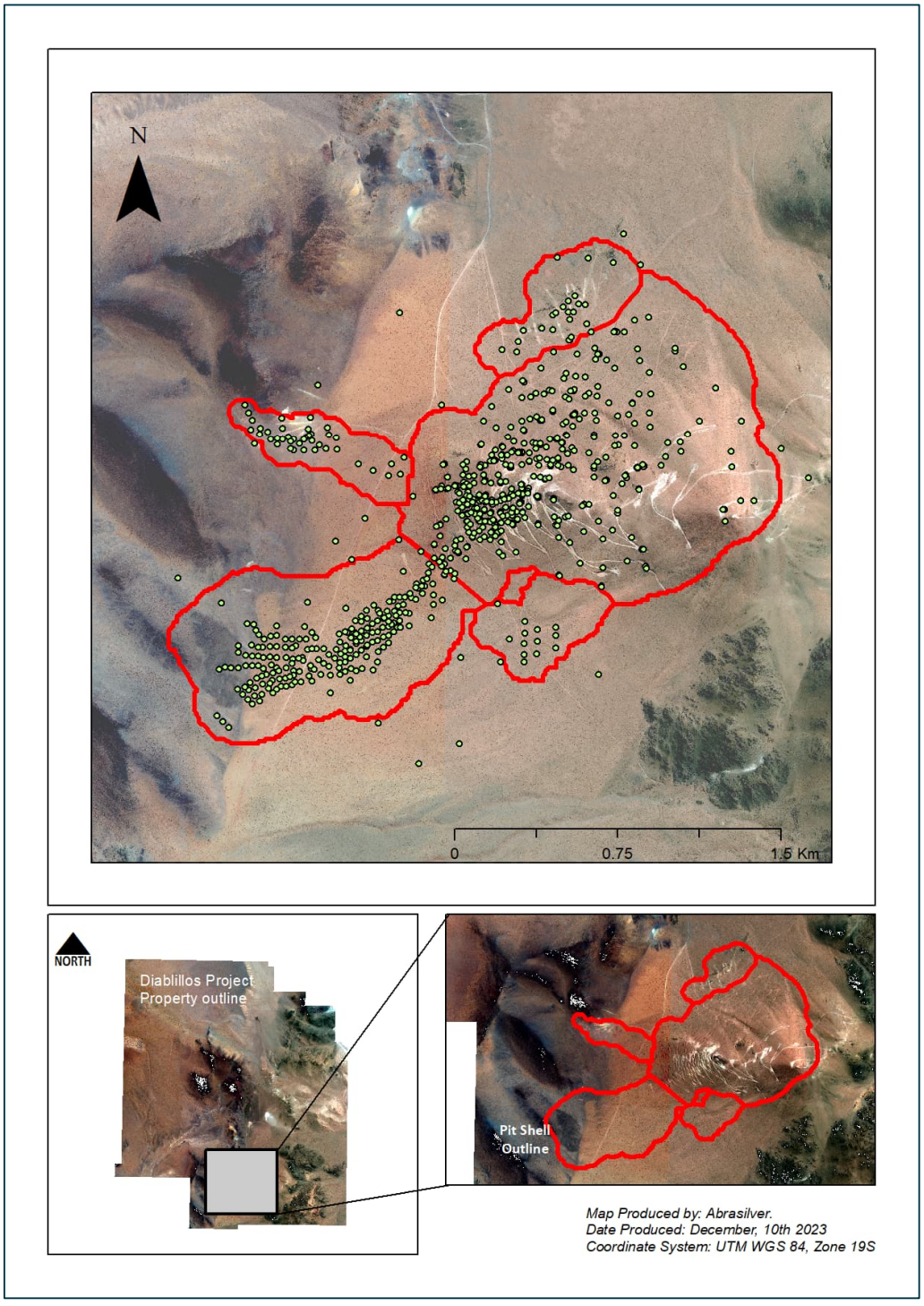
Drill holes used in the MRE have been summarized in Table 14-2. Table 14-3 shows a summary of excluded holes. Figure 14-1 and Figure 14-2 show the location of holes used in the MRE by drilling type and company.

Table 14-2: Summary of Drill Holes used in the Mineral Resource Estimate

Drilling Campaign	Type of Hole	Number of Holes	Meters Drilled	Average Meters Drilled	Min Metters Drilled	Max Metters Drilled
1990	RC	25	3,260	141	71	250
1993	DDH	5	1,225	200	146	254
1994	DDH	12	2,016	167	25	255
1996	RC	32	8,540	266	113	400
1997	RC	94	24,651	262	49	413
1997	DDH	19	4,558	239	31	380
1998	RC	24	7,547	314	220	370
1999	DDH	5	1,330	265	190	450
2003	RC	11	1,918	174	84	282
2005	RC	8	1,439	184	114	252
2007	DDH	46	9,804	213	31	365
2008	DDH	49	7,585	154	40	355
2012	DDH	7	659	95	41	125
2017	DDH	28	3,149	112	40	327
2019	DDH	2	844	421	380	464
2020	DDH	34	9,200	270	50	610
2021	DDH	61	14,872	243	50	451
2022	DDH	82	19,119	233	22	410
2023	DDH	80	14,521	181	122	245
2024	DDH	81	15,763	195	107	446
2025 (P - IV)	DDH	27	5,098	205	112	775
2025 (P - V)	DDH	58	13,270	228	122	1025
Subtotal	RC	194	47,355	224	109	328
Subtotal	DDH	596	123,013	213	92	394
Grand total		790	170,368	216	98	405

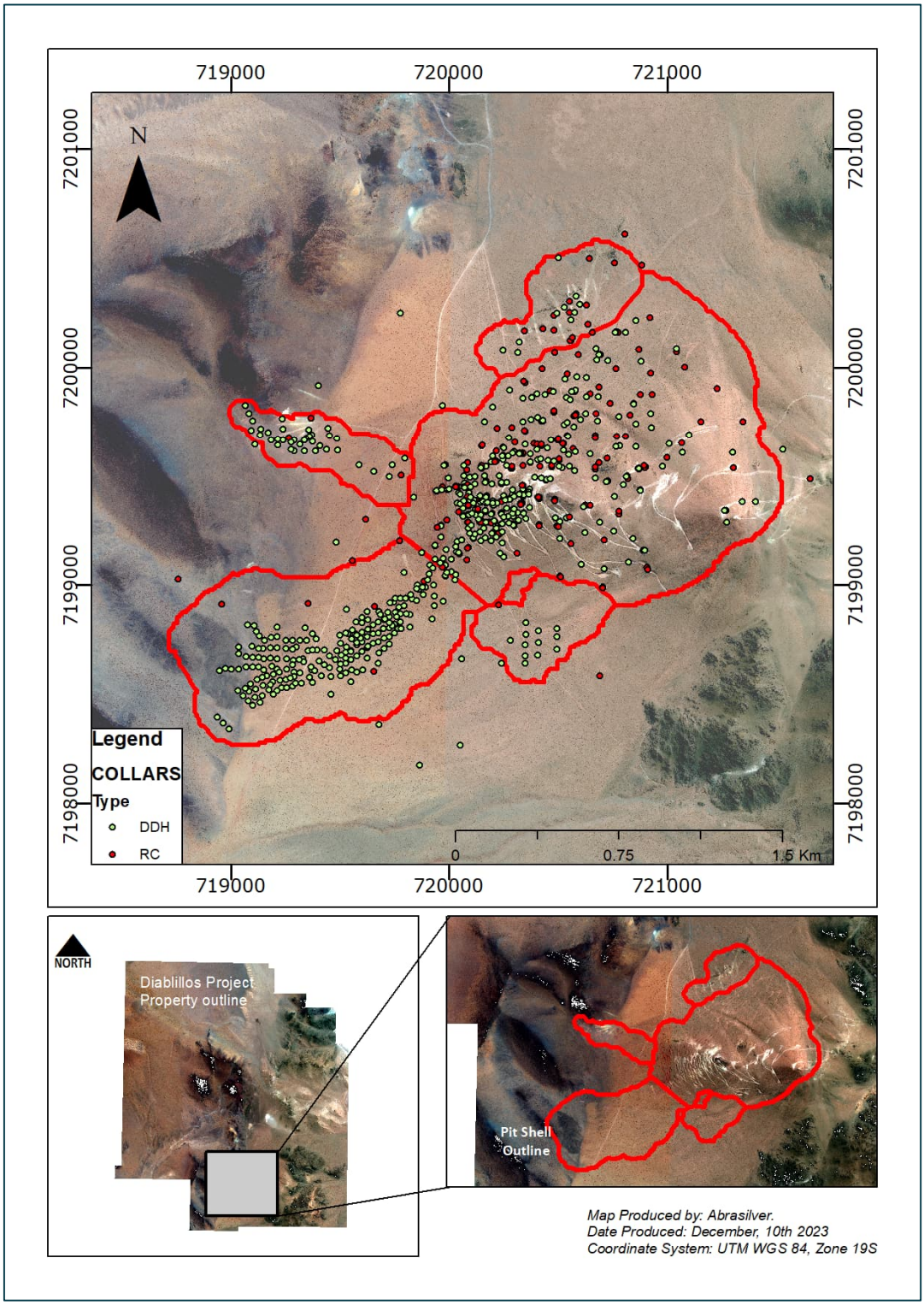
Table 14-3: Summary of Drill Holes Excluded from the Mineral Resource Estimate

Zone / Holes	N° Holes	Reason
Alpaca	14	Outside Diablillos Block Model
Cerro Blanco + Cerro Viejo Este + Cerro Viejo Oeste	33	
Corderos	18	
Jasperoide	13	
Northern Arc Valley Fill	5	
Pedernales	23	
Laderas	2	
Yolanda	2	
Oculto zone	22	Outside of block limit
Condemnation and other purpose drillholes	29	
Total Excluded	161	



Source: AbraSilver Resource Corp., 2025

Figure 14-1: Plan View of Drill Holes used in the Estimation of Mineral Resources Coloured by Drill Type



Source: AbraSilver Resource Corp., 2025

Figure 14-2: Plan View of Drill Holes used in the Estimation of Mineral Resources, colored by type.

14.3 Geological Model

The Diablillos deposit is an epithermal silver-gold deposit with complex mineralization that has strong alteration and structural stratigraphic control. The structures (main direction N 45°E and cross direction N 85°E) are steeply dipping feeders where the mineralizing fluids ascended and migrated laterally along permeable horizons. Fluid flow was primarily at the contact zone between the volcanic rock and underlying basement rocks. The Oculto zone is strongly oxidized to depths of 300 m to 400 m. Beneath which this mineralization grades into a transitional zone of oxide and sulphide. At the Oculto and JAC deposits the oxide/sulphide interface has been well defined by drilling.

A high-grade zone of silver mineralisation approximately 20 m thick occurs at around 100m depth and is believed to be a supergene enrichment zone. A broadly horizontal zone of higher-grade gold mineralization occurs at or near the contact between the volcanic rock and the underlying basement rock. This zone is approximately 30 m thick. In places it correlates well with an erosion over the basement, described as a regolith.

Relogging was completed for historical 2007 – 2008 drilling to ensure consistency in lithology, alteration, and oxidation state codes. This was developed in conjunction with Mr. Peralta (“QP”) and covers the highest density of recent drilling and re-logging of nearby historical drill holes. The uncertainty of the model outside this zone is increased and mineralization controls are based only on geological components. The QP defined the estimation domains primarily based on a combination of alteration and lithology. Subdomains are based on the oxidization level, as are a subset between oxide and sulphides. As no correlation between lithology and alteration types was found for the JAC, Fantasma and Sombra zones, a simple grade shell of 5 g/t AgEq cut-off defined most of the mineralized zones.

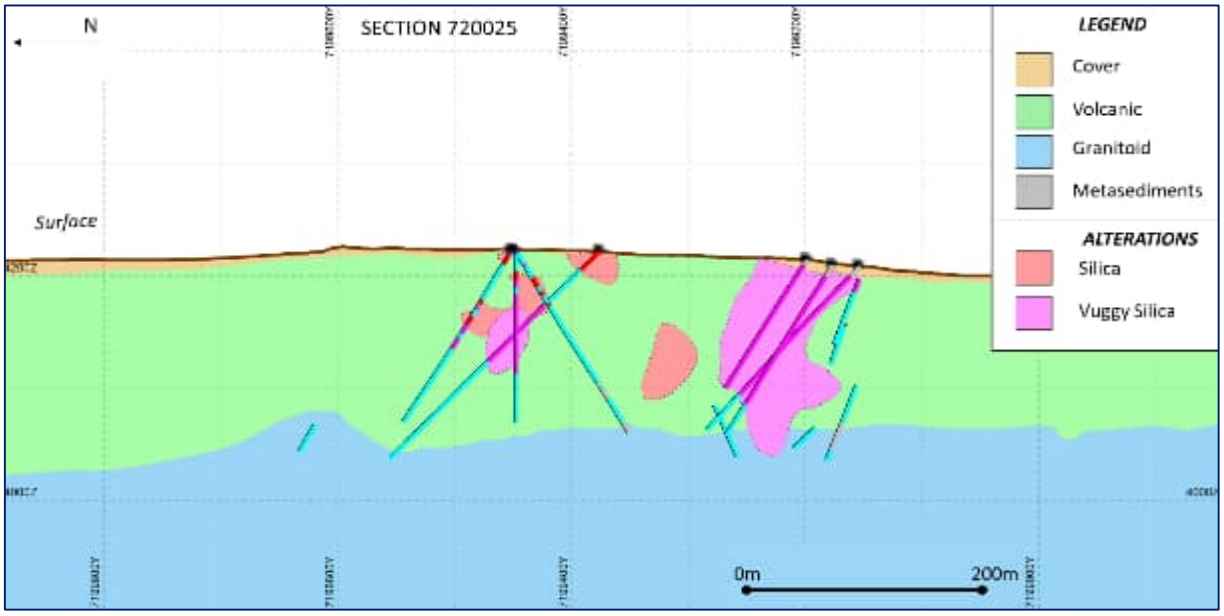
The boundary between oxide and sulphides zones was constructed using relogged historical and recent drilling.

The three main lithologies present in the Diablillos Project are volcanic rocks, metasediments rocks and granitic intrusions. As described in Section 7, hydrothermal alteration types include argillic, silica and vuggy silica. The applied domain coding has been detailed in Table 14-4. Oxidized rock was coded (1) and deep sulphides (2). For the JAC, Fantasma and Sombra zone, domains were coded inside the grade shell (510-610-810-910) and outside the grade shell as 520-620-720-820-920-1020-1120.

Table 14-4: Estimation Domains and Coding Used

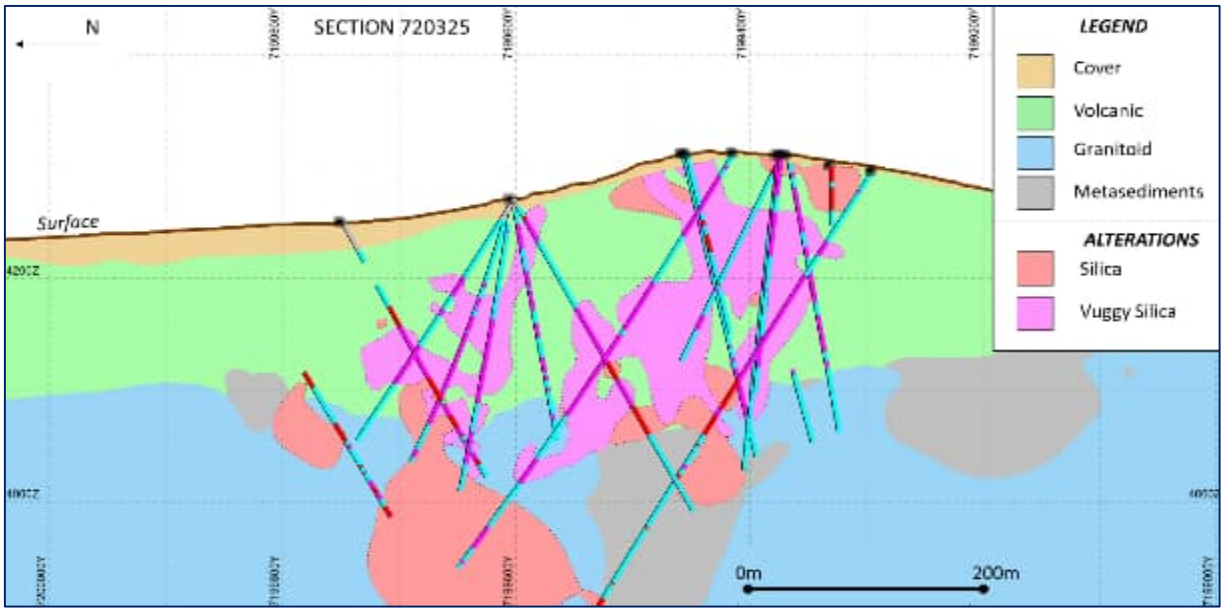
Target	Lithology (L)	Code	Alteration (A)	Code	Lithology + Alteration	Code	Ox/Su	Code	LA2	Code
	Cover	100	Unaltered	0	Cover + Unalt	10		0	Cover	10
OCULTO	Volc	200	Arg	10	Volc + Arg	210	Su	1	Vc+Ar+Su	211
		200		10		210	Ox	2	Vc+Ar+Ox	212
		200	Silica	20	Volc + Silica	220	Su	1	Vc+Si+Su	221
		200		20			Ox	2	Vc+Si+Ox	222
		200	Vuggy	30	Volc + Vuggy	230	Su	1	Vc+Vg+Su	231
		200		30			Ox	2	Vc+Vg+Ox	232
	Meta	300	Arg	10	Meta + Arg	310	Su	1	Mt+Ar+Su	311
		300		10			Ox	2	Mt+Ar+Ox	312
		300	Silica	20	Meta + Silica	320	Su	1	Mt+Si+Su	321
		300		20			Ox	2	Mt+Si+Ox	322
		300	Vuggy	30	Meta + Vuggy	330	Su	1	Mt+Vg+Su	331
		300		30			Ox	2	Mt+Vg+Ox	332
	Granite	400	Arg	10	Granite + Arg	410	Su	1	Gr+Ar+Su	411
		400		10			Ox	2	Gr+Ar+Ox	412
		400	Silica	20	Granite + Silica	420	Su	1	Gr+Si+Su	421
		400		20			Ox	2	Gr+Si+Ox	422
		400	Vuggy	30	Granite + Vuggy	430	Su	1	Gr+Vg+Su	431
		400		30			Ox	2	Gr+Vg+Ox	432
JAC	Volc	500	IGS	10	Volc + IGS	510	Su	1	JAC-Vc+IGS+Su	511
		500	IGS	10		510	Ox	2	JAC-Vc+IGS+Ox	512
		500	OGS	20	Volc + OGS	520	Su	1	JAC-Vc+OGS+Su	521
		500	OGS	20		520	Ox	2	JAC-Vc+OGS+Ox	522
	Granite	600	IGS	10	Granite + IGS	610	Su	1	JAC-Gr+IGS+Su	611
		600	IGS	10		610	Ox	2	JAC-Gr+IGS+Ox	612

Target	Lithology (L)	Code	Alteration (A)	Code	Lithology + Alteration	Code	Ox/Su	Code	LA2	Code
		600	OGS	20	Granite + OGS	620	Su	1	JAC-Gr+OGS+Su	621
		600	OGS	20		620	Ox	2	JAC-Gr+OGS+Ox	622
FAN	Volc	700	IGS	10	Volc + IGS	710	Su	1	FAN-Vc+IGS+Su	711
		700	IGS	10		710	Ox	2	FAN-Vc+IGS+Ox	712
		700	OGS	20	Volc + OGS	720	Su	1	FAN-Vc+OGS+Su	721
		700	OGS	20		720	Ox	2	FAN-Vc+OGS+Ox	722
	Granite	800	IGS	10	Granite + IGS	810	Su	1	FAN-Gr+IGS+Su	811
		800	IGS	10		810	Ox	2	FAN-Gr+IGS+Ox	812
		800	OGS	20	Granite + OGS	820	Su	1	FAN-Gr+OGS+Su	821
		800	OGS	20		820	Ox	2	FAN-Gr+OGS+Ox	822
SOM	Volc	900	IGS	10	Volc + IGS	910	Su	1	SOM-Vc+IGS+Su	911
		900	IGS	10		910	Ox	2	SOM-Vc+IGS+Ox	912
		900	OGS	20	Volc + OGS	920	Su	1	SOM-Vc+OGS+Su	921
		900	OGS	20		920	Ox	2	SOM-Vc+OGS+Ox	922
	Granite	1000	OGS	20	Gra + OGS	1020	Su	1	SOM-Gr+OGS+Su	1021
		1000	OGS	20		1020	Ox	2	SOM-Gr+OGS+Ox	1022
	Meta	1100	OGS	20	Meta + Arg	1120	Su	1	SOM-Mt+OGS+Su	1121
		1100	OGS	20		1120	Ox	2	SOM-Mt+OGS+Ox	1122



Source: AbraSilver Resource Corp., 2025

Figure 14-3: Vertical Cross Section N-S 720025, showing the Lithology Domains with Superimposed Alteration Domains (JAC Deposit)



Source: AbraSilver Resource Corp., 2025

Figure 14-4: Vertical Cross Section N-S 720325, showing the Lithology Domains with Superimposed Alteration Domains (Oculto deposit)

14.4 Exploratory Data Analysis

Gold and silver assay statistics and plots were examined in relation to lithology and alteration. Domains were classified to determine the most suitable approach for grade estimation.

Gold and silver grade statistics by lithology and alteration are presented in Table 14-5 and Table 14-6.

The table includes the coefficient of variation as a measure of grade variability. As a rule-of-thumb, CVs of composited samples should be ≤ 2 for typical linear estimation techniques, while CVs will be reduced slightly by compositing and treatment of the extreme high grade (top cut). In certain domains the CV is higher than 2 because the CV is strictly related to the arithmetic mean. When the arithmetic mean is close to zero or at zero, the CV loses meaning. It can give very large values which do not necessarily imply a large dispersion of data.

Table 14-5: Gold Grade Statistics by Lithological and Alteration Combination

Statistics/Code	211	212	221	222	231	232
Valid Data	267	29805	no data	5573	4	18825
Minimum	0.00	0.00		0.00	0.01	0.00
Maximum	5.01	42.40		10.60	0.01	54.71
Mean	0.072	0.075		0.128	0.010	0.518
Standard Deviation	0.38	0.47		0.46	0.00	1.58
Variance	0.144	0.223		0.208	0.000	2.499
Coefficient Of Variation	5.2	6.3		3.6	0.0	3.1
Code	311	312	321	322	331	332
Valid Data	1410	4533	1068	3559	204	1921
Minimum	0.00	0.00	0.00	0.00	0.01	0.00
Maximum	7.99	21.48	116.00	31.19	4.02	23.34
Mean	0.17	0.32	0.45	0.55	0.25	0.71
Standard Deviation	0.48	0.88	3.69	1.50	0.55	1.39
Variance	0.23	0.78	13.59	2.24	0.30	1.94
Coefficient Of Variation	2.9	2.8	8.1	2.7	2.2	2.0
Code	411	412	421	422	431	432
Valid Data	6611	14163	1386	3804	463	3613
Minimum	0.00	0.00	0.00	0.00	0.00	0.00
Maximum	47.06	134.55	29.53	64.62	24.67	32.30
Mean	0.21	0.21	0.43	0.31	0.83	0.58
Standard Deviation	1.20	1.88	1.47	1.52	2.07	1.67
Variance	1.44	3.55	2.15	2.30	4.29	2.79
Coefficient Of Variation	5.7	9.0	3.4	4.8	2.5	2.9
Code	511	512	521	522	531	532
Valid Data	101	6682	524	4769	no data	no data

Minimum	0.01	0.01	0.01	0.00		
Maximum	13.29	44.49	0.02	0.15		
Mean	0.26	0.06	0.01	0.01		
Standard Deviation	1.54	0.60	0.00	0.00		
Variance	2.36	0.36	0.00	0.00		
Coefficient Of Variation	5.9	9.5	0.0	0.4		
Code	611	612	621	622	711	712
Valid Data	80	3373	424	1901	no data	791
Minimum	0.01	0.01	0.01	0.01		0.00
Maximum	0.01	0.14	0.02	0.07		0.04
Mean	0.01	0.01	0.01	0.01		0.00
Standard Deviation	0.00	0.01	0.00	0.00		0.01
Variance	0.00	0.00	0.00	0.00		0.00
Coefficient Of Variation	0.0	0.6	0.0	0.3		1.6
Code	721	722	821	912	922	1012
Valid Data	287	1709	14	349	277	no data
Minimum	0.00	0.00	0.00	0.01	0.00	
Maximum	0.06	0.08	0.01	2.14	0.03	
Mean	0.01	0.01	0.00	0.06	0.01	
Standard Deviation	0.01	0.01	0.00	0.20	0.00	
Variance	0.00	0.00	0.00	0.04	0.00	
Coefficient Of Variation	1.3	1.4	3.7	3.2	0.2	
Code	1021	1022	1112	1121	1122	
Valid Data	166	231	216	24	580	
Minimum	0.00	0.00	0.01	0.00	0.00	
Maximum	0.81	0.51	0.65	0.04	0.28	
Mean	0.04	0.03	0.03	0.01	0.01	
Standard Deviation	0.10	0.07	0.06	0.01	0.02	
Variance	0.01	0.01	0.00	0.00	0.00	
Coefficient Of Variation	2.3	2.1	2.1	0.6	1.7	

Table 14-6: Statistics for Silver by Combined Lithological and Alteration Domain

Statistics / Code	211	212	221	222	231	232
Valid Data	267	29805	No Data	5573	4	18825
Minimum	0.05	0.05		0.05	0.60	0.05
Maximum	142.00	11304.50		4754.45	4.30	11550.12
Mean	4.597	14.952		18.722	2.450	74.982
Standard Deviation	14.66	149.18		98.41	1.51	301.26
Variance	214.997	22254.255		9684.376	2.283	90754.939
Coefficient Of Variation	3.2	10.0		5.3	0.6	4.0
Code	**311**	**312**	**321**	**322**	**331**	**332**
Valid Data	1410	4533	1068	3559	204	1921
Minimum	0.05	0.25	0.20	0.05	0.30	0.25
Maximum	893.00	331.00	327.00	610.00	100.00	352.00
Mean	6.39	9.50	5.48	13.05	5.17	16.31
Standard Deviation	31.67	16.05	12.82	20.44	9.89	20.94
Variance	1003.29	257.44	164.43	417.75	97.88	438.55
Coefficient Of Variation	5.0	1.7	2.3	1.6	1.9	1.3
Code	**411**	**412**	**421**	**422**	**431**	**432**
Valid Data	6611	14163	1386	3804	463	3613
Minimum	0.00	0.05	0.25	0.05	0.05	0.05
Maximum	7606.00	2720.00	312.80	860.00	11236.76	2685.00
Mean	5.57	8.15	6.45	9.90	99.27	48.69
Standard Deviation	96.00	53.96	14.33	33.78	725.02	127.58
Variance	9216.40	2911.24	205.42	1140.80	525650.96	16276.86
Coefficient Of Variation	17.2	6.6	2.2	3.4	7.3	2.6
Code	**511**	**512**	**521**	**522**	**531**	**532**
Valid Data	101	6682	524	4769	No Data	No Data
Minimum	0.30	0.25	0.30	0.25		
Maximum	2867.00	32480.56	8.80	32.80		
Mean	135.52	99.20	0.42	1.30		
Standard Deviation	463.41	503.23	0.57	1.83		
Variance	214746.96	253236.09	0.33	3.34		
Coefficient Of Variation	3.4	5.1	1.4	1.4		
Code	**611**	**612**	**621**	**622**	**711**	**712**
Valid Data	80	3373	424	1901	No Data	791
Minimum	0.30	0.30	0.30	0.30		0.90
Maximum	696.00	2312.93	8.40	17.30		1237.00
Mean	50.24	66.75	0.74	1.67		51.94
Standard Deviation	109.79	127.57	0.96	1.30		97.45
Variance	12054.46	16273.06	0.91	1.68		9496.98

Coefficient Of Variation	2.2	1.9	1.3	0.8		1.9
Code	721	722	821	912	922	1012
Valid Data	287	1709	14	349	277	No Data
Minimum	0.10	0.10	0.10	1.00	0.30	
Maximum	116.00	136.00	0.10	343.61	65.60	
Mean	1.57	2.06	0.10	21.10	1.85	
Standard Deviation	7.09	6.47	0.00	34.87	5.35	
Variance	50.29	41.86	0.00	1216.01	28.64	
Coefficient Of Variation	4.5	3.1	0.0	1.7	2.9	
Code	1021	1022	1112	1121	1122	
Valid Data	166	231	216	24	580	
Minimum	0.25	0.25	1.00	1.00	0.25	
Maximum	80.30	54.10	69.30	6.80	85.40	
Mean	3.61	2.92	5.54	2.33	2.55	
Standard Deviation	8.30	5.69	6.90	1.36	4.84	
Variance	68.88	32.41	47.63	1.86	23.46	
Coefficient Of Variation	2.3	2.0	1.2	0.6	1.9	

The QP concluded that gold and silver grade domains based on modelled geologic variables was the most representative way to define and evaluate the deposit. Factors considered included metal content, specific gravity, geotechnical characteristics, metallurgical characteristics, and other mining parameters related to the host rocks or hydrothermal alteration.

The definition of estimation domains was carried out based on a combination of alteration and lithology, depending on the number of samples. These domains more adequately capture different populations of grades in relation to the two main structural deposit orientations.

Some domains with few samples have been merged with the closest domain in terms of geological characteristics.

14.5 Treatment of Missing/Absent Samples

The proportion of sampled intervals within the block model limits is consistently high across all lithological domains. In the principal mineralized domains, sampling coverage is close to complete, generally representing more than 95% of the total drilled meters and reaching virtually 100% in most cases. The limited unsampled intervals are predominantly associated with barren sections, unaltered host rocks, or operational constraints during drilling and sampling. Consequently, these omissions are not considered material and are not expected to have a significant impact on the Mineral Resource estimation or the representativeness of the assay database.

14.6 Compositing

The drill hole database has been coded with the estimation domains (Lith+Alt domain from 211 to 1122) The drill hole intervals were composited to 1 meter down hole while honouring the estimation domain boundary.

A residual retention routine has been used where residuals are added back to the next adjacent interval Figure 14-5 and Figure 14-6.

Summary statistics for global population non-composite and composite are presented in Table 14-7.

Table 14-7: Summary Statistics, Global Population for Non-Composite and Composite Data

Statistics	Length	
	Uncomposited	Composited
Valid Data	123276	142737
Minimum	0.01	0.01
Maximum	134.55	134.55
Mean	0.22	0.21
Variance	1.43	1.25
Standard Deviation	1.20	1.12
Coefficient Of Variation	5.34	5.30

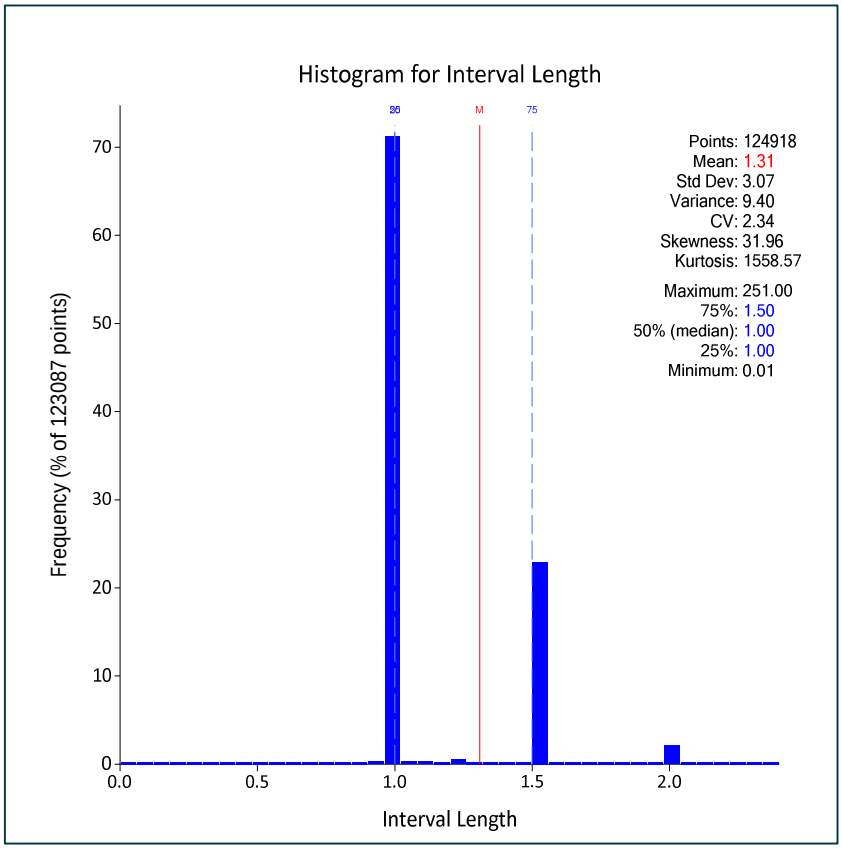


Figure 14-5: Non-Composited Sample Data - Samples Length

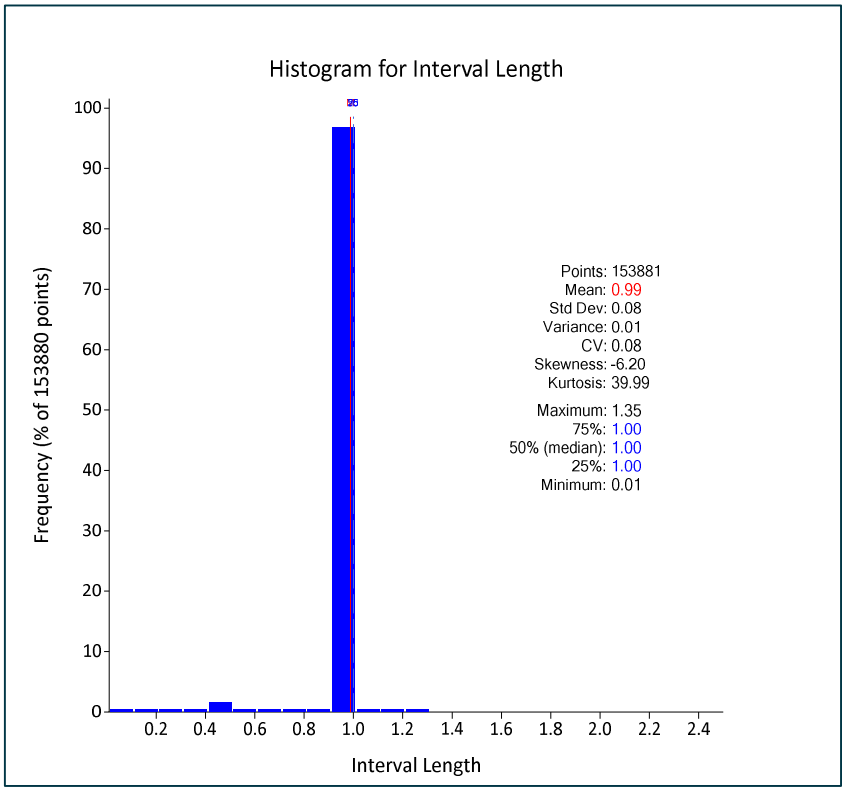


Figure 14-6: 1 m Composite Data - Sample Intervals

Summary statistics for raw data weighted by length (non-composited or raw) and composited sample intervals by estimation domains are presented in Table 14-8 and Table 14-9.

Table 14-8: Summary Statistics for Each Gold Domain of Composite - Au g/t.

Au Domain	Number of Samples		Mean Grade			Std Dev		Coeff Variation	
	Raw	Compos	Raw	Compos	Diff	Raw	Compos	Raw	Compos
211	267	361	0.07	0.06	-0.02	0.38	0.33	5.24	5.82
212	29,805	35,282	0.08	0.07	0.00	0.47	0.44	6.27	6.18
222	5,573	6,609	0.13	0.12	-0.01	0.46	0.44	3.56	3.57
231	4	4	0.01	0.01	0.00	0.00	0.00	0.00	0.00
232	18,825	22,100	0.52	0.49	-0.03	1.58	1.49	3.05	3.03
311	1,410	1,587	0.17	0.17	0.01	0.48	0.50	2.88	2.88
312	4,533	4,824	0.32	0.32	0.00	0.88	0.86	2.76	2.73
321	1,068	1,260	0.45	0.43	-0.03	3.69	3.40	8.10	7.94
322	3,559	4,117	0.55	0.53	-0.02	1.50	1.43	2.73	2.69
331	204	249	0.25	0.23	-0.02	0.55	0.50	2.18	2.21
332	1,921	2,087	0.71	0.70	-0.01	1.39	1.36	1.96	1.93
411	6,611	7,560	0.21	0.19	-0.01	1.20	1.15	5.75	5.93
412	14,163	15,298	0.21	0.20	0.00	1.88	1.79	9.03	8.81
421	1,386	1,567	0.43	0.39	-0.04	1.47	1.35	3.41	3.49
422	3,804	4,092	0.31	0.31	-0.01	1.52	1.47	4.84	4.78
431	463	560	0.83	0.70	-0.12	2.07	1.90	2.51	2.70
432	3,613	3,855	0.58	0.59	0.00	1.67	1.66	2.87	2.84
511	101	119	0.26	0.22	-0.04	1.54	1.42	5.86	6.33
512	6,682	7,279	0.06	0.06	0.00	0.60	0.47	9.49	7.45
521	524	777	0.01	0.01	0.00	0.00	0.00	0.04	0.04
522	4,769	5,977	0.01	0.01	0.00	0.00	0.00	0.39	0.36
611	80	97	0.01	0.01	0.00	0.00	0.00	0.00	0.00
612	3,373	3,860	0.01	0.01	0.00	0.01	0.01	0.59	0.60
621	424	622	0.01	0.01	0.00	0.00	0.00	0.05	0.04
622	1,901	2,538	0.01	0.01	0.00	0.00	0.00	0.25	0.24

Au Domain	Number of Samples		Mean Grade			Std Dev		Coeff Variation	
	Raw	Compos	Raw	Compos	Diff	Raw	Compos	Raw	Compos
712	791	917	0.00	0.00	0.00	0.01	0.01	1.57	1.60
721	287	469	0.01	0.01	0.00	0.01	0.01	1.26	1.34
722	1,709	2,450	0.01	0.01	0.00	0.01	0.01	1.38	1.29
821	14	27	0.00	0.00	0.00	0.00	0.00	3.74	3.60
912	349	373	0.06	0.06	0.00	0.20	0.18	3.21	3.06
922	277	314	0.01	0.01	0.00	0.00	0.00	0.18	0.17
1012	2	3	0.01	0.01	0.00	0.00	0.00	0.00	0.00
1021	166	185	0.04	0.04	0.00	0.10	0.10	2.32	2.40
1022	231	275	0.03	0.03	0.00	0.07	0.07	2.11	2.20
1112	216	219	0.03	0.03	0.00	0.06	0.07	2.06	2.07
1121	24	28	0.01	0.01	0.00	0.01	0.01	0.58	0.56
1122	580	632	0.01	0.01	0.00	0.02	0.02	1.71	1.66

Table 14-9: Summary Statistics for each Silver Domain of Composite - Ag g/t.

Ag Domain	Number of Samples		Mean Grade			Std Dev		Coeff Variation	
	Raw	Compos	Raw	Compos	Diff	Raw	Compos	Raw	Compos
211	267	361	4.60	3.82	-0.78	14.66	12.62	3.19	3.31
212	29,805	35,282	14.95	14.37	-0.58	149.18	138.94	9.98	9.67
222	5,573	6,609	18.72	18.25	-0.47	98.41	90.80	5.26	4.98
231	4	4	2.45	2.45	0.00	1.51	1.51	0.62	0.62
232	18,825	22,100	74.98	73.49	-1.49	301.26	288.72	4.02	3.93
311	1,410	1,587	6.39	6.13	-0.26	31.67	30.12	4.95	4.91
312	4,533	4,824	9.50	9.44	-0.06	16.05	15.80	1.69	1.67
321	1,068	1,260	5.48	5.36	-0.12	12.82	11.92	2.34	2.22
322	3,559	4,117	13.05	12.82	-0.24	20.44	19.99	1.57	1.56

Ag Domain	Number of Samples		Mean Grade			Std Dev		Coeff Variation	
	Raw	Compos	Raw	Compos	Diff	Raw	Compos	Raw	Compos
331	204	249	5.17	4.76	-0.41	9.89	9.17	1.91	1.93
332	1,921	2,087	16.31	16.20	-0.11	20.94	20.64	1.28	1.27
411	6,611	7,560	5.57	5.22	-0.35	96.00	89.92	17.23	17.24
412	14,163	15,298	8.15	8.18	0.03	53.96	53.83	6.62	6.58
421	1,386	1,567	6.45	5.99	-0.46	14.33	13.51	2.22	2.26
422	3,804	4,092	9.90	9.98	0.09	33.78	32.96	3.41	3.30
431	463	560	99.27	109.27	10.00	725.02	750.02	7.30	6.86
432	3,613	3,855	48.69	49.95	1.27	127.58	128.25	2.62	2.57
511	101	119	135.52	117.64	-17.88	463.41	428.82	3.42	3.65
512	6,682	7,279	99.20	97.92	-1.28	503.23	473.73	5.07	4.84
521	524	777	0.42	0.42	-0.01	0.57	0.51	1.36	1.24
522	4,769	5,977	1.30	1.24	-0.06	1.83	1.69	1.41	1.36
611	80	97	50.24	46.01	-4.23	109.79	101.35	2.19	2.20
612	3373	3,860	66.75	62.37	-4.38	127.57	118.99	1.91	1.91
621	424	622	0.74	0.76	0.02	0.96	0.97	1.29	1.28
622	1,901	2,538	1.67	1.59	-0.08	1.30	1.21	0.78	0.76
711	19	19	69.71	69.71	0.00	125.07	125.07	1.79	1.79
712	791	917	51.94	53.16	1.22	97.45	95.11	1.88	1.79
721	287	469	1.57	1.18	-0.39	7.09	5.59	4.51	4.73
722	1,709	2,450	2.06	1.72	-0.33	6.47	5.48	3.15	3.18
821	14	27	0.10	0.10	0.00	0.00	0.00	0.00	0.00
912	349	373	21.10	24.26	3.17	34.87	40.37	1.65	1.66
922	277	314	1.85	1.83	-0.02	5.35	4.85	2.89	2.65
1012	2	3	1.00	1.00	0.00	0.00	0.00	0.00	0.00
1021	166	185	3.61	3.33	-0.28	8.30	7.91	2.30	2.37
1022	231	275	2.92	2.56	-0.35	5.69	5.28	1.95	2.06
1112	216	219	5.54	5.52	-0.02	6.90	6.88	1.25	1.25

Ag Domain	Number of Samples		Mean Grade			Std Dev		Coeff Variation	
	Raw	Compos	Raw	Compos	Diff	Raw	Compos	Raw	Compos
1121	24	28	2.33	2.28	-0.05	1.36	1.26	0.59	0.55
1122	580	632	2.55	2.53	-0.02	4.84	4.65	1.90	1.84

14.7 Top Cutting

Top cutting, or capping of outlier grades, was determined for each estimation domain. The top cutting assessment considered the following:

- Review of the composite data by examining the cumulative distribution.
- Comparison of the percentage of metal and data of the Coefficient of Variation ("CV") affected by top cutting.
- Visual 3D review to assess the clustering of the high-grade composite data.
- Based on the assessment, appropriate top cuts were determined for each estimation domain. The application of top cuts resulted in minor reductions in the mean gold and silver grades.

Table 14-10 and Table 14-11 summarize uncut and top cut gold and silver statistics of composites for each estimation domain. Examples of top cut analysis have been provided in Figure 14-7 to Figure 14-10.

Table 14-10: Top Cut Statistics by Gold Domain – Au g/t Composite Data

Au Domain	Number of Samples		Mean Grade			Top-Cut Value	Std Dev		Coeff Variation		Max Un-Cut Value	Top Cut %ile
	Un-Cut	Top-Cut	Un-Cut	Top-Cut	% Diff		Un-Cut	Top-Cut	Un-Cut	Top-Cut		
211	928	927	0.04	0.04	0.0%	4.61	0.22	0.21	5.14	4.96	5.01	1.0%
212	35101	35099	0.07	0.07	0.0%	30.42	0.46	0.40	6.70	5.95	30.42	1.0%
222	6357	6353	0.14	0.13	-7.1%	7.18	0.47	0.45	3.50	3.35	10.60	1.0%
232	20349	20338	0.54	0.53	-1.9%	26.78	1.64	1.52	3.04	2.85	54.71	1.0%
311, 321, 331	1146	1145	0.42	0.42	0.0%	111.10	3.56	3.42	8.44	8.19	116.00	1.0%
312	4267	4265	0.29	0.29	0.0%	12.20	0.80	0.74	2.73	2.54	21.48	1.0%
322	3120	3118	0.64	0.63	-1.6%	18.73	1.55	1.45	2.43	2.29	31.19	1.0%
332	1740	1738	0.75	0.74	-1.3%	14.70	1.40	1.31	1.87	1.77	23.34	1.0%
411, 421. 431	9687	7556	0.26	0.25	-3.8%	29.87	1.25	0.79	4.89	3.17	47.06	1.0%
412	11251	11248	0.17	0.17	0.0%	16.15	0.64	0.58	3.86	3.54	30.83	1.0%
422	4092	4091	0.31	0.31	0.0%	64.62	1.47	1.47	4.78	4.78	64.65	1.0%
432	2606	2602	0.68	0.67	-1.5%	14.08	1.60	1.52	2.36	2.26	27.66	1.0%
511, 521	896	118	0.00	0.00	0.0%	12.59	0.52	0.50	12.52	13.25	12.59	0.6%
512	7290	7289	0.06	0.06	0.0%	23.91	0.47	0.47	7.45	7.45	23.91	1.0%
522	5055	5053	0.01	0.01	0.0%	0.08	0.00	0.00	0.37	0.31	0.09	0.2%
611, 621	719	97	0.01	0.01	0.0%	0.02	0.01	0.01	0.04	0.02	0.04	0.0%
612	3860	3857	0.01	0.01	0.0%	0.14	0.01	0.01	0.60	0.60	0.14	0.1%
622	2538	2537	0.01	0.01	0.0%	0.06	0.00	0.00	0.24	0.24	0.06	0.0%
712	940	926	0.01	0.01	0.0%	0.02	0.01	0.01	1.50	1.47	0.03	1.0%
721	469	466	0.01	0.01	0.0%	0.03	0.01	0.01	1.34	1.28	0.03	0.0%
722	2454	2450	0.01	0.01	0.0%	0.04	0.01	0.01	1.29	1.28	0.08	1.0%
912	599	597	0.05	0.05	0.0%	1.44	0.15	0.15	3.07	2.98	1.74	1.0%

Au Domain	Number of Samples		Mean Grade			Top-Cut Value	Std Dev		Coeff Variation		Max Un-Cut Value	Top Cut %ile
	Un-Cut	Top-Cut	Un-Cut	Top-Cut	% Diff		Un-Cut	Top-Cut	Un-Cut	Top-Cut		
922	314	313	0.01	0.01	0.0%	0.03	0.00	0.00	0.17	0.16	0.03	1.0%
1021, 1121	234	232	0.04	0.04	0.0%	0.43	0.07	0.07	2.03	1.98	0.51	1.0%
1022, 1122	234	232	0.04	0.04	0.0%	0.42	0.07	0.07	2.03	1.98	0.52	1.0%

Table 14-11: Top Cut Statistics by Gold Domain – Au g/t Composite Data

Ag Domain	Number of Samples		Mean Grade			Top-Cut Value	Std Dev		Coeff Variation		Max Un-Cut Value	Top Cut %ile
	Un-Cut	Top-Cut	Un-Cut	Top-Cut	% Diff		Un-Cut	Top-Cut	Un-Cut	Top-Cut		
211	384	382	3.65	3.58	-1.9%	118	12.25	11.50	3.36	3.22	134	2.0%
212	35324	35986	14.33	14.04	-2.0%	5908	138.80	123.30	9.69	8.78	11305	2.0%
222	6358	6357	19.28	19.08	-1.0%	3481	87.13	77.13	4.52	4.04	4754	1.0%
232	22102	20332	73.49	71.99	-2.0%	6840	288.70	286.60	3.93	3.98	11550	2.0%
311, 321, 331	3097	1585	5.72	5.60	-2.1%	570	23.01	17.41	4.03	3.11	893	2.0%
312	4823	4822	9.42	9.42	0.0%	306	15.78	15.68	1.67	1.66	331	1.0%
322	31221	3116	15.44	15.29	-1.0%	170	21.59	15.29	1.40	1.24	610	1.0%
332	1738	1734	16.11	15.94	-1.1%	224	21.90	19.78	1.36	1.24	352	1.0%
411, 421. 431	9687	7559	11.36	11.04	-2.8%	4525	198.50	49.85	17.48	4.52	11237	2.8%
412	11247	11245	7.58	7.50	-1.1%	1477	40.51	37.18	5.35	4.96	1968	1.0%
422	2044	2041	12.50	12.38	-1.0%	514	35.20	33.30	2.82	2.60	596	1.0%
432	2602	2596	76.86	76.11	-1.0%	1652	170.61	161.80	2.22	2.13	2685	1.0%
511, 521	896	118	15.98	15.66	-2.0%	2580	16.70	154.40	10.05	9.86	2867	2.0%
512	5616	5614	108.90	107.80	-1.0%	2651	545.70	485.00	5.01	4.50	32481	1.0%

Ag Domain	Number of Samples		Mean Grade			Top-Cut Value	Std Dev		Coeff Variation		Max Un-Cut Value	Top Cut %ile
	Un-Cut	Top-Cut	Un-Cut	Top-Cut	% Diff		Un-Cut	Top-Cut	Un-Cut	Top-Cut		
522	3984	3976	1.34	1.32	-1.5%	26	2.40	2.20	1.80	1.67	53	1.0%
611, 621	719	717	6.86	6.24	-9.0%	392	6.86	6.24	5.85	5.00	696	1.9%
612	2807	2801	60.26	59.66	-1.0%	930	108.00	101.00	1.79	1.71	1370	1.0%
622	606	605	0.75	0.75	0.0%	8	0.96	0.96	1.28	1.28	8	0.0%
712	940	938	53.45	52.95	-0.9%	854	95.74	90.14	1.79	1.70	1237	1.0%
721	325	324	0.82	0.81	-1.2%	112	6.66	6.43	8.08	7.92	116	1.5%
722	314	313	1.83	1.83	0.0%	62	4.85	4.84	2.65	2.64	62	0.0%
912	517	514	14.19	14.05	-1.0%	277	32.66	31.34	2.30	2.20	343	1.0%
922	314	313	1.83	1.83	0.0%	62	4.85	4.84	2.65	2.64	62	0.0%
1021, 1121	213	211	3.19	3.04	-4.7%	51	7.39	6.00	2.31	1.97	80	4.7%
1022, 1122	907	905	2.54	2.49	-2.0%	47	4.85	4.16	1.91	1.67	85	2.0%

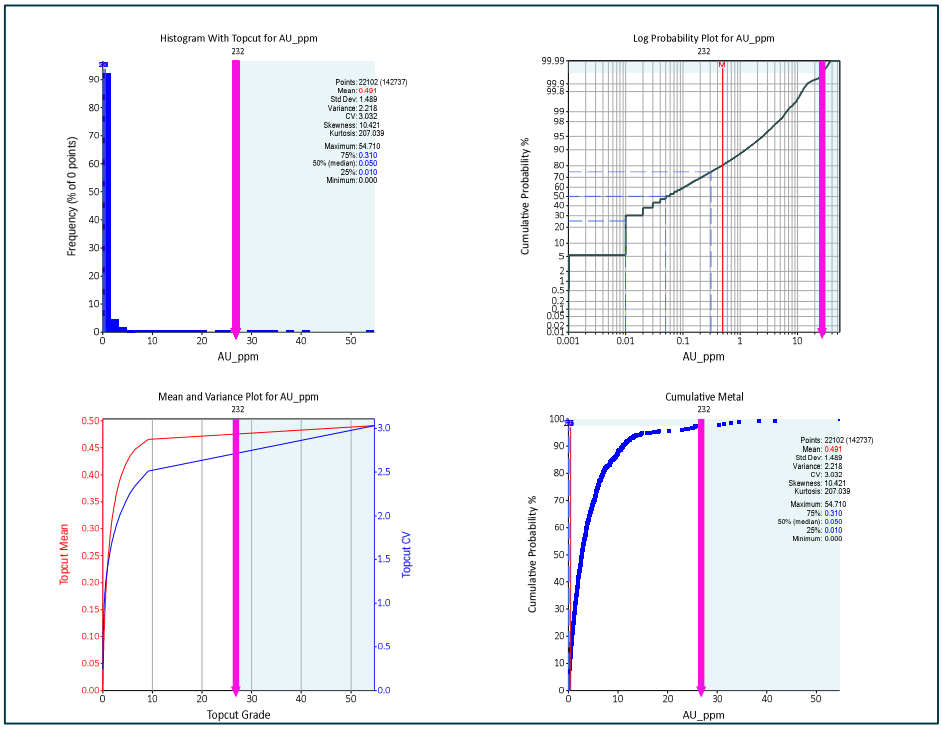


Figure 14-7: Example of the Top Cut Analysis – Gold Domain 232

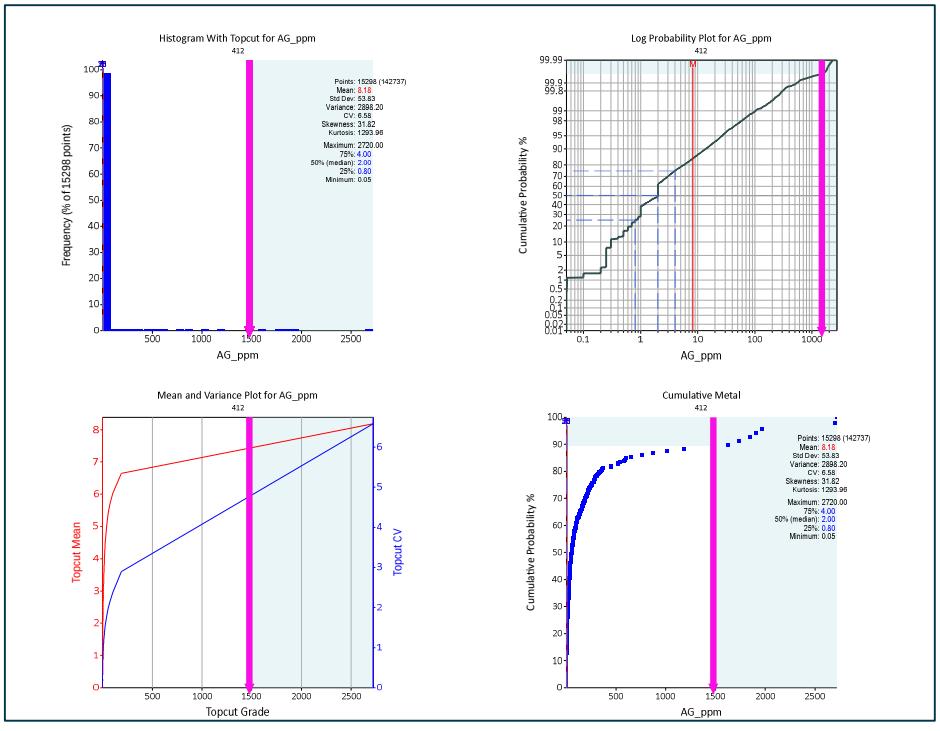


Figure 14-8: Example of the Top Cut Analysis – Silver Domain 412

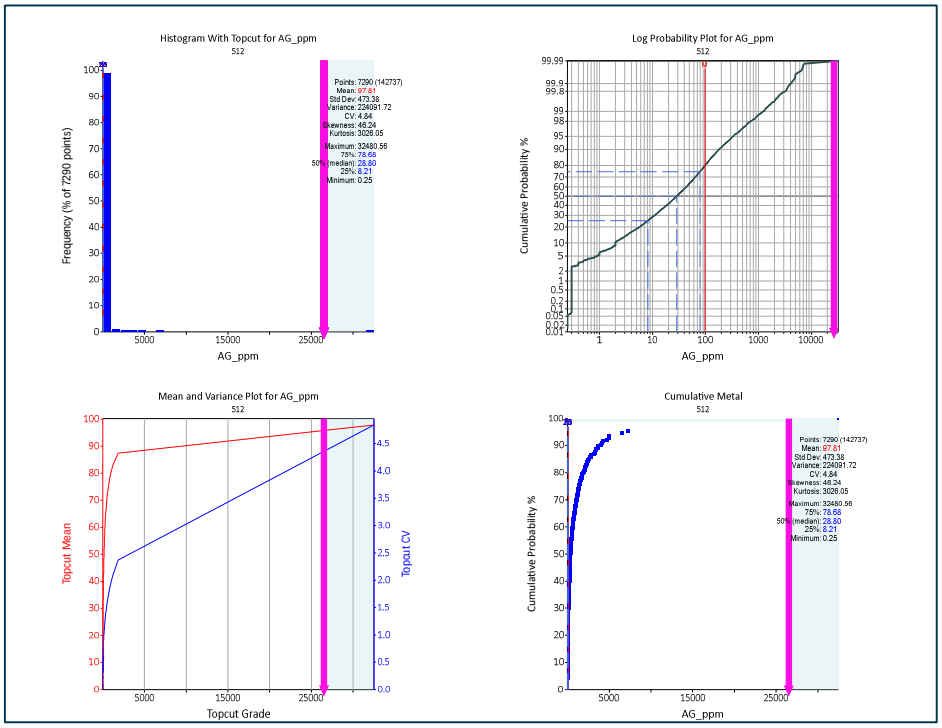


Figure 14-9: Example of the Top Cut Analysis – Silver Domain 512

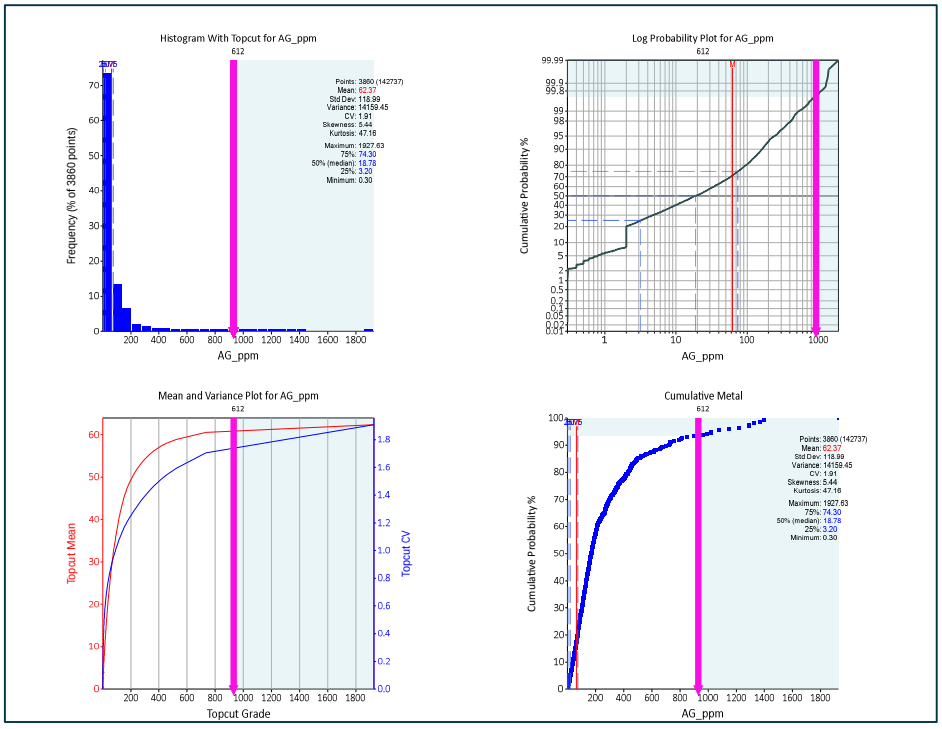


Figure 14-10: Example of the Top Cut Analysis – Silver Domain 612

14.8 Bulk Density Determination

A revised dry bulk density model was constructed using 8,451 samples taken over drilling performed from 2019 – 2025. This was measured at site by AbraSilver geologists and performed by the unwrapped core method.

This Unwrapped Core method is outlined below:

4. Dry samples are weighed in air (Mdry).
5. Dry samples are weighed immediately upon submersion in water (Mini).
6. The sample is left submerged and weighed again sometime later (Msat in water).
7. The sample is removed from the water and immediately weighed (Msat).

The in-situ bulk density is determined from the following formula:

Mass of contained water (Mwater) = Msat - Mdry

Volume of contained water (Vwater) = Mwater / Density of water (ρW)

Volume of sample (Vsamp) = Vwater + ((Mdry – Msat in water) / ρW)

Bulk Density = Mdry / Vsamp

The QP considers that the sample sets selected are appropriate to determine the in-situ bulk density of the Diablillos mineralization. A summary of the bulk densities applied to each code domain are shown in Table 14-12 and Figure 14-11.

Table 14-12: In-Situ Bulk Density Applied into the Block Model

Domain	Samples	Minimum	Maximum	Standard deviation	CV	Variance	Mean
10	61	1.85	2.72	0.21	0.09	0.04	2.25
211	4	2.32	2.72	0.17	0.07	0.03	2.48
212	1869	1.03	3.04	0.23	0.1	0.05	2.2
222	545	1.63	2.74	0.19	0.08	0.04	2.29
232	2252	1.52	3.5	0.18	0.08	0.03	2.29
311	87	1.54	3.32	0.23	0.09	0.05	2.68
312	154	1.06	3.27	0.21	0.08	0.04	2.45
321	157	2.31	3.43	0.16	0.06	0.03	2.72
322	236	2.11	3.21	0.19	0.07	0.04	2.54
331	17	2.53	3.01	0.11	0.04	0.01	2.68
332	243	1.46	2.89	0.16	0.07	0.03	2.42
411	215	1.06	3.19	0.18	0.07	0.03	2.55
412	476	1.01	3.2	0.21	0.09	0.05	2.42
421	79	2.17	2.85	0.15	0.06	0.02	2.56

Domain	Samples	Minimum	Maximum	Standard deviation	CV	Variance	Mean
422	262	1.66	4.77	0.25	0.1	0.06	2.45
431	49	2.04	2.74	0.17	0.07	0.03	2.45
432	181	1.8	3.26	0.18	0.07	0.03	2.36
511	8	1.72	2.46	0.24	0.11	0.06	2.2
512	417	1.03	5.14	0.29	0.14	0.08	2.05
521	34	2.04	2.67	0.19	0.08	0.04	2.34
522	243	1.48	4.48	0.32	0.14	0.1	2.33
611	3	2.15	2.26	0.06	0.03	0	2.2
612	213	1.05	4.18	0.26	0.12	0.07	2.18
621	34	2.32	6.09	0.6	0.22	0.37	2.74
622	112	1.16	3.81	0.33	0.14	0.11	2.37
711	1	2.12	2.12	0	0	0	2.12
712	31	1.83	2.45	0.17	0.08	0.03	2.18
721	3	2.36	2.56	0.1	0.04	0.01	2.45
722	69	1.9	2.63	0.15	0.06	0.02	2.37
912	39	1.97	2.82	0.17	0.07	0.03	2.41
922	16	1.76	2.67	0.26	0.12	0.07	2.24
1021	4	2.55	2.86	0.13	0.05	0.02	2.69
1022	5	2.4	2.67	0.1	0.04	0.01	2.54
1121	1	2.64	2.64	0	0	0	2.64
1122	19	2.04	2.69	0.15	0.06	0.02	2.5

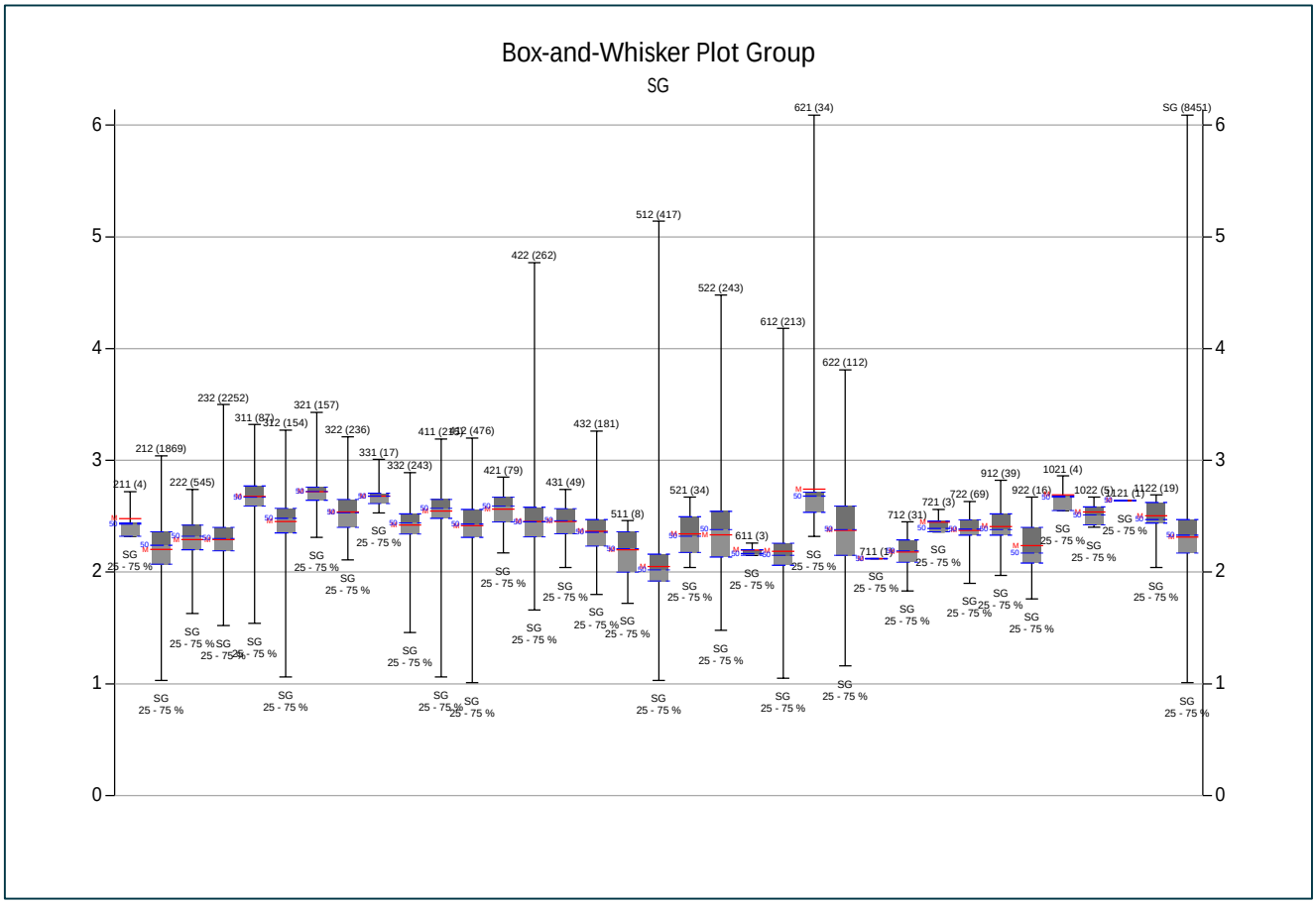


Figure 14-11: Box and Whisker Plot for Specific Gravity per Domain Distribution

14.9 Variography

Normal Score Variograms were chosen to model the gold and silver grade continuity as they were found to give better structures. The Normal Score Variograms assume that the variable being modelled has a normal (Gaussian) distribution, in which the data is transformed into normal scores. A monotonic transformation is performed where each data value is replaced by the quantile of a standard normal distribution that corresponds to its cumulative probability in the data distribution. Once the data is in normal score space, the variogram is computed, a back transformation is applied after model is fitted to preserve both the spatial correlation structure and the original histogram.

The Snowden Supervisor software was employed to generate variogram maps. Considering a 2 or 3 structured spherical model and nugget effect, recreating the spatial continuity and knowledge of the deposit geology. The nugget effect and sill contributions were derived from down-hole experimental variograms followed by final model fitting on directional variogram plots.

The normal score variogram for gold and silver was modelled for each estimation domain. Table 14-13 and Table 14-14.

An example of the normal score variogram models (gold domain 232, silver domain 232, silver domain 512 and silver domain 612) (Figure 14-12 to Figure 14-15).

Table 14-13: Variogram Models used For Gold Domains – Summary

Domain	Variogram Orientation			Ranges			Variogram Parameters			Anisotropy Ratios			Anisotropy Ratios		
	Bearing	Plunge	Dip	A1	A2	A3	Nugget	S1	S2	Major	Semi-Major	Minor	Major	Semi-Major	Minor
212	50.00	3.00	-19.00	150.00	88.00	76.00	0.10	0.71	0.20	14.00	16.00	6.00	150.00	88.00	76.00
222	-40.00	-40.00	0.00	125.00	150.00	77.00	0.09	0.83	0.09	41.00	55.00	16.00	125.00	150.00	77.00
232	15.00	0.00	-75.00	120.00	76.00	60.00	0.16	0.58	0.27	20.00	21.00	26.00	120.00	76.00	60.00
211, 221, 231	-100.00	82.00	44.00	68.00	81.00	74.00	0.12	0.76	0.12	50.00	58.00	29.00	68.00	81.00	74.00
312	85.00	-9.00	-60.00	180.00	110.00	278.00	0.12	0.63	0.25	50.00	32.00	145.00	180.00	110.00	278.00
322	65.00	0.00	0.00	77.00	121.00	84.00	0.08	0.80	0.12	42.00	38.00	22.00	77.00	121.00	84.00
332	120.00	3.00	-19.00	78.00	126.00	80.00	0.13	0.63	0.24	43.00	57.00	26.00	78.00	126.00	80.00
311, 321, 331	-53.00	65.00	10.00	60.00	85.00	62.00	0.49	0.48	0.03	17.00	32.00	39.00	60.00	85.00	62.00
412	-22.00	46.00	-60.00	163.00	151.00	90.00	0.46	0.51	0.04	46.00	95.00	53.00	163.00	151.00	90.00
422	-11.00	-42.00	-132.00	240.00	349.00	139.00	0.18	0.71	0.11	85.00	85.00	138.00	240.00	349.00	139.00
432	14.00	-13.00	38.00	69.00	89.00	57.00	0.12	0.82	0.07	28.00	41.00	15.00	69.00	89.00	57.00
411, 421, 431	-33.00	24.00	7.00	107.00	40.00	40.00	0.12	0.62	0.00	44.00	20.00	20.00	44.00	40.00	40.00
512	98.00	19.00	7.00	67.00	149.00	54.00	0.18	0.75	0.07	33.00	31.00	30.00	67.00	149.00	54.00
522	-30.00	49.00	40.00	48.00	120.00	51.00	0.25	0.73	0.02	45.00	35.00	19.00	48.00	120.00	51.00
511, 521	-30.00	0.00	0.00	48.00	51.00	40.00	0.00	0.96	0.04	31.00	21.00	20.00	48.00	51.00	40.00
612	60.00	2.00	-10.00	191.00	140.00	69.00	0.17	0.47	0.37	23.00	60.00	31.00	191.00	140.00	69.00
622	47.00	-10.00	-28.00	73.00	52.00	18.00	0.11	0.85	0.04	52.00	40.00	6.00	73.00	52.00	18.00
611, 621	2.00	-25.00	51.00	147.00	67.00	40.00	0.00	0.98	0.02	104.00	41.00	20.00	147.00	67.00	40.00
712	70.00	60.00	-90.00	41.00	97.00	88.00	0.04	0.36	0.59	16.00	61.00	32.00	41.00	97.00	88.00
721	89.00	-54.00	-126.00	113.00	94.00	64.00	0.00	0.99	0.01	25.00	50.00	14.00	113.00	94.00	64.00
722	-70.00	20.00	0.00	159.00	84.00	115.00	0.03	0.27	0.70	84.00	50.00	77.00	156.00	84.00	115.00
912	35.00	-60.00	0.00	49.00	77.00	70.00	0.18	0.56	0.26	12.00	20.00	20.00	49.00	77.00	70.00
922	140.00	80.00	180.00	40.00	40.00	40.00	0.00	0.79	0.21	20.00	20.00	20.00	40.00	40.00	40.00
1021, 1121	5.00	70.00	90.00	25.00	91.00	84.00	0.00	0.88	0.12	4.00	20.00	20.00	25.00	91.00	84.00

Domain	Variogram Orientation			Ranges			Variogram Parameters			Anisotropy Ratios			Anisotropy Ratios		
	Bearing	Plunge	Dip	A1	A2	A3	Nugget	S1	S2	Major	Semi-Major	Minor	Major	Semi-Major	Minor
1022, 1122	36.00	23.00	68.00	111.00	40.00	235.00	0.09	0.62	0.29	58.00	20.00	141.00	111.00	40.00	235.00

Table 14-14: Variogram Models used for Silver Domains – Summary

Domain	Variogram Orientation			Ranges			Variogram Parameters			Anysotropy Ratios			Anysotropy Ratios		
	Bearing	Plunge	Dip	A1	A2	A3	Nugget	S1	S2	Major	Semi-Major	Minor	Major	Semi-Major	Minor
212	37.00	11.00	-146.00	305.00	351.00	99.00	0.64	0.33	0.03	73.00	80.00	36.00	305.00	351.00	99.00
222	20.00	-40.00	0.00	216.00	264.00	81.00	0.33	0.61	0.06	125.00	95.00	26.00	216.00	264.00	81.00
232	54.00	13.00	-38.00	444.00	163.00	53.00	0.25	0.56	0.20	56.00	31.00	52.00	444.00	163.00	53.00
211, 221, 231	-176.00	52.00	-143.00	43.00	82.00	65.00	0.02	0.94	0.04	16.00	45.00	39.00	43.00	82.00	65.00
312	130.00	5.00	-8.00	359.00	129.00	115.00	0.25	0.53	0.22	58.00	67.00	75.00	359.00	129.00	115.00
322	-13.00	-62.00	-136.00	134.00	163.00	155.00	0.00	0.72	0.29	52.00	50.00	69.00	134.00	163.00	155.00
332	-3.00	34.00	-23.00	132.00	142.00	138.00	0.00	0.69	0.31	34.00	31.00	51.00	132.00	142.00	138.00
311, 321, 331	158.00	-72.00	55.00	86.00	82.00	66.00	0.97	0.04	0.00	86.00	82.00	66.00	0.00	0.00	0.00
412	10.00	70.00	0.00	354.00	158.00	220.00	0.20	0.78	0.02	21.00	30.00	70.00	354.00	158.00	220.00
422	-85.00	-49.00	12.00	73.00	110.00	118.00	0.45	0.48	0.07	57.00	45.00	25.00	73.00	110.00	118.00
432	47.00	-9.00	70.00	274.00	55.00	107.00	0.14	0.52	0.34	59.00	15.00	27.00	274.00	55.00	107.00
411, 421, 431	-25.00	70.00	0.00	102.00	30.00	20.00	0.74	0.26	0.00	102.00	30.00	22.00	0.00	0.00	0.00
512	49.00	22.00	-62.00	155.00	111.00	61.00	0.26	0.71	0.04	43.00	51.00	40.00	155.00	111.00	61.00
522	50.00	0.00	-60.00	91.00	159.00	218.00	0.18	0.44	0.39	32.00	52.00	43.00	91.00	159.00	218.00
511, 521	43.00	13.00	59.00	115.00	77.00	40.00	0.35	0.57	0.08	45.00	31.00	20.00	115.00	77.00	40.00
612	104.00	4.00	14.00	218.00	209.00	113.00	0.12	0.84	0.04	44.00	45.00	31.00	218.00	209.00	113.00
622	124.00	7.00	7.00	110.00	88.00	115.00	0.16	0.72	0.13	49.00	66.00	42.00	110.00	88.00	115.00
611, 621	120.00	0.00	-10.00	77.00	95.00	14.00	0.00	0.73	0.27	30.00	53.00	3.00	77.00	95.00	14.00
712	70.00	-20.00	-90.00	84.00	33.00	116.00	0.06	0.60	0.34	50.00	25.00	49.00	84.00	33.00	116.00

Domain	Variogram Orientation			Ranges			Variogram Parameters			Anysotropy Ratios			Anysotropy Ratios		
	Bearing	Plunge	Dip	A1	A2	A3	Nugget	S1	S2	Major	Semi-Major	Minor	Major	Semi-Major	Minor
721	0.00	90.00	0.00	117.00	211.00	44.00	0.69	0.25	0.06	111.00	67.00	24.00	117.00	211.00	44.00
722	63.00	-63.00	20.00	110.00	106.00	49.00	0.36	0.45	0.19	38.00	67.00	41.00	110.00	106.00	49.00
912	-35.00	-5.00	0.00	63.00	66.00	40.00	0.00	0.60	0.40	22.00	20.00	20.00	63.00	66.00	40.00
922	-60.00	-15.00	0.00	264.00	109.00	40.00	0.00	0.25	0.75	20.00	20.00	20.00	264.00	109.00	40.00
1021, 1121	-70.00	69.00	-45.00	40.00	115.00	40.00	0.00	0.84	0.16	8.00	20.00	20.00	40.00	115.00	40.00
1022, 1122	50.00	75.00	90.00	30.00	40.00	40.00	0.00	0.97	0.03	6.00	20.00	20.00	30.00	40.00	40.00

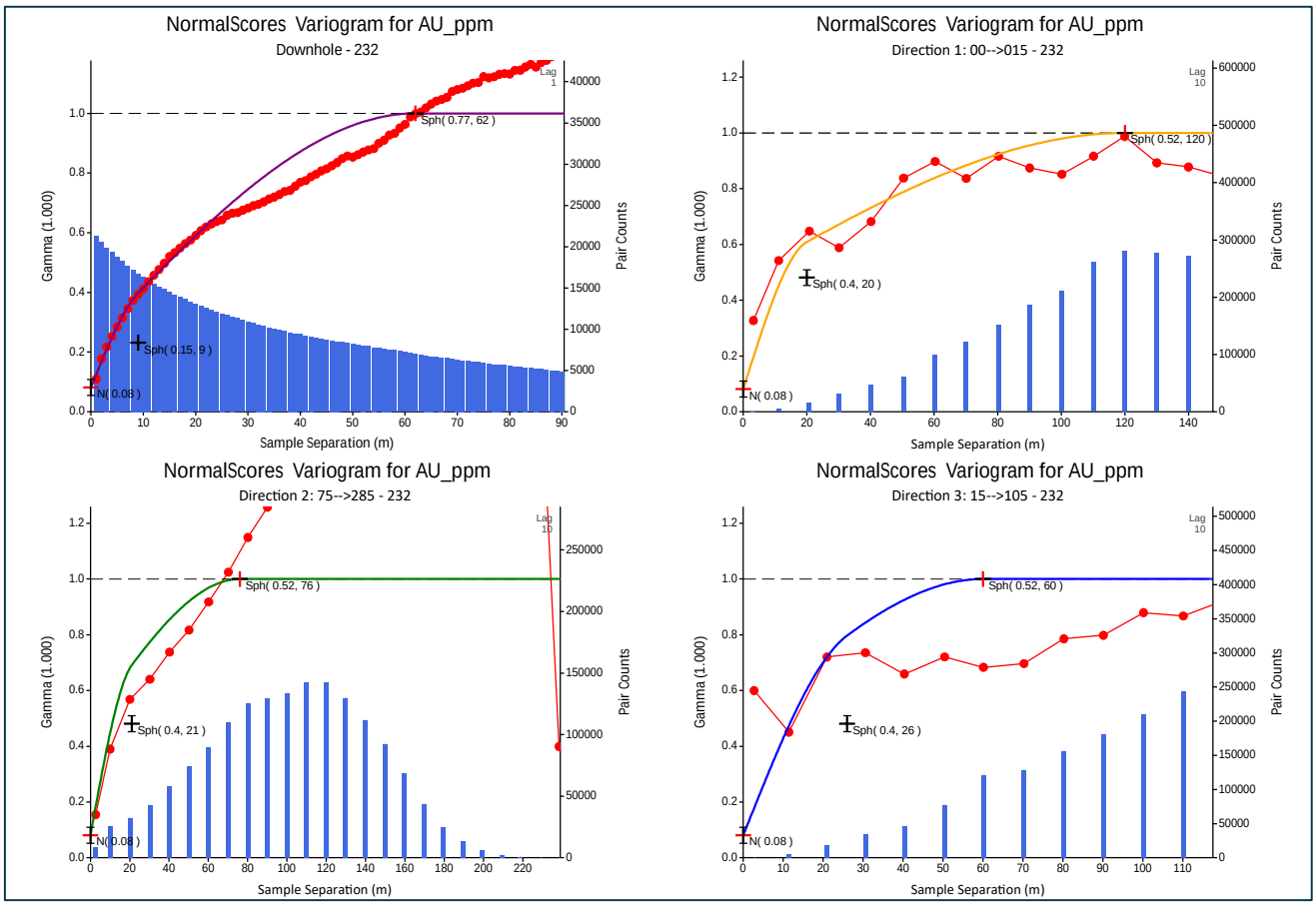


Figure 14-12: Gold Domain 232 – Normal Score Variogram

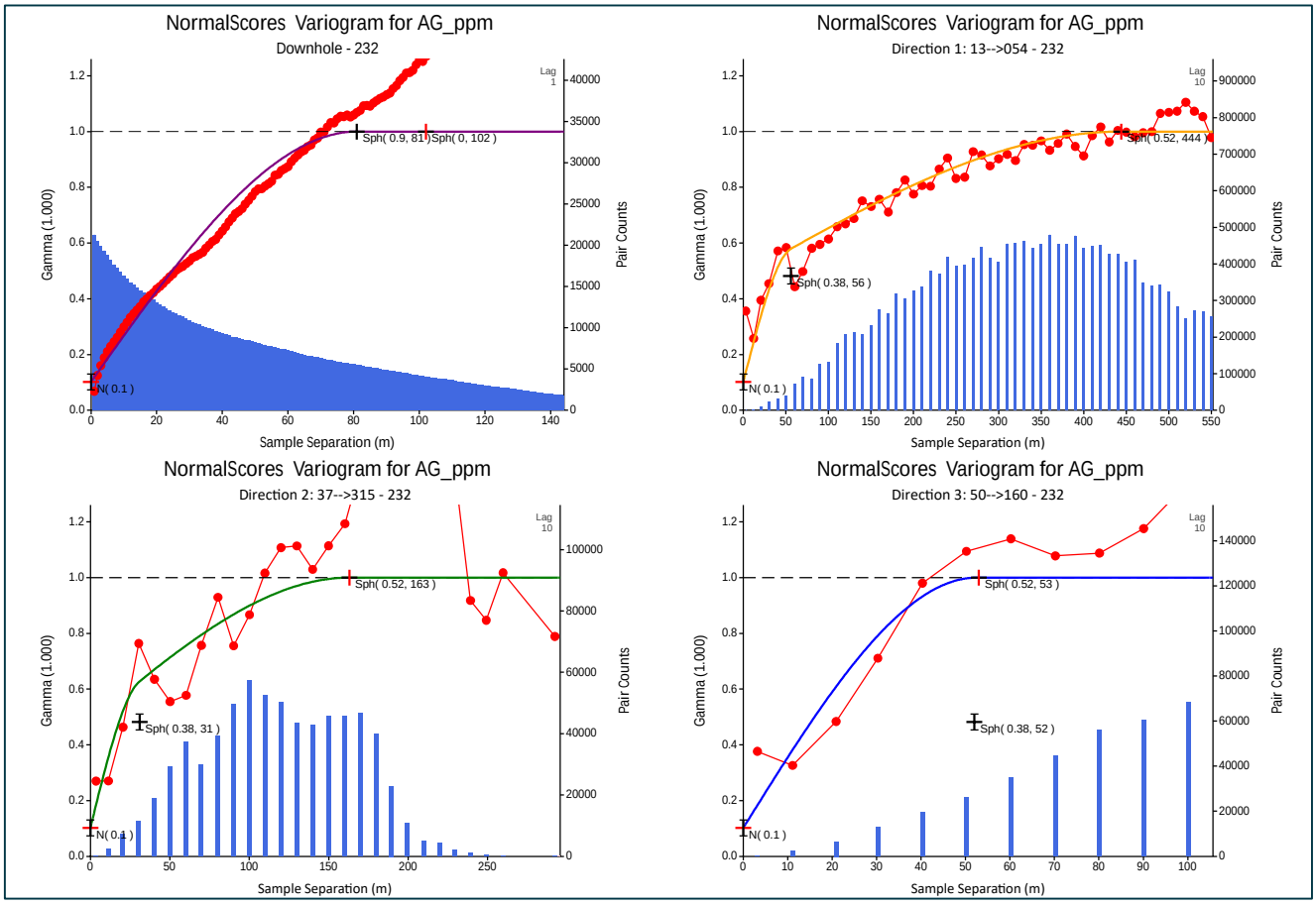


Figure 14-13: Silver Domain 232 – Normal Score Variogram

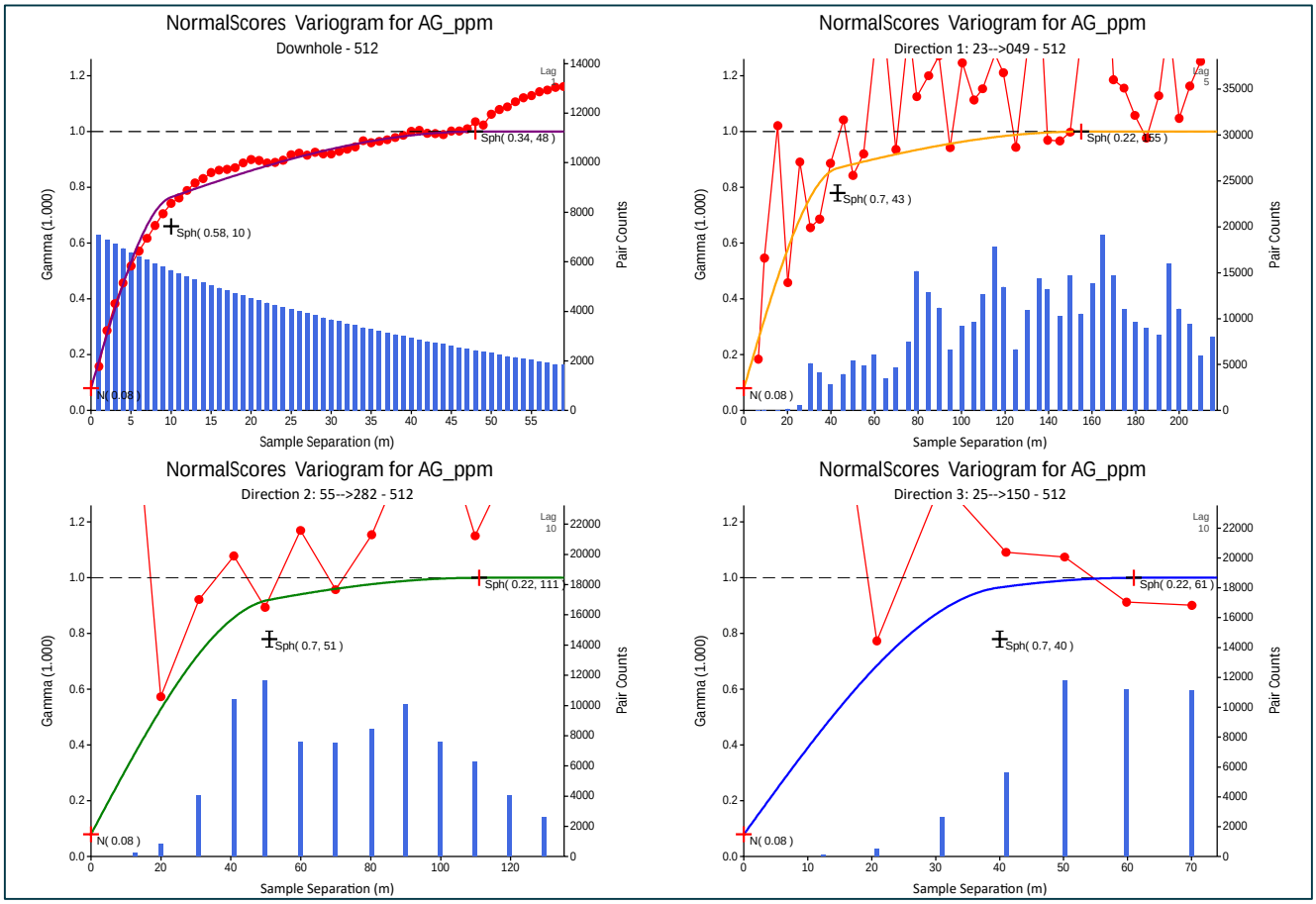


Figure 14-14: Silver Domain 512 – Normal Score Variogram

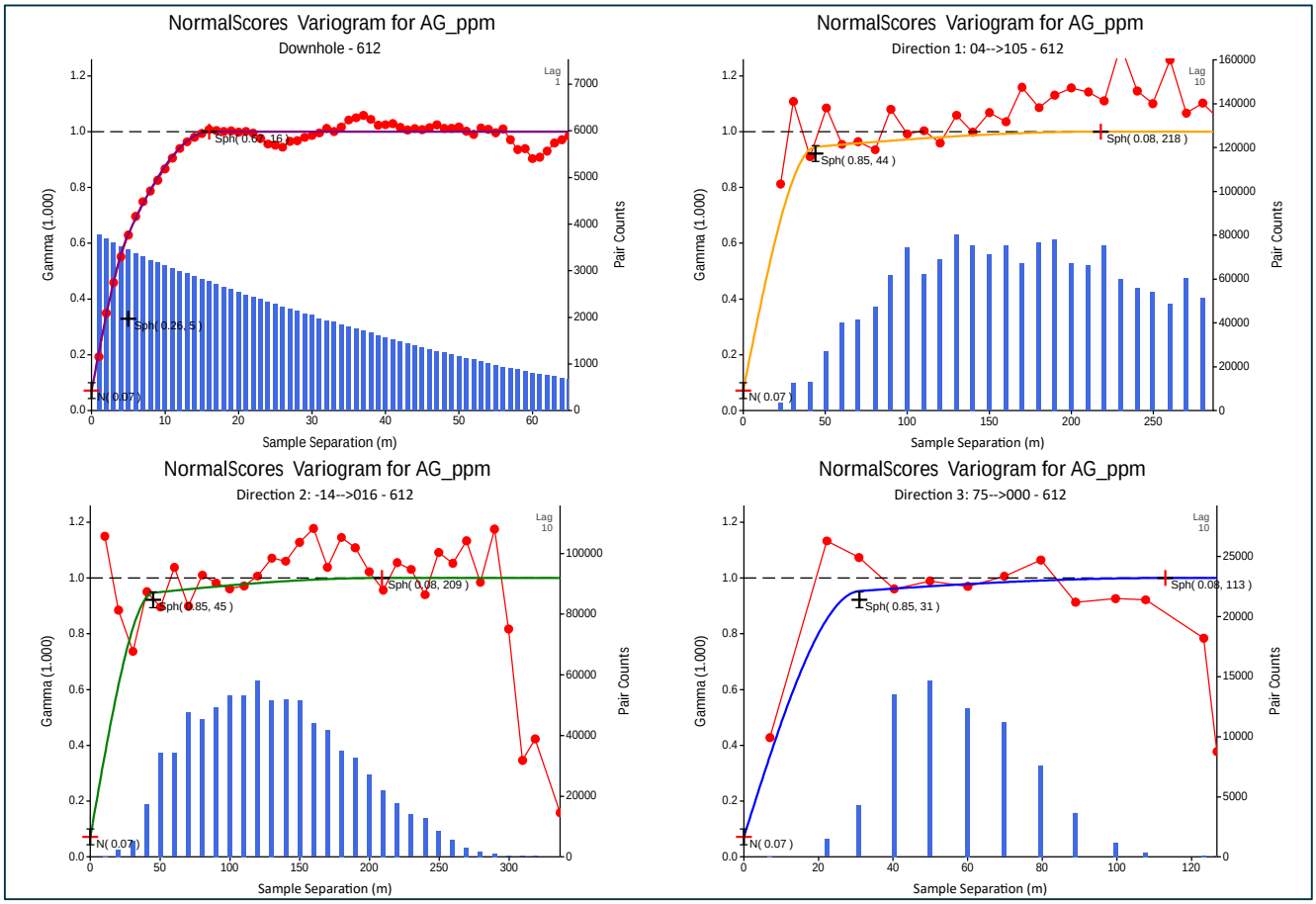


Figure 14-15: Silver Domain 612 – Normal Score Variogram

14.10 Block Model

A three-dimensional block model was constructed, covering all interpreted mineralized zones. The model dimensions included waste material and topsoil material to aid any future optimization studies.

14.11 Model Construction and Parameters

The MS MinePlan commercial mining software package was used for block model construction. The selected block size was based on the geometry of interpreted domains, data configuration and expected mining method. A parent cell size of 10 mE x 10 mN x 10 mRL was selected with no sub-blocking. Sufficient variables were included in the block model (DFS15.dat) construction to enable grade estimation. No block rotation was used.

The block model was considered suitable for reporting estimated Mineral Resources. A topographic surface was used to constrain the upper extent of the block model. The block model construction parameters are displayed in Table 14-15.

Table 14-15: Variogram Models used for Silver Domains – Summary.

	X	Y	Z
Block Model Origin	718710	7197840	3800
Block Model Maximum	722020	7201010	4750
Total extent (m)	3310	3170	950
Block Size	10	10	10
Number of Blocks	331	317	95

14.12 Grade Estimation

Grade estimation was performed using the Ordinary Kriging ("OK") function provided with MS MinePlan.

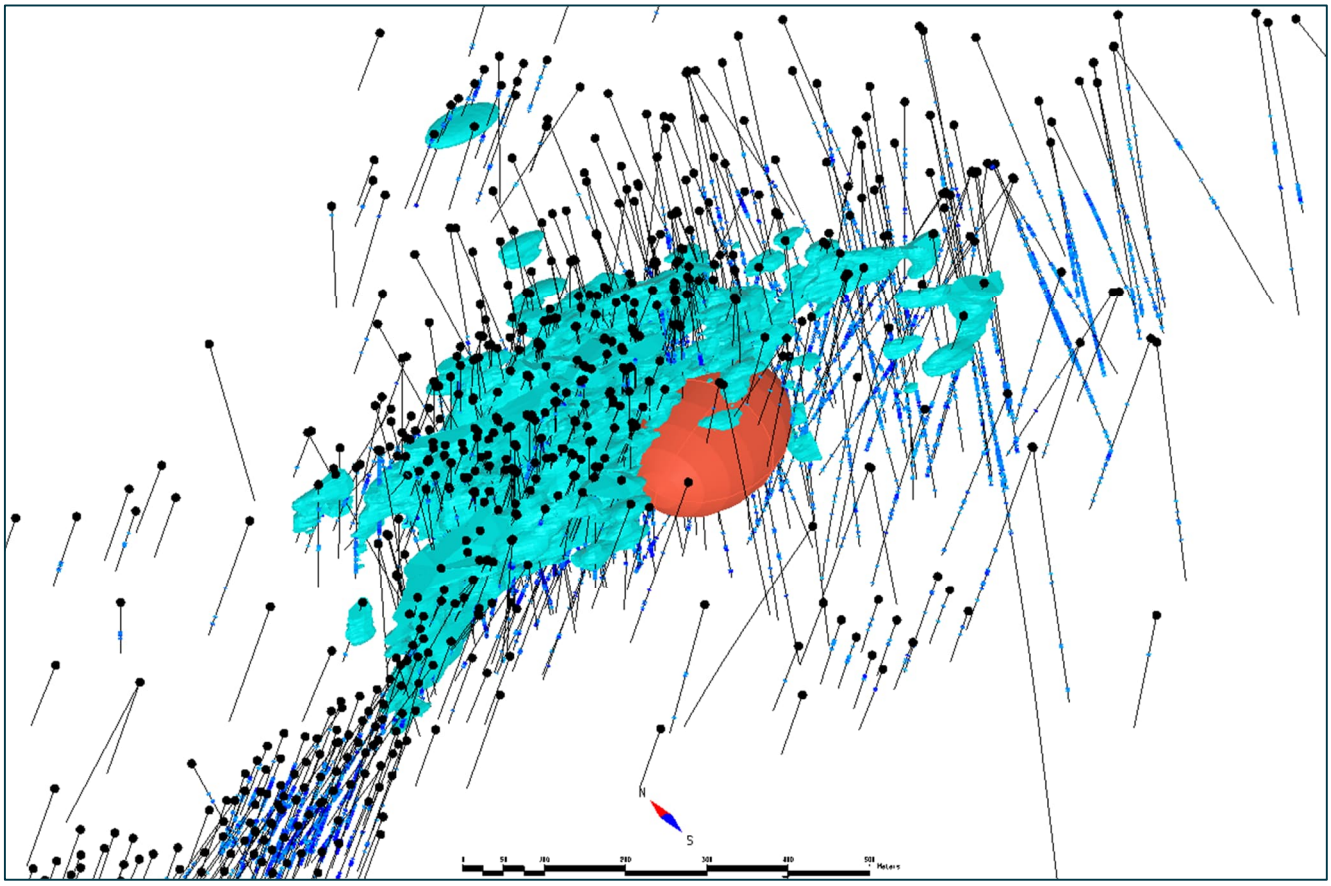
The block model was coded from geological solids with codes assigned to each estimation domain. The average distance to composites, Kriging Variance and Estimation Pass was stored in the block model to aid Mineral Resource classification.

14.13 Estimation Methods

The sample search strategy was based upon analysis of the normal score variogram model anisotropy, mineralization geometry and data distribution.

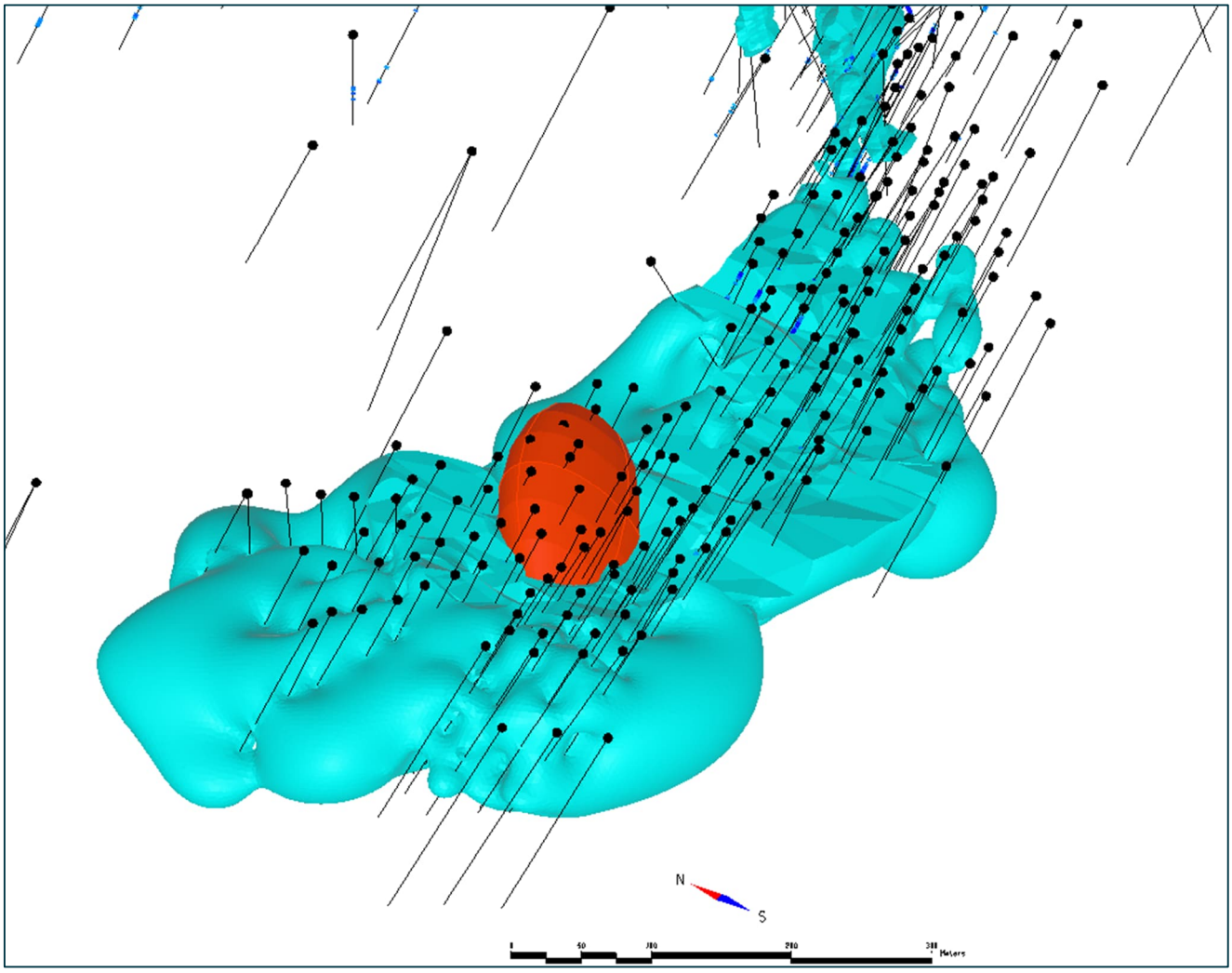
The first pass range was calculated based on the ratios 2 : 1, of average range of sill 0.7 (close to 30m in the strike). This was obtained as the average of the major range in all domain models of the combined mineralized domain. The QP Mr. Peralta considers that 30 meters is a common range in this type of precious metal deposit.

- The search strategy used in the block model is described in Table 14-12 and Table 14-13, additionally the following is noted:
- For all estimated domains, no octant search was applied.
- For all estimated domains, no coordinate transformation was applied.
- A minimum of 1 composite per block was used.
- A maximum of 5 composites per drill hole was used.
- A high-grade restriction in the second pass was applied, restricting the number of composites to be used per block to 15 composites.
- In the case of the cover domain (10), some economic grades were presented that are part of the mineralized structure. However, this was interpreted as loose material without transport, so no estimation was performed for this domain.



Source: AbraSilver Resource Corp., 2026

Figure 14-16: Domain 232 – 3D View of Search Ellipse Model for Silver



Source: AbraSilver Resource Corp., 2026

Figure 14-17: Domain 512 – 3D View of Search Ellipse Model for Silver

14.14 Metal Risk Review

No metal risk analysis was performed to evaluate the impact of metal loss due to the capping of extreme gold and silver grades. The capping average for all estimation domains was performed simultaneously with the analysis. The metal loss calculated over the composite represents no more than 1.5 %.

14.15 Parent Cell Size Sensitivity

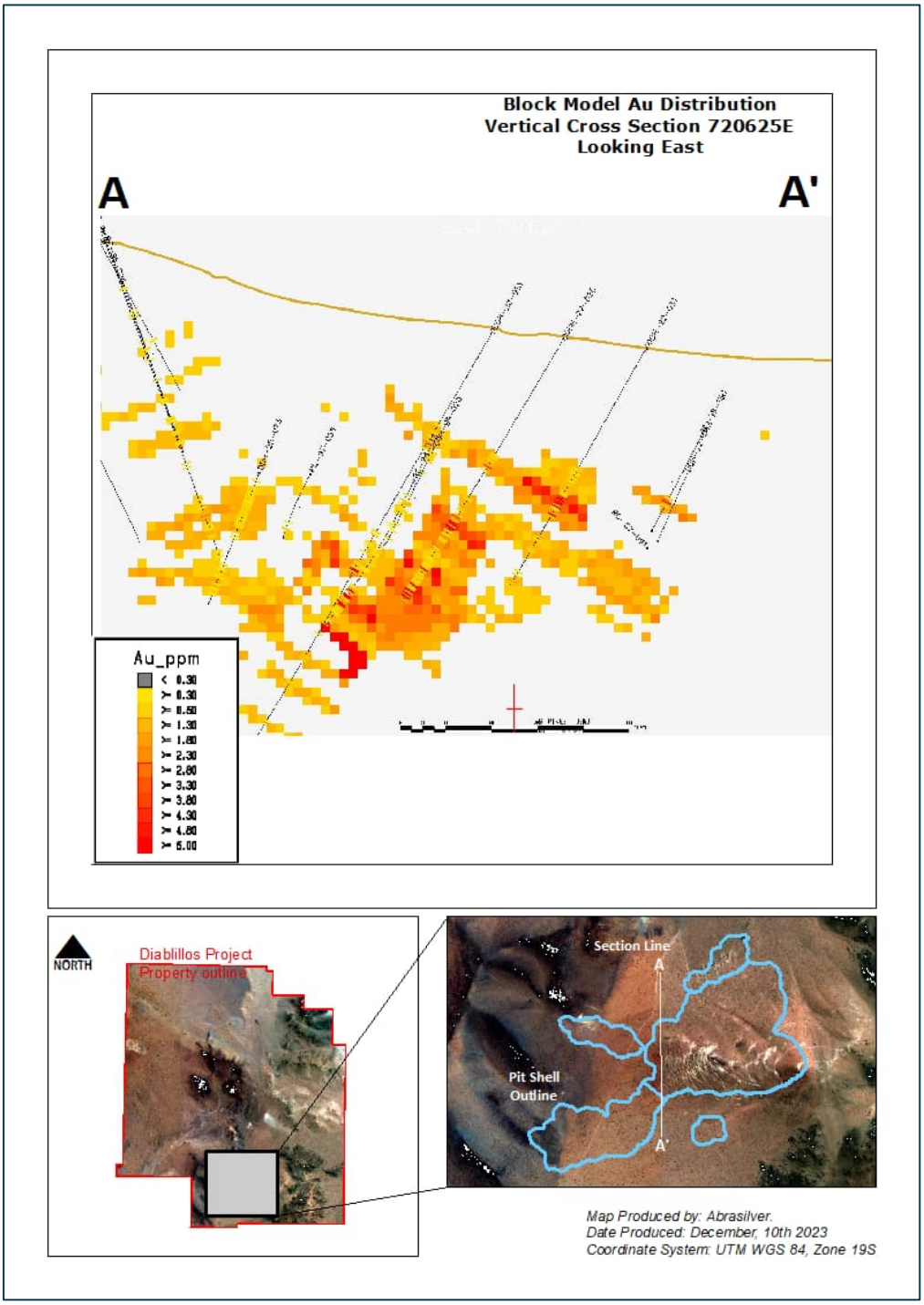
No parent cell size sensitivity was performed at this time since it was completed in a previous NI 43-101 by Mining Plus, 2022. The author considers that the conclusion of that analysis was correct. It is important to mention that the dilution impact of a 10 mE x 10 mN x 10 mRL block

model for MRE is acceptable for an open pit and is similar to other projects with similar characteristics.

14.16 Model Validation

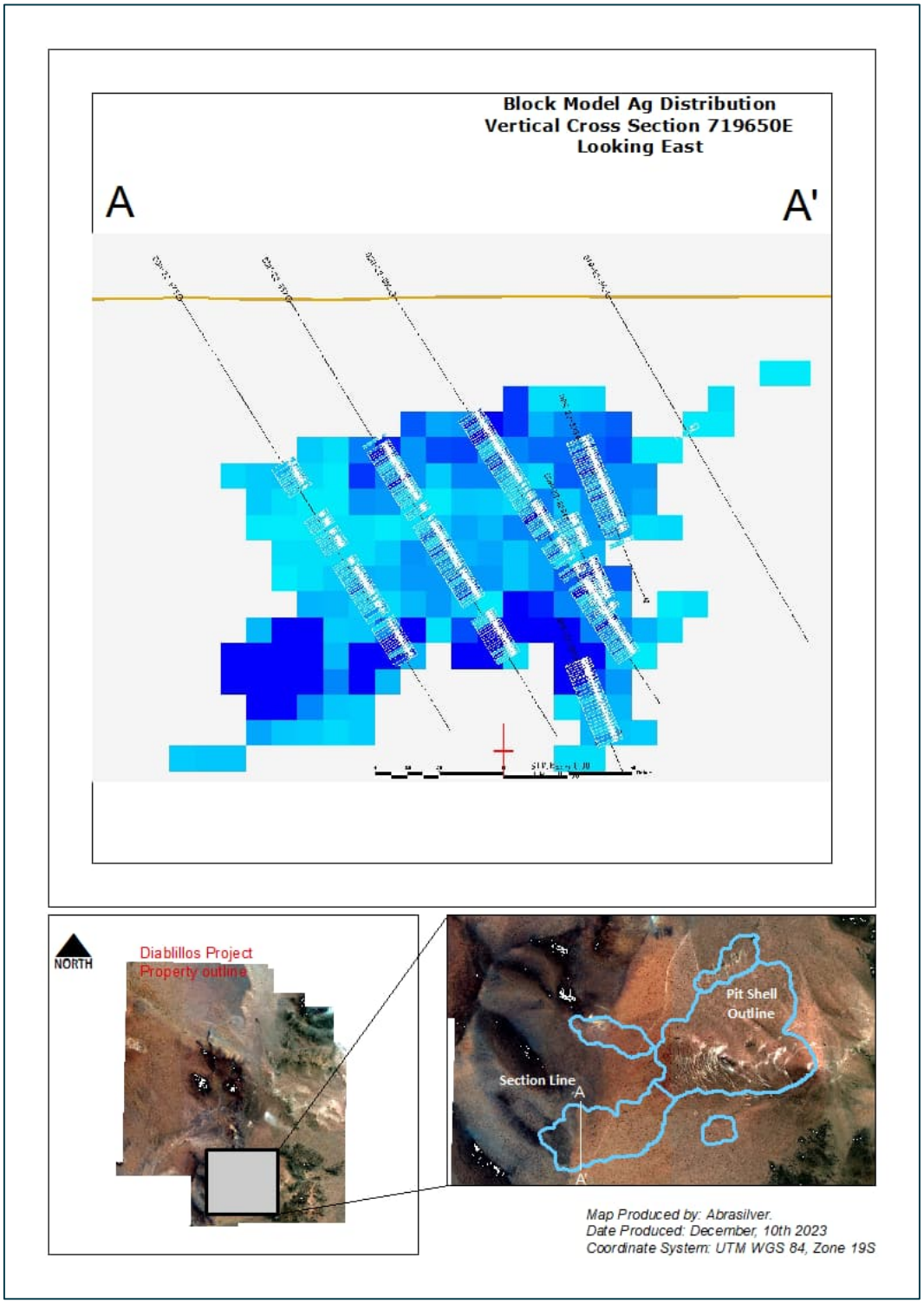
14.16.1 Visual Inspection

Block grades were compared visually with supporting drill data on section and plan maps. A good fit with the composites was observed. An example section of block grades and composites is included in Figure 14-18 and Figure 14-19.



Source: AbraSilver Resource Corp., 2026

Figure 14-18: Section 720625- E with Block Model 10 mE x 10 mN x 10 mRL and Composite for Gold



Source: AbraSilver Resource Corp., 2026

Figure 14-19: Section 719650- E with Block Model 10 mE x 10 mN x 10 mRL and Composite for Silver

14.16.2 Trend Plots Validation

Validation trend plots or swath plots are presented to graphically display a comparison of the mean grade of the estimate compared with the composites. Block model slices were aligned on Easting, Northing and RL with average grade calculated for the various domains.

Figure 14-20 and Figure 14-21 shows the Ordinary Kriged estimate compares favorably to the native composite data.

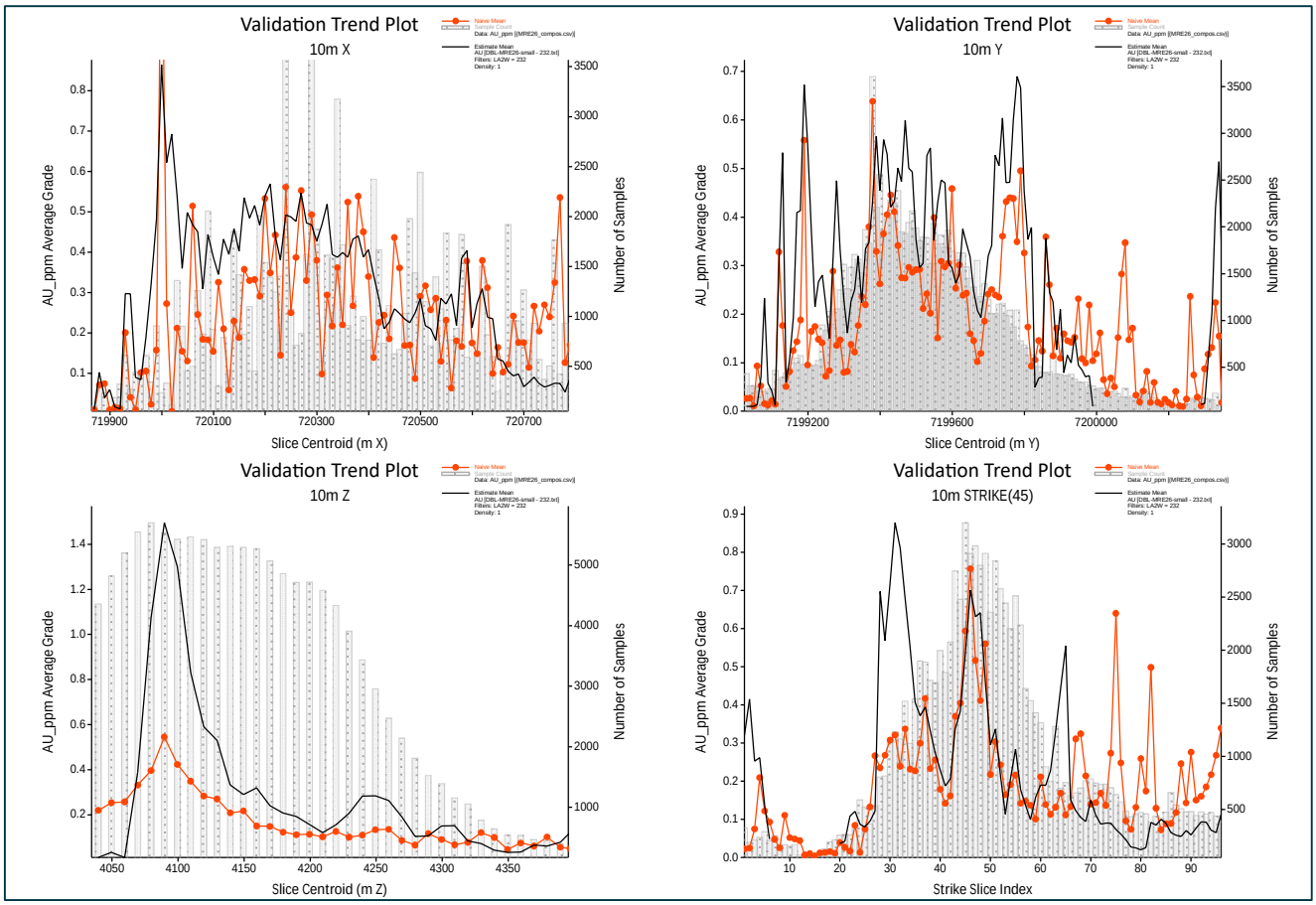


Figure 14-20: Swath Plots comparing Native Data versus Estimated Data. Estimates for Gold Domains.

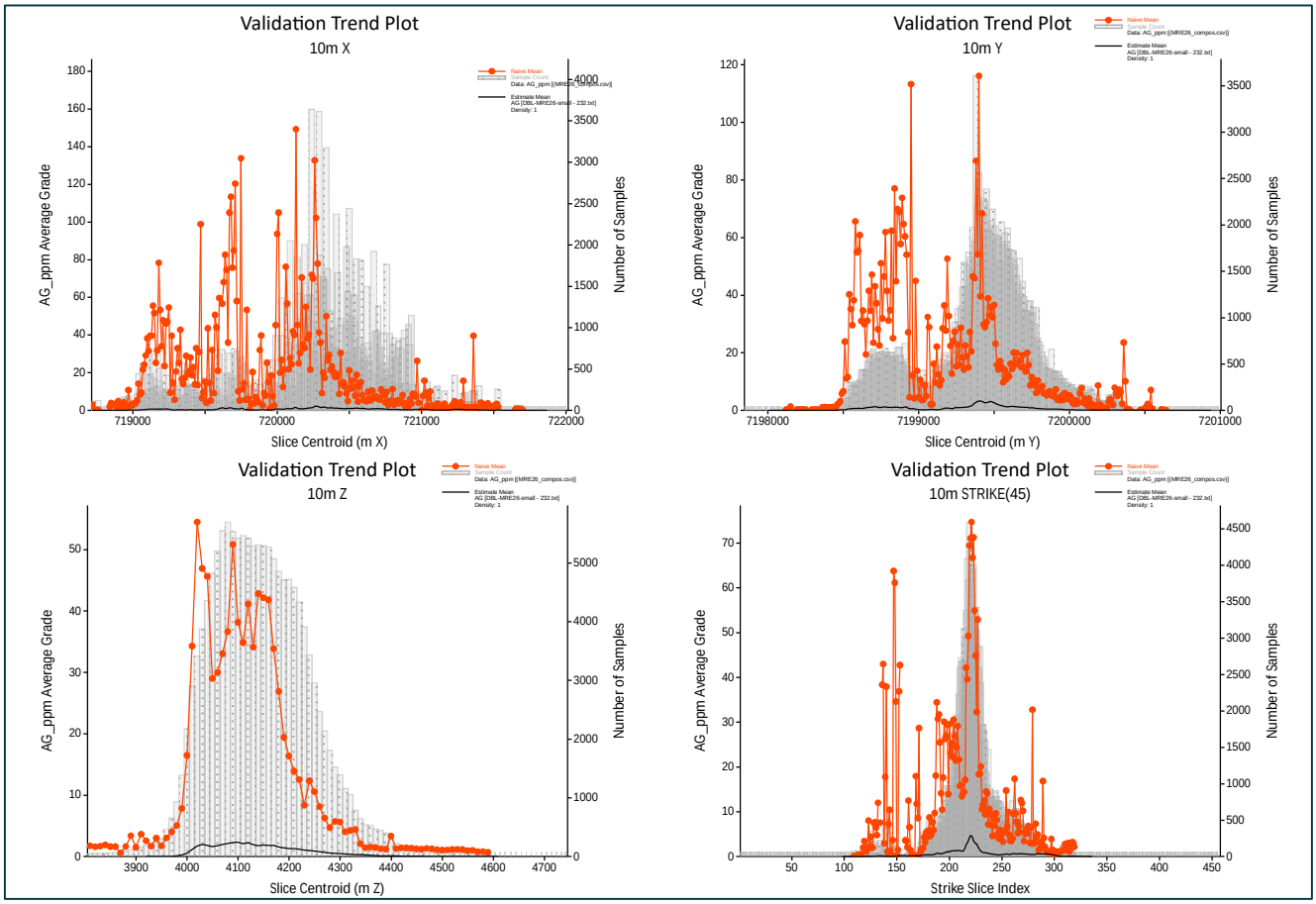


Figure 14-21: Swath Plots comparing Native Data versus Estimated Data. Estimates for Silver Domains

14.17 Mineral Resource Classification and Criteria

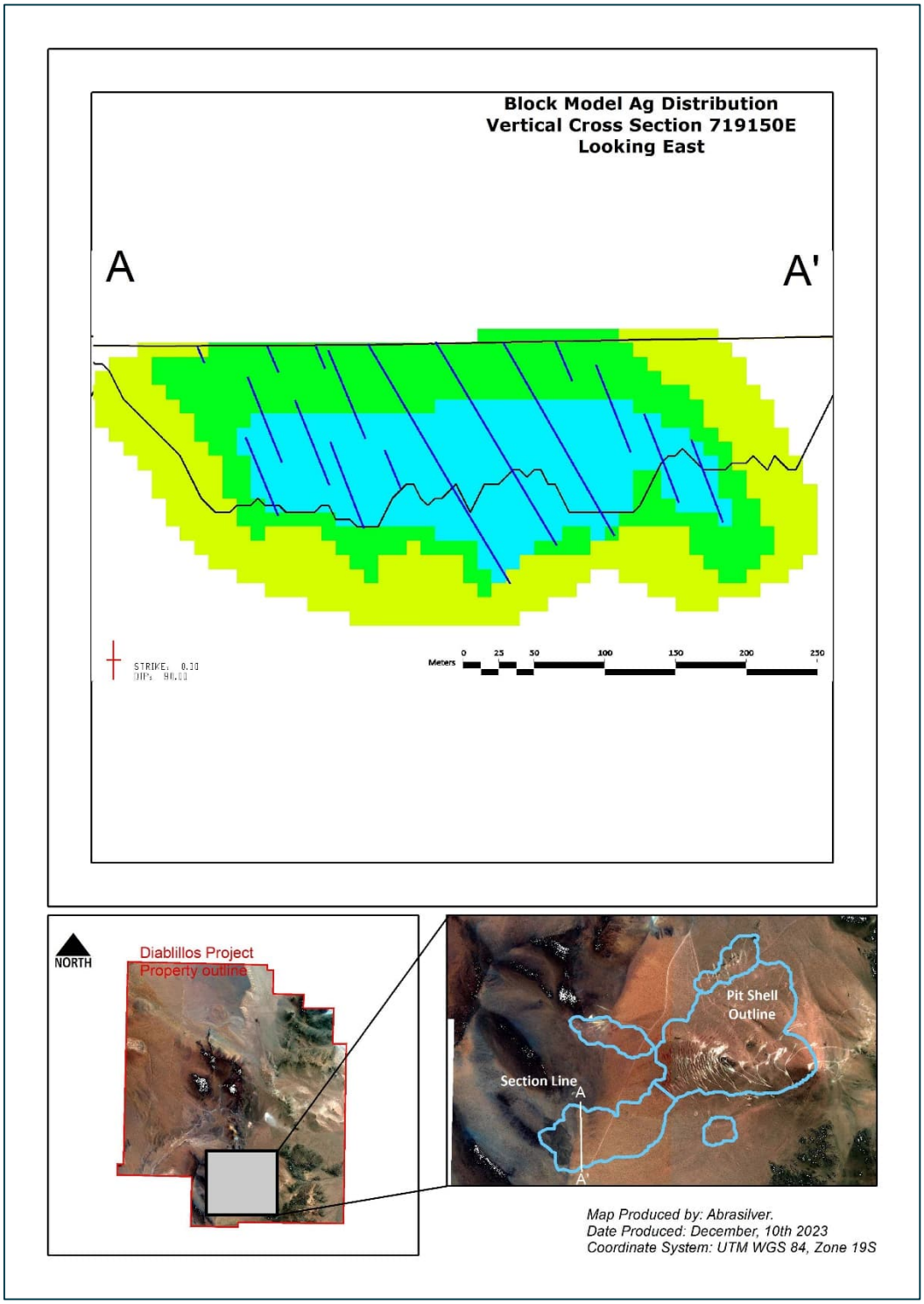
The Mineral Resource has been categorized into Measured, Indicated and Inferred, reflecting uncertainty in geological evidence, hole spacing, and data quality. Mineral Resources were classified considering the following information:

- Confidence of geological information.
- QA / QC results, hole deviation measurements for historical and recent holes.
- Hole spacing.
- Estimate search passes.
- Wireframe to restrict the estimation passes.

After visual inspection of these models, a set of solids were built to code the block model. A smoothing step then allowed classification as:

- Measured: Blocks within a halo of 30 m around the AbraSilver holes and relog holes that contain more than 3 downhole survey records. These blocks are seen in recent drilling by AbraSilver. A good proportion of historical holes have deviation measurements.
- Indicated: Blocks within a halo of 70 m around AbraSilver holes and relog holes that contain more than 3 downhole survey records. These blocks are also well seen in recent drilling by AbraSilver, but the nominal drill spacing exceeds 30m.
- Inferred: Blocks within a halo of 120m around the AbraSilver holes. Some mineralized areas are seen in recent drilling by AbraSilver, but the nominal drill spacing exceeds 70m.
- Blocks based on isolated holes, broadly spaced holes at depth, and blocks beyond the end of the hole were classified as geological exploration potential (outside of wireframes) and are not included in the reported Mineral Resources.

Figure 14-22 shows the wireframe used to restrict the classification of Mineral Resources in vertical cross section.



Source: AbraSilver Resource Corp., 2026

Figure 14-22: Vertical Cross Section 719150- E showing the Wireframe used to Categorize the Block Model

14.18 Mineral Resource Statement

The MRE for the Diablillos Project, with an effective date of as May 6th, 2026, has been estimated and classified based on the CIM's Estimation of Mineral Resource and Mineral Reserves Best Practices Guidelines (CIM, 2019) and is reported in accordance with the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101").

Mineral Resources at the Diablillos Project are considered as potentially mineable by an open pit method. They are estimated based on drilling conducted prior to AbraSilver and more recent drilling by AbraSilver between 2019 and 2025. The Mineral Resource includes the Oculto, JAC, Sombra, Laderas and Fantasma zones. The Mineral Resource is reported inside an open pit shell based on a Net Value per Block valuation method. The average cut-off grade is 39 g/t silver equivalent for tank leaching and 15 g/t for the heap leaching, based on a gold price of US$3,200/oz and a silver price of US$34.50/oz. Mining costs and metallurgical recoveries were provided by AbraSilver.

The Qualified Person ("QP") for the MRE is Mr. Luis Rodrigo Peralta, B.Sc., FAusIMM CP (Geo), an INSA Consultora Managing Principal Geologist, and an Independent Qualified Person under NI 43-101.

The following is a summary of the estimation process:

- Grades for diamond and reverse circulation drill holes (790 drill holes) were composited to 1.0 meter.
- Estimation domains were defined using a combination of alteration and lithology domains, defining a subset domain for gold and silver.
- Grade capping has been applied to composited grade intervals for each domain.
- The normal score variograms for gold and silver were modeled for estimation domains with sufficient data.

The Mineral Resource was estimated with Ordinary Kriging ("OK") and bias was reviewed using Inverse Distance squared (ID2) for comparison.

The estimation used a block model constructed in MS MinePlan mining software.

- Grades were estimated for parent cells of 10 mE x 10 mN x 10 mRL.
- The bulk density applied to the block model was based on 7,654 drill core samples. The average number of samples contained in each domain has been assigned to each wireframe and the block model.
- The final block model is 10 mE x 10 mN x 10 mRL for Mineral Resource optimization and reporting.

The MRE comprises Measured, Indicated and Inferred Mineral Resources as summarized in Table 14-16. The block model "DFS15.dat" was used to report with constraints fields: "ORE = 1", and "CLASS = 1, 2 and 3" with the proportion of the model below the topographical surface.

Mineral Resource Estimate by deposit is shown in Table 14-17.

Table 14-16: Total Diablillos Mineral Resource Summary (Tank & Heap Leach) – As of April 30, 2026.

Zone	Zone	Category	Tonnes (000 t)	Ag (g/t)	Au (g/t)	AgEq (g/t)	Contained Ag (000 Oz Ag)	Contained Au (000 Oz Ag)	Contained AgEq (000 Oz Ag)
Tank Leaching	Oxides	Measured	41042	100	0.68	159	131 668	896	209 281
		Indicated	60978	41	0.58	92	81 060	1 143	180 078
		Measured & Indicated	102021	65	0.62	119	212 728	2 039	389 359
		Inferred	14400	25	0.57	74	11 468	262	34 187
Heap Leaching	Oxides	Measured	25469	13	0.09	19	10 997	76	15 425
		Indicated	104491	7	0.13	15	24 328	428	49 342
		Measured & Indicated	129960	8	0.12	16	35 325	503	64 767
		Inferred	34947	6	0.14	14	6 939	158	16 153
Total	Oxides	Measured	66512	67	0.45	105	142 665	971	224 706
		Indicated	165469	20	0.30	43	105 388	1 570	229 420
		Measured & Indicated	231981	33	0.34	61	248 053	2 542	454 127
		Inferred	49347	12	0.26	32	18 406	420	50 340

Notes for Mineral Resource Estimate:

1. Mineral Resources are not Mineral Reserves and have not demonstrated economic viability.
2. The formula for calculating AgEq is as follows: Silver Eq Oz = Silver Oz + Gold Oz x (Gold Price/Silver Price) x (Gold Recovery/Silver Recovery).
3. The Mineral Resource model was populated using Ordinary Kriging grade estimation within a three-dimensional block model and mineralized zones defined by wireframed solids, which are a combination of lithology and alteration domains. The 1m composite grades were capped where appropriate.

4. The Mineral Resource is reported inside a conceptual open pit shell derived using US$ 34.50/oz Ag price, US $3,200/oz Au price, 86.6% process recovery for Au, and 80.9% process recovery for Ag, for the tank leaching and 74.3% process recovery for Au, and 46.8% process recovery for Ag, for the secondary heap leaching.
5. Open pit optimization was constrained using a dual-process approach, with tank leaching as the primary process (total opex of US$32.30/t) and heap leaching as the secondary process (total opex of US$7.00/t).
6. The MRE has been categorized in accordance with the CIM Definition Standards (CIM, 2014).
7. A Net Value per block [NVB] calculation was used to constrain the Mineral Resource, determine the "Benefits = Revenue-Cost", where, Revenue = [(Au Selling Price (US$/oz) - Au Selling Cost (USD/Oz)) x (Au grade (g/t)/31.1035)) x Au Recovery (%)] + [(Ag Selling Price (US$/oz) - Ag Selling Cost (USD/Oz)) x (Ag grade (g/t)/31.1035)) x Ag Recovery (%)] and Cost = Mining Cost (US$/t) + Process Cost (US$/t) + Transport Cost (US$/t) + G&A Cost (US$/t) + [Royalty Cost (%) x Income]
8. The Mineral Resource is sub-horizontal with sub-vertical feeders and has a reasonable prospect for eventual economic extraction by open pit methods.
9. In-situ bulk densities were assigned to each model domain, according to samples averages for each lithology domain, separated by alteration zones and subset by oxidation.
10. All tonnages reported are dry metric tonnes and ounces of gold and silver contained are troy ounces.
11. Mining recovery and dilution factors have not been applied to the Mineral Resource estimates.
12. The Mineral Resource was estimated by Luis Rodrigo Peralta, B.Sc., FAusIMM CP (Geo), an INSA Consultora Managing Principal Geologist, and an Independent Qualified Person under NI 43-101.
13. Mr. Peralta is not aware of any environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues that could materially affect the potential development of the Mineral Resource.
14. All figures are rounded to reflect the relative accuracy of the estimates. Minor discrepancies may occur due to rounding to appropriate significant figures.

Table 14-17: Total Diablillos Mineral Resource Summary by Zone (Tank & Heap Leach) – As of April 30, 2026.

Zone	Zone	Category	Tonnes (000 t)	Ag (g/t)	Au (g/t)	AgEq (g/t)	Contained Ag (000 Oz Ag)	Contained Au (000 Oz Au)	Contained AgEq (000 Oz Ag)
Oculto	Oxides	Measured	53747	54	0.55	102	93 891	944	173 602
		Indicated	146634	16	0.32	44	75 326	1 503	194 188
		Measured & Indicated	200380	26	0.38	59	169 217	2 447	367 791
		Inferred	42950	9	0.28	33	12 147	389	41 783
JAC	Oxides	Measured	12765	119	0.07	125	48 774	27	51 104
		Indicated	7863	84	0.03	87	21 286	8	21 963
		Measured & Indicated	20628	106	0.05	110	70 060	36	73 067
		Inferred	1561	79	0.01	80	3 967	1	4 005
Fantasma	Oxides	Measured	-	-	-	-	-	-	-
		Indicated	2293	63	0.01	63	4 621	0	4 653
		Measured & Indicated	2293	63	0.01	63	4 149	0	4 176
		Inferred	416	66	0.01	66	881	0	889
Laderas	Oxides	Measured	-	-	-	-	-	-	-
		Indicated	5678	9	0.27	32	1 583	50	5 409
		Measured & Indicated	5678	9	0.27	32	1 583	50	5 409
		Inferred	4135	8	0.22	28	1 115	30	3 362
Sombra	Oxides	Measured	-	-	-	-	-	-	-
		Indicated	3001	27	0.09	35	2 572	9	3 207
		Measured & Indicated	3001	27	0.09	35	2 572	9	3 207

Zone	Zone	Category	Tonnes (000 t)	Ag (g/t)	Au (g/t)	AgEq (g/t)	Contained Ag (000 Oz Ag)	Contained Au (000 Oz Au)	Contained AgEq (000 Oz Ag)
		Inferred	284	32	0.01	33	297	0	300
Total	Oxides	Measured	66512	67	0.45	106	142 665	971	224 706
		Indicated	165469	20	0.30	45	105 388	1 570	229 420
		Measured & Indicated	231981	33	0.34	63	248 053	2 542	454 127
		Inferred	49063	12	0.27	31	18 406	420	50 340

Table 14-18: Diablillos Mineral Resource Summary by Deposit (Tank Leach Process) – As of April 30, 2026.

Zone	Zone	Category	Tonnes (000 t)	Ag (g/t)	Au (g/t)	AgEq (g/t)	Contained Ag (000 Oz Ag)	Contained Au (000 Oz Au)	Contained AgEq (000 Oz Ag)
Oculto	Oxides	Measured	31443	85	0.86	159	85 507	870	160 890
		Indicated	51617	34	0.66	91	55 745	1 099	150 983
		Measured & Indicated	83059	53	0.74	117	141 252	1 969	311 872
		Inferred	12145	17	0.62	71	6 705	244	27 810
JAC	Oxides	Measured	9600	150	0.08	157	46 161	26	48 392
		Indicated	5192	115	0.04	118	19 125	7	19 729
		Measured & Indicated	14792	137	0.07	143	65 286	33	68 120
		Inferred	874	121	0.01	122	3 407	0	3 432
Fantasma	Oxides	Measured	-	-	-	-	-	-	-
		Indicated	1738	74	0.01	75	4 149	0	4 176
		Measured & Indicated	1738	74	0.01	75	4 149	0	4 176

Zone	Zone	Category	Tonnes (000 t)	Ag (g/t)	Au (g/t)	AgEq (g/t)	Contained Ag (000 Oz Ag)	Contained Au (000 Oz Au)	Contained AgEq (000 Oz Ag)
		Inferred	337	75	0.01	76	814	0	822
Laderas	Oxides	Measured	-	-	-	-	-	-	-
		Indicated	1520	15	0.66	72	726	32	3 501
		Measured & Indicated	1520	15	0.66	72	726	32	3 501
		Inferred	988	14	0.57	63	435	18	2 017
Sombra	Oxides	Measured	-	-	-	-	-	-	-
		Indicated	912	45	0.15	58	1 314	4	1 689
		Measured & Indicated	912	45	0.15	58	1 314	4	1 689
		Inferred	56	59	0.00	59	106	0	106
Total	Oxides	Measured	41042	100	0.68	159	131 668	896	209 281
		Indicated	60978	41	0.58	92	81 060	1 143	180 078
		Measured & Indicated	102021	65	0.62	119	212 728	2 039	389 359
		Inferred	14400	25	0.57	74	11 468	262	34 187

Table 14-19: Diablillos Mineral Resource Summary by Deposit (Heap Leach Process) – As of April 30, 2026.

Zone	Zone	Category	Tonnes (000 t)	Ag (g/t)	Au (g/t)	AgEq (g/t)	Contained Ag (000 Oz Ag)	Contained Au (000 Oz Au)	Contained AgEq (000 Oz Ag)
Oculto	Oxides	Measured	22304	12	0.10	18	8 384	74	12 712
		Indicated	95017	6	0.13	14	19 580	404	43 206
		Measured & Indicated	117321	7	0.13	15	27 965	478	55 918
		Inferred	30805	5	0.15	14	5 442	146	13 973
JAC	Oxides	Measured	3165	26	0.02	27	2 613	2	2 713
		Indicated	2671	25	0.01	26	2 161	1	2 234
		Measured & Indicated	5836	25	0.02	26	4 774	3	4 947
		Inferred	688	25	0.01	26	560	0	573
Fantasma	Oxides	Measured	-	-	-	-	-	-	-
		Indicated	555	26	0.01	27	472	0	477
		Measured & Indicated	555	-	-	-	-	-	-
		Inferred	79	26	0.00	26	66	0	67
Laderas	Oxides	Measured	-	-	-	-	-	-	-
		Indicated	4159	6	0.13	14	856	18	1 908
		Measured & Indicated	4159	6	0.13	14	856	18	1 908
		Inferred	3147	7	0.11	13	680	11	1 345
Sombra	Oxides	Measured	-	-	-	-	-	-	-
		Indicated	2090	19	0.07	23	1 258	4	1 517
		Measured & Indicated	2090	19	0.07	23	1 258	4	1 517
		Inferred	229	26	0.01	26	191	0	194

Zone	Zone	Category	Tonnes (000 t)	Ag (g/t)	Au (g/t)	AgEq (g/t)	Contained Ag (000 Oz Ag)	Contained Au (000 Oz Au)	Contained AgEq (000 Oz Ag)
Total	Oxides	Measured	25469	13	0.09	19	10 997	76	15 425
		Indicated	104491	7	0.13	15	24 328	428	49 342
		Measured & Indicated	129960	8	0.12	15	34 854	503	64 291
		Inferred	34947	6	0.14	18	6 939	158	16 153

14.19 Reasonable Prospects for Eventual Economic Extraction Requirement

An open pit optimization was conducted using the MS MinePlan Pit Optimizer module. This was used to constrain the MRE and demonstrate “reasonable prospects for eventual economic extraction”, satisfying the requirement of NI 43-101 and the Mineral Resource and Mineral Reserves Best Practices Guidelines (CIM, 2019).

The oxide material has reasonable prospects of economic extraction based on Net Value per Block. This assumes mine operating costs of USD 28.23/t for tank leaching process and USD 11.31/t for heap leaching process. An average cut-off grade of approximately 39 g/t AgEq for the tank leaching process and 22g/t AgEq for the heap leaching process. The “Net Value per Block” methodology used the single next formula below:

Net Value per Block (NVB) = Income - Cost

Being positive where:

Income = [(Au Selling Price (USD/oz) - Au Selling Cost (USD/oz)) x (Au grade (g/t)/31.1035)) x Au Recovery (%)] + [(Ag Selling Price (USD/oz) - Ag Selling Cost (USD/oz)) x (Ag grade (g/t)/31.1035)) x Ag Recovery (%)]

And

Cost = Mining Cost (USD/t) + Process Cost (USD/t) + Transport Cost (USD/t) + G&A Cost (USD/t) + [Royalty Cost (%) x Revenue]

Inputs for the formula are listed in Table 14-20.

Table 14-20: Optimization Parameters

OP Optimization Parameters		Unit	Value (9,000 tpd)
Overall Pit Slope Angle - Oculto		Degrees	55
Tank Leaching	Mining cost	$ per tonne	2.23
	Processing cost	$ per tonne	23.84
	G&A cost	$ per tonne	6.51
Heap Leaching	Processing cost	$ per tonne	7.00
Mining Recovery		%	100
Mining Dilution		%	0
Gold metal price		$/oz	3200
Silver metal price		$/oz	34.5
Cut-off grade (based on Net Value per Block) - Tank		Ageq	39
Cut-off grade (based on Net Value per Block) - Heap		Ageq	15

OP Optimization Parameters		Unit	Value (9,000 tpd)
Transport cost		$ per tonne	0.00
Tank Leaching	Metallurgical Recovery for Gold	%	86.6
	Metallurgical Recovery for Silver	%	80.9
Heap Leaching	Metallurgical Recovery for Gold	%	74.4
	Metallurgical Recovery for Silver	%	46.8
Royalties (over incomes)		%	3
Block size			10 x 10 x10

Metallurgical recoveries used in the open pit optimization are based on the metallurgical model described in the report Section 13. The average silver recovery was 80.9% and an average gold recovery was 86.6% for the primary process tank leaching and the average silver recovery was 46.8% and an average gold recovery was 74.4% for the secondary process heap leaching.

AbraSilver has conducted a detailed analysis of metallurgy to have confidence in the estimation domains. Mr. Peralta agrees that for this stage of study, confidence, and the level of information of metallurgical testing is enough. However, with new information a detailed geo-metallurgical model should be updated.

Geotechnical sectors have been defined based on four geotechnical drilling campaigns performed by AbraSilver during the years 2008, 2022, 2023 and 2024. The overall slopes angles defined in Table 14-21 have been defined based on detailed geotechnical logs using the method RMR from Bieniawsky (1989) combined with a series of laboratory testing.

Mr. Peralta agrees that for this stage of study, confidence and the level of information in geotechnical testing is enough. However, with new information a detailed geotechnical model should be updated.

Table 14-21: Optimization Parameters

GEOTS	Overall Slope Angle	Comments
1	35.7	OCULTO
2	47.9	OCULTO
3	40.4	OCULTO
4	39.8	OCULTO
5	57.9	OCULTO
6	40	OCULTO
7	36	JAC-rock
8	53	FAN
9	46.1	OCULTO 2.2
10	38.1	COVER en S7

GEOTS	Overall Slope Angle	Comments
20	42	Default = Undef

14.20 Mineral Resource Estimate Sensitivity

The QP also evaluated the optimized open pit constrained Measured & Indicated Mineral Resource estimate for Oculto, JAC, Sombra, Laderas and Fantasma at a range of operational costs. This ranged from +25% and -25%, and an approximate tank leach cut-off grade between 29 g/t AgEq and 49 g/t AgEq. This sensitivity analysis was done for both processes simultaneously. (Table 14-22, Figure 14-23, and Figure 14-24).

Table 14-22: NVB and Tank & Heap Leach Cut-Off Grade Sensitivity of Measured & Indicated Mineral Resources.

Delta	OPEX Tank L.	OPEX Heap L.	Approx Cut Off (g/t AgEq)	Tonnage Oxides (000 t)	Silver Grade (g/t)	Gold Grade (g/t)	Silver Equivalent Grade (g/t)
-25%	24.43	5.25	29	260873	30	0.31	57
-20%	26.06	5.60	31	255619	31	0.31	58
-15%	27.68	5.95	33	248796	31	0.32	59
-10%	29.31	6.30	35	243328	32	0.33	60
-5%	30.94	6.65	37	237066	33	0.33	62
0%	32.57	7.00	39	231981	33	0.34	63
5%	34.20	7.35	41	226704	34	0.35	64
10%	35.83	7.70	43	221328	34	0.35	65
15%	37.46	8.05	45	216416	35	0.36	66
20%	39.08	8.40	47	211272	36	0.37	68
25%	40.71	8.75	49	206939	36	0.37	69

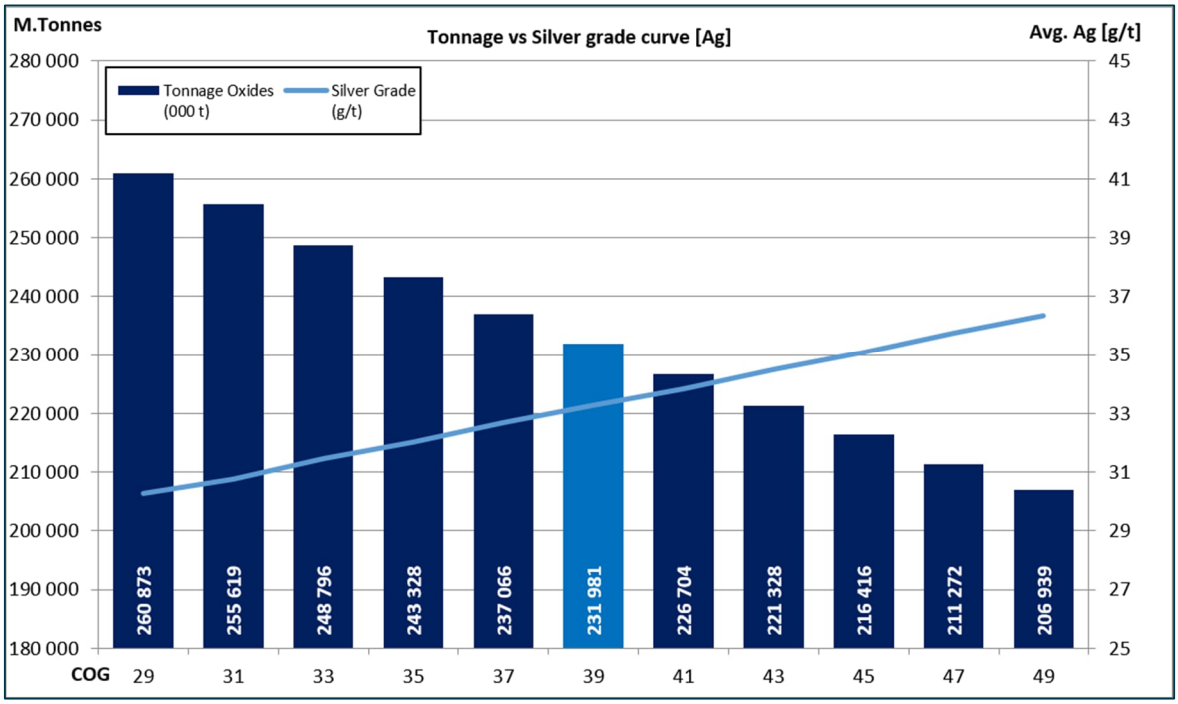


Figure 14-23: Cut-Off Sensitivity Analysis for Measured & Indicated Category, Silver Grade.

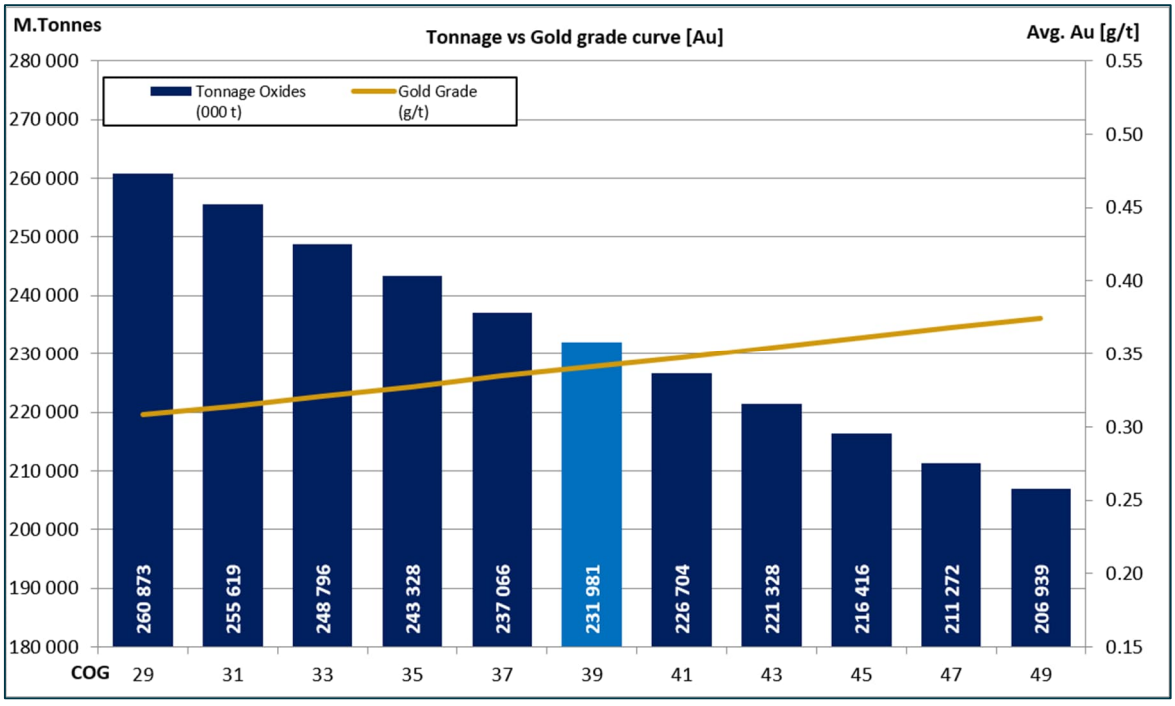


Figure 14-24: Cut-Off Sensitivity Analysis for Measured & Indicated Category, Gold Grade.

The sensitivity tables presented above demonstrate that changes to key parameters such as operating costs have a limited effect on the contained metal in the Mineral Resource. A 20% increase in the NVB operating costs and cut-off grades resulted in a less than 10% loss of contained silver and gold metal.

This demonstrates the robust and high-grade mineralization present at Diablillos. The base case employs an operating cost estimate of USD 32.58/tonne for the primary tank leaching and USD 7.00/tonne for the secondary heap leaching, resulting in a cut-off grade of approximately 39 g/t AgEq and 15 g/t AgEq respectively and it is believed that at all the sensitivities presented a significant Mineral Resource with reasonable prospects for eventual economic extraction exists.

14.21 Comparison with Previous Oculto Mineral Resource Estimate

The current Oculto, JAC, Sombra, Fantasma and Laderas MRE is not directly comparable to the previous Mineral Resource estimate by this author (MRE23) due to:

- New holes in the Oculto NE and JAC zones intercepting additional mineral resources.
- New drillholes corroborating previously estimated mineralization allowing Measured and Indicated classification.
- This Mineral Resource estimate represents an extension of the old Oculto open pit shell to the northeast and a wide extension to the southwest of the JAC zone.
- Different metal price assumptions, in 2025 MRE: silver price at USD 27.00/oz & gold price at USD 2,4000/oz compared to 2026 MRE: silver price at USD 34.50/oz & gold price: USD3,200/oz.

Mr. Peralta compared both Mineral Resources, directly and mathematically. The result of this comparison is detailed in Table 14-23. He also compared Mineral Resources for the tank leaching process alone from the previous Mineral Resource estimate to the current, Table 14-19.

In Mr. Peralta's opinion, the primary cause of the change has been the new drilling that has been determined by:

- The May 2026 Mineral Resource Estimate shows a substantial increase in total mineralized tonnage across the project compared to the July 2025 estimate, reflecting the successful incorporation of additional mineralized material through recent drilling and updated geological modeling.
- At the project level, Measured and Indicated resources increased by approximately 40% in tonnage, while Inferred resources increased by approximately 49%, demonstrating significant resource growth in both categories of confidence.
- Although average silver grades decreased in most deposits due to the inclusion of larger volumes of mineralization, the increase in tonnage more than compensated for this effect, resulting in higher contained metal.
- Contained gold ounces in the Measured and Indicated category increased by approximately 31%, while contained silver-equivalent ounces increased by approximately 19%.

- Oculto remains the dominant contributor to global resources and accounts for the largest absolute increase in contained metal, with Measured and Indicated tonnage increasing by 45% relative to the previous estimate. In addition, an extension to the Oculto NE is driven by recent drillholes with economic grades (moderate to high grades) of gold in the NE zone of the open pit shell.
- Laderas recorded the strongest relative growth, particularly in the Inferred category, where tonnage increased by more than eightfold, reflecting successful delineation of previously unmodeled mineralization and an increase driven primarily to higher metal prices assumptions and a lower cut-off grade.
- JAC continued to demonstrate remarkable grade consistency, maintaining nearly unchanged silver grades while increasing total tonnage, resulting in a meaningful increase in contained metal.
- Fantasma also exhibited significant resource growth, with Measured and Indicated tonnage increasing by 66% while maintaining comparable average grades. The increase in metal price assumptions has upgraded material previously classified as waste into lower grade mineralization, enhancing the size of the Mineral Resource in this area.
- At Sombra, although still at an early stage with 13 drillholes, shows significant potential as mineralization lies beneath only 35 metres of easily mineable unconsolidated colluvial cover.
- Overall, the updated Mineral Resource Estimate indicates that the exploration and resource definition programs have successfully expanded the project's mineral inventory while maintaining economically attractive grades, leading to a significant increase in contained silver-equivalent ounces available for future mine planning and economic evaluation.

Table 14-23: Difference between prior Mineral Resources Estimate and Current Mineral Resource Estimate

Parameter	Category	Tonnes (000 t)	Ag (g/t)	Au (g/t)	Contained Ag (k oz Ag)	Contained Au (k oz Au)	Contained AgEq (k oz Ag)
Current Resource	*Measured & Indicated*	*231981*	*33*	*0.34*	*248053*	*2542*	*454127*
	Inferred	49347	12	0.26	18406	420	50340
Prior Resource	*Measured & Indicated*	*103904*	*59*	*0.51*	*198643*	*1715*	*349927*
	Inferred	19628	21	0.38	13427	241	33496
Variance %	*Measured & Indicated*	*123%*	*-44%*	*-33%*	*25%*	*48%*	*30%*
	Inferred	151%	-45%	-31%	37%	74%	50%

Table 14-24: Difference between Prior Mineral Resources Estimate and Current Mineral Resource Estimate, by the Primary Tank Leaching Process and by Zone.

Deposit		Category	Tonnes (000 t)	Ag (g/t)	Au (g/t)	Contained Ag (k oz Ag)	Contained Au (k oz Ag)	Contained AgEq (k oz Ag)
OCULTO	Current Resource	Measured & Indicated	83059	53	0.74	141252	1969	311872
		Inferred	12145	17	0.62	6705	244	27810
	Prior Resource	Measured & Indicated	57382	67	0.82	123321	1505	260051
		Inferred	8026	27	0.67	6898	173	22663
	Variance %	Measured & Indicated	45%	-21%	-10%	15%	31%	20%
		Inferred	51%	-36%	-7%	-3%	40%	23%
JAC	Current Resource	Measured & Indicated	14792	137	0.07	65286	33	68120
		Inferred	874	121	0.01	3407	0	3432
	Prior Resource	Measured & Indicated	13134	139	0.06	58492	27	60965
		Inferred	1036	77	0.01	2558	0	2602
	Variance %	Measured & Indicated	13%	-1%	7%	12%	20%	12%
		Inferred	-16%	58%	-30%	33%	0%	32%
Fantasma	Current Resource	Measured & Indicated	1738	74	0.01	4149	0	4176
		Inferred	337	75	0.01	814	0	822
	Prior	Measured &	1049	72	0.01	2436	0	2455

Deposit		Category	Tonnes (000 t)	Ag (g/t)	Au (g/t)	Contained Ag (k oz Ag)	Contained Au (k oz Ag)	Contained AgEq (k oz Ag)
	Resource	Indicated						
		Inferred	475	64	0.01	978	0	986
	Variance %	Measured & Indicated	66%	3%	0%	70%	0%	70%
		Inferred	-29%	17%	0%	-17%	0%	-17%
Laderas	Current Resource	Measured & Indicated	1520	15	0.66	726	32	3501
		Inferred	988	14	0.57	435	18	2017
	Prior Resource	Measured & Indicated	806	17	0.67	428	17	2014
		Inferred	104	15	0.68	51	2	259
	Variance %	Measured & Indicated	88%	-10%	-3%	70%	84%	74%
		Inferred	848%	-10%	-16%	756%	699%	680%
Sombra	Current Resource	Measured & Indicated	912	45	0.15	1314	4	1689
		Inferred	56	59	0.00	106	0	106
	Prior Resource	Measured & Indicated	758	54	0.12	1317	3	1594
		Inferred	51	80	0.04	131	0	137
	Variance %	Measured & Indicated	20%	-17%	18%	0%	42%	6%
		Inferred	10%	-27%	-98%	-20%	-98%	-23%
All deposits	Current	Measured &	102021	65	0.62	212728	2039	389359

Deposit		Category	Tonnes (000 t)	Ag (g/t)	Au (g/t)	Contained Ag (k oz Ag)	Contained Au (k oz Ag)	Contained AgEq (k oz Ag)
(Only tank Leaching process)	Resource	Indicated						
		Inferred	14400	25	0.57	11468	262	34187
	Prior Resource	Measured & Indicated	73129	79	0.66	185994	1553	327078
		Inferred	9693	34	0.57	10616	176	26647
	Variance %	Measured & Indicated	40%	-18%	-6%	14%	31%	19%
		Inferred	49%	-27%	0%	8%	49%	28%

Notes to Mineral Comparison Table:

- Key Assumptions in May 2026 MRE:
  - Ag price: $ 34.50/oz & Au price: $3,200/oz.
  - Average recovery rates (tank leach): 80.9% Ag and 86.6% Au.
  - Cut-off grade: based on Net Value per Block, with an average cut-off grade equivalent to ~39 g/t AgEq.
  - Open pit optimization parameters: Mining cost; $2.23/t; Processing cost; $23.84/t; G&A cost $6.51/t.
- Key Assumptions in July 2025 MRE:
  - Ag price: $ 27.50/oz & Au price: $2,400/oz
  - Average recovery rates (tank leach): 82.6% Ag and 86.5% Au
  - Cut-off grade: based on Net Value per Block, with an average cut-off grade equivalent to ~41 g/t AgEq.
  - Open pit optimization parameters: Mining cost; $1.94/t; Processing cost; $22.97/t; G&A cost $3.32/t

For additional details, please refer to “NI 43-101 Technical Report, Mineral Resource Estimate, Diablillos Ag-Au Project” with an effective date of July 29, 2025, and available on www.sedarplus.ca.

14.22 Mineral Resource Risk Assessment

Mr. Peralta has not detected any significant risks that could impact the Mineral Resources in a material way. However, the following minor risks are mentioned:

- Historical drilling does not have logs consistent with the current drilling which generates imprecision in areas with little or no recent drilling.
- It is recommended that work is conducted on copper in the sulphides zone to understand its potential impact on metallurgical recovery and potential to contribute to the economics of the project. Particularly to understand the impact on metallurgical recoveries and potentially recoverable Mineral Resources. Nevertheless, this mineral resource estimate includes only oxidized materials.
- Other elements such as arsenic, bismuth, and antimony are present. Their impact should continue to be reviewed in future metallurgical studies.

The price of metals and variation in production costs is considered a risk inherent to any mining project.

15. Mineral Reserve Estimates

The following section has been taken in its entirety from the previous NI 43.101 Technical Report Updated Pre-Feasibility Study, effective December 3rd, 2024, and report date January 17th, 2025.

This section provides details of the Mineral Reserve estimation methodology for the Diablillos Property. Dated January 2024, based on the updated Mineral Resources Estimate with effective date November 22nd, 2023. Adjustments were made to account for updated metallurgical test work, a new geo-metallurgical model, geotechnical evaluation, mining and processing operating costs.

Mr. Fuentealba, on behalf of Bmining, was contracted by AbraSilver to conduct the mine engineering and Mineral Reserve estimation for the Diablillos Project. Mr Fuentealba adopted standard mine planning processes to perform a Mineral Reserve estimate for this surface mining project. The following describes the processes, inputs, and the Mineral Reserve estimate.

Mineral Resources considered the three categories of Measured, Indicated, and Inferred. The Mineral Reserve estimate has reported only Measured and Indicated Mineral Resources given sufficient geological confidence in line with CIM, 2014 definitions. Measured Mineral Resources can be converted into Proven Mineral Reserves, while Indicated Resources can be converted into Probable Reserves.

15.1 Introduction

The Diablillos project is at PFS stage based on conventional open pit, truck and loader operation feeding a silver/gold concentrator. The Mineral Reserve is supported by the PFS Study, and the Project is progressing to Feasibility Stage. To support the Mineral Reserve evaluation within the PFS a Whittle open pit optimisation evaluation was completed with no value given to the inferred classified material within the mineral resource for all deposits. Following this a PFS level mine design, mine scheduling, mining costing and over economic evaluation was completed to confirm positive economic outcomes from the Mineral Reserve.

This report is based on information gathered by the QP during a site visit conducted from October 19 – 21, 2024. The following information was noted:

- Discussions with Abra Silver personnel.
- Inspection of the Diablillos Project area including drill core sheds, current roads and future infrastructure locations.
- Weekly meetings with the Whittle Consulting team to modify the prior mining sequence.
- Weekly meetings with the Mining Plus team to update mine OPEX and CAPEX.

15.2 Mine Design Input Parameters

This section remains unchanged in comparison with the previously developed pre-feasibility study dated May 29th, 2024. It describes the pit optimization and mine design input parameters, considering that ore is an estimate of the tonnage mined and processed from stockpiles.

15.2.1 Metal Prices

Metal sales prices used for the pit optimization stage correspond to US$1,750/oz for gold and US$22.50 for silver. It should however be noted that price forecasting has elements of uncertainty due to global economic fluctuations. Historical prices can be found below in Figure 15-1 and Figure 15-2.



Source: Goldprice.com

Figure 15-1: Gold Price



Source: Goldprice.com

Figure 15-2: Silver Price

Sales prices have the greatest impact on the estimation process. Income from gold sales corresponds to 42% and income from silver sales accounts for the remaining 58%.

15.2.2 Mineral Resource Model

A resource block model in *.csv format was received from QP Rodrigo Peralta, called "DBL23-BM-.csv". It is a regular, non-rotated model with the following characteristics in Table 15-1:

Table 15-1: Block Model Dimensions

Item	Unit	X	Y	Z
Block Model Origin	m	718,710	7,197,840	3,800
Block Model Maximum	m	722,020	7,201,010	4,750
Parent Block Size	m	10	10	10

The model contains relevant information such as densities, lithology, alteration, grades, metallurgical recovery, and other fields necessary to estimate Mineral Resources (Table 15-2).

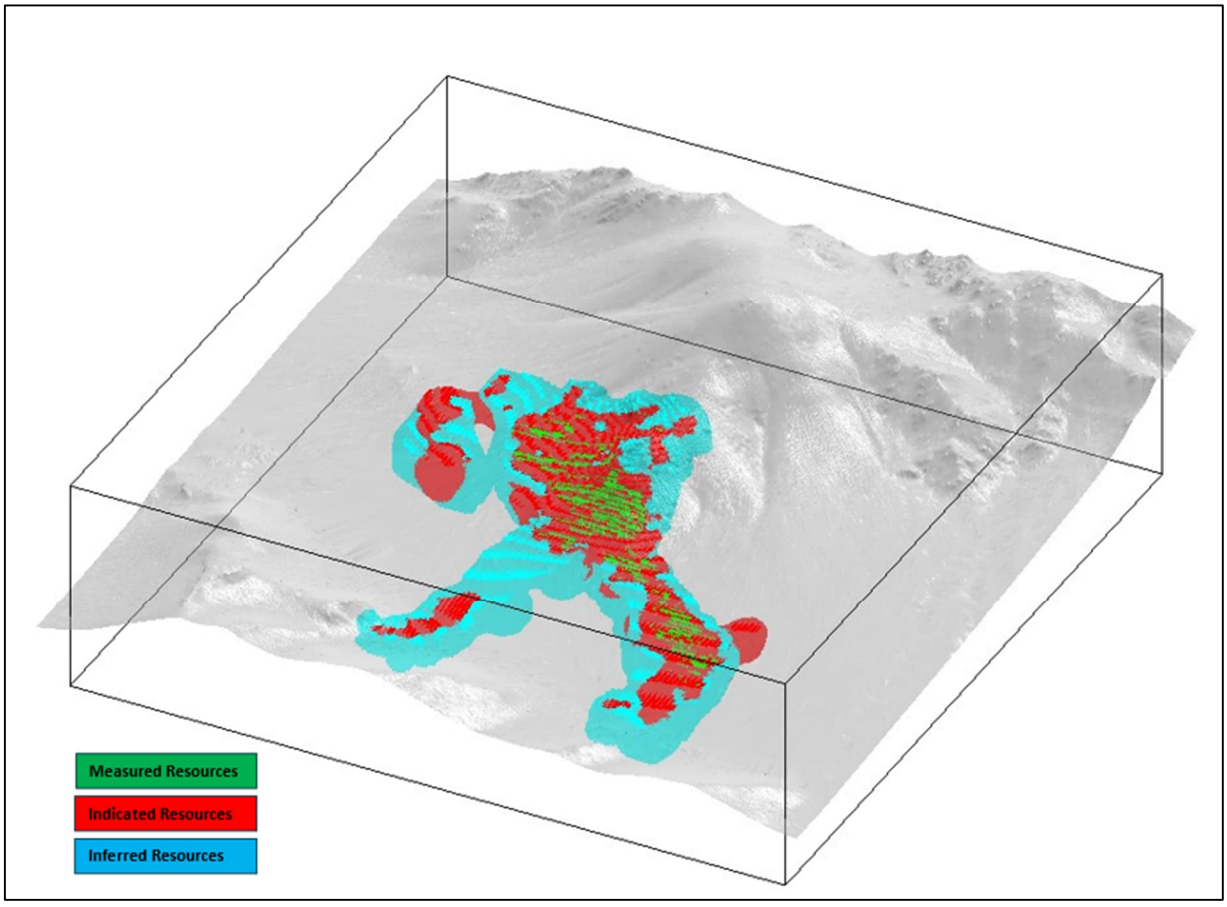
Table 15-2: Block Model Fields

Variable	Description	Min	Max	Precision
XC	X coordinate			
YC	Y coordinate			
ZC	Z coordinate			
AU	Final gold in g/t	0	999	0.001
AUOK	Gold estimate with OK, capping applied	0	999	0.001
AUID	Gold estimate with ID, capping applied	0	999	0.001
AG	Final silver in g/t	0	99999	0.01
AGOK	Silver estimate with OK, capping applied	0	99999	0.01
AGID	Silver estimate with ID, capping applied	0	99999	0.01
AGEQ	Silver Equivalent (AU*70+AG)	0	99999	0.01
AUEQ	Gold Equivalent (AG/70+AU)	0	999	0.001
SG	Bulk density	0	50	0.01
CUOK	Copper estimate with OK, no capping applied			
CLASS	Category: 1=Measured, 2=Indicated and 3=Inferred 4=Uncathegorized	0	5	1
P22OX	0=Outside Oxide Pit Shell, 1=Inside Oxide Pitshell 2=Transition 3=Sulphide	0	5	1
P22OS	0=Oxide, 1=Sulphide	0	10	1
LITW	Lithological Code 1=Cover 2=Volcanic 3=Metasedi 4=Granitoides	0	10	1
ALTW	Alteration Code: 1=Argillic 2=Silica 3=Vuggy	0	10	1
OX_ST	Oxidation State: 1=Oxide 2=Transition 3=sulphides	0	10	1
NSAM	Number of samples used for estimation	0	25	1
ORE	ORE: 0=waste 1=ore; if NVBR1>0	0	10	1
WASTY	Waste type: 1=A 2=B 3=C	0	10	1
RQD	Rock Quality	0	100	1
RMR	Rock Mass rating	0	100	1
SMR	Slope Mass Rating	0	100	1
GEOT	Final Wall: -30:20 Default value set to 10	0	100	1
GEOTS	Geotech Sector: S1=1 S2=2 S3=3 S4=4 S5=5 S6=6	0	100	1
LAW	Combination of Lith_Code +Alt_Code	0	100	1
LA2W	Combination of Lith_Code +Alt_Code + Oxidation State	0	1000	1
KVAU	Block variance (AUOKC)	0	25	0.01
KVAG	Block variance (AGOKC)	0	25	0.01
TOPO	Topo			
RCOD	Recovery domain Shallow=1Silver=2 Deep=3 NE=4 JAC=5 Fan=6	0	10	1
DEST	Destiny Waste=1 Tank Leach Process Plant=2 Heap Leach=3	0	10	1
NVBR1	Net Value per Block for Resources	0	100000	0.01
INC	Income	0	99999	0.01

Variable	Description	Min	Max	Precision
COST	Mining Cost+Processing Cost+G&A+ Etc.	0	99999	0.01
AUREC	Gold metal recovery, based on Geomet domain.	0	99999	0.01
AGREC	Silver metal recovery, based on Geomet domain.	0	99999	0.01

An inspection of the block model was conducted by plotting sections with the estimated Mineral Resource categories, grades and drill holes in the vicinity. According to this general review, the model is suitable for use. An isometric view of the block model with the Mineral Resource categories is shown in Figure 15-3. Green cells represent Measured Mineral Resources, red cells represent Indicated Mineral Resources, and light blue cells represent Inferred Mineral Resources.

The density variable corresponds to the SG field shown in Table 15-2 and ranges from 0 to 2.7 depending on the type of rock. In the case of air block density is 0.

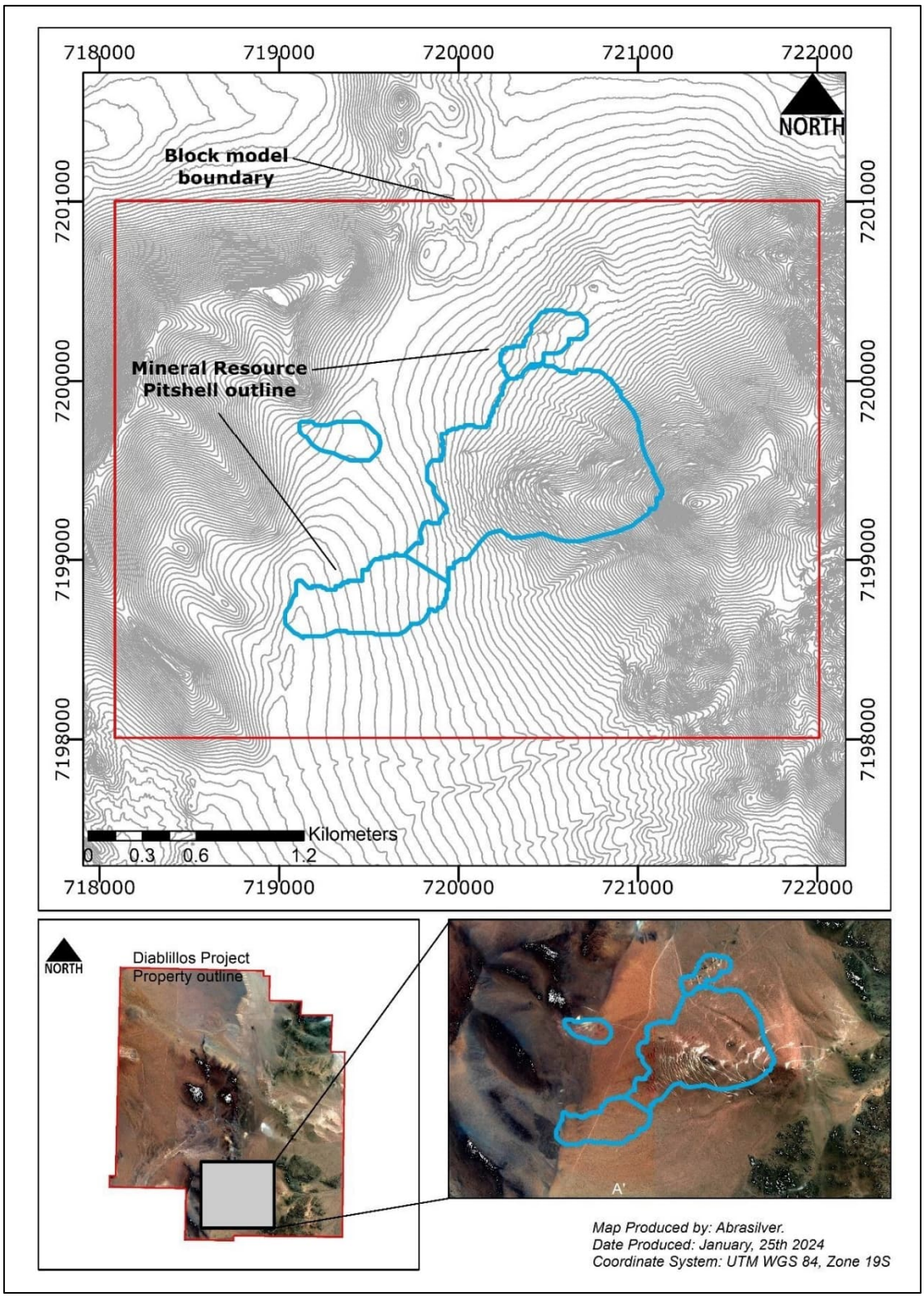


Source: AbraSilver Resource Corp., 2023

Figure 15-3: Overview of the Block Model

15.2.3 Topography

The initial topography of the project remains unchanged with regard to the previous pre-feasibility study. There was no material movement during the current year. It is the file “DTM-5m.msr” dated 09/13/2023. The topography is shown in Figure 15-4.



Note: The red rectangle delimits the boundaries of the block model

Source: AbraSilver Resource Corp., 2024

Figure 15-4: Starting Topography of the Project

15.2.4 Geotechnical Pit Slope Assessment and Design Guidance

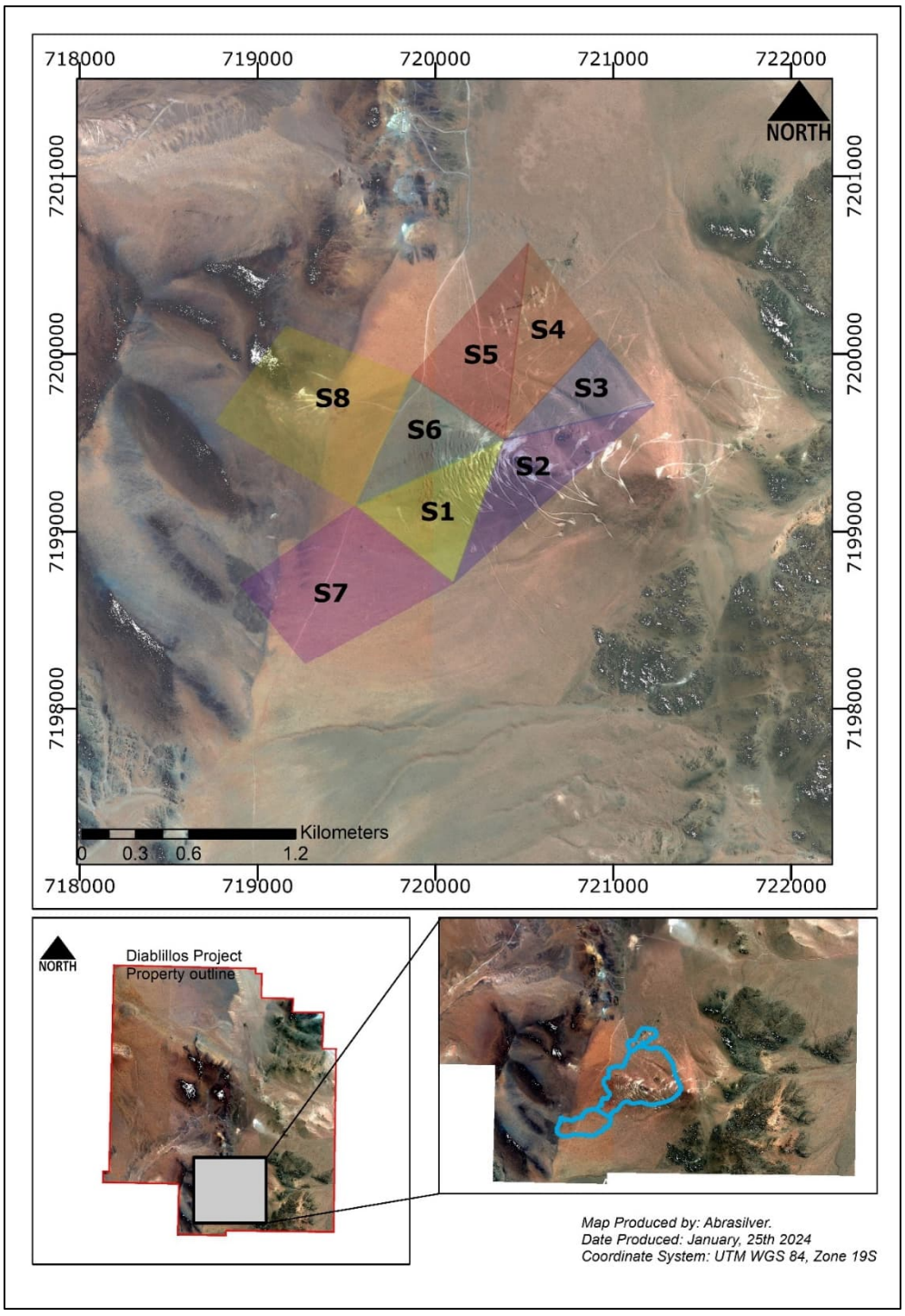
Geotechnical information was received from AbraSilver which includes geotechnical rock testing as well as geotechnical logs gathered simultaneously with geological core logging. This information was used to compile a recommendation of overall slope angles to be used in the open pit optimization process. The working area has been divided into 8 geotechnical domains by AbraSilver. Each of these areas incorporates different structural, lithological and alteration characteristics.

Figure 15-5 details these geotechnical domains. It should be noted that blocks lacking geotechnical zone coding were coded as code 11. In addition, an overall angle of 53° is assigned to this geotechnical zone as the hard rock assumption for the Mineral Resource open pit optimization. More conservative angles were in cover material. The overall angles for each geotechnical zone are presented in Table 15-3.

Table 15-3: Overall Slope Angle by Geotechnical Domain

Geotechnical Zone	Overall Angle (°)
1	60
2	49
3	54
4	53
5	53
6	53
7	60
8	53
10	37
11 (No data)	53

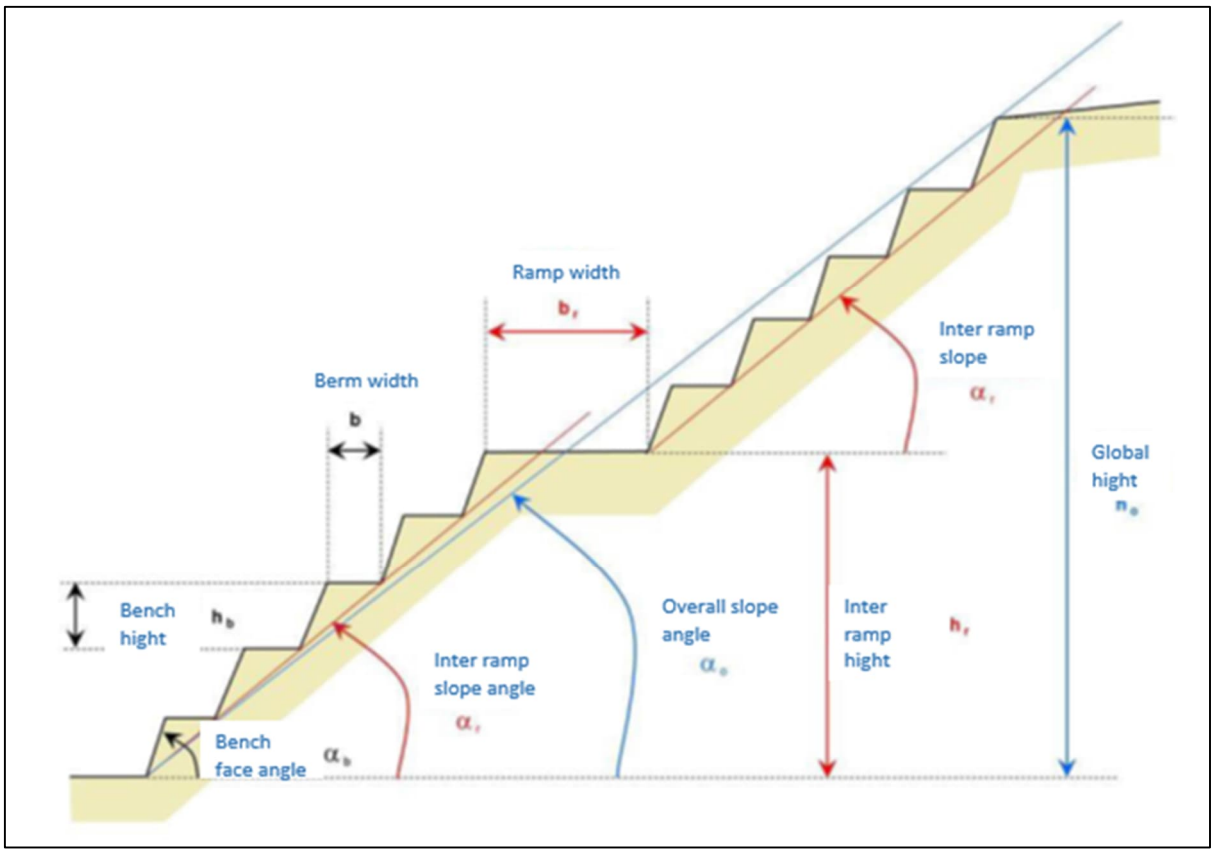
The open pit shell generated during the Mineral Resource estimate was reviewed in conjunction with Mr. Peralta and AbraSilver personnel. It was observed that certain geotechnical domains present a height greater than 350 meters (domains S2 and S3). As per the recommendation of Mr. Fuentealba and a geotechnical expert, it was recommended that at least one ramp passes through each pit wall.



Source: AbraSilver Resource Corp., 2024

Figure 15-5: Geotechnical Domains

The general parameters used for the open pit design are shown schematically in Figure 15-6. Table 15-4 shows the detail design parameters.



Source: AbraSilver Resource Corp., 2024

Figure 15-6: Schematic Open Pit Design Parameters

Table 15-4: Open Pit Design Parameters

Geotechnical Zone	Bench height (m)	Bench face angle (°)	Ramp width (m)	Slope ramp (%)	Geotechnical berm (m)	Berm width (m)	Inter-ramp slope angle (°)	Geotechnical height (m)
S1	10	80	22	10%	12	4.5	57.6	120
S2	10	60	22	10%	13	5	45.0	120
S3	10	75	22	10%	12	4.5	54.5	120
S4	10	75	22	10%	12	4.5	53.4	120
S5	10	75	22	10%	12	4.5	53.4	120
S6	10	75	22	10%	12	4.5	53.4	120
S7	10	75	22	10%	12	4.5	57.8	120
S8	10	75	22	10%	12	4.5	53.4	120
10	10	75	22	10%	12	11.76	34.7	120
11 (No information)	10	80	22	10%	12	4.5	53.4	120

According to the stability analysis described in Section 16.9.2, the overall slope angles could be higher in some areas. More geotechnical tests are recommended to validate these assumptions.

Some phases were designed to the final pit wall, in cover material. It will be necessary to validate this with assays from a geotechnical campaign.

15.2.5 Mineral Processing Methods

The background of the process was obtained from the document “DSP-PFS-GEN-0000-DC-009-B - Criterio de Diseño – Proceso.docx”. All metallurgical tests indicate that gold and silver can utilise the Merrill-Crowe zinc precipitation process. The process begins with the size reduction of the ore using a primary jaw crusher and a semi-autogenous grinding (SAG) mill. A pebble crusher and a ball mill circuit to reduce the ore size 80 percent down to size I (P80 of 150 microns).

The treatment capacity will be 9,000 metric tonnes per day. The design itself however could allow for a significant increase in treatment capacity. This however would require a second process, such as heap leaching or an increase of current throughput (currently limited by water availability).

Ore will be leached with cyanide in a series of eight hammer tanks. The rich solution will be recovered through a counter current decant (CCD) circuit and then processed through the Merrill-Crowe process.

The average recoveries over life of mine are 83.6% for silver and 86.8% for gold.

15.2.6 Operating Cost Used for Block Value

The economic parameters to define the final operating limit were provided by AbraSilver. They are presented in the Table 15-5 below:

Table 15-5: Economics Parameters for Optimization

Variable	Value	Unit
Mine Cost		
Base	1.94	USD/t
Cover*	1.73	USD/t
MCAF	MB Variable	-
Plant Cost		
Process	22.97	USD/t min
G&A	3.32	USD/t min
Metals Price		
Au	1,750	USD/toz
Ag	22.5	USD/toz
Other Parameters		

Variable	Value	Unit
Au Recovery	MB Variable	%
Ag Recovery	MB Variable	%
Royalty	3%*Revenues	USD

* Note: The "cover" mine cost refers to a type of material characterized in the block model. This will not require drilling and blasting operations and therefore, has a lower mining cost.

To capture the effect of increasing mine cost with depth, an adjustment factor (MCAF) was applied as a function of elevation.

The factors to be used in the MCAF calculation equation are based on a base elevation of 4,280 masl (if above or below this elevation). The calculation is shown in Table 15-6:

Table 15-6: Incremental Mine Cost

Elevation	Mine Cost
>4.280	Mine Cost*(((Elevation – 4,280) /10) *0.008+1)
4,280	Mine Cost
<4,280	Mine Cost*(((4,280 - Elevation) /10) *0.005+1)

15.2.7 Mining Dilution

Diablillos Project is a disseminated orebody with relatively clear ore/waste contacts. In consideration of blast movement and ore boundary, the shape of the deposit and the structurally constrained grade distribution for each lithology and sector (Oculto, JAC, Fantasma, Laderas) large dilutions are not expected in the opinion of Mr. Fuentealba; however, a dilution factor has been applied using a methodology in conjunction with Mr. Peralta considering 7.5% dilution for each face ore block in contact with face waste block.

For this PFS, the following methodology has been developed to estimate the dilution (Figure 15-7).

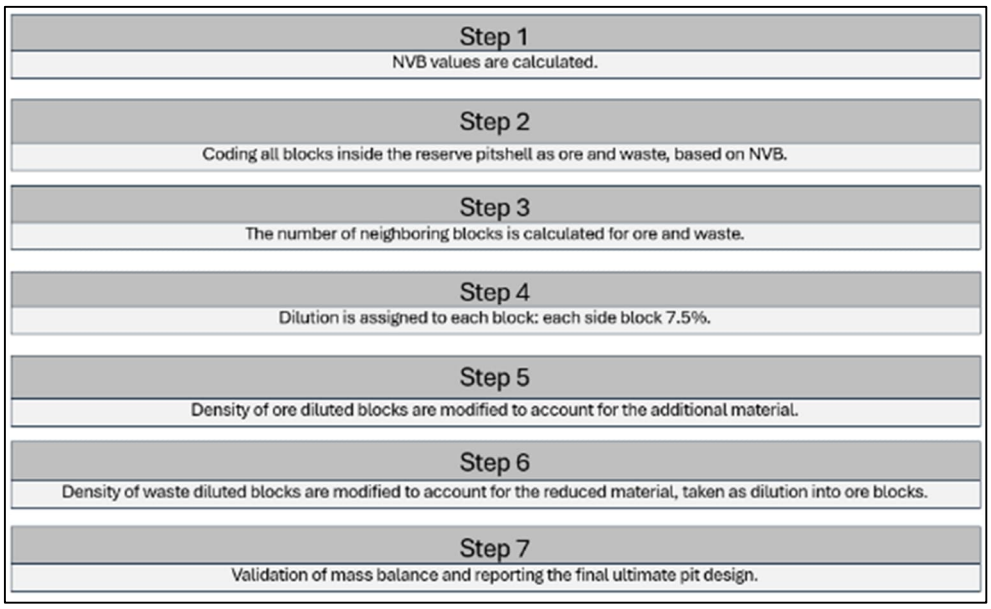


Figure 15-7: Dilution Methodology

15.2.8 Ore Loss

This engineering stage has considered full mining recovery of 100%. Future work should consider the application of losses due to operational deviations and human error.

15.2.9 Gold and Silver Recovery

The recovery of Gold and Silver for the current stage of the project is defined by the "recovery domain" (RCOD) variable of the block model.

The following are the recovery values for each domain:

Table 15-7: Gold (Au) and Silver (Ag) Recovery

RCOD	Variable	Value	Unit
1 (Shallow gold domain)	Au Recovery	88.25	%
	Ag Recovery	56.77	%
2 (Silver Enrichment domain)	Au Recovery	86.17	%
	Ag Recovery	85.62	%
3 (Deep Gold domain)	Au Recovery	83.73	%
	Ag Recovery	82.59	%
4 (Northeast domain)	Au Recovery	89.30	%
	Ag Recovery	81.69	%
5 (JAC domain)	Au Recovery	84.29	%
	Ag Recovery	88.18	%

RCOD	Variable	Value	Unit
6 (Fantasma domain)	Au Recovery	84.29	%
	Ag Recovery	88.18	%

In addition, a recovery value of 0% Gold and Silver is considered for all blocks that have no RCOD set.

15.2.10 Block Valuation and Cut-off Grade

Prior to determining the variable that will define the blocks that can be processed and those that cannot, each block must be valued to differentiate blocks that provide a positive profit.

The valuation of each block is explained below:

Revenue:

$$Revenues = \frac{(P_{Au} - CV_{Au}) * G_{Au} * R_{Au} + (P_{Ag} - CV_{Ag}) * G_{Ag} * R_{Ag}}{31.1035}$$

Where:

- PAu: Price of Gold (US$/toz)
- CVAu: Gold Cost of Sales (US$/toz)
- GAu: Gold Grade (g/t)
- RAu: Gold Recovery (Decimal)
- PAg: Silver Price (US$/toz)
- CVAg: Silver Cost of Silver (US$/toz)
- GAg: Silver Grade (g/t)
- RAg: Silver Recovery (Decimal)

Cost:

$$Cost = CM + CP + G\&A + Royalty * Revenue$$

Where:

- MC: Min Cost (US$/t)
- PC: Plant Cost (US$/t min)
- G&A: Administrative Cost (US$/t min)
- Royalty: 3%

Profit:

$$Profit = Revenue - Cost$$

Profit Considerations:

Negative block profit is defined as overburden, in the following cases described in Table 15-8:

Table 15-8: Blocks Classified and Costed as Waste Rock

Case	Profit
Block with Inferred category or no data	Negative mining cost
Blocks with no data	Negative mining cost
Block classified as sulphide	Negative mining cost
Block coded as "Cover" type	Negative mining cost

A reference cut-off grade of 53 gr/t AgEq can be determined as the minimum grade to send for processing.

General Cut-Off Grade Formula:

$$Cut-off\ Grade = \frac{Operating\ Cost\ per\ Ton\ of\ Ore\ (\frac{USD}{ton})}{Metal\ Price\ per\ Unit\ of\ Metal\ x\ Metallurgical\ Recovery\ (\%)}$$

Ag	g/t	28.2
Au	g/t	0.32
AgEQ	g/t	53
Silver Price	USD/oz	22.5
Gold Price	USD/oz	1,750
Silver Selling	USD/oz	15
Gold Selling	USD/oz	0.45
Silver Recovery	%	75.00%
Gold Recovery	%	80.00%
Net Revenue	USD/t ore	29.27
Net Revenue	USD/g	0.55
Mining Cost	USD/t ore	1.94
Processing Cost	USD/t ore	22.97
Royalti	%	3%
G&A Cost	USD/t ore	3.35
Operating Cost	USD/t ore	29.13
CoG Marginal AgEq	g/t	53

In addition, for reporting purposes, a marginal cut-off grade of 15 gr/t AgEq was used to identify and stockpile material that could be treated in the future.

15.3 Open Pit Optimization

The process of defining the final extraction limit is carried out using Whittle Four X software, and the Lerchs & Grossman algorithm. This software generates a set of three-dimensional envelopes (nested pits) using Revenue Factors (RF), associated to increasing prices of the elements of interest (Silver and Gold).

The ultimate pit was calculated by considering the category, oxidation state, recovery domain, lithology, and profit. This is presented in Table 15-9.

Table 15-9: Optimization Variable Definition (OREW)

OREW	Category	Oxidation State	RCOD	Lithology	Block Profit
1	Meas + Ind	Oxide	1 – 6	All except cover	>0
0	Block that fails to meet at least 1 of the above conditions to be sent for processing.				

The results of the Resource Optimization are presented in Table 15-10.

Table 15-10: Pit Optimization Results

Pit	Minimum RF	Maximum RF	Total Rock (000 t)	Ore (000 t)	Au (g/t)	Ag (g/t)	AgEq (g/t)
1	0.11	0.12	2,387	26	0.16	3343	3356
2	0.13	0.13	2,389	28	0.15	3116	3128
3	0.14	0.14	12,960	456	1.43	978	1100
4	0.15	0.15	22,984	1,039	1.15	772	866
5	0.16	0.16	27,993	1,469	1.32	660	767
6	0.17	0.17	36,610	2,149	1.52	577	699
7	0.18	0.18	46,674	2,857	1.52	535	659
8	0.19	0.19	47,046	3,061	1.48	512	634
9	0.20	0.20	48,688	3,383	1.48	483	604
10	0.21	0.21	49,223	3,583	1.46	466	586
11	0.22	0.22	49,731	3,756	1.44	453	571
12	0.23	0.23	52,425	4,050	1.38	441	555
13	0.24	0.24	63,588	5,173	1.46	390	509
14	0.25	0.25	64,788	5,464	1.44	377	495
15	0.26	0.26	65,208	5,696	1.41	367	483
16	0.27	0.27	68,277	6,126	1.36	356	468
17	0.28	0.28	71,478	6,498	1.30	349	456
18	0.29	0.29	73,864	6,951	1.28	336	441
19	0.30	0.30	76,114	7,206	1.25	331	434
20	0.31	0.31	78,148	7,610	1.24	320	422
21	0.32	0.32	79,306	7,943	1.21	312	412

Pit	Minimum RF	Maximum RF	Total Rock (000 t)	Ore (000 t)	Au (g/t)	Ag (g/t)	AgEq (g/t)
22	0.33	0.33	79,953	8,177	1.19	307	405
23	0.34	0.34	83,979	8,682	1.19	296	394
24	0.35	0.35	88,662	9,257	1.16	288	383
25	0.36	0.36	90,042	9,674	1.13	280	373
26	0.37	0.37	94,059	10,262	1.12	270	364
27	0.38	0.38	100,280	10,964	1.13	260	353
28	0.39	0.39	101,159	11,315	1.11	255	347
29	0.40	0.40	101,904	11,619	1.09	251	341
30	0.41	0.41	102,205	11,917	1.08	247	336
31	0.42	0.42	119,232	13,677	1.13	224	317
32	0.43	0.43	119,822	14,010	1.12	220	312
33	0.44	0.44	157,586	17,853	1.24	181	285
34	0.45	0.45	159,340	18,388	1.23	178	280
35	0.46	0.46	165,637	19,398	1.19	174	273
36	0.47	0.47	167,070	19,997	1.17	171	268
37	0.48	0.48	168,560	20,582	1.16	168	264
38	0.49	0.49	169,626	21,138	1.14	164	260
39	0.50	0.50	180,936	22,602	1.14	157	252
40	0.51	0.51	181,193	23,057	1.13	155	249
41	0.52	0.52	182,491	23,572	1.12	153	246
42	0.53	0.53	183,687	24,152	1.11	150	243
43	0.54	0.54	183,957	24,662	1.09	148	239
44	0.55	0.55	187,141	25,500	1.08	145	235
45	0.56	0.56	188,063	26,014	1.07	143	232
46	0.57	0.57	195,979	27,029	1.07	139	228
47	0.58	0.58	197,928	27,709	1.06	137	225
48	0.59	0.59	199,215	28,357	1.05	134	222
49	0.60	0.60	199,294	28,772	1.04	133	220
50	0.61	0.61	202,986	29,470	1.03	131	217
51	0.62	0.62	206,594	30,311	1.02	129	214
52	0.63	0.63	206,827	30,783	1.01	128	212
53	0.64	0.64	207,503	31,311	1.00	126	210
54	0.65	0.65	208,374	31,822	0.99	125	208
55	0.66	0.66	209,074	32,305	0.98	124	206
56	0.67	0.67	209,467	32,763	0.97	123	204
57	0.68	0.68	210,409	33,235	0.97	121	202
58	0.69	0.69	211,503	33,756	0.96	120	200
59	0.70	0.70	212,197	34,277	0.95	119	198

Pit	Minimum RF	Maximum RF	Total Rock (000 t)	Ore (000 t)	Au (g/t)	Ag (g/t)	AgEq (g/t)
60	0.71	0.71	212,620	34,661	0.94	118	197
61	0.72	0.72	214,673	35,290	0.94	116	195
62	0.73	0.73	215,090	35,751	0.93	115	193
63	0.74	0.74	215,306	36,201	0.92	114	191
64	0.75	0.75	215,490	36,606	0.92	113	190
65	0.76	0.76	238,262	38,923	0.92	108	185
66	0.77	0.77	238,693	39,374	0.91	107	184
67	0.78	0.78	239,790	39,874	0.91	106	182
68	0.79	0.79	240,632	40,390	0.90	105	181
69	0.80	0.80	250,363	41,357	0.90	104	179
70	0.81	0.81	250,845	41,826	0.90	103	178
71	0.82	0.82	251,215	42,249	0.89	102	176
72	0.83	0.83	251,358	42,684	0.88	101	175
73	0.84	0.84	251,649	43,140	0.88	100	174
74	0.85	0.85	252,130	43,572	0.87	100	173
75	0.86	0.86	252,360	43,983	0.87	99	172
76	0.87	0.87	253,637	44,513	0.86	98	170
77	0.88	0.88	255,995	45,103	0.86	97	169
78	0.89	0.89	260,863	45,905	0.85	96	167
79	0.90	0.90	260,949	46,306	0.85	96	166
80	0.91	0.91	261,388	46,769	0.84	95	165
81	0.92	0.92	261,562	47,181	0.84	94	164
82	0.93	0.93	261,833	47,587	0.83	94	163
83	0.94	0.94	263,854	47,922	0.83	93	163
84	0.95	0.95	264,829	48,017	0.83	93	163
85	0.96	0.96	264,966	48,071	0.83	93	162
86	0.97	0.97	267,703	48,217	0.83	93	162
87	0.98	0.98	268,381	48,289	0.83	93	162
88	0.99	0.99	268,993	48,383	0.83	93	162
89	1.00	1.00	269,368	48,450	0.83	93	162

*Note: Tonnages and grades reported are cumulative and do not represent the incremental contribution of each pit.

The pit-by-pit graph for each nested pit and its corresponding cash flow is shown in Figure 15-8.

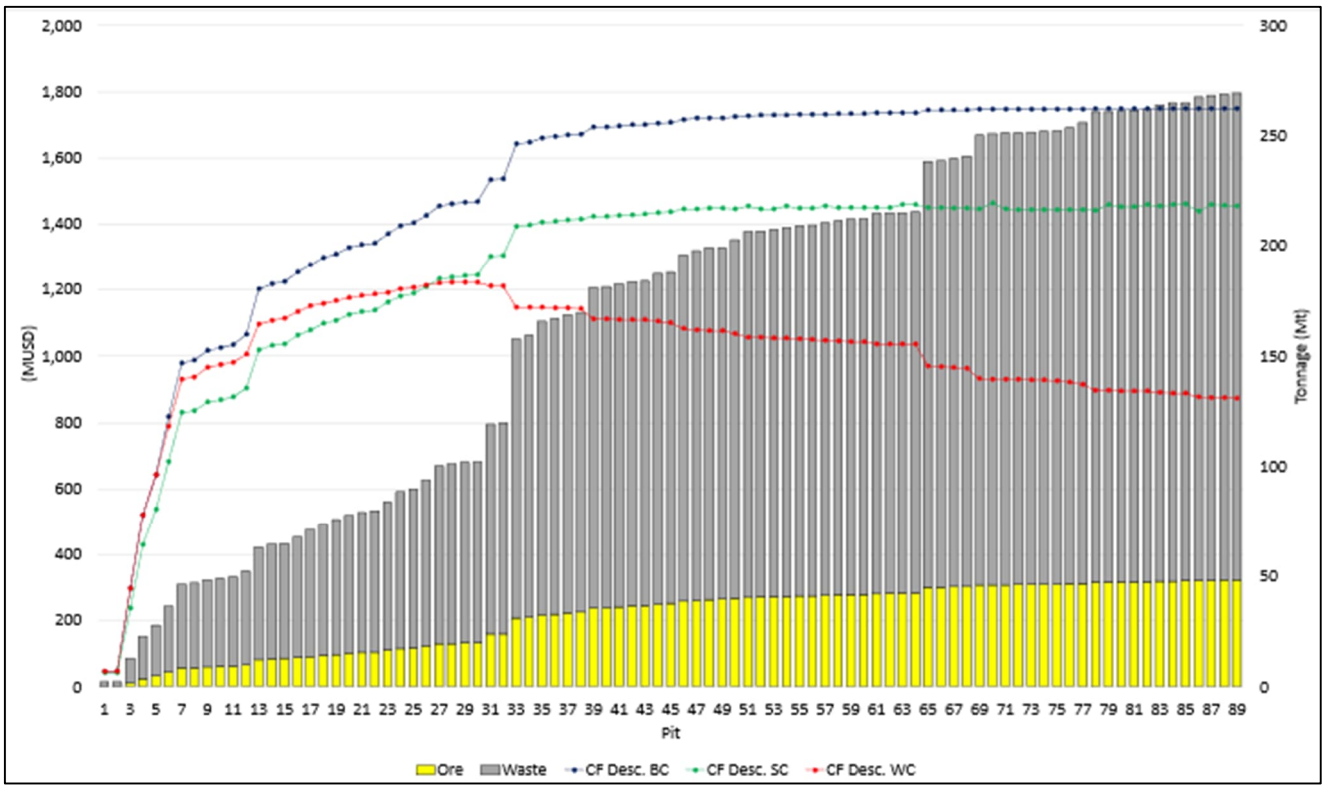


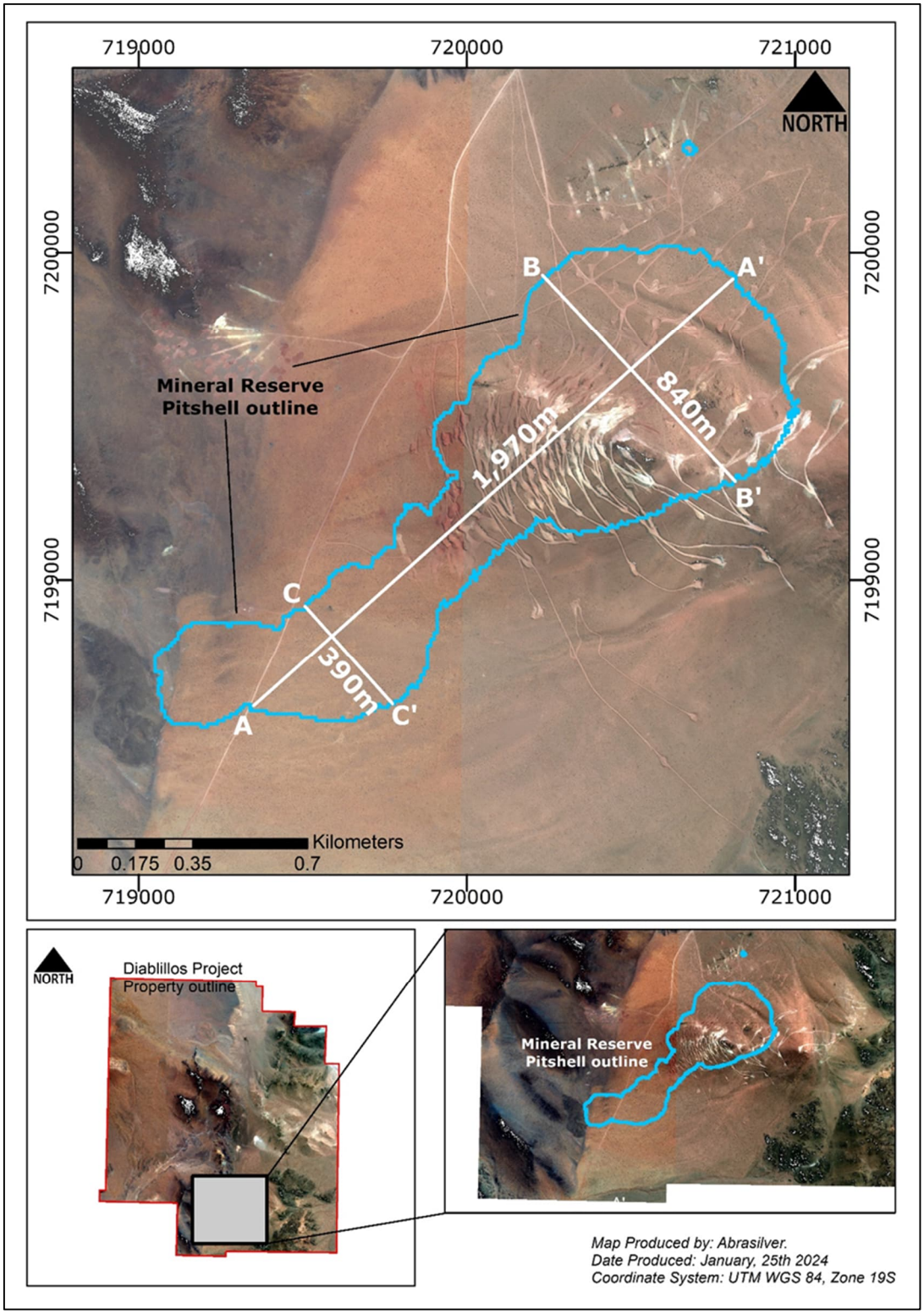
Figure 15-8: Pit by Pit Optimization

Measured and Indicated Mineral Resources within the economic pitshell (using Revenue Factor 1) amount to 48.5 Mt grading 0.83 g/t Au and 93.2 g/t Ag, while contained ounces are 3.1 M Ozt AuEq and 253 M Ozt AgEq as shown below in Table 15-11.

Table 15-11: Measured and Indicated Resources in Optimal Pitshell

Resources	Tonnage (000 t)	Au (g/t)	Ag (g/t)	AgEq (000 Oz)
Measured	13,557	0.87	119	83,837
Indicated	34,894	0.81	82	169,086
Measured + Indicated	48,450	0.83	93	252,924

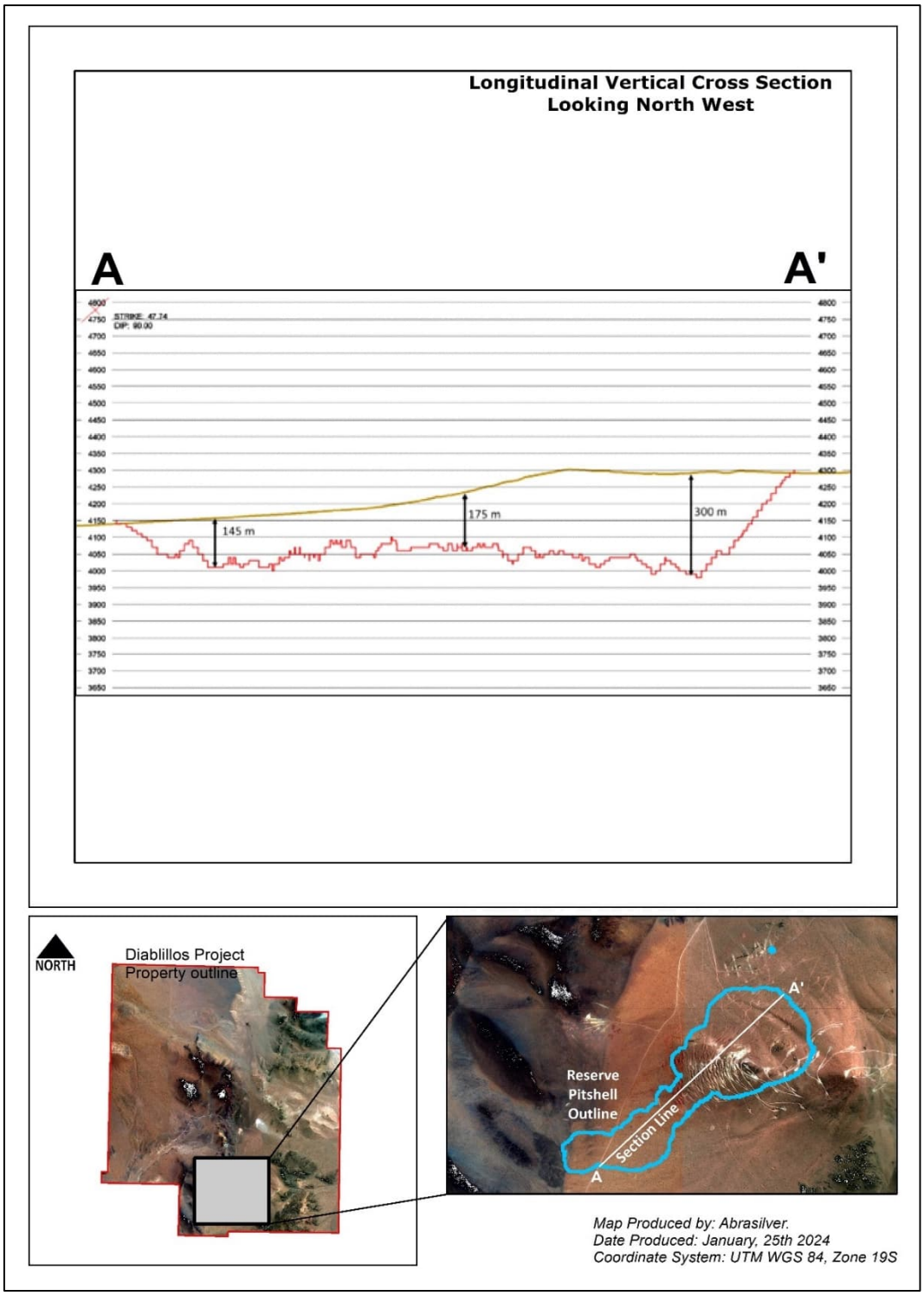
As can be seen in cyan line in Figure 15-9, only material from the Oculto, Ladera, and JAC sectors lies within the economic envelope.



Source: AbraSilver Resource Corp., 2024

Figure 15-9: Solid Used for Cubic Capacity Estimates of Meas + Ind Resources

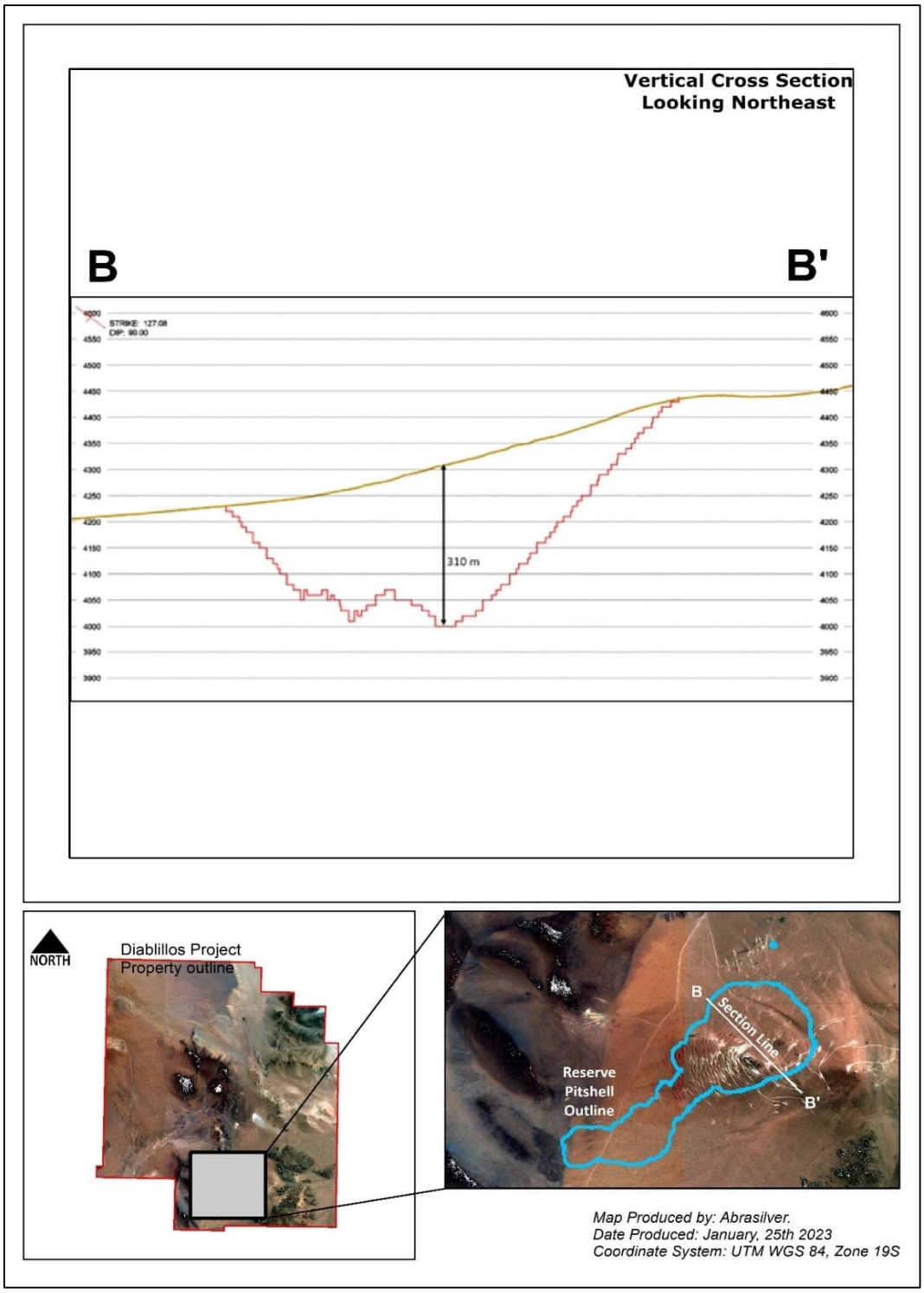
Section AA' in Figure 15-10 shows that the ultimate pit is deeper towards A'.



Source: AbraSilver Resource Corp., 2024

Figure 15-10: Final Pit Section AA'

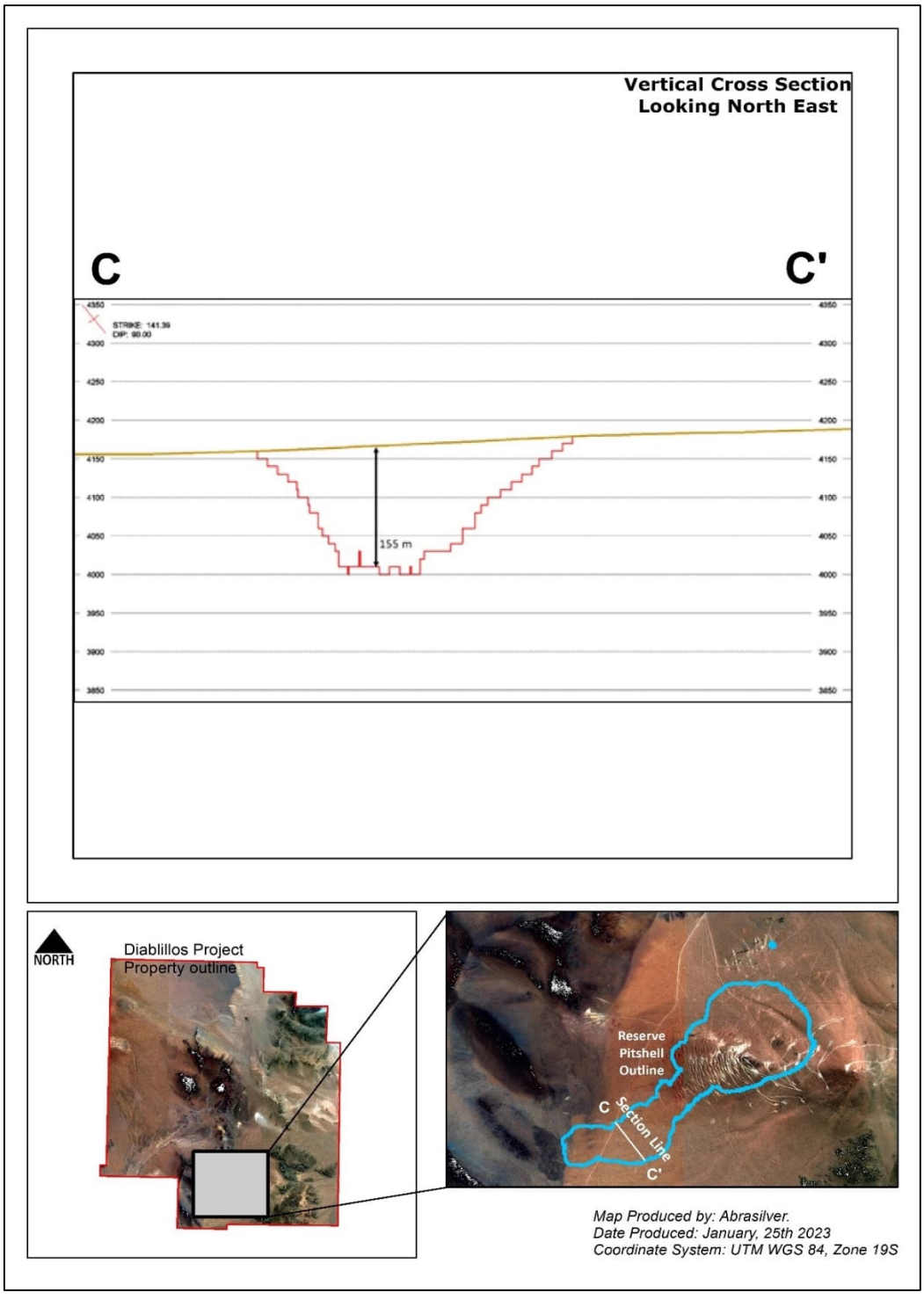
Section BB' in Figure 15-11 shows that the ultimate pit is deeper towards B'.



Source: AbraSilver Resource Corp., 2024

Figure 15-11: Final Pit Section BB'

Section CC' in Figure 15-12 shows that the ultimate pit has constant depth.



Source: AbraSilver Resource Corp., 2024

Figure 15-12: Final Pit Section CC'

15.4 Mineral Reserve Pit Design

15.4.1 Parameters Relevant to Mine Design

15.4.1.1 Selective Mining Unit Size

During the 2020 PEA, Mining Plus conducted a heterogeneity study on drill core data to assess the impact of block size/mining scale on dilution. The conclusion of the study was that the Diablillos Project was relatively insensitive to mining scale, with at most 6% waste in ore at a bench height of 15 meters. Smaller benches reduced dilution, but only by two to four percent. Aligning with the productivity necessary for a medium to large operation like Diablillos.

This study was not repeated at the PFS stage, but the results were reviewed. A bench height of 10 m has been selected.

15.4.1.2 Geotechnical Pit Wall Design

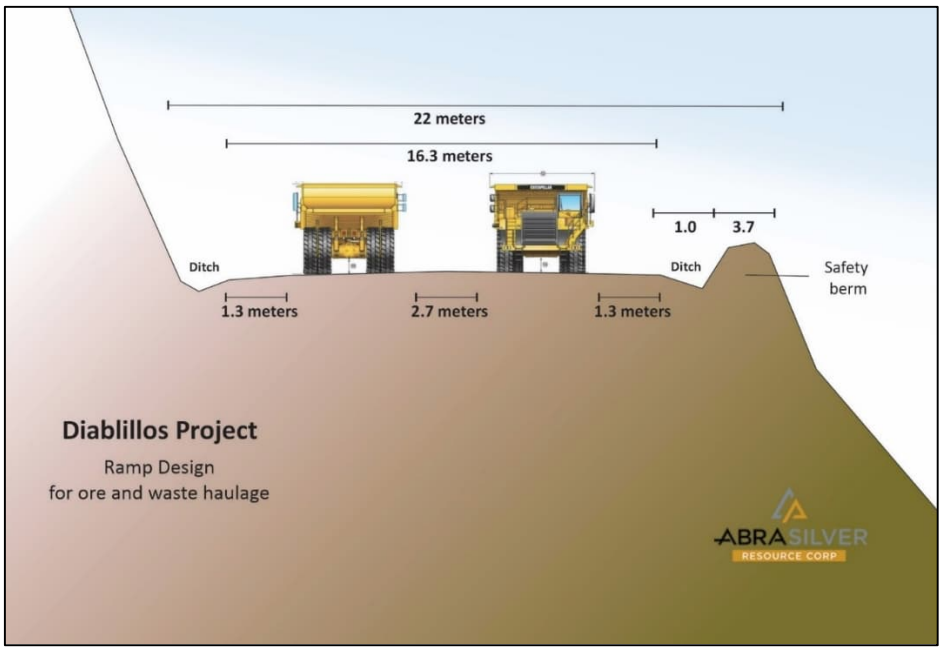
The PFS pit wall design criteria was provided above in Section 15.2.4 and Figure 15-6. These criteria have been followed in open pit design.

15.4.1.3 Haul Road Design

Haul roads within the open pit were designed according to international mine design standards for safe and productive haulage. The road is 3.0 times the width of the widest vehicle. A safety berm on the downslope side of the road is designed to be 75% of the height of the tire of the largest vehicle. An additional allowance is left on the upslope side of the road for a drainage ditch, utility lines and potential pit dewatering pipes. The parameters used in haul road design are summarized in Table 15-12. A typical road cross section is illustrated in Figure 15-13. Haul roads are designed at a maximum gradient of 10% throughout most of the pit.

Table 15-12: Two-Way Haul Road Design Parameters

Parameter	Units	Value
Haul truck model	-	CAT 777B
Haul truck width	Meters	5.5
Tire specifications	-	27.00R49
Tire outer diameter	Meters	2.7
Berm height	Meters	2.2
Safety Berm width	Meters	4.7
Ditch width	Meters	1.0
Safety Margin	Meters	1.3
Central Spacing	Meters	2.7
Total design width	Meters	22.0



Source: AbraSilver Resource Corp., 2023

Figure 15-13: Typical Haul Road Cross-Section

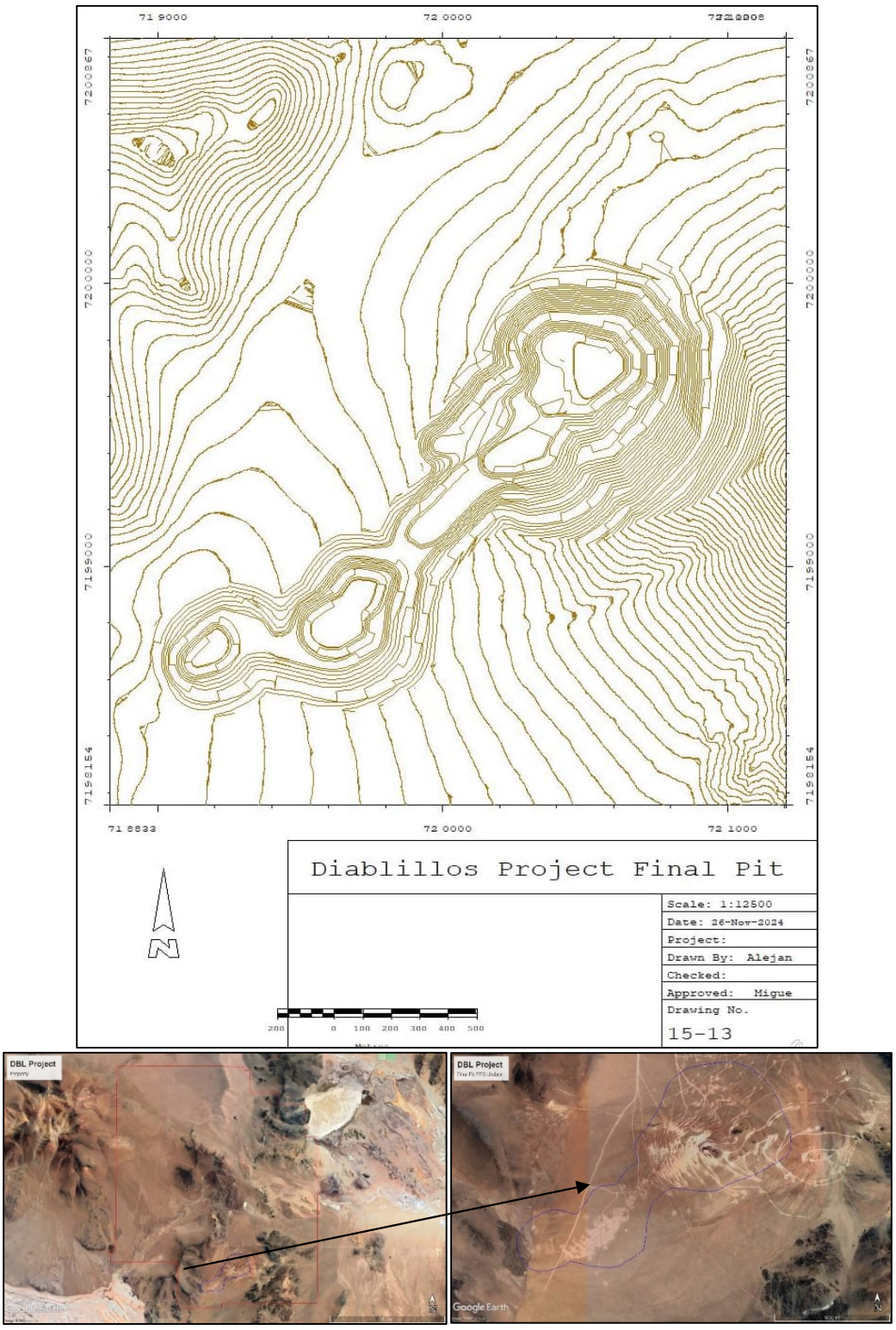
15.4.2 Mineral Reserve Pit Design

Mr. Fuentealba designed the ultimate open pit for Diablillos Project Mineral Reserves in alignment with pit shell 89 from the open pit optimization analysis. The design is provided in Figure 15-14.

Compared to pit shell #89 (RF = 1), the ultimate open pit design has 17% less contained silver equivalent metal (Table 15-3). This is considered an acceptable variance in such design. The main reason for the decrease was the incorporation of ramps in the operational phase designs.

Table 15-13: Mineral Resources Measured + Indicated

Resources Measured + Indicated	Tonnage (000 t)	Au (g/t)	Ag (g/t)	AgEq (t oz)
Pitshell 89 (RF=1)	48,450	0.83	93.00	252,294
Operatives Phase Designs	42,294	0.81	90.81	209,619
Difference				-17%



Source: AbraSilver Resource Corp., 2024

Figure 15-14: Diablillos Project Ultimate Pit Design

15.5 Mining Reserve Estimate

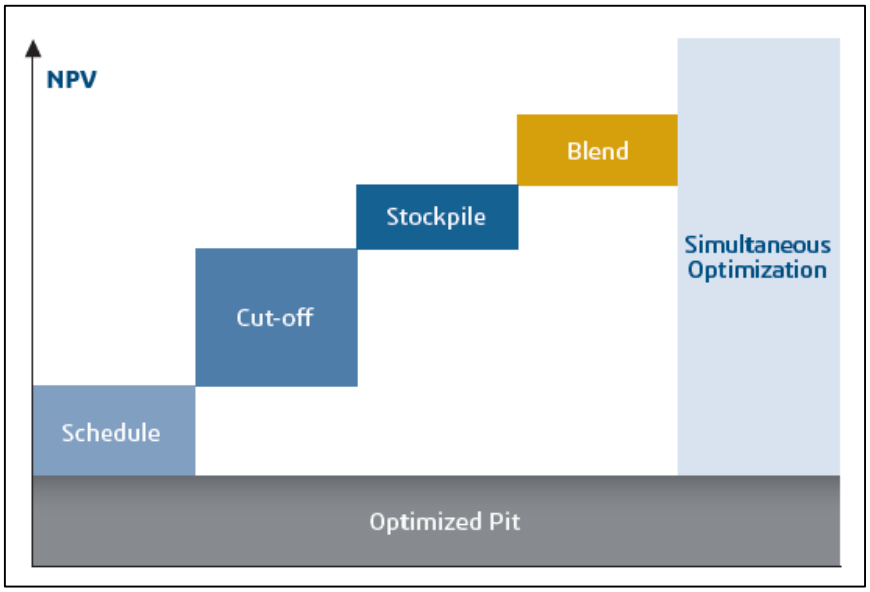
15.5.1 Strategic Mine Planning

The same ultimate pit was used as in the previous stage; however, an optimization of the mining sequence was performed using Whittle's SIMO (Simultaneous Optimization) module.

SIMO optimizes a schedule for a given mining operation from resource through to market, including mining schedule, cut-off grades, blending, stockpiling, processing, product and logistics. It integrates Activity-Based cost drivers to estimate the real costs and constraints mining, processing and product to ensure the optimization results are practical.

This uses the ProberB engine developed by Whittle Consulting to identify an optimal mining sequence as expressed below and illustrated in Figure 15-15:

- Add blocks in pushbacks, panels, and stockpiles to reduce problem size.
- Express the sequence to be solved by linear programming and use an iterative approach to optimise.
- Iterate semi-randomly through the solution space to ensure the sequence obtained is close to the optimal solution.



Source: Whittle Consulting, 2024

Figure 15-15: Simultaneous Optimization

Several strategic mine planning scenarios were developed to meet cover material needs during construction of infrastructure such as the ROM pad, TSF wall, camp, truck shop, and process plant. In addition, construction of the road to the waste dump and main roads will use cover material and waste rock.

Finally, preference was given to stockpiling below cut-off grade mineralisation that could be processed in the future. A stockpile was designed to store approximately 20 million metric tonnes separate it from the rest of pure waste material. Further studies are required to determine if below cut-off grade mineralisation can be treated economically.

15.5.2 Selection of Theoretical Phases

The selection of theoretical phases was made based on a proposed processing capacity of 9,000 tpd or 3,150,000 tonnes per year. This considers an assumption of 350 plant operating days per year.

The criterion adopted was for each phase to contain enough ore to feed the plant for 2-3 years. A minimum loading width of 25 meters was selected according to the smallest space necessary to carry out efficient loading. Corresponding to the sum of the distances for maneuvering equipment. The ends of the benches have smaller widths, so excavation in these sectors will be carried out by smaller equipment such as hydraulic excavators.

The Table 15-10 corresponds to the ore tonnage contained in each nested open pit considering the price of the elements multiplied by the pit Revenue Factor.

The Table 15-14 shows the adjusted tonnage capacity estimate of the nested pits:

Table 15-14: Nested Pit Tonnage Capacity Estimates at Planning Prices

Pit	Total Rock (000 t)	Ore (000 t)	Waste (000 t)	Strip Ratio	Au (g/t)	Ag (g/t)	AgEq (g/t)
1	2,387	81	2,306	28.46	0,20	1123	1139
2	2,389	83	2,306	27.71	0,20	1104	1120
3	12,961	1,698	11,263	6.63	0,90	309	395
4	22,986	3,728	19,257	5.17	0,60	275	329
5	27,995	4,967	23,027	4.64	0,66	254	311
6	36,612	6,783	29,829	4.40	0,76	237	302
7	46,676	8,627	38,049	4.41	0,79	230	298
8	47,047	8,712	38,335	4.40	0,79	229	298
9	48,690	9,174	39,515	4.31	0,80	225	294
10	49,225	9,332	39,892	4.27	0,80	224	292
11	49,732	9,528	40,204	4.22	0,79	222	290
12	52,427	9,980	42,446	4.25	0,78	221	287
13	63,589	12,450	51,139	4.11	0,80	203	270
14	64,789	12,735	52,054	4.09	0,80	201	269
15	65,210	12,856	52,353	4.07	0,80	200	268
16	68,278	13,464	54,814	4.07	0,78	199	264
17	71,480	13,789	57,691	4.18	0,77	199	264

Pit	Total Rock (000 t)	Ore (000 t)	Waste (000 t)	Strip Ratio	Au (g/t)	Ag (g/t)	AgEq (g/t)
18	73,866	14,415	59,451	4.12	0,77	195	259
19	76,116	14,551	61,565	4.23	0,77	195	260
20	78,150	15,226	62,924	4.13	0,76	191	255
21	79,308	15,529	63,779	4.11	0,76	189	253
22	79,955	15,649	64,305	4.11	0,75	189	252
23	83,981	16,689	67,292	4.03	0,75	182	245
24	88,664	17,367	71,296	4.11	0,74	180	242
25	90,044	17,700	72,343	4.09	0,74	178	240
26	94,061	18,554	75,506	4.07	0,74	174	237
27	100,283	19,581	80,702	4.12	0,75	169	233
28	101,161	19,790	81,371	4.11	0,75	169	232
29	101,907	19,964	81,942	4.10	0,75	168	231
30	102,208	20,095	82,112	4.09	0,75	167	230
31	119,235	23,834	95,401	4.00	0,78	149	214
32	119,825	24,012	95,812	3.99	0,78	148	213
33	157,588	31,360	126,228	4.03	0,86	121	193
34	159,343	31,707	127,636	4.03	0,86	120	193
35	165,640	32,749	132,891	4.06	0,84	120	191
36	167,073	33,010	134,063	4.06	0,84	120	190
37	168,563	33,439	135,124	4.04	0,84	119	189
38	169,629	33,843	135,785	4.01	0,84	118	188
39	180,939	35,931	145,007	4.04	0,84	113	184
40	181,196	36,039	145,156	4.03	0,84	113	184
41	182,494	36,327	146,166	4.02	0,84	112	183
42	183,690	36,765	146,924	4.00	0,84	112	182
43	183,960	36,878	147,082	3.99	0,84	111	182
44	187,143	37,580	149,563	3.98	0,84	110	180
45	188,066	37,921	150,144	3.96	0,84	109	180
46	195,982	39,051	156,930	4.02	0,84	107	178
47	197,931	39,399	158,531	4.02	0,84	106	177
48	199,218	39,722	159,496	4.02	0,84	106	176
49	199,297	39,763	159,533	4.01	0,84	106	176
50	202,989	40,191	162,797	4.05	0,84	105	176
51	206,597	40,683	165,913	4.08	0,83	105	175
52	206,830	40,764	166,066	4.07	0,83	105	175
53	207,506	40,892	166,614	4.07	0,83	104	175
54	208,377	41,030	167,347	4.08	0,83	104	174
55	209,077	41,238	167,839	4.07	0,83	104	174

Pit	Total Rock (000 t)	Ore (000 t)	Waste (000 t)	Strip Ratio	Au (g/t)	Ag (g/t)	AgEq (g/t)
56	209,470	41,340	168,129	4.07	0,83	104	174
57	210,412	41,505	168,906	4.07	0,83	104	173
58	211,506	41,773	169,733	4.06	0,83	103	173
59	212,200	41,936	170,264	4.06	0,83	103	172
60	212,623	42,031	170,591	4.06	0,83	103	172
61	214,676	42,337	172,339	4.07	0,83	102	172
62	215,093	42,449	172,643	4.07	0,82	102	171
63	215,310	42,528	172,781	4.06	0,82	102	171
64	215,493	42,613	172,879	4.06	0,82	102	171
65	238,265	45,182	193,083	4.27	0,83	97	167
66	238,696	45,322	193,373	4.27	0,83	97	167
67	239,793	45,443	194,350	4.28	0,83	97	167
68	240,635	45,565	195,070	4.28	0,83	97	166
69	250,366	46,231	204,135	4.42	0,83	96	166
70	250,848	46,294	204,553	4.42	0,83	95	166
71	251,218	46,360	204,858	4.42	0,83	95	165
72	251,361	46,405	204,956	4.42	0,83	95	165
73	251,653	46,466	205,186	4.42	0,83	95	165
74	252,133	46,569	205,564	4.41	0,83	95	165
75	252,363	46,644	205,719	4.41	0,83	95	165
76	253,641	46,789	206,851	4.42	0,83	95	165
77	255,999	47,004	208,994	4.45	0,83	95	164
78	260,866	47,395	213,471	4.50	0,83	94	164
79	260,952	47,430	213,522	4.50	0,83	94	164
80	261,391	47,524	213,867	4.50	0,83	94	164
81	261,566	47,580	213,986	4.50	0,83	94	163
82	261,837	47,636	214,201	4.50	0,83	94	163
83	263,857	47,922	215,935	4.51	0,82	93	163
84	264,833	48,017	216,815	4.52	0,82	93	163
85	264,970	48,071	216,898	4.51	0,82	93	162
86	267,706	48,217	219,489	4.55	0,82	93	162
87	268,385	48,289	220,096	4.56	0,82	93	162
88	268,997	48,383	220,613	4.56	0,82	93	162
89	269,372	48,450	220,921	4.56	0,82	93	162

*Note: Tonnages and grades reported are totals, i.e., cumulative and do not represent the incremental contribution of each open pit.

**Note: All figures are rounded to reflect the relative accuracy of the estimates. Minor discrepancies may occur due to rounding to appropriate significant figures.

Based on the tonnage estimate and the phase selection criteria, the open pits chosen as theoretical phases can be visualized in Table 15-15.

Table 15-15: Tonnage capacity estimate of theoretical phases

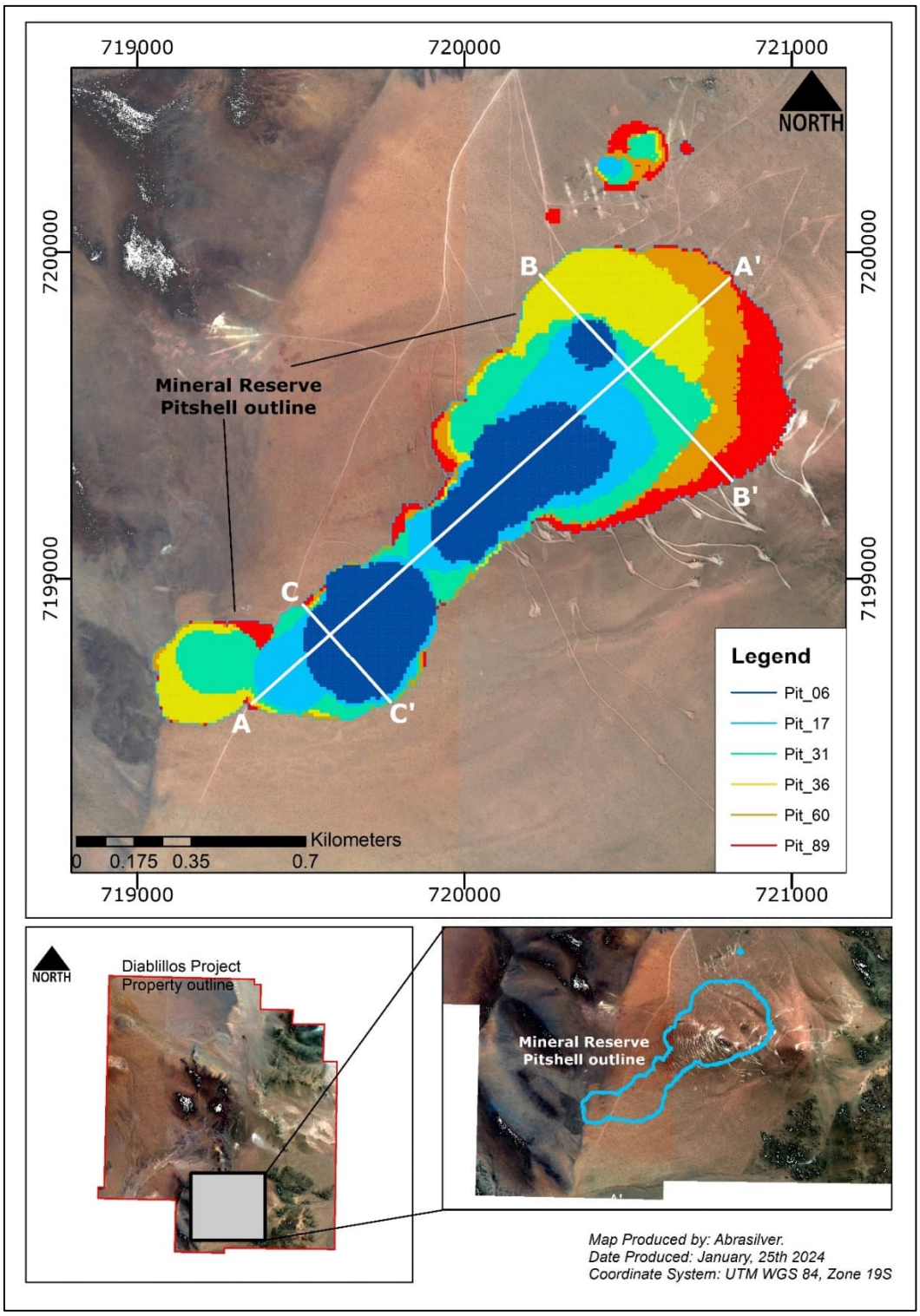
Pit	Total Rock (000 t)	Ore (000 t)	Overburden (000 t)	Feed Years	Au (g/t)	Ag (g/t)	AgEq (g/t)
6	36,612	6,783	29,829	2.2	0.77	237	302
17	34,867	7,006	27,861	2.2	0.78	162	227
31	47,754	10,044	37,710	3.2	0.79	80	145
36	47,838	9,176	38,662	2.9	1.00	44	128
60	45,549	9,021	36,528	2.9	0.79	41	107
89	56,748	6,418	50,330	2.0	0.80	27	94

A series of vertical cross sections have been prepared to display their extents. Found below in Figure 15-16 to Figure 15-19.

The first theoretical phase is observed to have 2 independent cones. The mining growth trend of the following phases is towards the northeastern sector of mineralization.

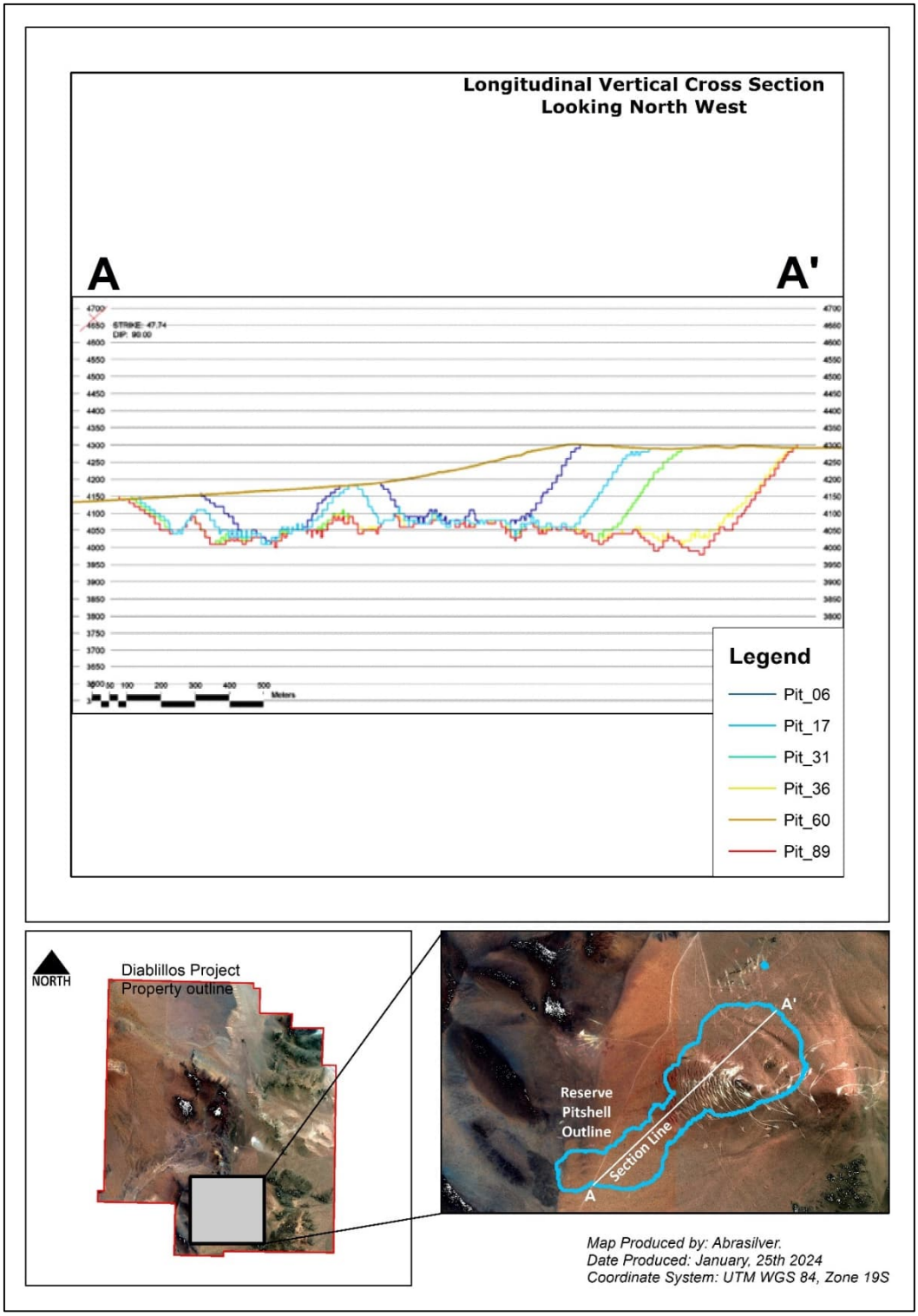
In section AA', horizontal growth of the theoretical phases is observed in greater proportion than vertical growth in Oculto and JAC. In section BB', horizontal and vertical growth of the theoretical phases is observed in Oculto. No significant horizontal and vertical mining growth of theoretical phases is observed in section CC' in JAC.

It is important to mention that theoretical phases are a guide for the operationalization of the open pit. However, the boundaries of each theoretical phase may not coincide with the operational phase. Primarily due to required widths for operation.



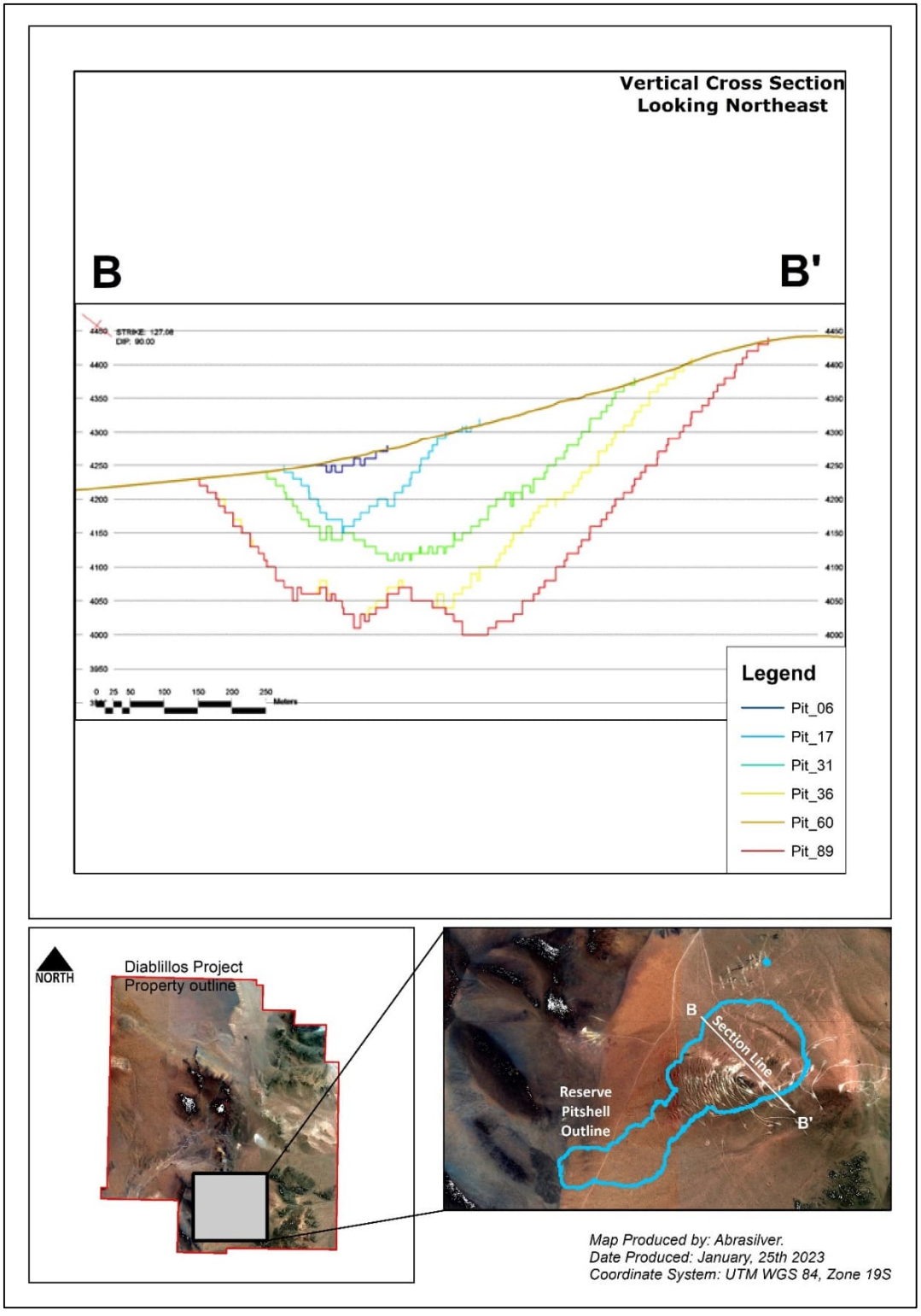
Source: AbraSilver Resource Corp., 2024

Figure 15-16: Theoretical Phases



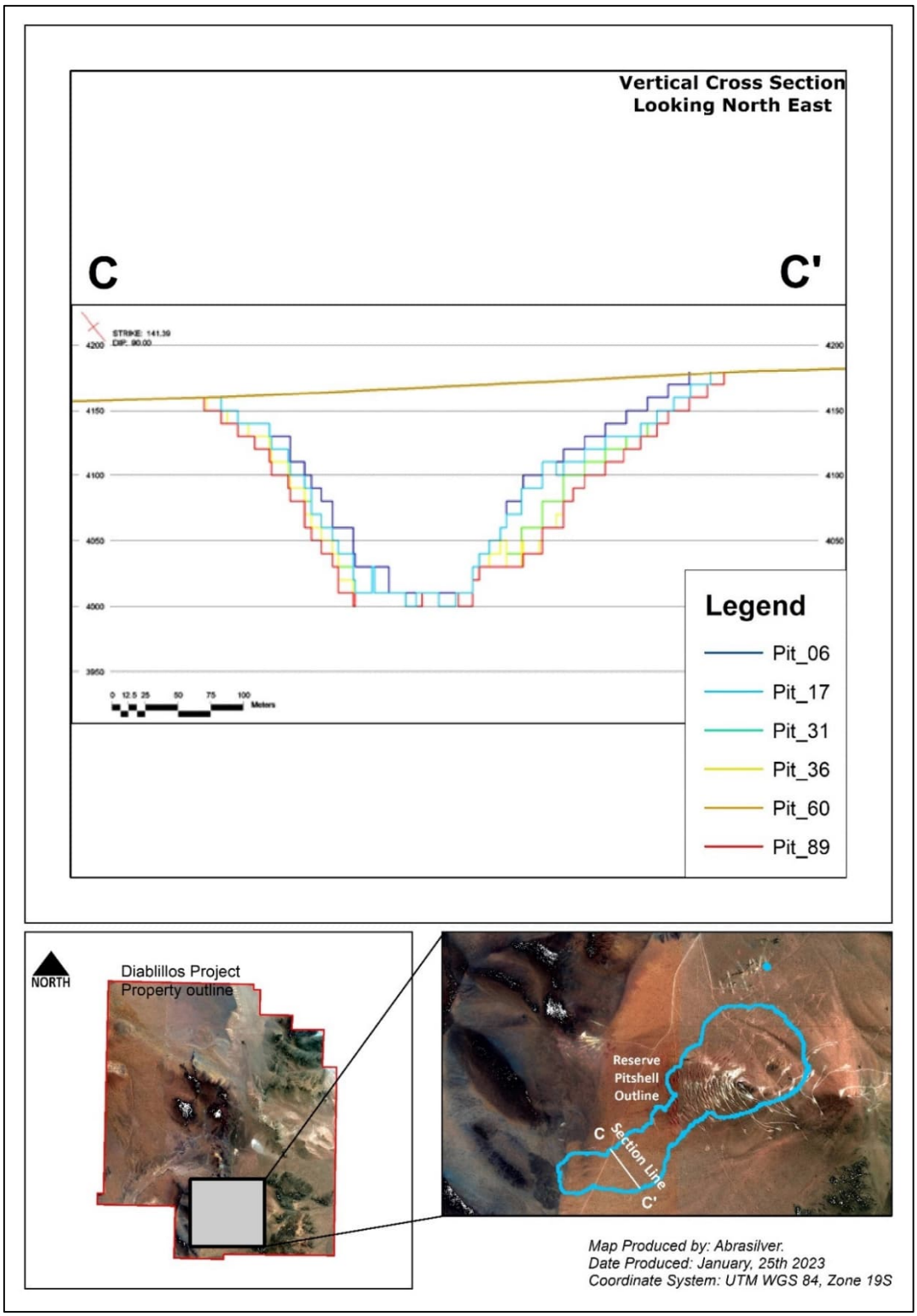
Source: AbraSilver Resource Corp., 2024

Figure 15-17: Theoretical Phases, Vertical Section AA'



Source: AbraSilver Resource Corp., 2024

Figure 15-18: Theoretical Phases, Vertical Section BB'



Source: AbraSilver Resource Corp., 2024

Figure 15-19: Theoretical Phases, Vertical Section CC'

15.6 Mineral Reserve Statement

The Mineral Reserve estimate for the Diablillos Project shown in Table 15-16 has been estimated by Mr. Miguel Fuentealba, MAusIMM, P. Eng (QP). It is based on the Mineral Resource block model documented in Section 14 of this report, entitled “Mineral Resource Estimates”.

The Mineral Reserve estimate is calculated using a combination of the final pit (Section 15.4.2) and the production plan (Section 16.2).

The following Table 15-16 shows a summary of the estimated Mineral Reserves as of March 7, 2024.

Table 15-16: Proved and Probable Mineral Reserves as of March 7th, 2024

Mineral Reserve (all domains)	Tonnage (Mt)	Au (g/t)	Ag (g/t)	AgEq (g/t)	Contained Ag (000 oz Ag)	Contained Au (000 oz Au)	Contained AgEq (000 oz AgEq)
Proven	12.4	0.86	118	185	46,796	341	73,352
Probable	29.9	0.80	80	142	76,684	766	136,267
Total Proven and Probable	42.3	0.81	91	154	123,480	1,107	209,619

Notes for Mineral Reserve Estimate:

1. The Mineral Reserves have an effective date of March 07, 2024.
2. The Qualified Person for the Mineral Reserves Estimate is Mr. Miguel Fuentealba, P. Eng.
3. Mineral Reserves were estimated using the Canadian Code of the Canadian Institute of Mining, Metallurgy, and Petroleum (CIM), Definition of Standards for Mineral Resources and Mineral Reserves, prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.
4. The Mineral Reserves are based on a pit design which is aligned with the ultimate pit selected during the optimization process performed at Whittle.
5. Estimated reserves at a sale price of US$1,750 per ounce of Au and US$22.5 per ounce of Ag.
6. Variable mining cost per bench and type of material was applied. Average cost of 1.94 US$/t for all lithologies, except for cover, where a cost of 1.73 US$/t is applied.
7. Processing cost for all zones is US$22.97/t.
8. Infrastructure, general, and administrative costs amount to US$3.32/ton.
9. The average pit angles range from 37° to 60°, depending on the geotechnical zone domain.
10. The average recovery is estimated at 82.6% for silver and 86.5% for gold.
11. The formula for calculating Ageq is Ag eq oz = Ag oz + Au oz x (Au Price/Ag Price).
12. Mineral Reserves Estimates have been categorized according to CIM Standard definitions (CIM, 2014).
13. A net value per block (“NVB”) was used to restrict the Mineral Reserves within the pitshell. The NVB is based on “Profit = Revenue - Costs”, being positive, where, Revenue = [(Au Price (US$/oz) - Au Sales Cost (US$/oz) x (Au Grade (g/t)/31.1035))x Au Recovery (%)]+[(Ag Price (US$/oz) - Ag Sales Cost (US$/oz) x (Ag Grade (g/t)/31. 1035) x Ag Recovery (%) and cost as, Cost = Mine Cost (US$/t) + Process Cost (US$/t) + Transportation Cost (US$/t) + General and Administrative Costs (US$/t) + [Royalty Cost (%) x Revenue]. The NVB method matches an equivalent cut-off grade of 46 g/t Ageq.
14. In situ density is read directly from the block model, previously assigned for each domain during the mineral resource estimation process, according to averages of samples from each lithological domain, separated by alteration and oxidation zones.
15. All tonnages reported are in dry metric tons and ounces of gold and silver contained in troy ounces.
16. Mining plan considers 298 Mt of waste rock, corresponding to a waste to ore ratio of 6.7. Waste rock from period -2 is considered CAPEX.
17. Ore is considered to be that which complies with profit greater than zero, measured or indicated resource, is within the designed operating phases, oxidation state is oxide, any lithology except cover.

18. Recoveries were estimated by geometallurgical domain within each block.
19. Figures are approximate due to rounding.
20. Au and Ag metal is considered in situ.

15.7 Factors Affecting Mineral Reserve Estimates

Various technical and economic factors can affect a Mineral Reserve estimate. These include dilution, metal prices, off-site costs, metallurgical recoveries, open pit slope designs, capital, and operating cost estimates. In the opinion of Mr. Fuentealba (QP), the factors affecting the Mineral Reserve have been adequately considered. The main factors affecting the Mineral Reserve reported in this section are:

- Metal sales prices, particularly silver price, accounting for a 58% share of total revenues.
- Open pit design parameters that should be validated by a future geotechnical study. This will reduce risk of a decrease in strip ratio.

The Mineral Reserve estimate has considered all known legal, political, environmental, or other risks that could materially affect potential development as discussed in this Technical Report.

15.8 Comments on Section 15 from the QP

This document represents technical and economic conditions based on the data received and generated during the prefeasibility stage of the Project. This document is valid as of December 31, 2024. Given the nature of the mining business, the aforementioned conditions may be change due to advances in technology and/or the global economic situation. In this case the results of the study may vary significantly.

A margin of error must also be considered due to the rounding of figures used in the process; however, in the opinion of the QP, these are negligible.

Neither Bmining nor the QP are privileged persons, associates, or affiliates of AbraSilver, or its respective subsidiaries.

The project has many possibilities to expand its Mineral Reserves by investing in geotechnical testing, exploration and in-fill campaigns. There are sectors of the pit that can be modified with a higher overall slope angle, potentially improving strip ratio.

There is the possibility of converting some marginal resources by exploring blending or other processes such as heap leaching.

16. Mining Methods

The following section has been taken in its entirety from the previous NI 43.101 Technical Report Updated Pre-Feasibility Study, effective December 3rd, 2024, and report date January 17th, 2025.

Being a large, near-surface orebody, the Diablillos Project will be developed as an open pit mining operation. Waste and ore will be drilled, blasted, loaded by hydraulic shovels/loaders, and transported by haul trucks. Either to external waste storage facilities (WSF), long-term stockpiles, or a run-of-mine (ROM) pad to feed a primary crusher for mineral processing.

An optimal processing throughput rate of 9,000 tonnes per day (tpd) was established on raw water availability from the explored Barranquillas district. This plant throughput rate became the basis of all subsequent mine planning. The project has several stockpile facilities to allow processing of higher-grade material while stockpiling low-grade material.

Ore control technicians will oversee grade control; this is the procedure to differentiate ore from waste. In the Diablillos Project, this will require the sampling of blast holes and the classification of bench materials. Being primarily classified as ore, low-grade ore or waste. In some cases, grade control could also involve the sampling of truck or loader to ensure material is assigned to the correct stockpile or waste dump. The objective is to ensure mining closely follows the mineralized zone and that overbreak is kept to a minimum.

The PFS considers only one mineral processing flowsheet, which is comminution, gravity recovery followed by intense cyanidation, tank leaching, Merrill-Crowe recovery and foundry. Below cut-off grade marginal mineralization is currently being treated as waste. A preliminary internal study suggested it may be amenable to other low-cost processing technologies, which could be confirmed by further test work. The current PFS calls for grade control to only differentiate waste from ore and direct feed ore from stockpiled ore. Given insufficient data to define another extraction process at this stage, a specific waste rock dump area has been reserved to stockpile low grade mineralization for future use.

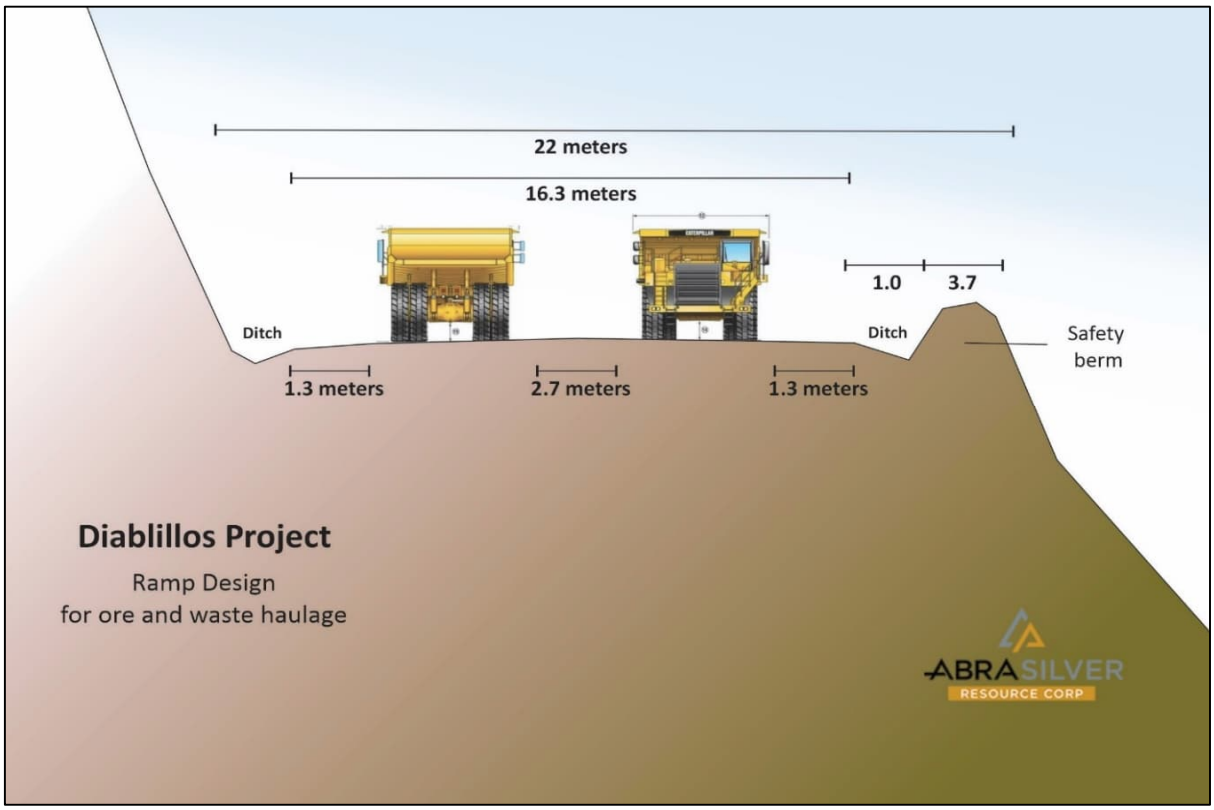
16.1 Introduction

This section describes the operational design of the mining phases, and mine planning that underpins the Mineral Reserves of the Diablillos Project. The Diablillos ore bodies are feasible to mine through open pit mining, and the Merrill-Crowe method is suitable for processing.

16.2 Mine Design Criteria

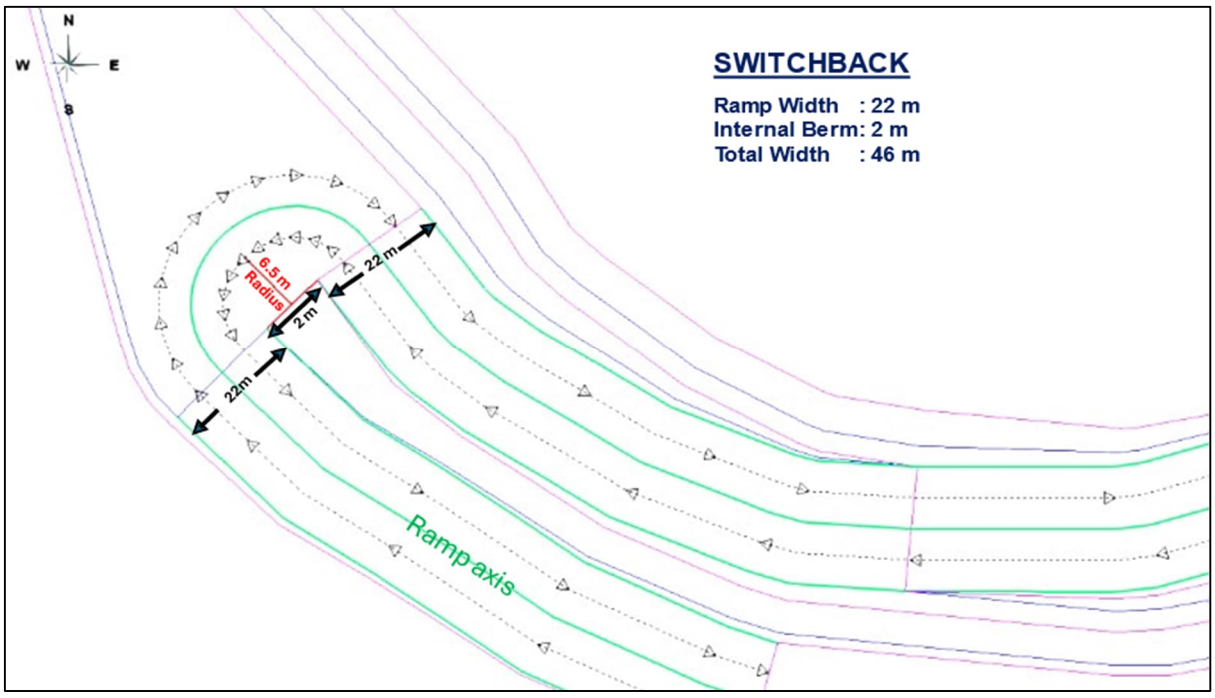
Benches in each phase will be mined to a height of 10 meters for all types of material. A 22-meter-wide transport ramp with a 10% slope will be used to allow off-road trucks of 100 metric tonnes to travel in both directions (double track). This width considers safety berms, rainwater ditches, and minimum centre separation.

Figure 16-1 shows the profile of the internal mine haulage ramp and Figure 16-2 shows the diagram for the directional turns. A 13-meter decoupling berm is considered every 90 meters of slope height when no ramp passes through the wall. The average inter-ramp angles for the different pit walls range from 33 to 54 degrees. A summary of the pit design parameters is shown in Table 16-1.



Source: AbraSilver Resource Corp., 2023

Figure 16-1: Haulage Ramp Design Parameters



Source: AbraSilver Resource Corp., 2023

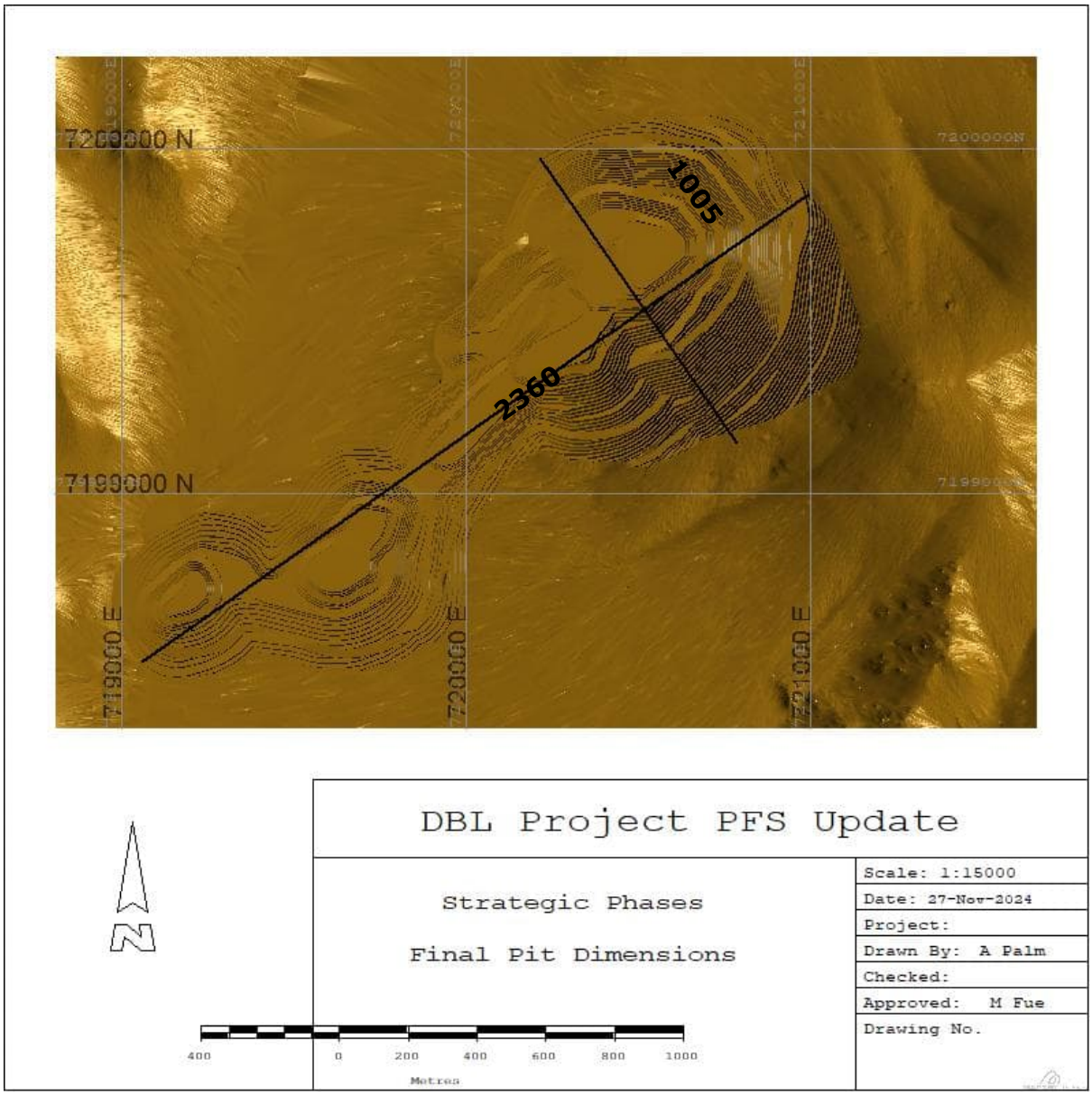
Figure 16-2: Haulage Ramp Turning Radius

Table 16-1: Pit Design Parameters

Parameter	Units	Value	Comments
Bench height	Meters	10	
Bench face angle	Degrees	60-80	According to geotechnical zone
Berm width	Meters	4.5-11.76	According to geotechnical zone
Geotechnical berm	Meters	12-13	Each 120m of high
Inter ramp slope angle	Degrees	34.7-57.8	
Ramp width	Meters	22	Allows for double lane traffic
Maximum slope	%	10	

16.2.1 Final Open Pit Design

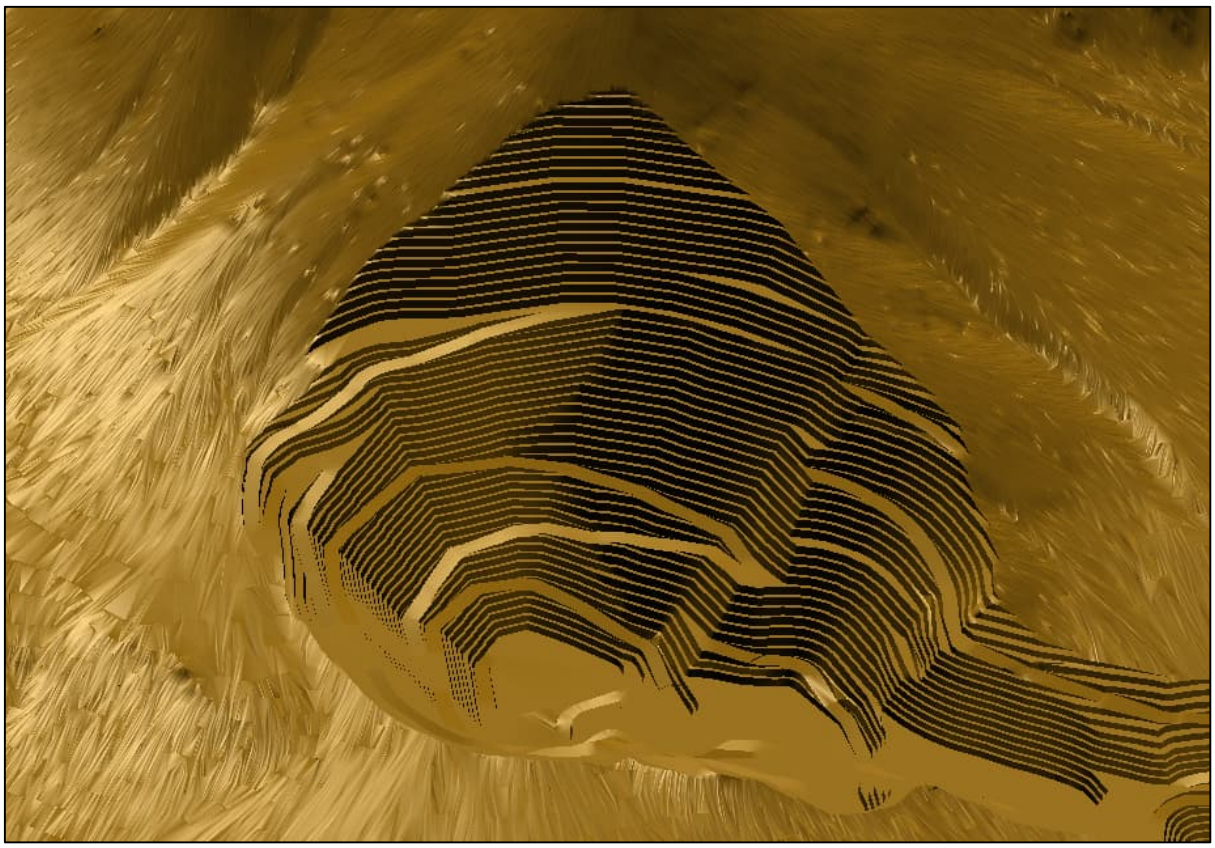
The ultimate designed operational pit has an approximate azimuth of 49 degrees in a southwest to northeast direction. Its dimensions are 2,360 meters long by 1,005 meters wide. Its deepest part reaches 4,000 masl (meters above sea level) in the sector known as Oculto, followed in depth by the JAC sector at 4,020 masl. The highest part of the slope is found at approximately 600 meters. The ultimate pit, shown in Figure 16-3, also considers a minimum of two pit exits in any period to comply with local legislation.



Source: AbraSilver Resource Corp., 2023

Figure 16-3: Final Pit Dimensions

Figure 16-4 shows the Oculto sector, which, due to its height, involves 6 geotechnical decoupling berms and 5 ramp passes. The highest wall contains a total of 64 benches.

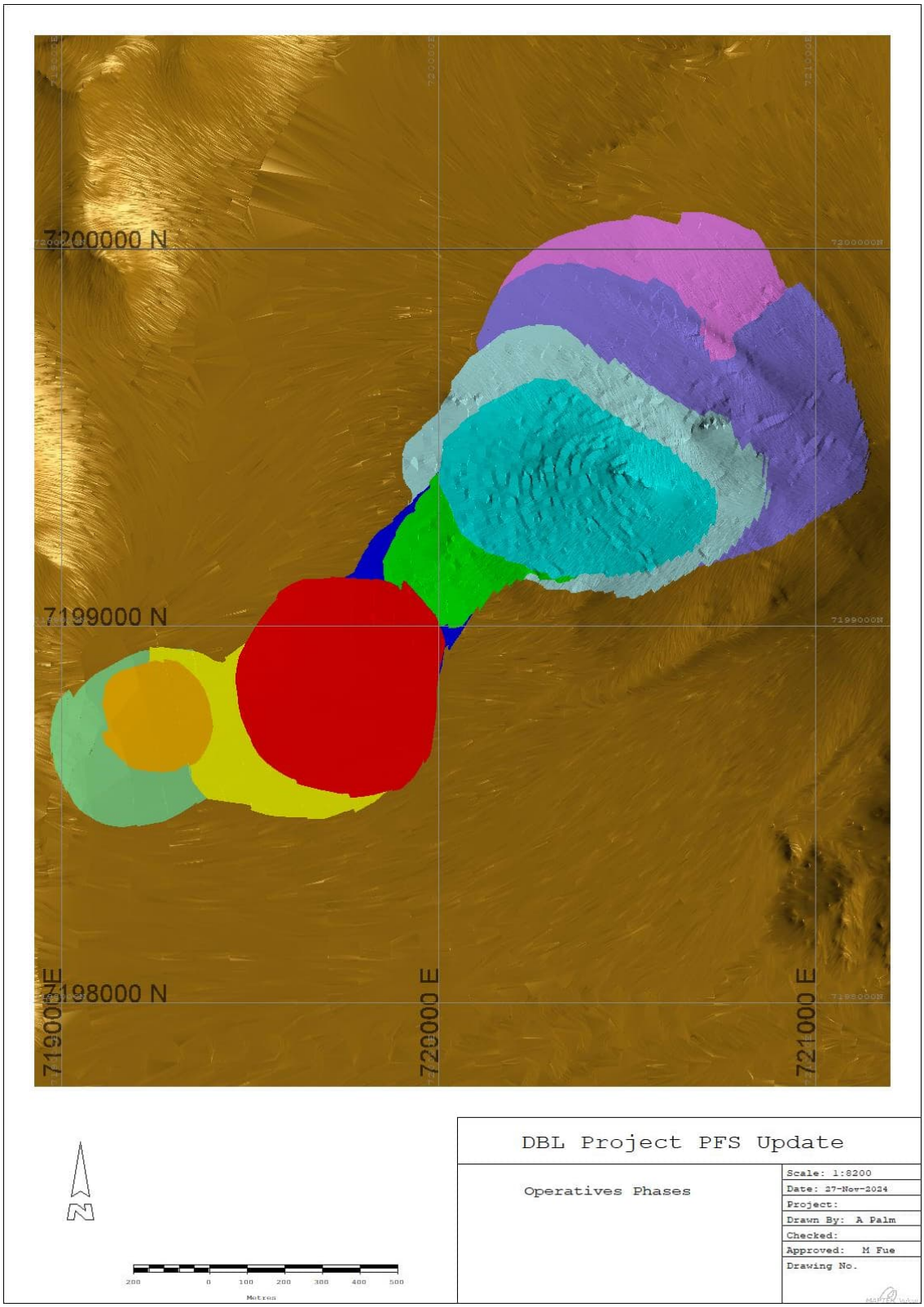


Source: AbraSilver Resource Corp., 2023

Figure 16-4: Oculto Sector

16.2.2 Operating Phase Design

As mentioned in Chapter 15, after selecting theoretical phases ten operating phases of the project were designed. The first phases were made to align with the construction plan and to have necessary material to build platforms for main infrastructure. The operating phases are shown in Figure 16-5 below.



Source: AbraSilver Resource Corp., 2024

Figure 16-5: Operating Phases

The cubic capacity estimates of material that meets the conditions set forth in Table 15-9 of Chapter 15 are the proven reserves plus probable reserves of the project shown in Table 16-2.

Table 16-3 shows the waste rock of each phase. Figure 16-6 shows the sinking ratio.

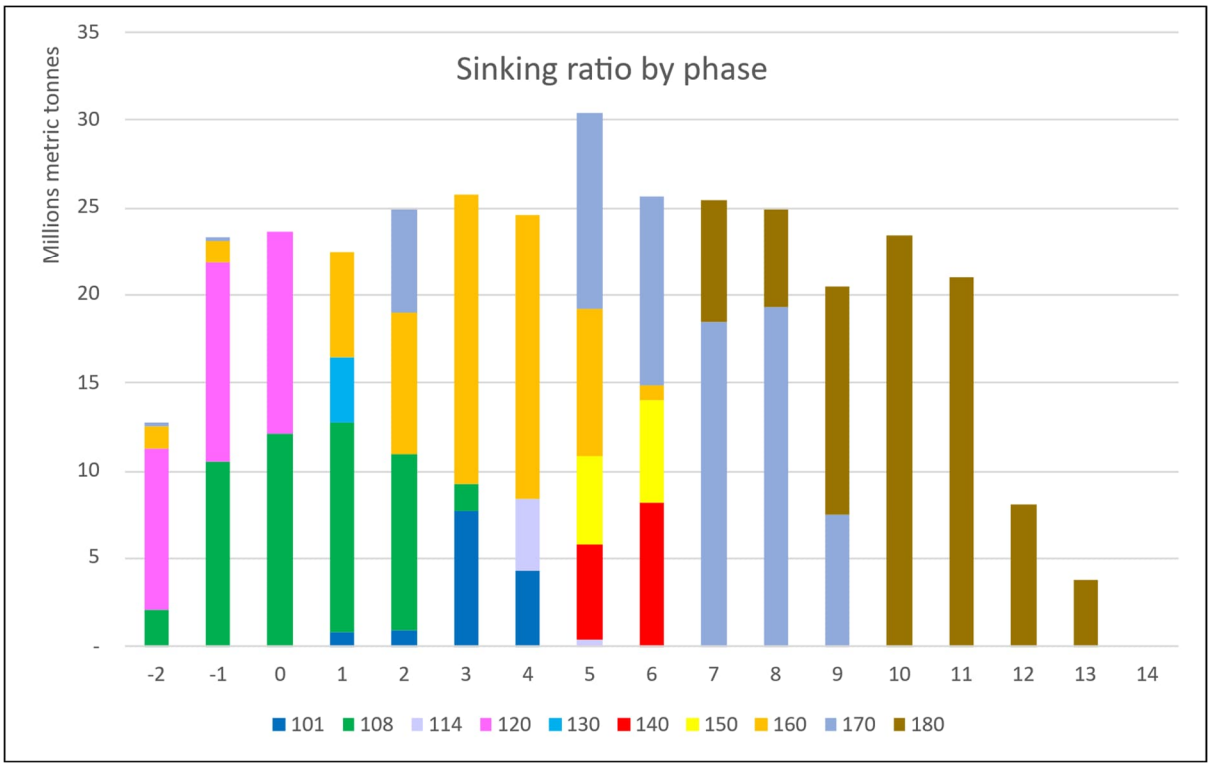
Table 16-2: Proven and Probable Reserves by Phase

Proven Reserves						
Phase	Ore tonnes (kt)	Au (g/t)	Ag (g/t)	Aueq (g/t)	Ageq (g/t)	Ageq (toz)
F02_108	2,978	0.84	171	3.03	236	22,592
F04_101	831	0.98	173	3.20	249	6,647
F05_120	1,451	0.19	220	3.02	235	10,966
F06_140	286	0.09	171	2.29	178	1,639
F08_160	3,148	0.92	92	2.10	163	16,504
F09_170	1,736	1.02	48	1.63	127	7,078
F10_180	1,827	1.26	24	1.58	123	7,200
F11_114	108	0.44	175	2.68	209	725
Total Proven Reserves	12,364	0.86	118	2.37	185	73,352
Probable Reserves						
Phase	Ore tonnes (kt)	Au (g/t)	Ag (g/t)	Aueq (g/t)	Ageq (g/t)	Ageq (toz)
F02_108	2,738	0.82	150	2.74	213	18,780
F03_130	1	0.05	357	4.65	361	9
F04_101	1,529	0.70	158	2.74	213	10,463
F05_120	1,634	0.15	219	2.97	231	12,137
F06_140	507	0.10	218	2.90	226	3,680
F07_150	646	0.03	133	1.74	135	2,811
F08_160	7,102	0.71	85	1.80	140	32,023
F09_170	7,346	0.84	47	1.45	113	26,615
F10_180	8,233	1.08	24	1.39	108	28,672
F11_114	195	0.38	142	2.20	172	1,076
Total Probable Reserves	29,930	0.80	80	1.82	142	136,267

Proven + Probable Reserves						
Phase	Ore tonnes (kt)	Au (g/t)	Ag (g/t)	Aueq (g/t)	Ageq (g/t)	Ageq (toz)
F02_108	5,716	0.83	161	2.89	225	41,372
F03_130	1	0.05	357	4.65	361	9
F04_101	2,359	0.80	163	2.90	226	17,110
F05_120	3,085	0.17	220	3.00	233	23,103
F06_140	793	0.10	201	2.68	209	5,319
F07_150	646	0.03	133	1.74	135	2,811
F08_160	10,250	0.78	87	1.89	147	48,528
F09_170	9,082	0.87	47	1.48	115	33,694
F10_180	10,059	1.11	24	1.43	111	35,872
F11_114	303	0.40	154	2.38	185	1,801
Total Proven + Probable	42,294	0.81	91	1.98	154	209,619

Table 16-3: Waste by Phase

Phase	Waste Cover (kt)	Waste Other (kt)	Total (kt)
F02_108	5,987	42,272	48,259
F03_130	3,633	10	3,642
F04_101	1,999	11,612	13,611
F05_120	18,054	14,128	32,182
F06_140	8,004	5,667	13,670
F07_150	5,774	5,107	10,881
F08_160	4,278	54,111	58,389
F09_170	9,600	64,075	73,674
F10_180	5,551	76,333	81,884
F11_114	654	3,840	4,494
Total	63,534	277,153	340,687



Source: AbraSilver Resource Corp., 2024

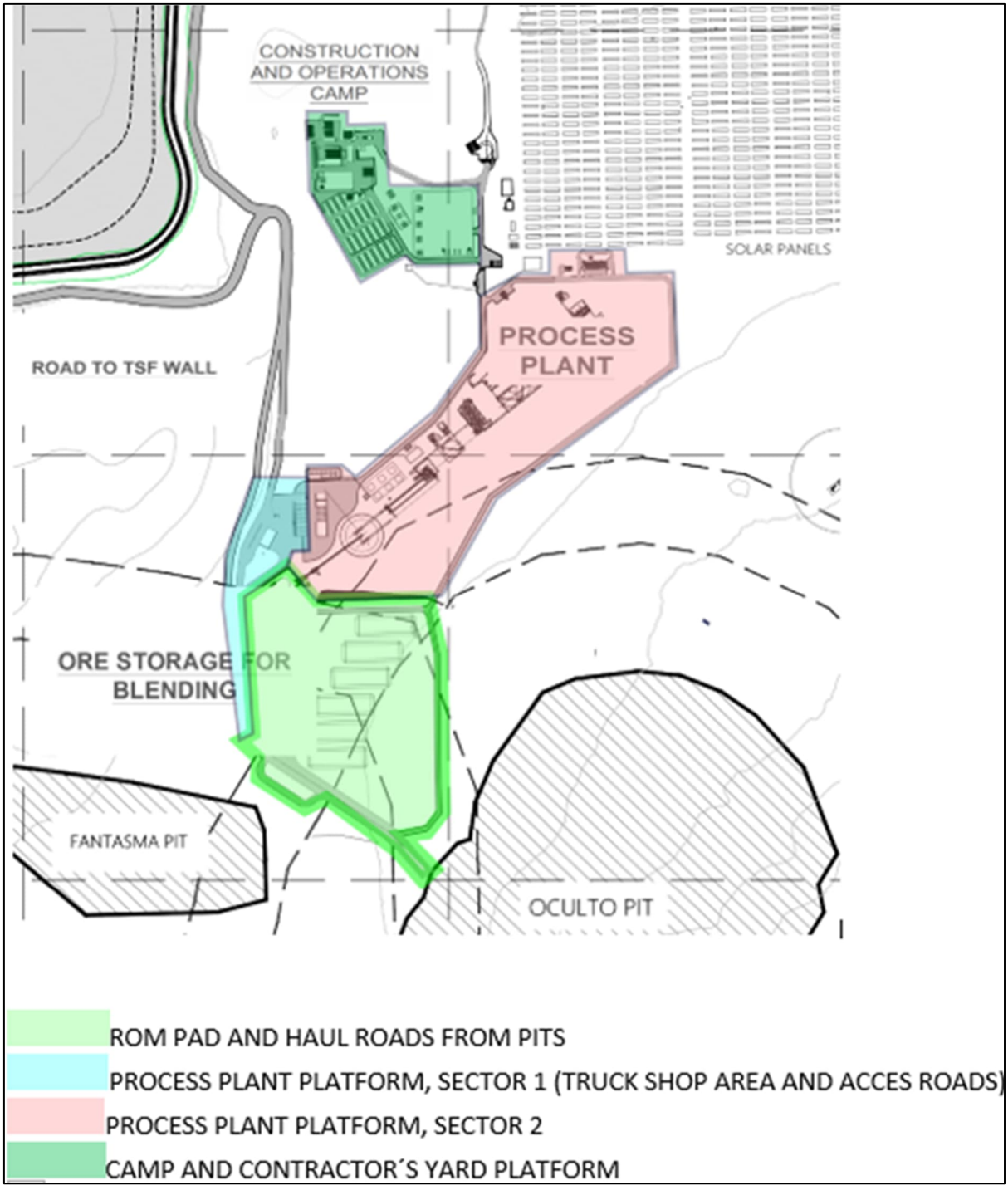
Figure 16-6: Sinking Ratio by Phase

16.3 Mining Plan

The useful life of the mine will be 17 years, including 2 pre-production years called the -2 and -1 periods. Feed to the process plant will begin during period -1 beginning a ramp up to full production. An average mining rate of between 20 and 25 million metric tonnes is considered over the mine life.

16.3.1 Pre-Production Activities

During the pre-production stage, main infrastructure platforms for the camp, contractor's yard, truck shop platform north dump road, tailings dam road and ore stockpile platform for primary crushing will be built (Figure 16-7).



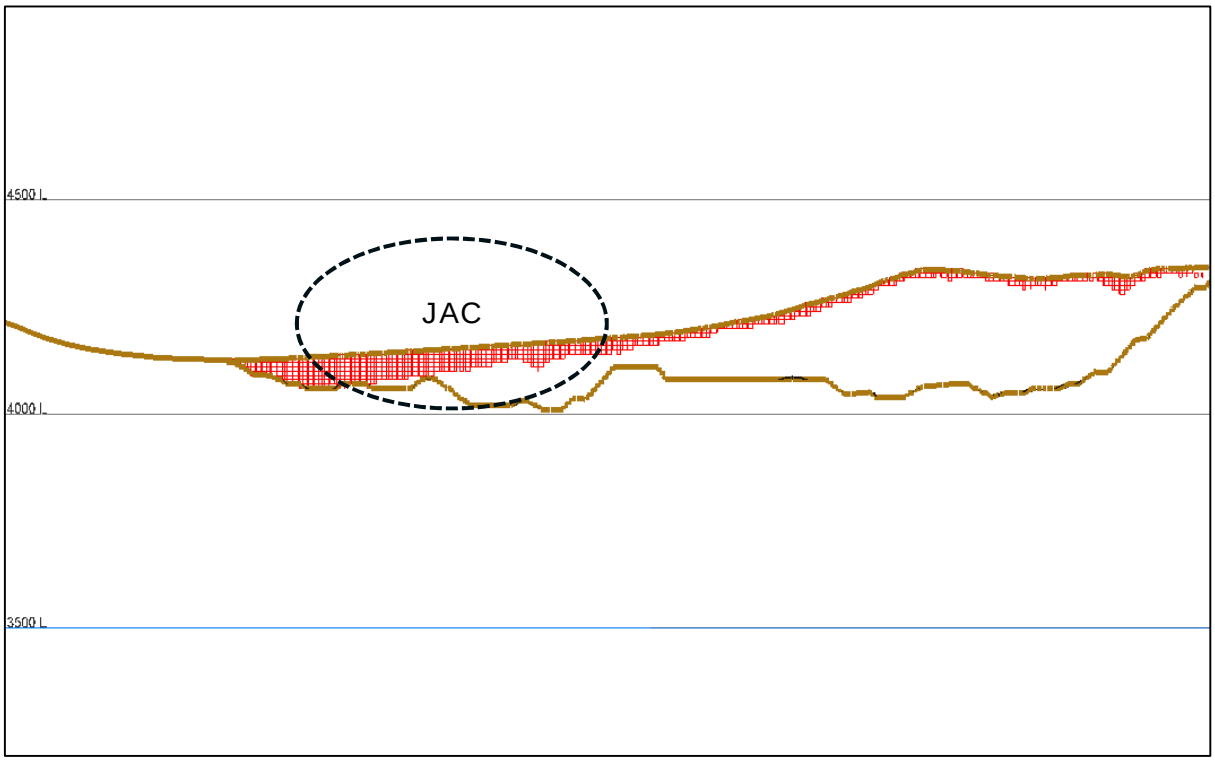
Source: AbraSilver Resource Corp., 2023

Figure 16-7: Platforms for Facilities, Infrastructure and Roads

Cover material requirements:

- Contractors' camp and yard 88,000 m^3
- Truck shop and main access road 310,000 m^3
- Road to north waste dump and tailings dam wall 2,819,504 m^3
- Rom pad platform 1,880,000 m^3
- Process plant platform 915,000 m^3

The demand for cover material for the pre-production stage is largely covered by the cover material present in the phases of the southwest sector of the deposit (Figure 16-8).

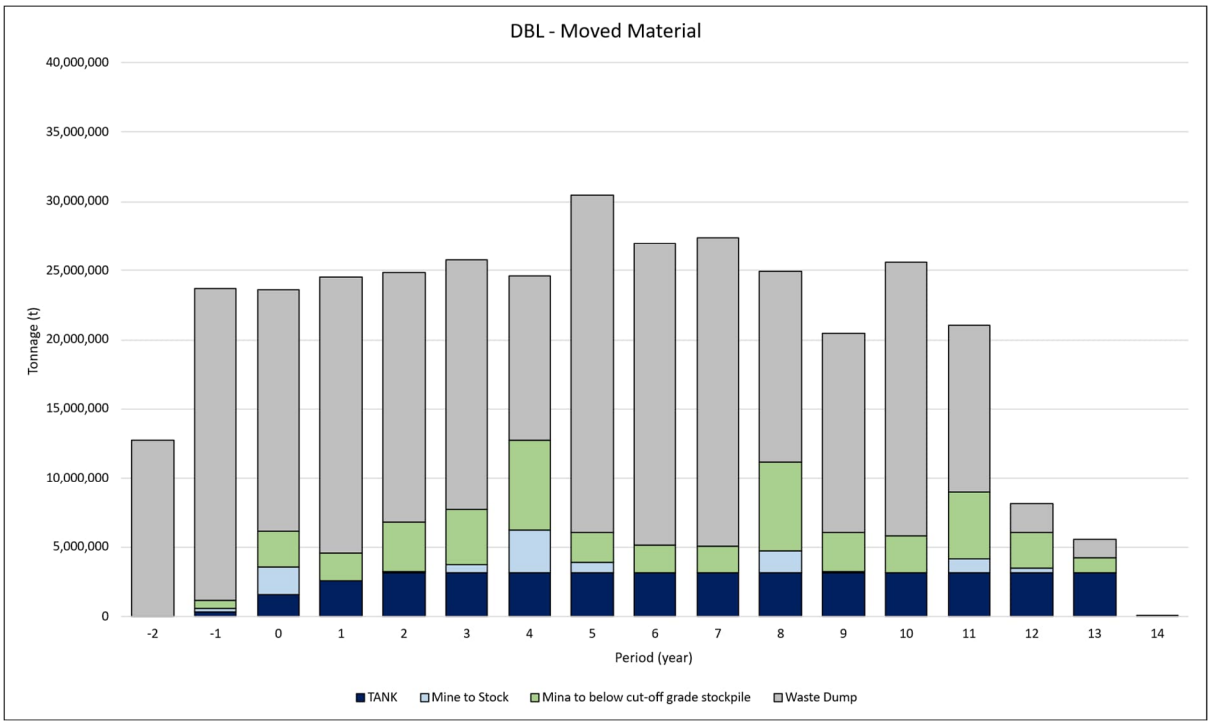


Source: AbraSilver Resource Corp., 2024

Figure 16-8: Cover in Final Pit Increase in JAC

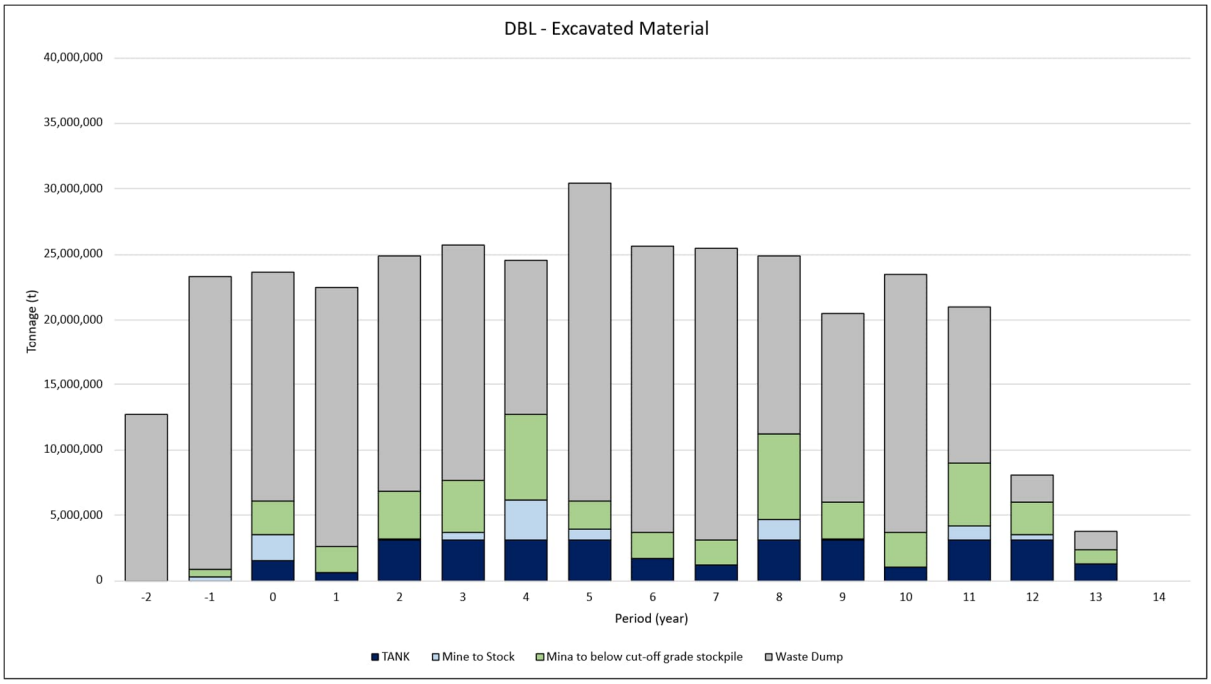
16.3.2 Mine Plan

The production plan considers a strategy that generates sufficient cover material during the first two years. Period -2 will be purely for uncovering waste rock. Ore will be sent to the plant towards the end of period -1 for commissioning. There will then be a ramp up to maximum plant capacity. Figure 16-9 shows the material moved by period while Figure 16-10 shows the material excavated by period.



Source: AbraSilver Resource Corp., 2024

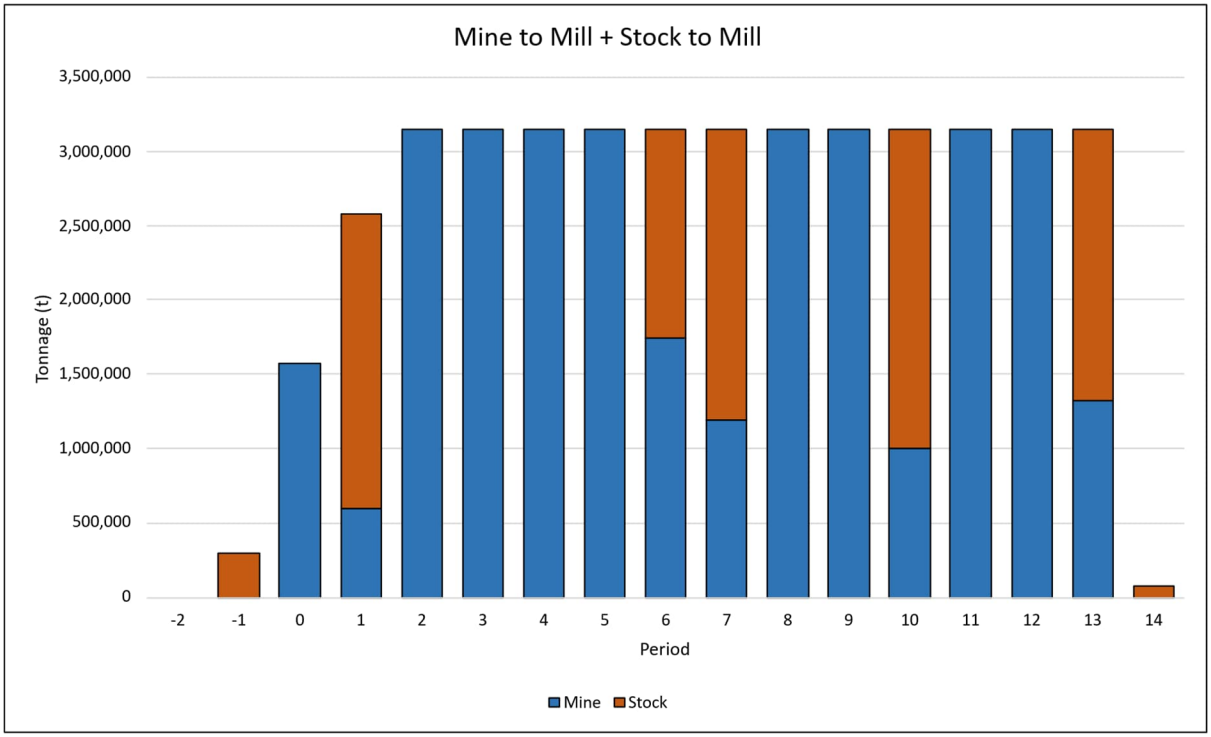
Figure 16-9: Movement Material by Period



Source: AbraSilver Resource Corp., 2024

Figure 16-10: Mined Material by Period

The feed to the process plant is generally sent from the mine. However, there are years in which it will be necessary to stockpile ore to maintain maximum plant capacity. Figure 16-11 shows that in periods 1, 6, 7, 10, 13, and 14 it will be necessary to utilise stockpiles.



Source: AbraSilver Resource Corp., 2024

Figure 16-11: Source of plant feed

Table 16-4 shows total movement per period while Table 16-5 shows the production plan in detail. The ore from the different geometallurgical domains is shown separately due to differing metallurgical recoveries for gold and silver.

Table 16-4: Movement by Period

Period	TANK			Mine to Stock	Mine to Marginal Stockpile	Stock to TANK	Waste Dump	Total Rock
	Tonnage (Mt)	Au (g/t)	Ag (g/t)	Tonnage (Mt)	Tonnage (Mt)	Tonnage (Mt)	Tonnage (Mt)	Tonnage (Mt)
-2	-	-	-	-	0.0	-	12.7	12.8
-1	0.3	0.67	77	0.3	0.6	0.3	22.5	23.6
0	1.6	0.49	90	2.0	2.6	-	17.5	21.7
1	2.6	0.34	229	-	2.0	2.0	19.9	26.5
2	3.1	0.47	204	0.1	3.6	-	18.0	24.8

Period	TANK			Mine to Stock	Mine to Marginal Stockpile	Stock to TANK	Waste Dump	Total Rock
	Tonnage (Mt)	Au (g/t)	Ag (g/t)	Tonnage (Mt)	Tonnage (Mt)	Tonnage (Mt)	Tonnage (Mt)	Tonnage (Mt)
3	3.2	0.85	126	0.6	4.0	-	18.0	25.1
4	3.2	0.58	88	3.1	6.5	-	11.8	21.5
5	3.1	1.23	82	0.8	2.2	-	24.4	29.7
6	3.2	0.50	153	-	2.0	1.4	21.9	28.4
7	3.2	0.61	123	-	1.9	2.0	22.3	29.4
8	3.2	0.58	53	1.5	6.5	-	13.7	23.4
9	3.2	1.33	26	0.1	2.8	-	14.4	20.4
10	3.2	0.64	56	-	2.7	2.1	19.8	27.7
11	3.2	1.01	29	1.0	4.8	-	12.0	20.0
12	3.1	1.35	18	0.4	2.5	-	2.1	7.7
13	3.2	1.15	20	-	1.1	1.8	1.4	7.4
14	0.1	1.24	22	-	-	0.1	0.0	0.2
Total	42.3	0.81	91	9.7	45.8	9.7	252.6	350.4

The "Mine to Marginal Stockpile" column shows material with marginal grade. A sector to the west of Jac will be available for stockpiling, with a capacity of 22 million metric tonnes. The waste rock to ore ratio is 8.28, which could decrease if a secondary heap leach process was to materialize. Stability studies may also increase overall angles on some walls.

Table 16-5: Mine Movement

Material/ Period	-2	-1	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	Total
Ore Total to Plant (Mt)	-	0.30	1.57	2.58	3.15	3.15	3.15	3.15	3.15	3.15	3.15	3.15	3.15	3.15	3.15	3.15	0.08	42.33
Au (g/t)	-	0.67	0.49	0.34	0.47	0.85	0.58	1.23	0.50	0.61	0.58	1.33	0.64	1.01	1.35	1.15	1.24	0.81
Ag(g/t)	-	77	90	229	204	126	88	82	153	123	53	26	56	29	18	20	22	91
Mine to Plant (Mt)	-	-	1.6	0.6	3.2	3.2	3.2	3.2	1.7	1.2	3.2	3.2	1.0	3.2	3.2	1.3	-	32.6
Au (g/t)	-	-	0.49	0.68	0.47	0.85	0.58	1.23	0.27	0.45	0.58	1.33	0.48	1.01	1.35	1.06	-	0.84
Ag(g/t)	-	-	90	55	204	126	88	82	151	107	53	26	49	29	18	15	-	80
Stock to Plant (Mt)	-	0.3	-	2.0	-	-	-	-	1.4	2.0	-	-	2.1	-	-	1.8	0.1	9.7
Au (g/t)	-	0.67	-	0.24	-	-	-	-	0.78	0.71	-	-	0.72	-	-	1.22	1.24	0.72
Ag(g/t)	-	77	-	281	-	-	-	-	155	133	-	-	59	-	-	24	22	127
Ore Mine to Stock (Mt)	-	0.3	2.0	-	0.1	0.6	3.1	0.8	-	-	1.5	0.1	-	1.0	0.4	-	-	9.7
Au (g/t)	-	0.67	0.24	-	0.68	0.35	0.70	0.95	-	-	0.88	0.89	-	1.46	1.04	-	-	0.72
Ag(g/t)	-	77	281	-	31	86	147	99	-	-	32	25	-	20	13	-	-	127
Waste to Waste Dump (Mt)	12.8	23.1	20.1	21.9	21.6	22.0	18.4	26.5	23.9	24.3	20.2	17.3	22.4	16.8	4.6	2.4	-	298.4
SR	-	79.2	14.4	9.3	8.0	8.2	7.9	9.1	8.2	8.3	8.5	6.4	8.0	6.9	2.3	1.1	-	8.1
Total Movement (Mt)	12.8	23.6	23.6	24.5	24.9	25.7	24.6	30.5	27.0	27.4	24.9	20.5	25.6	21.0	8.1	5.6	0.1	350.4

16.4 Waste Dumps

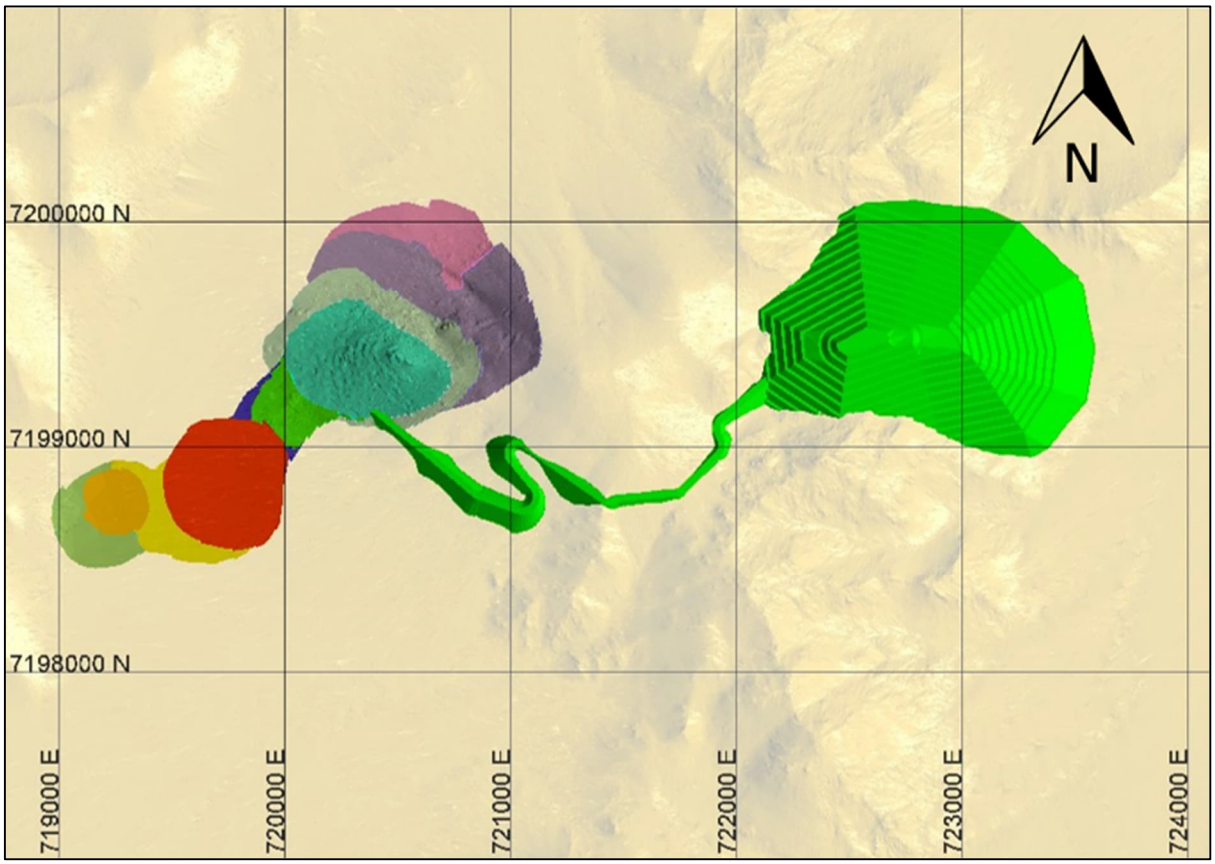
Two waste rock dumps were considered. The main one will be in the southeast sector of the Property while the secondary one will be in the north, after the tailings dam. The mining plan shows a total dump requirement of 298 million metric tons including marginal material. The capacity would be covered with a design of both dumps totalling 326 Mt. Table 16-6 shows the design parameters used.

Table 16-6: Waste Dump Design Parameters

Waste Dump Design Parameter	Value	Unit
Capacity	326	Mt
Hight Lift	20	m
Slope Angle Lift	37	degrees
Overall Slope Angle	27	degrees
Berm Width	40	m
Ramp Width	30	m
Density	1.8	t/m3
Slope Ramp	4 a 10	%

16.4.1 East Main Dump

The East Main Dump has a capacity of 309 million metric tons which assures 100% of the required capacity (Figure 16-2). The road from the mine to this dump will be built during the pre-production stage. 9.7 Mt of waste rock is considered for construction. The main dump will receive waste rock until period 13, filling its capacity to 96%. At a later stage, a stability analysis should be performed.

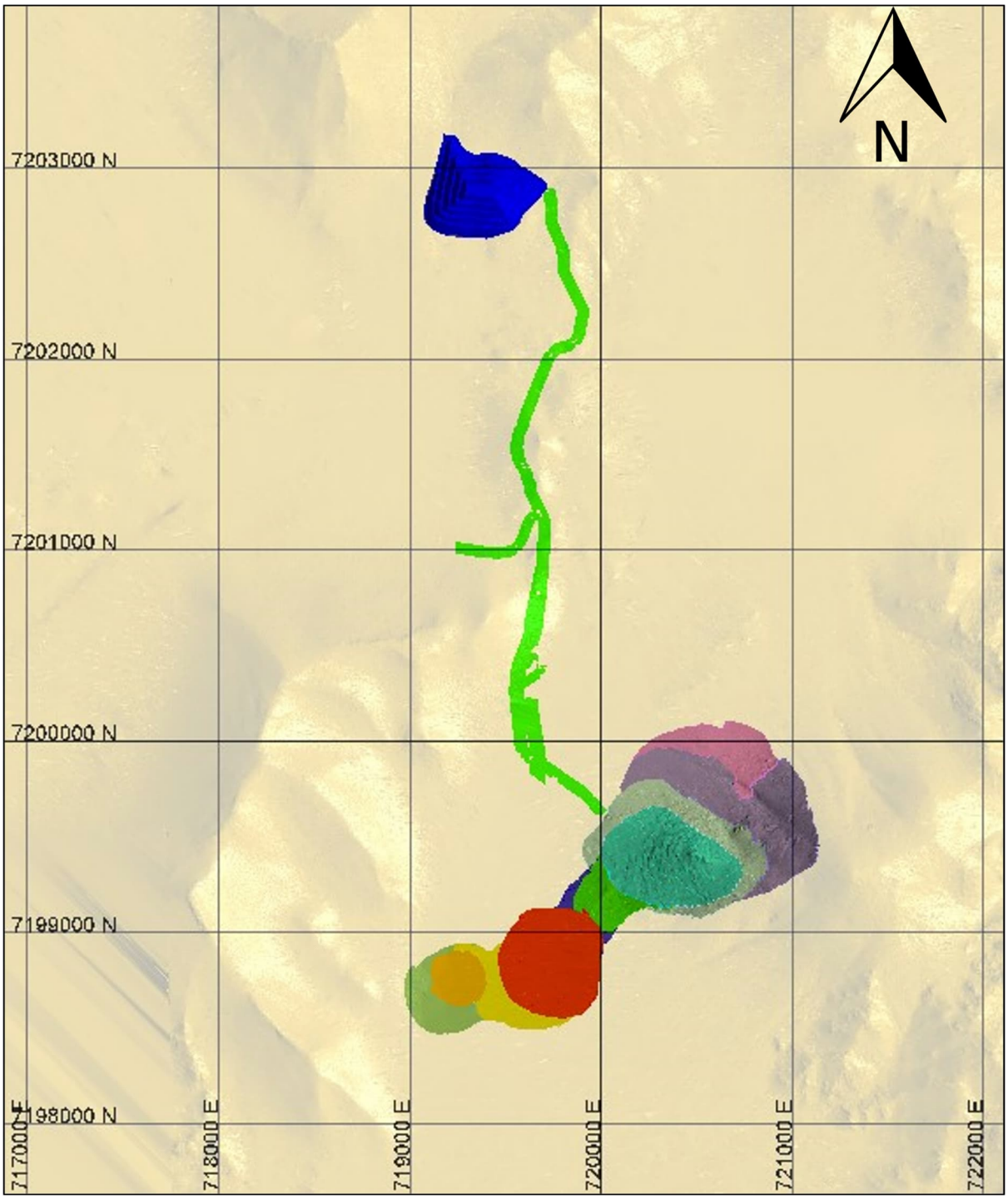


Source: AbraSilver Resource Corp., 2023

Figure 16-12: East Main Waste Dump

16.4.2 North Secondary Waste Dump

The North Secondary Dump has a capacity of 17 Mt, or 3% of the required capacity (Figure 16-13). At a later stage, a stability analysis should be performed.



Source: AbraSilver Resource Corp., 2023

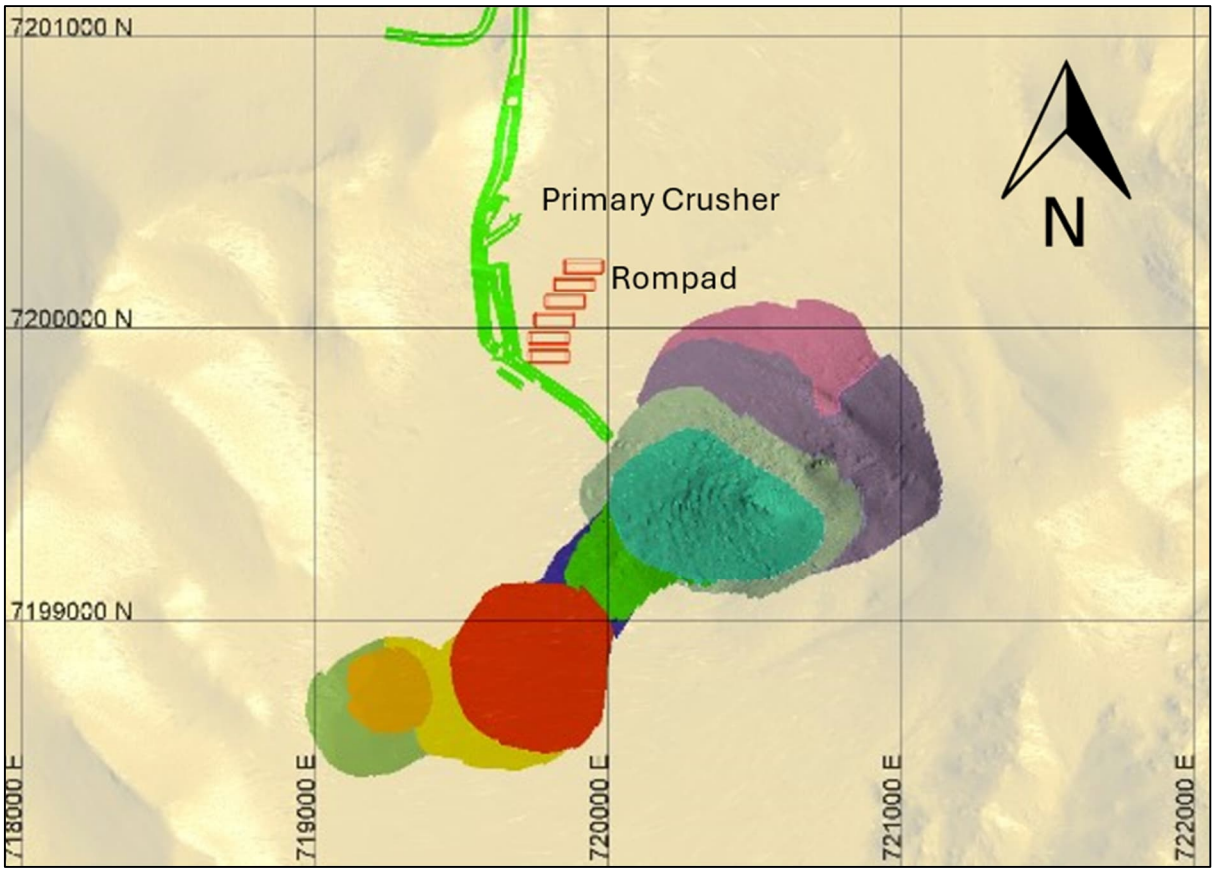
Figure 16-13: North Auxiliary Waste Dump

16.5 Stocks

Two stockpiles are considered, one for stockpiling high-grade ore called "Stock Rom Pad" and the other for stockpiling low grade marginal material called "Stock Marginal".

16.5.1 ROM Pad Stock

This stockpile will be used to stockpile ore to be fed to the tank in leach (TIL) process. Figure 16-14 shows the design considering six 10-meter-high rectangular stockpiles with a capacity of 720,00 tons of ore.

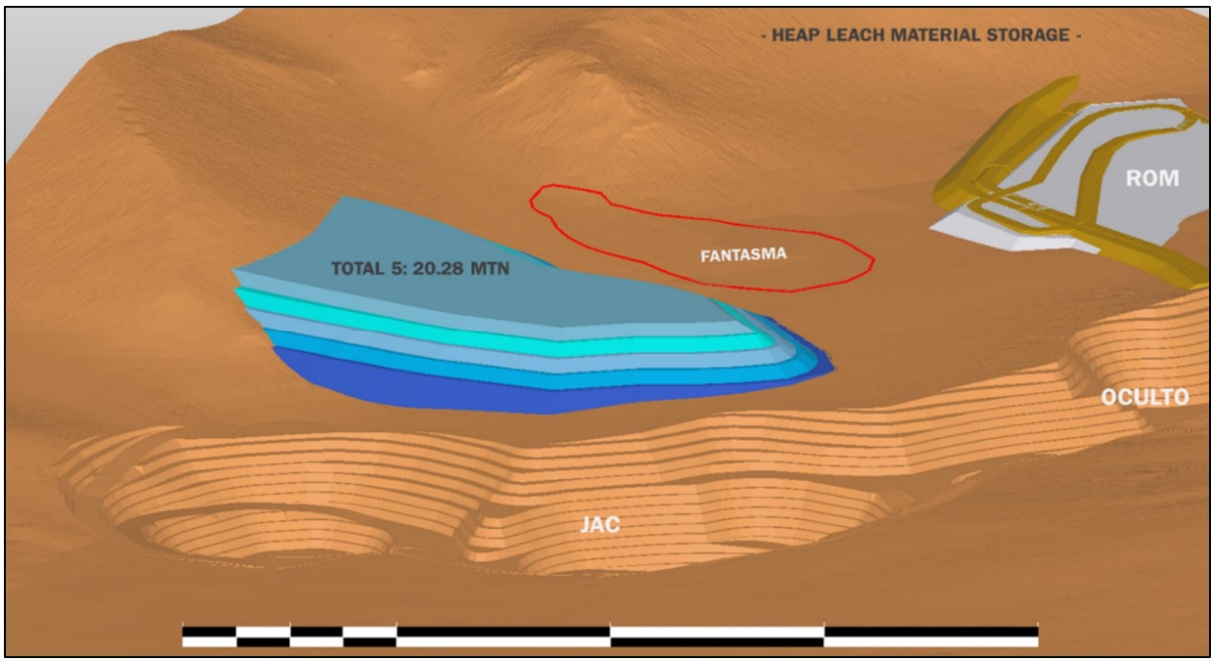


Source: AbraSilver Resource Corp., 2023

Figure 16-14: Stock Rom Pad

16.5.2 Marginal Stock

A stockpile of marginal grade mineralized material will be located in the southwest sector of the Property and the west side of JAC. It will have a total capacity of 22.28 million metric tonnes and this location was chosen so that it will not increase transport distance to the main landfill and will not be far from the crushing plant. The location can be seen in Figure 16-15.



Source: AbraSilver Resource Corp., 2023

Figure 16-15: Stock Marginal

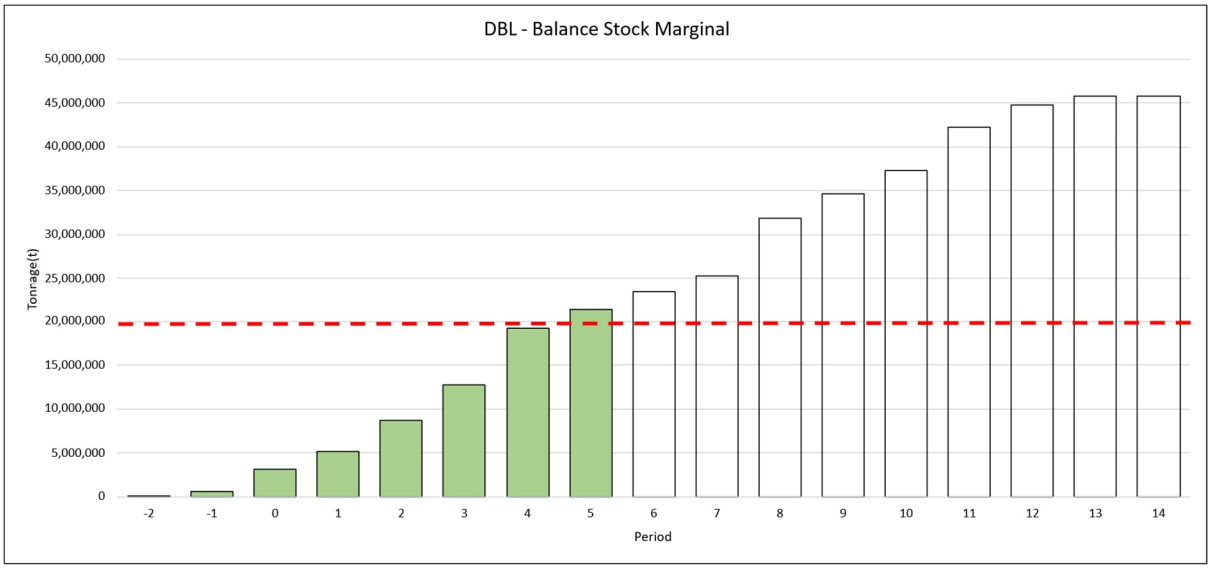
The strategy for moving this stock is to receive marginal material until period 4 or 5, filling its 20 Mt capacity. It is assumed any process strategies will be understood by that time.

The stock balance is shown in Table 16-7 and Figure 16-16. Showing that although from year 6 onwards this material continues to be mined. If a second process methodology began it would no longer require greater stock capacity.

Table 16-7: Balance Marginal Stock

Period	Mine to marginal Stockpile Tonnage (000 t)	Cumulative Marginal Stockpile Tonnage (000 t)
-2	2	2
-1	579	582
0	2,564	3,145
1	2,007	5,152
2	3,594	8,746
3	3,964	12,710
4	6,550	19,260
5	2,159	21,419
6	1,957	23,376
7	1,942	25,317
8	6,509	31,827
9	2,836	34,663

Period	Mine to marginal Stockpile Tonnage (000 t)	Cumulative Marginal Stockpile Tonnage (000 t)
10	2,686	37,349
11	4,823	42,172
12	2,528	44,701
13	1,071	45,772
Total	45,772	



Source: AbraSilver Resource Corp., 2024

Figure 16-16: Need for Marginal Stockpile

16.6 Fleet Estimation

The project schedule indicates that the mine will operate 360 days of the year with two 12-hour shifts. Below is the fleet estimate calculated using haulage roads carried out in MineHaul to determine the flat, uphill, and downhill distances considering origin and destinations. This includes the contour perimeters of each phase and bench to determine the required pre-cut.

The following Table 16-8 shows a summary of the baseline information used to estimate the fleet.

Table 16-8: Input for Fleet Estimation

MINE PLAN AND MAIN REQUIREMENTS

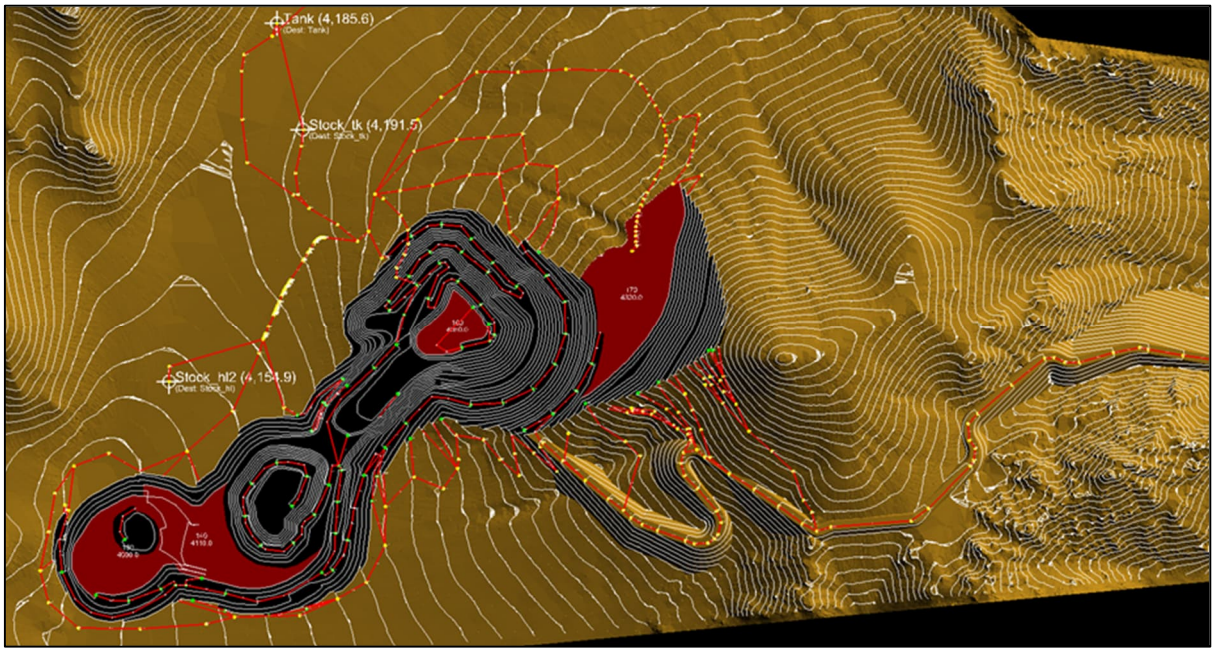
PERIOD	MATERIAL	FROM-TO	TONNAGE (kt)	DISTANCES - 10% (m)	DISTANCES +10% (m)	DISTANCES HZ EXPIT (m)	PERIMETERS
PER-2	Waste	Mina a Dump	2,119.22	1,080.90	2,477.20	1,032.30	6,839.79
PER-2	Waste	Mina a Dump	10,631.95	1,080.90	2,477.20	1,032.30	
PER-2	Ore	Mina a HL	2.19	2,327.30	100.50	1,301.40	
PER-1	Waste	Mina a Dump	12,688.21	1,077.10	2,456.60	1,043.70	15,963.43
PER-1	Waste	Mina a Dump	9,784.14	1,077.10	2,456.60	1,043.70	
PER-1	Ore	Mina a HL	579.37	1,107.50	351.20	829.60	
PER-1	Ore	Mina a Stock	298.32	672.30	664.20	1,020.90	
PER-1	Ore	Stock a Proceso	298.32	136.36	113.77	722.55	
PER00	Waste	Mina a Dump	14,965.12	1,049.40	2,868.90	998.00	13,586.07
PER00	Waste	Mina a Dump	2,552.29	1,049.40	2,868.90	998.00	
PER00	Ore	Mina a HL	2,563.58	828.70	867.20	823.10	
PER00	Ore	Mina a Stock	1,988.17	155.80	1,656.30	1,024.50	
PER00	Ore	Mina a Proceso	1,575.00	652.60	969.80	1,270.90	
PER01	Waste	Mina a Dump	14,237.82	1,085.70	2,429.50	1,100.20	14,136.02
PER01	Waste	Mina a Dump	5,685.34	1,085.70	2,429.50	1,100.20	
PER01	Ore	Mina a HL	2,006.59	632.40	108.00	891.10	
PER01	Ore	Mina a Proceso	592.23	359.80	181.30	1,035.70	
PER01	Ore	Stock a Proceso	1,988.17	136.36	113.77	722.55	
PER02	Waste	Mina a Dump	14,951.91	1,321.90	2,031.60	1,349.60	16,279.56
PER02	Waste	Mina a Dump	3,097.76	1,321.90	2,031.60	1,349.60	
PER02	Ore	Mina a HL	3,594.49	676.50	438.80	863.80	
PER02	Ore	Mina a Stock	72.25	346.60	78.50	774.40	
PER02	Ore	Mina a Proceso	3,150.00	332.10	720.80	1,023.20	
PER03	Waste	Mina a Dump	15,333.44	1,139.50	2,756.00	1,542.80	15,393.46
PER03	Waste	Mina a Dump	2,700.68	1,139.50	2,756.00	1,542.80	
PER03	Ore	Mina a HL	3,964.07	593.10	255.10	981.70	
PER03	Ore	Mina a Stock	571.43	368.70	496.20	829.00	
PER03	Ore	Mina a Proceso	3,150.00	339.90	525.50	1,113.80	
PER04	Waste	Mina a Dump	1,254.00	1,107.70	3,238.60	2,020.40	16,670.97
PER04	Waste	Mina a Dump	10,580.34	1,107.70	3,238.60	2,020.40	
PER04	Ore	Mina a HL	6,549.64	571.30	692.30	920.20	
PER04	Ore	Mina a Stock	3,051.93	315.00	1,139.80	897.60	
PER04	Ore	Mina a Proceso	3,150.00	347.70	626.80	1,044.20	
PER05	Waste	Mina a Dump	19,509.80	1,173.80	2,518.70	1,313.70	17,496.13
PER05	Waste	Mina a Dump	4,879.21	1,173.80	2,518.70	1,313.70	
PER05	Ore	Mina a HL	2,158.95	632.20	1,251.30	942.20	
PER05	Ore	Mina a Stock	756.96	319.10	1,635.00	861.40	
PER05	Ore	Mina a Proceso	3,150.00	349.20	1,299.30	1,069.90	
PER06	Waste	Mina a Dump	20,058.38	1,400.30	3,191.80	1,214.00	18,244.38
PER06	Waste	Mina a Dump	1,858.73	1,400.30	3,191.80	1,214.00	
PER06	Ore	Mina a HL	1,956.73	488.60	948.80	1,476.40	
PER06	Ore	Mina a Proceso	1,739.49	293.80	1,507.20	2,087.50	
PER06	Ore	Stock a Proceso	1,410.51	136.36	113.77	722.55	
PER07	Waste	Mina a Dump	15,987.64	2,135.60	3,285.30	1,998.10	12,048.31
PER07	Waste	Mina a Dump	6,356.59	2,135.60	3,285.30	1,998.10	
PER07	Ore	Mina a HL	1,941.79	758.70	159.50	933.60	
PER07	Ore	Mina a Proceso	1,193.97	502.30	210.70	1,089.90	
PER07	Ore	Stock a Proceso	1,956.03	136.36	113.77	722.55	
PER08	Waste	Mina a Dump	11,556.80	2,032.20	3,678.90	1,955.70	11,739.45
PER08	Waste	Mina a Dump	2,157.13	2,032.20	3,678.90	1,955.70	
PER08	Ore	Mina a HL	6,509.33	696.80	887.90	1,317.30	
PER08	Ore	Mina a Stock	1,546.23	825.40	1,710.50	2,532.90	
PER08	Ore	Mina a Proceso	3,150.00	516.10	960.70	1,517.90	
PER09	Waste	Mina a Dump	11,906.60	2,177.00	3,677.70	2,458.30	10,166.81
PER09	Waste	Mina a Dump	2,541.21	2,177.00	3,677.70	2,458.30	
PER09	Ore	Mina a HL	2,836.37	892.90	1,593.10	2,293.60	
PER09	Ore	Mina a Stock	51.98	825.40	1,710.50	2,532.90	
PER09	Ore	Mina a Proceso	3,150.00	860.60	2,019.30	2,873.10	
PER10	Waste	Mina a Dump	19,539.46	1,923.20	1,946.90	3,062.30	9,577.95
PER10	Waste	Mina a Dump	212.92	1,923.20	1,946.90	3,062.30	
PER10	Ore	Mina a HL	2,685.91	835.90	653.30	1,171.40	
PER10	Ore	Mina a Proceso	1,002.89	531.00	652.00	1,244.50	
PER10	Ore	Stock a Proceso	2,147.11	136.36	113.77	722.55	
PER11	Waste	Mina a Dump	12,016.71	2,141.60	4,630.30	2,688.70	9,171.91
PER11	Ore	Mina a HL	4,823.06	787.70	1,295.50	1,181.50	
PER11	Ore	Mina a Stock	1,010.23	457.20	1,636.50	972.50	
PER11	Ore	Mina a Proceso	3,150.00	516.50	1,303.80	1,288.90	
PER12	Waste	Mina a Dump	2,065.56	2,240.70	5,374.30	2,884.10	3,681.40
PER12	Ore	Mina a HL	2,528.48	787.70	1,868.90	1,232.80	
PER12	Ore	Mina a Stock	355.96	457.20	2,080.30	1,075.10	
PER12	Ore	Mina a Proceso	3,150.00	516.50	1,875.90	1,347.00	
PER13	Waste	Mina a Dump	1,358.67	2,240.70	5,735.20	2,994.50	2,766.14
PER13	Ore	Mina a HL	1,071.39	787.70	2,201.20	1,330.90	
PER13	Ore	Mina a Proceso	1,322.90	516.50	2,201.90	1,439.20	
PER13	Ore	Stock a Proceso	1,827.10	136.36	113.77	722.55	
PER14	Ore	Stock a Proceso	76.23	136.36	113.77	722.55	

16.6.1 Haulage (Ore & Waste)

The mine loading equipment selected for the project is the CAT992G. The Table 16-9 shows the loading equipment requirements for the life of mine.

Simulation of travel times per period was performed using the MineHaul software (Figure 16-17). The software is loaded with operational phase designs, the project production plan, and the characteristics of the equipment (speed vector, capacity). For each phase – bench, the path followed by the truck from inside the mine to its possible destination (dump, stockpile, process plant) is traced until the phase is mined in full.

Performing the process for each phase yields travel times that can be combined with fixed times to determine the cycle time. This estimates the fleet required to execute the production plan.



Source: AbraSilver Resource Corp., 2024

Figure 16-17: Routes in Minehaul

The mine haulage equipment selected for the project is a CAT777D. A business case using autonomous trucks (Autonomous Haulage System, AHS) is intended to be developed in later engineering stages.

Table 16-9 shows the life-of-mine haulage equipment requirements and the parameters used.

Table 16-9: Loading Requirements

LOADERS REQUERIMENT

			PER-2A01 PER-2 A01	PER-1A01 PER-1 A01	PER00A01 PER00 A01	PER01A01 PER01 A01	PER02A01 PER02 A01	PER03A01 PER03 A01	PER04A01 PER04 A01	PER05A01 PER05 A01	PER06A01 PER06 A01	PER07A01 PER07 A01	PER08A01 PER08 A01	PER09A01 PER09 A01	PER10A01 PER10 A01	PER11A01 PER11 A01	PER12A01 PER12 A01	PER13A01 PER13 A01	PER14A01 PER14 A01
Ldr1																			
Operational Hours Req	hrs		14,997	27,458	27,804	26,484	29,241	30,244	28,911	35,813	30,119	29,963	29,303	24,090	27,565	24,694	9,525	2,858	
Efectives Hours Req	hrs		14,997	27,458	27,804	26,484	29,241	30,244	28,911	35,813	30,119	29,963	29,303	24,090	27,565	24,694	9,525	2,858	
Calendar Hours	hrs		8,664	8,640	8,640	8,640	8,664	8,640	8,640	8,640	8,664	8,640	8,640	8,640	8,664	8,640	8,640	8,640	8,664
Mechanical Availability	%		85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
Maintenance	hrs		1,300	1,296	1,296	1,296	1,300	1,296	1,296	1,296	1,300	1,296	1,296	1,296	1,300	1,296	1,296	1,296	1,300
Available Hous	hrs		7,364	7,344	7,344	7,344	7,364	7,344	7,344	7,344	7,364	7,344	7,344	7,344	7,364	7,344	7,344	7,344	7,364
Production Delays	hrs																		
Hours Used	hrs		7,364	7,344	7,344	7,344	7,364	7,344	7,344	7,344	7,364	7,344	7,344	7,344	7,364	7,344	7,344	7,344	7,364
Operational Factor	%		100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%
Utility	%		85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
Times of Inefficiency	hrs		1,105	1,102	1,102	1,102	1,105	1,102	1,102	1,102	1,105	1,102	1,102	1,102	1,105	1,102	1,102	1,102	1,105
Productive Hours	hrs		6,260	6,242	6,242	6,242	6,260	6,242	6,242	6,242	6,260	6,242	6,242	6,242	6,260	6,242	6,242	6,242	6,260
Units Req	un		2.4	4.4	4.5	4.2	4.7	4.8	4.6	5.7	4.8	4.8	4.7	3.9	4.4	4.0	1.53	0.5	
Units On Site	un		3.0	5.0	5.0	5.0	5.0	5.0	5.0	6.0	5.0	5.0	5.0	4.0	5.0	4.0	2.0	1.0	
Smooth	un		3.0	5.0	5.0	5.0	5.0	5.0	5.0	6.0	6.0	6.0	6.0	6.0	6.0	4.0	2.0	1.0	
Usefull Life	hrs																		
Max Equipments	un	6																	

Table 16-10: Hauling Requirements

HAULAGE REQUIREMENTS

			PER-2A01 PER-2 A01	PER-1A01 PER-1 A01	PER00A01 PER00 A01	PER01A01 PER01 A01	PER02A01 PER02 A01	PER03A01 PER03 A01	PER04A01 PER04 A01	PER05A01 PER05 A01	PER06A01 PER06 A01	PER07A01 PER07 A01	PER08A01 PER08 A01	PER09A01 PER09 A01	PER10A01 PER10 A01	PER11A01 PER11 A01	PER12A01 PER12 A01	PER13A01 PER13 A01	PER14A01 PER14 A01
Trk1																			
Operational Hours Req	hrs		92,006	166,155	163,416	157,917	157,165	174,620	160,474	215,806	214,531	251,053	207,204	210,812	209,695	203,404	66,195	40,802	202
Efectives Hours Req	hrs		92,006	166,155	163,416	157,917	157,165	174,620	160,474	215,806	214,531	251,053	207,204	210,812	209,695	203,404	66,195	40,802	202
Total Efectives Hours Req	hrs		92,006	166,155	163,416	157,917	157,165	174,620	160,474	215,806	214,531	251,053	207,204	210,812	208,646	202,387	65,864	40,598	201
Calendar Hours	hrs		8,664	8,640	8,640	8,640	8,664	8,640	8,640	8,640	8,664	8,640	8,640	8,640	8,664	8,640	8,640	8,640	8,664
Mechanical Availability	%		85%	85%	85%	85%	85%	90%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
Maintenance	hrs		1299.6	1296	1296	1296	1299.6	864	1296	1296	1299.6	1296	1296	1296	1299.6	1296	1296	1296	1299.6
Available Hous	hrs		7,364	7,344	7,344	7,344	7,364	7,776	7,344	7,344	7,364	7,344	7,344	7,344	7,364	7,344	7,344	7,344	7,364
Production Delays	hrs																		
Hours Used	hrs		7,364	7,344	7,344	7,344	7,364	7,776	7,344	7,344	7,364	7,344	7,344	7,344	7,364	7,344	7,344	7,344	7,364
Operational Factor	%		100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%
Utility	%		85%	85%	85%	82%	78%	90%	85%	85%	85%	85%	85%	85%	85%	91%	85%	85%	85%
Times of Inefficiency	hrs		1,105	1,102	1,102	1,351	1,605	778	1,102	1,102	1,105	1,102	1,102	1,102	1,105	661	1,102	1,102	1,105
Productive Hours	hrs		6,260	6,242	6,242	5,993	5,759	6,998	6,242	6,242	6,260	6,242	6,242	6,242	6,260	6,683	6,242	6,242	6,260
Units Req	un		14.7	26.6	26.2	26.4	27.3	25.0	25.7	34.6	34.3	40.2	33.2	33.8	33.3	30.3	10.6	6.5	0.0
Units On Site	un		15.0	27.0	27.0	27.0	28.0	25.0	26.0	35.0	35.0	41.0	34.0	34.0	34.0	31.0	11.0	7.0	1.0
Smooth	un		15.0	27.0	27.0	27.0	28.0	25.0	26.0	35.0	35.0	41.0	34.0	34.0	34.0	31.0	11.0	7.0	1.0
Operational Factor	%		85%	85%	85%	82%	78%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
Usefull Life	hrs																		
Max Eq On site	un	41																	

16.6.2 Drilling

The drilling fleet will consist of two types of machines. One for production and another for pre-cutting. For production and contour drilling, the Sandvik DR412i2 drill will be used with a distribution of 70% for production and 30% for contouring. The Sandvik DI650i drill will be used for pre-cutting. The drill and blast assumptions considered can be seen in Table 16-11.

Table 16-11: Drilling Parameters

DRILL & BLAST - OPERATIONAL PARAMETERS

			Production	Buffer	Pre-split
1	**Equipment**		**Prf1**	**Prf2**	**Prf7**
2	Model		DR412i2	DR412i2	DI650i
3	Type		Prod	Prod	Sopo
4	Grid Type		Medio	Medio	PS
5	Profile	%	70%	30%	100%
6	Diámeter	inch	7	7	5
7	Diámeter	mm	177.8	177.8	127.0
8	UGT				
9	Higth Bench	m	10.0	10.0	10.0
10	Sub-Drilling	m	1.0	1.0	
11	Total Length	m	11.0	11.0	10.0
12	Stemming	m	3.5	3.5	3.5
13	Burden	m	7.0	7.0	7.0
14	Spacing	m	8.0	8.0	8.0
16	Drill product	m/hr	23	23	17
17	Hole Volume	m3	560.0	560.0	560.0
18	Rock Volume	m3/m	50.9	50.9	56.0
19	Blasted Volume by Blast	m3	100,000	100,000	50,000
20	Holes by Blast	un	178.6	178.6	89.3
21	Insitu Density	t/m3	2.18	2.18	2.18
22	Rock Type	-			
23	Explosive Type	-			
24	Explosive Density	gr/m3	1,300,000	1,300,000	
81					
82	Drills Operational Parameters				
83	Avalability	%	80%	80%	75%
84	Utitlity	%	75%	70%	100%
85	Operational Factor	%	100%	100%	50%
87	Usefull Life Equipment	hr	60,000	60,000	60,000
88	Diesel Consume	l/hr	107.00	107.00	105.00

Table 16-12, Table 16-13 and Table 16-14 show the drilling equipment requirements per period in line with requirements for production, contouring, and pre-cutting. During the -2-pre-production period, no drilling equipment is required because all the excavated material is cover.

Table 16-12: Production Drilling Requirements

DRILL & BLAST REQUIREMENT

			PER-2A01 PER-2 A01	PER-1A01 PER-1 A01	PER00A01 PER00 A01	PER01A01 PER01 A01	PER02A01 PER02 A01	PER03A01 PER03 A01	PER04A01 PER04 A01	PER05A01 PER05 A01	PER06A01 PER06 A01	PER07A01 PER07 A01	PER08A01 PER08 A01	PER09A01 PER09 A01	PER10A01 PER10 A01	PER11A01 PER11 A01	PER12A01 PER12 A01	PER13A01 PER13 A01
Medium Type Rock	kt		2,121	13,566	21,092	16,837	21,769	23,019	14,006	25,576	23,755	19,123	22,762	17,945	23,228	21,000	8,100	3,753
Production Drill																		
Operational Hours Req	hrs		579	3,703	5,758	4,596	5,942	6,284	3,823	6,982	6,485	5,220	6,214	4,899	6,341	5,733	2,211	1,024
Efective Hours Req	hrs		579	3,703	5,758	4,596	5,942	6,284	3,823	6,982	6,485	5,220	6,214	4,899	6,341	5,733	2,211	1,024
Drilled	m		13,319	85,174	132,426	105,710	136,675	144,525	87,935	160,578	149,144	120,067	142,914	112,668	145,840	131,849	50,856	23,563
Holes	un		1,211	7,743	12,039	9,610	12,425	13,139	7,994	14,598	13,559	10,915	12,992	10,243	13,258	11,986	4,623	2,142
Tonnage	kt		1,485	9,496	14,764	11,786	15,238	16,113	9,804	17,903	16,628	13,386	15,934	12,561	16,260	14,700	5,670	2,627
Calendar Hours	hrs		8,664	8,640	8,640	8,640	8,664	8,640	8,640	8,640	8,664	8,640	8,640	8,640	8,664	8,640	8,640	8,640
Mechanical Availability	%		80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%
Maintenance	hrs		1,733	1,728	1,728	1,728	1,733	1,728	1,728	1,728	1,733	1,728	1,728	1,728	1,733	1,728	1,728	1,728
Available Hours	hrs		6,931	6,912	6,912	6,912	6,931	6,912	6,912	6,912	6,931	6,912	6,912	6,912	6,931	6,912	6,912	6,912
Production Delays	hrs		1,733	1,728	1,728	1,728	1,733	1,728	1,728	1,728	1,733	1,728	1,728	1,728	1,733	1,728	1,728	1,728
Used Hours	hrs		5,198	5,184	5,184	5,184	5,198	5,184	5,184	5,184	5,198	5,184	5,184	5,184	5,198	5,184	5,184	5,184
Utility	%		75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%
Operational Factor	%		100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%
Productives Hours	hrs		5,198	5,184	5,184	5,184	5,198	5,184	5,184	5,184	5,198	5,184	5,184	5,184	5,198	5,184	5,184	5,184
Units Req	un		0.1	0.7	1.1	0.9	1.1	1.2	0.7	1.3	1.2	1.0	1.2	0.9	1.2	1.1	0.4	0.2
On Site Units	un		1.0	1.0	2.0	1.0	2.0	2.0	1.0	2.0	2.0	2.0	2.0	1.0	2.0	2.0	1.0	1.0
Smoot	un		1.0	1.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0	1.0
Usefull Life	hrs	60,000																
Max Eq	un	2																
In/Out Eq																		
1	un		1															
Usefull Life by Eq																		
1	hrs		579	4,282	7,161	9,459	12,430	15,572	17,484	20,975	24,217	26,827	29,934	32,383	35,554	38,420	40,631	41,656
Start Mechanical Avalabilit	%	80%																
Operative Units		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
In/Out		1	1															

Table 16-13: Buffer Drilling Requirements

DRILL & BLAST REQUIREMENT

			PER-2A01 PER-2 A01	PER-1A01 PER-1 A01	PER00A01 PER00 A01	PER01A01 PER01 A01	PER02A01 PER02 A01	PER03A01 PER03 A01	PER04A01 PER04 A01	PER05A01 PER05 A01	PER06A01 PER06 A01	PER07A01 PER07 A01	PER08A01 PER08 A01	PER09A01 PER09 A01	PER10A01 PER10 A01	PER11A01 PER11 A01	PER12A01 PER12 A01	PER13A01 PER13 A01
Medium Type Rock	kt		2,121	13,566	21,092	16,837	21,769	23,019	14,006	25,576	23,755	19,123	22,762	17,945	23,228	21,000	8,100	3,753
Buffer Drill																		
Operational Hours Req	hrs		248	1,587	2,468	1,970	2,547	2,693	1,639	2,992	2,779	2,237	2,663	2,099	2,718	2,457	948	439
Efective Hours Req	hrs		248	1,587	2,468	1,970	2,547	2,693	1,639	2,992	2,779	2,237	2,663	2,099	2,718	2,457	948	439
Drilled	m		5,708	36,503	56,754	45,304	58,575	61,939	37,686	68,819	63,919	51,457	61,249	48,286	62,503	56,507	21,795	10,098
Holes	un		519	3,318	5,159	4,119	5,325	5,631	3,426	6,256	5,811	4,678	5,568	4,390	5,682	5,137	1,981	918
Tonnage	kt		636	4,070	6,328	5,051	6,531	6,906	4,202	7,673	7,126	5,737	6,829	5,383	6,968	6,300	2,430	1,126
Calendar Hours	hrs		8,664	8,640	8,640	8,640	8,664	8,640	8,640	8,640	8,664	8,640	8,640	8,640	8,664	8,640	8,640	8,640
Mechanical Availability	%		80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%
Maintenance	hrs		1,733	1,728	1,728	1,728	1,733	1,728	1,728	1,728	1,733	1,728	1,728	1,728	1,733	1,728	1,728	1,728
Available Hours	hrs		6,931	6,912	6,912	6,912	6,931	6,912	6,912	6,912	6,931	6,912	6,912	6,912	6,931	6,912	6,912	6,912
Production Delays	hrs		2,079	2,074	2,074	2,074	2,079	2,074	2,074	2,074	2,079	2,074	2,074	2,074	2,079	2,074	2,074	2,074
Used Hours	hrs		4,852	4,838	4,838	4,838	4,852	4,838	4,838	4,838	4,852	4,838	4,838	4,838	4,852	4,838	4,838	4,838
Utility	%		70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%
Operational Factor	%		100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%
Productives Hours	hrs		4,852	4,838	4,838	4,838	4,852	4,838	4,838	4,838	4,852	4,838	4,838	4,838	4,852	4,838	4,838	4,838
Units Req	un		0.1	0.3	0.5	0.4	0.5	0.6	0.3	0.6	0.6	0.5	0.6	0.4	0.6	0.5	0.2	0.1
On Site Units	un		1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Smoot	un		1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Usefull Life	hrs	60,000																
Max Eq	un	1																
In/Out Eq																		
1	un		1															
Usefull Life by Eq																		
1	hrs		248	1,835	4,303	6,273	8,819	11,512	13,151	16,143	18,922	21,159	23,822	25,922	28,639	31,096	32,044	32,483
Start Mechanical Avalabilit	%	80%																
Disponibilidad Mecánica por Equipo																		
	%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%
Disponibilidad Mecánica por Flota	%		80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%
Operative Units		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
In/Out		1	1															

Table 16-14: Pre-Split Drilling Requirements

DRILL & BLAST REQUIREMENT

			PER-2A01 PER-2 A01	PER-1A01 PER-1 A01	PER00A01 PER00 A01	PER01A01 PER01 A01	PER02A01 PER02 A01	PER03A01 PER03 A01	PER04A01 PER04 A01	PER05A01 PER05 A01	PER06A01 PER06 A01	PER07A01 PER07 A01	PER08A01 PER08 A01	PER09A01 PER09 A01	PER10A01 PER10 A01	PER11A01 PER11 A01	PER12A01 PER12 A01	PER13A01 PER13 A01
Medium Type Rock	kt		2,121	13,566	21,092	16,837	21,769	23,019	14,006	25,576	23,755	19,123	22,762	17,945	23,228	21,000	8,100	3,753
Pre Split Drill																		
Operational Hours Req	hrs		4,023	9,390	7,992	8,315	9,576	9,055	9,806	10,292	10,732	7,087	6,906	5,980	5,634	5,395	2,166	1,627
Efective Hours Req	hrs		2,012	4,695	3,996	4,158	4,788	4,527	4,903	5,146	5,366	3,544	3,453	2,990	2,817	2,698	1,083	814
Drilled	m		34,199	79,817	67,930	70,680	81,398	76,967	83,355	87,481	91,222	60,242	58,697	50,834	47,890	45,860	18,407	13,831
Holes	un		3,420	7,982	6,793	7,068	8,140	7,697	8,335	8,748	9,122	6,024	5,870	5,083	4,789	4,586	1,841	1,383
Tonnage	kt																	
Calendar Hours	hrs		8,664	8,640	8,640	8,640	8,664	8,640	8,640	8,640	8,664	8,640	8,640	8,640	8,664	8,640	8,640	8,640
Mechanical Availability	%		75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%
Maintenance	hrs		2,166	2,160	2,160	2,160	2,166	2,160	2,160	2,160	2,166	2,160	2,160	2,160	2,166	2,160	2,160	2,160
Available Hours	hrs		6,498	6,480	6,480	6,480	6,498	6,480	6,480	6,480	6,498	6,480	6,480	6,480	6,498	6,480	6,480	6,480
Production Delays	hrs																	
Used Hours	hrs		6,498	6,480	6,480	6,480	6,498	6,480	6,480	6,480	6,498	6,480	6,480	6,480	6,498	6,480	6,480	6,480
Utility	%		100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%
Operational Factor	%		50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%
Tiempo de Ineficiencia	hrs		3,249	3,240	3,240	3,240	3,249	3,240	3,240	3,240	3,249	3,240	3,240	3,240	3,249	3,240	3,240	3,240
Productives Hours	hrs		3,249	3,240	3,240	3,240	3,249	3,240	3,240	3,240	3,249	3,240	3,240	3,240	3,249	3,240	3,240	3,240
Units Req	un		0.6	1.4	1.2	1.3	1.5	1.4	1.5	1.6	1.7	1.1	1.1	0.9	0.9	0.8	0.3	0.3
On Site Units	un		1.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0	1.0	1.0	1.0	1.0
Smoot	un		1.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0	1.0	1.0	1.0	1.0
Usefull Life	hrs	60,000																
Max Eq	un	2																
In/Out Eq																		
1	un		1								-1							
2	un		1															
Usefull Life by Eq																		
1	hrs	18,000	21,249	25,944	29,940	34,098	38,886	43,413	48,316	53,462								
2	hrs	18,000	21,249	25,944	29,940	34,098	38,886	43,413	48,316	53,462	58,828	62,372	65,825	71,805	77,439	82,835	85,000	86,627
Start Mechanical Avalabilit	%	75%																
Desgaste Mecánico Anual	%																	
Disponibilidad Mecánica por Equipo																		
1	%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%							
2	%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%
Disponibilidad Mecánica por Flota	%		75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%
Operative Units In/Out		2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1	1

16.6.3 Support Services

Support equipment consists of all equipment required for the mining operation. Bulldozers or crawler tractors provide the services of unclogging blasting, maintaining an adequate unloading point and cutting in non-rock terrain such as cover. The wheel dozer performs similar functions to the bulldozer but has the advantage of traveling at higher speeds without the need for a low bed. Excavators will play a fundamental role in the operation since they are in charge of profiling slopes to reach design lines and avoid rock falls. The motor grader performs the function of maintaining an optimal road surface for haulage. It will work-in conjunction with the roller and water truck. A CAT950 loader was considered to eventually feed from the rom pad stock to the primary crusher. Finally, tire handler and lowboy trucks are considered to move tracked equipment.

The Caterpillar line was considered for support and production support due to its massive use and good availability in Argentina. The estimation criteria for each support team are shown in Table 16-15.

Table 16-15: Support Equipment Criterion

Equipment	Model	Criterion
Bulldozer	CAT D8	0.5 equipment by active phase + 1 equipment by active waste dump.
Wheeldozer	CAT 834T	0.5 equipment by active phase
Motorgrader	CAT 160H	1 equipment each 10 trucks
Water Truck	CAT 777WT	2 equipments
Excavator	CAT 374	0.5 equipment y active loader.
Loader	CAT 950	1 equipment
Tire Handler	CAT TM	1 equipment
Lowboy Truck	Cama Baja	1 equipment
Compactor Roller	CAT CS79	1 equipment

16.7 Mine Personnel

The staffing required for the mining operation was estimated based on the fleet estimation. An absenteeism factor of 25% was considered only for operators; this factor was not applied to staff personnel. Detailed staffing numbers can be found below in Table 16-16.

Table 16-16: Required Mine Staffing

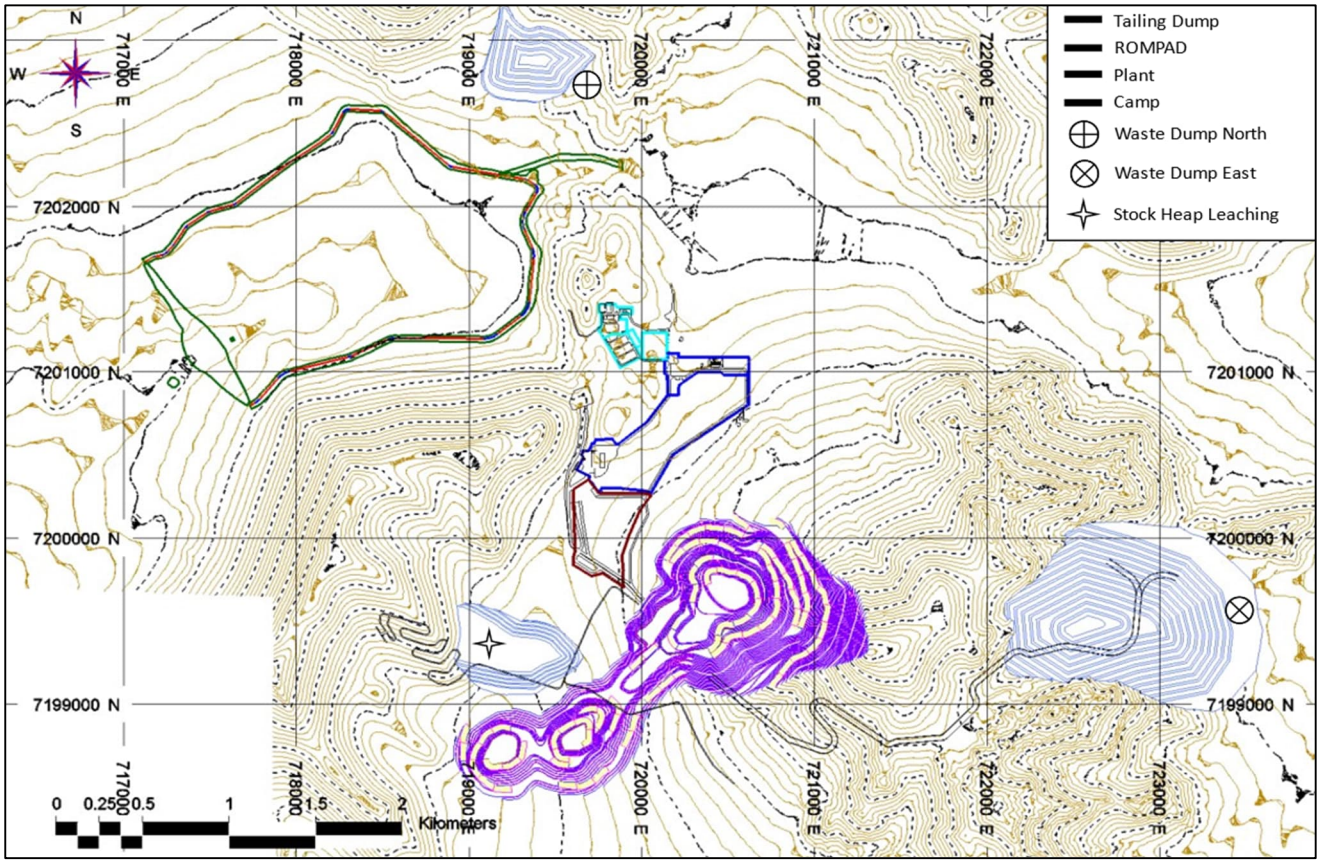
DIABLILLOS SILVER - GOLD PROJECT

INVESTMENT COST ESTIMATION BREAKDOWN FOR OPERATIONAL EXPENDITURE (OPEX) - 9.000 ton/p.d. PLANT THROUGHPUT

DESCRIPTION	UNIT	STAFFING (PAYROLL) PER YEAR																
		CONSTRUCTION		COMMISSIONING RAMP-UP	OPERATIONS													
		-2	-1	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14
HUMAN RESOURCES																		
Mining		115	157	158	166	166	166	166	166	166	166	166	166	166	166	166	166	166
Mine Superintendant	Staff/year	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Deputy Mine Superintendant	Staff/year	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Drilling Shift Supervisor	Staff/year	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Drilling - Operators	Staff/year	4	6	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Drilling - Assistance, helpers	Staff/year	12	18	24	24	24	24	24	24	24	24	24	24	24	24	24	24	24
Blasting - Shift Supervisor	Staff/year	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Blasting - Operators	Staff/year	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Blasting - ANFO truck driver	Staff/year	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Blasting - Powder magazine supervisor	Staff/year	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Blasting - Nitrate preparation field - Operator	Staff/year	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Blasting - Nitrate preparation field - Helper	Staff/year	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Fleet management - Fleet dispatcher	Staff/year	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Excavation & Hauling - Shift Supervisor	Staff/year	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Excavation & Hauling - Excavator - Operator	Staff/year	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Excavation & Hauling - Front end loader - Operator	Staff/year	6	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10
Excavation & Hauling - 100 ton Hauling truck driver	Staff/year	30	54	54	54	54	54	54	54	54	54	54	54	54	54	54	54	54
Excavation & Hauling - Wheel Bulldozer Operator	Staff/year	4	4	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Excavation & Hauling - Bulldozer Operator	Staff/year	8	8	6	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Waste & Ore rehandling - Shift Supervisor	Staff/year	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Waste & Ore rehandling - Bulldozer Operator	Staff/year	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Waste & Ore rehandling - Front end loader Operator	Staff/year	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Mine support & maintenance - Grader - Operator	Staff/year	4	8	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Mine support & maintenance - Front end loader - Operator	Staff/year	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine support & maintenance - Backhoe - Operator	Staff/year	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine support & maintenance - Bobcat Operator	Staff/year	2	2	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine support & maintenance - Fuel truck driver	Staff/year	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine support & maintenance - 100 ton Water truck driver	Staff/year	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Mine support & maintenance - 26 ton Water truck driver	Staff/year	2	2	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Mine support & maintenance - Low boy truck driver	Staff/year	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

16.8 General Mine Layout

Figure 16-18 shows the general layout of the main mine infrastructure. Highlighting is the TSF (Tailings Storage Facility), ROM pad, process plant, camp, east and north dumps and marginal material stockpile.



Source: AbraSilver Resource Corp., 2023

Figure 16-18: General Mine Layout

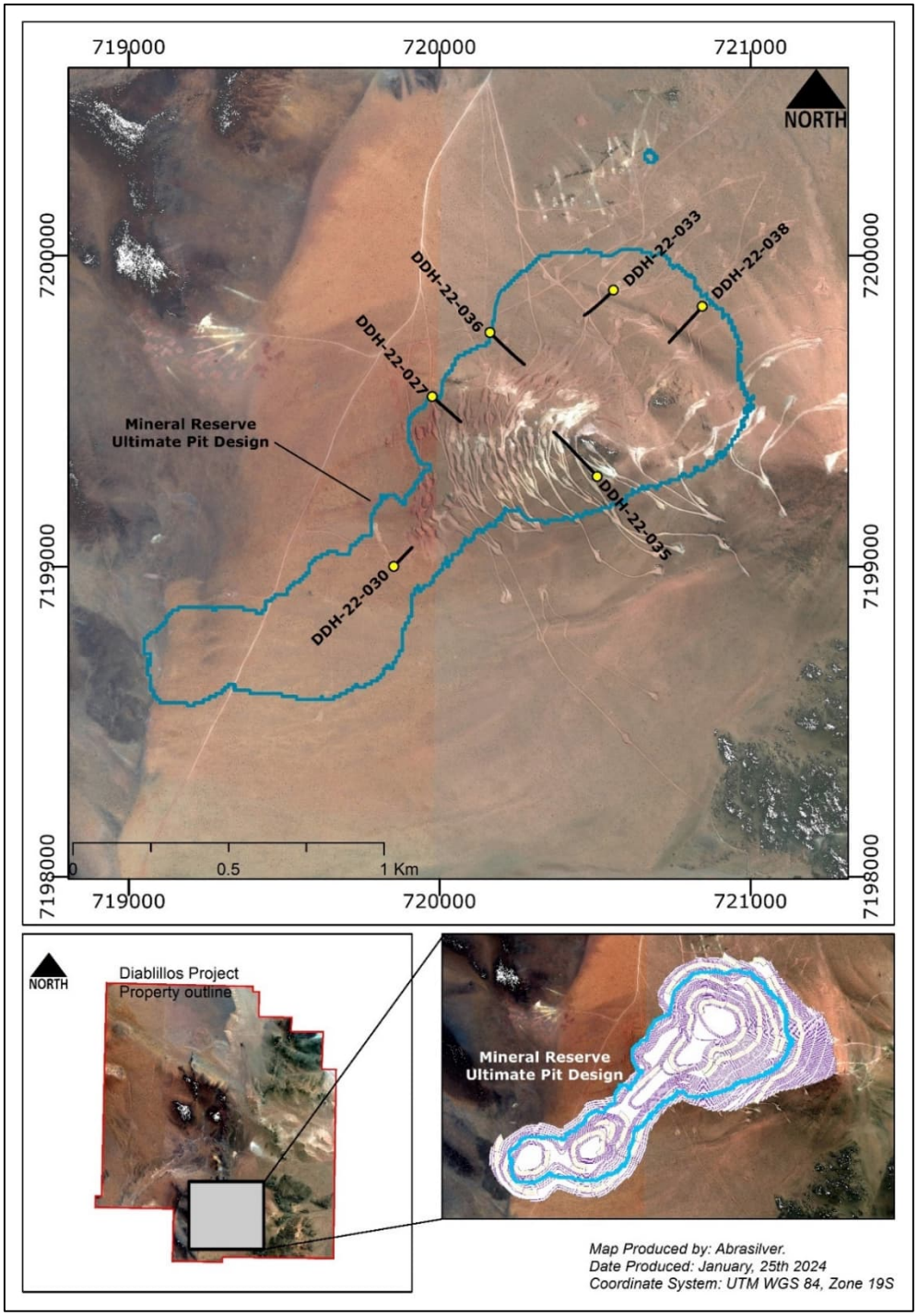
16.9 Stability Open Pit Analysis

During the 2022 drilling campaign, AbraSilver drilled 6 geotechnical wells as part of its exploration campaign. Six core-oriented HQ size holes were developed and geotechnically logged to support future mine designs. During December 2023 and January 2024, the author processed this background information with the assistance of a geomechanical specialist. The results and conclusions are presented in the following subsection.

16.9.1 Geotechnical Units

Geotechnical information provided by lithology allowed the definition of 6 Geotechnical Units (GU). These have been used for stability analysis.

The Rock Mass Rating (RMR) for each lithology was determined with the data provided by borehole logging every meter. The location of the boreholes is shown in Figure 16-19 and the RMR established by lithology that each drillhole intercepts are shown in Figure 16-20 to Figure 16-22.



Source: AbraSilver Resource Corp., 2023

Figure 16-19: Location of Boreholes

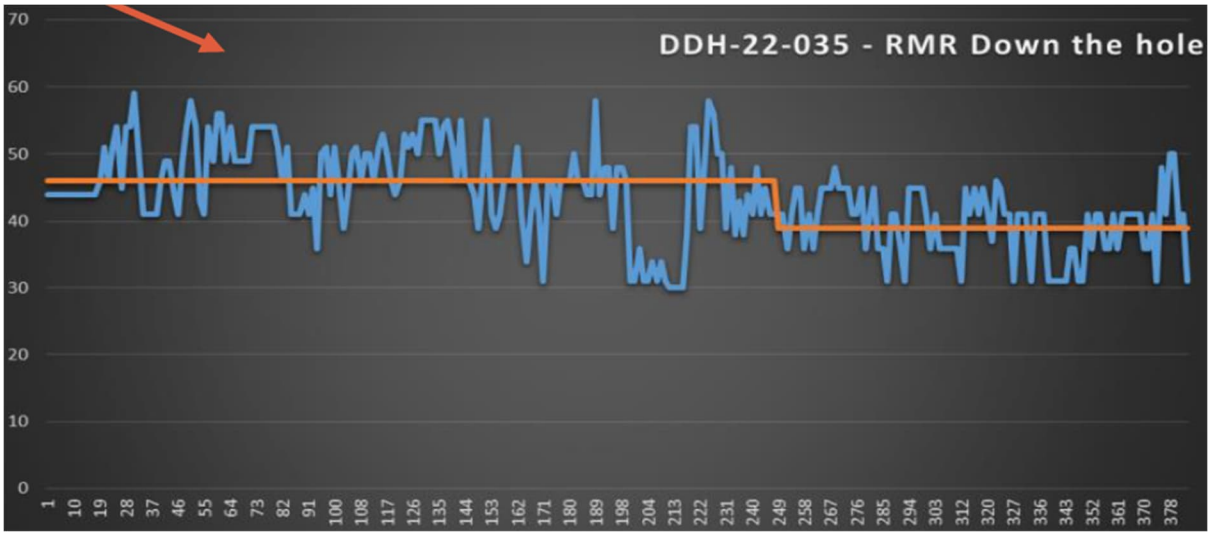


Figure 16-20: RMR According to Depth for Metasediment Lithology

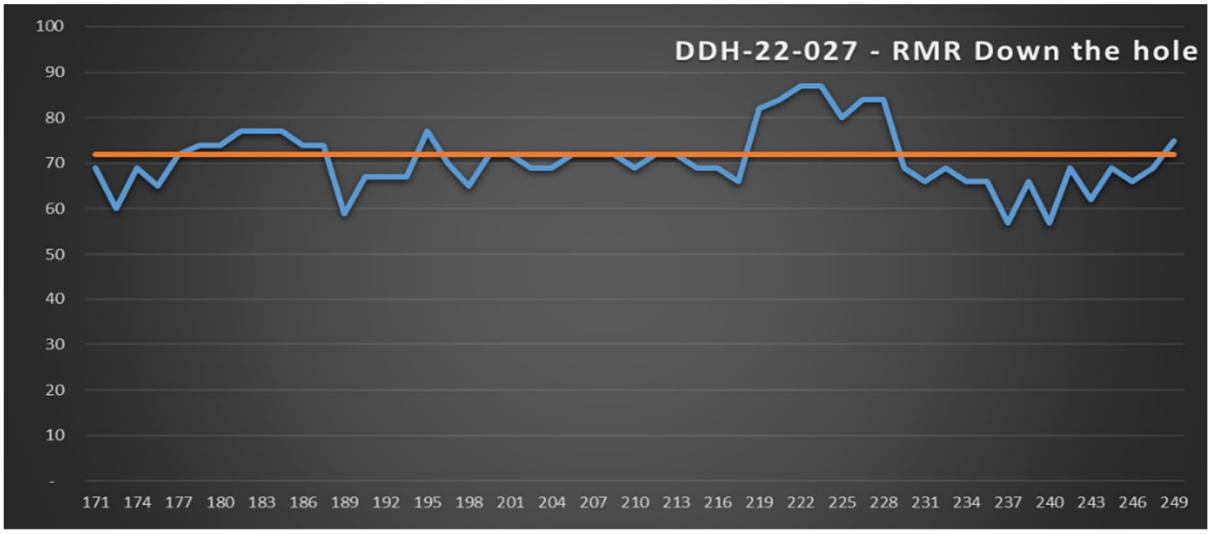
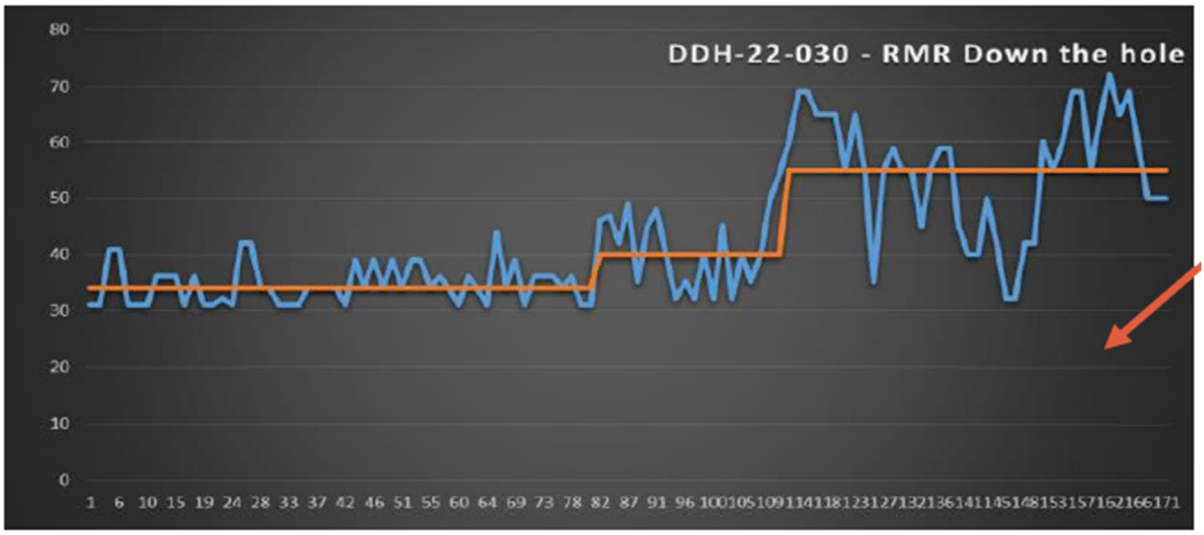


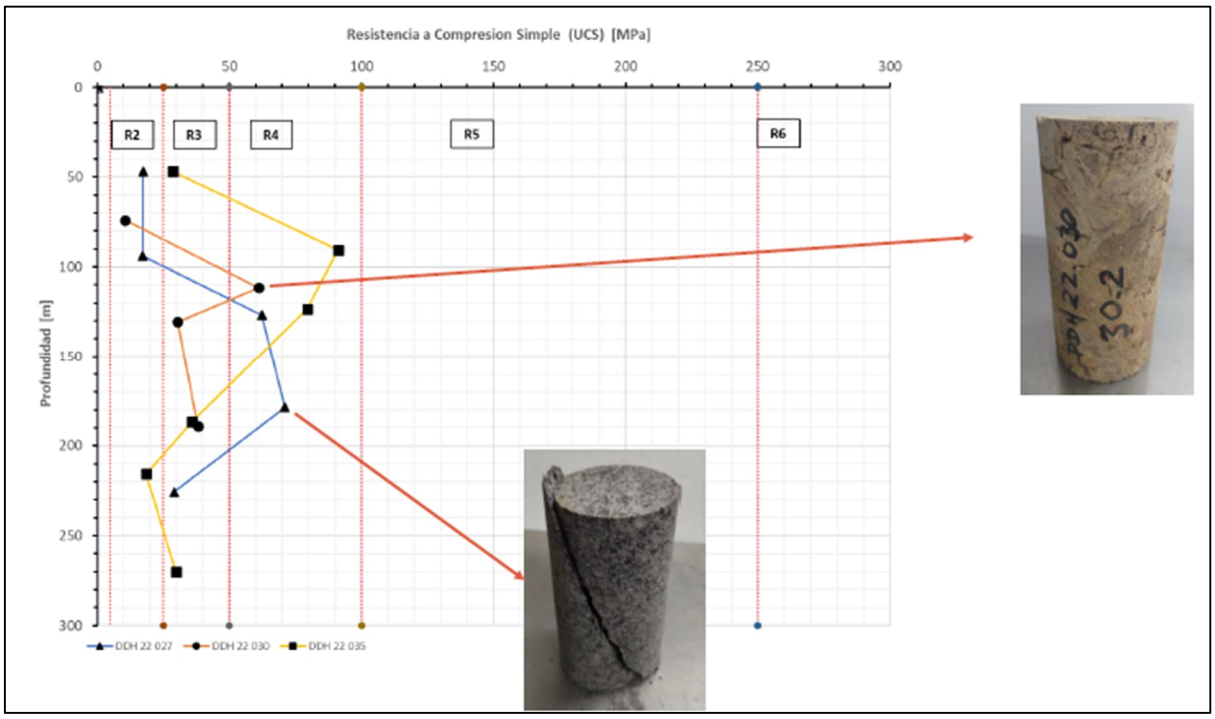
Figure 16-21: RMR According to Depth for Granite Lithology



Source: AbraSilver Resource Corp., 2023

Figure 16-22: According to Depth for Wall Rock-Andesite Lithology

The UCS at different depths by lithology was also determined from the data provided by the boreholes, see Figure 16-23.



Source: AbraSilver Resource Corp., 2023

Figure 16-23: UCS at Different Depths

The following parameters are obtained from the above information. The parameter for each type of rock present in the study area is shown in Table 16-17:

Table 16-17: Geotechnical Parameters by Lithology

Lithology	Depth [m]	UCS [Mpa]	mi	RMR	GSI
Metasediment	50	26	20	47	42
	100	85	20	47	42
	150	80	20	47	42
	200	30	20	47	42
	250	25	20	39	34
	300	25	20	39	34
	350	25	20	39	34
	400	25	20	39	34
	450	25	20	39	34
	Typical	25	20	43	40
Granite	50	27	19	71	66
	100	27	19	71	66
	150	27	19	71	66
	200	27	19	71	66
	250	27	19	71	66
	Typical	27	19	52	47
Andesite	50	18	6	34	29
	100	18	6	40	35
	150	67	6	55	50
	200	50	6	55	50
	230	30	6	55	50
	Typical	67	6	48	43

According to the model provided, there is a layer of surface material composed of semi-consolidated gravel-type backfill. The typical properties for this type of material C= 1.8 MPa and φ=12°.

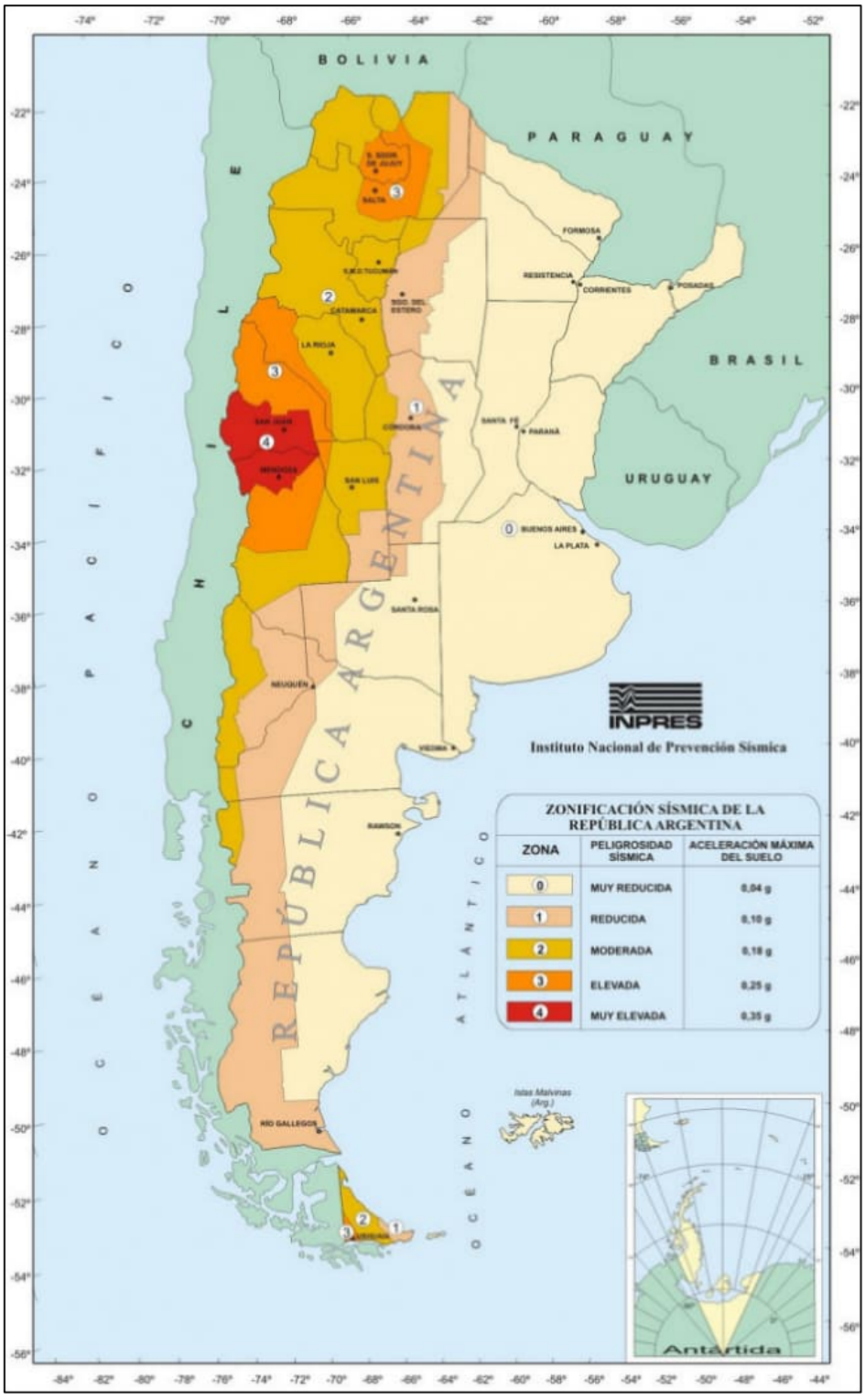
Given the uncertainty of the available geotechnical data, a 40% variation in the established parameters will be assumed. This translates into a coefficient of variation of 15%. This condition will be considered in failure probability analysis for the different scenarios simulated below.

The National Institute for Seismic Prevention (INPRES) of the Ministry of Public Works of Argentina proposes a seismic zoning. This is associated with seismic hazard and places the deposit in moderate seismic hazard “Zone 2” (Figure 16-24). There are no destructive earthquakes with a Mercalli intensity of at least VI in the historical seismic record for the years 1692–2021.

The events recorded in this region correspond to intraplate earthquakes. Caused by rupture within the South American tectonic plate from thrust produced by the subduction of the Nazca plate from the west.

The affected area corresponds to the Metán basin. A tectonic depression filled by sediments and volcanites since the Lower Cretaceous and structured by faults and folds. The seismogenic potential of these faults is evidenced in the morphologies of the terrain as fault scarps (steps in the surface). This affects alluvial levels of Quaternary age deposits. These structures are linked to growth of the easternmost sierras of the Andes (Santa Barbara System), forming the active orogenic front of the mountain range. The earthquakes that have occurred respond to the reactivation of some of these faults or other associated structures with little expression in the terrain.

Considering these conditions, a preliminary static stability analysis for the pit walls of the project is sufficient.



Source: INPRES

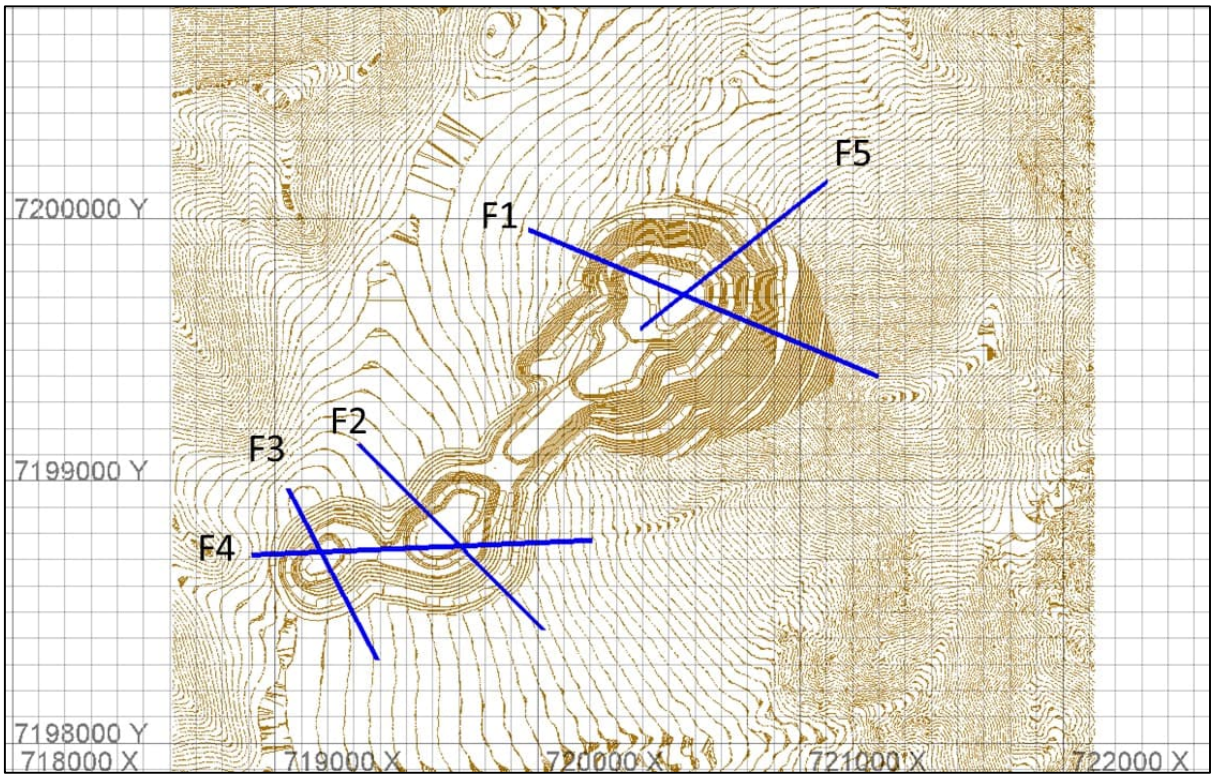
Figure 16-24: Argentine Seismic Zoning

16.9.2 Diablillos Final Pit Stability

The stability analysis was performed using Slide software, which allows us to obtain the static stability of the walls that comprise an open pit mine. but does not allow us to simulate the dynamic behaviour of the design. By using this methodology, stability factors and the possibility of failure that may exist in the field can be established.

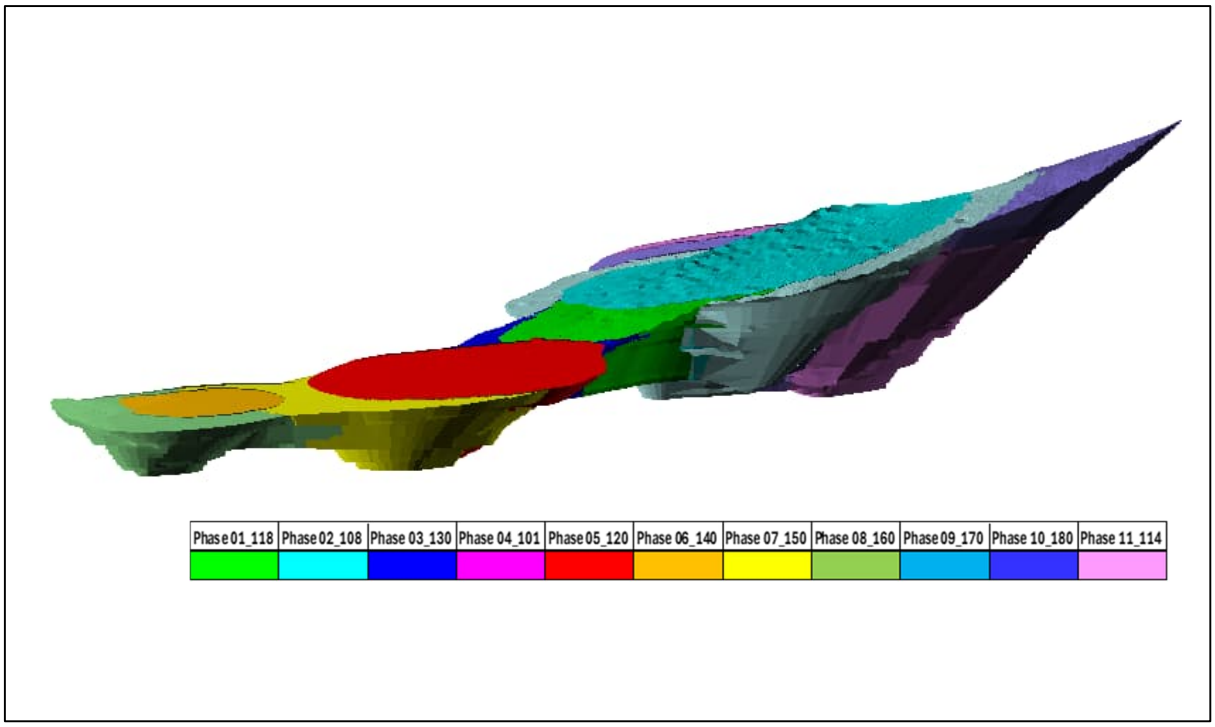
The Diablillos project consists of 10 mining phases, starting in the centre of the deposit, continuing an expansion towards the northeast and southwest, ending later with the high phases in the northeast sector as a final wall. Given this growth and individualization of the phases, the ultimate pit stability can be defined by combining the walls of F04_101, F05_120, F06_140, F07_150, F08_160, F09_170 and F10_180, F11_114, as shown in the figure below.

Five profiles were established on which the stability analysis of the final pit geometry was performed, as shown in Figure 16-25 and Figure 16-26.



Source: AbraSilver Resource Corp., 2024

Figure 16-25: Sections for Stability Analysis



Source: AbraSilver Resource Corp., 2023

Figure 16-26: Final Pit

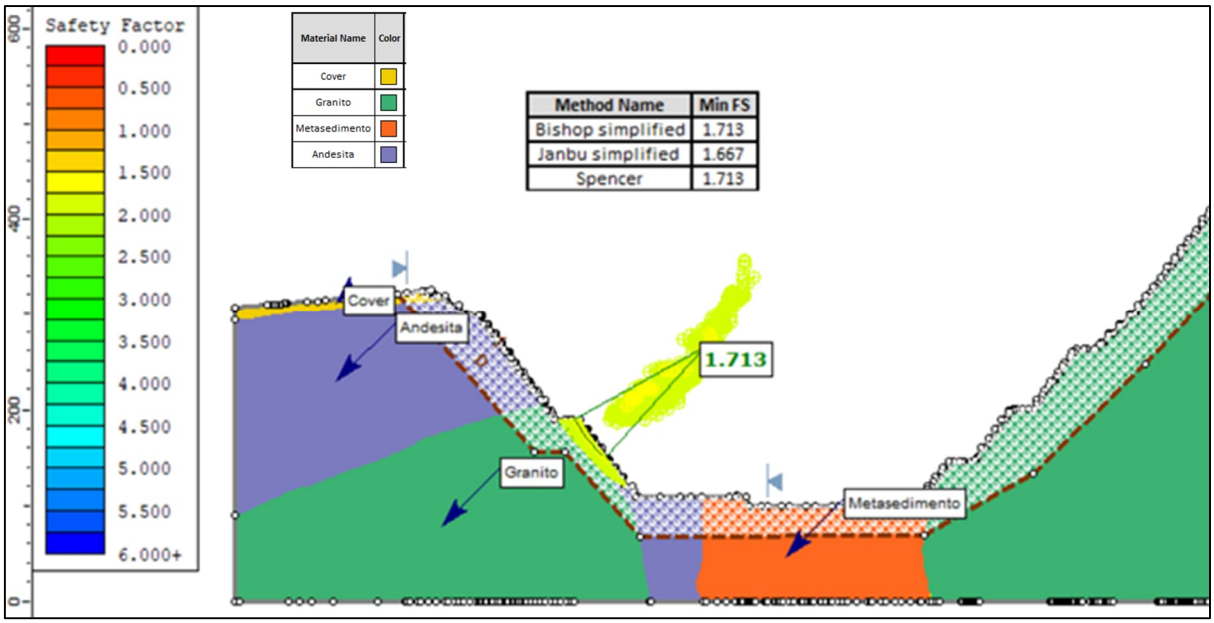
Each line represents a section view with its number, indicating the phase to which the profile belongs, starting from the left (L) to the other end of the line, which we will call right (R).

The walls defined to estimate the ultimate pit stability are shown in Table 16-18, which also includes the safety factor and probability of failure for that profile and wall.

Table 16-18: Summary with SF and PF Values for Final Pit

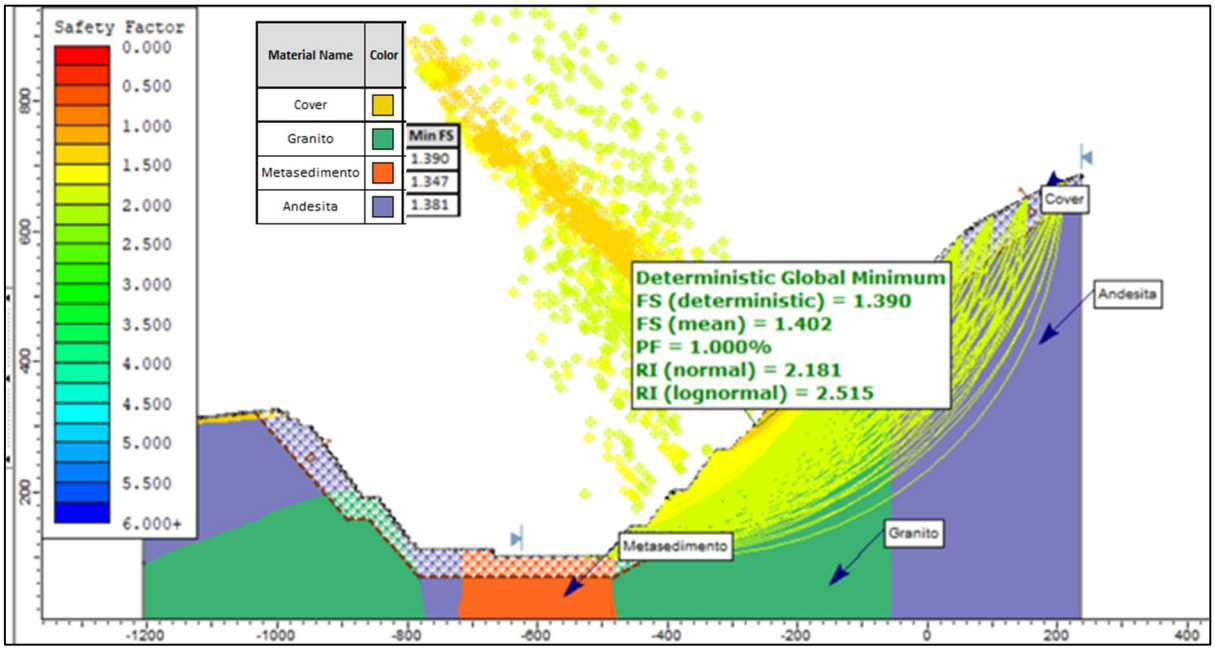
Profile	Wall Side			
	Right (R)		Left (L)	
	SF	PF	SF	PF
F1	1.39	1.00%	1.71	0.00%
F2	2.35	0.00%	2.70	0.00%
F3	1.86	0.00%	2.75	0.00%
F4	2.49	0.00%	3.73	0.00%
F5	-	-	1.36	2.00%

From the analyzed profiles, we can observe that there are 2 walls with a safety factor close to the minimum acceptable of 1.3, but their probability of failure is quite low. These critical cases are shown in the Figure 16-27 to Figure 16-35.



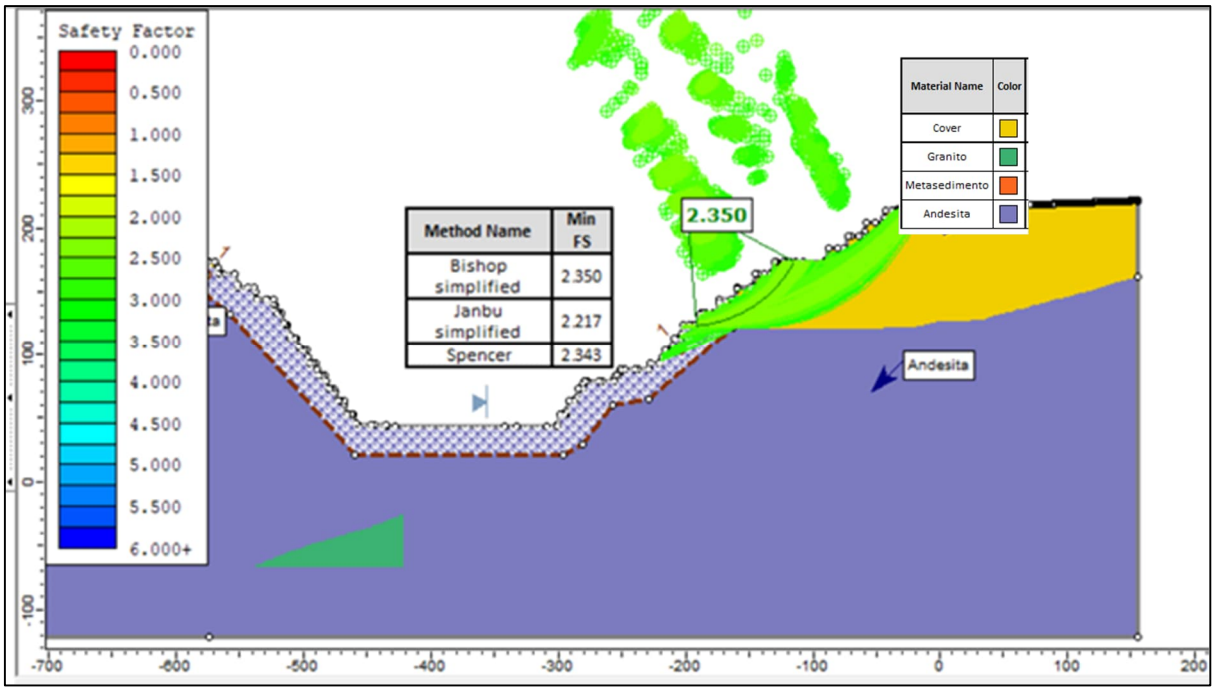
Source: AbraSilver Resource Corp., 2024

Figure 16-27: Section F1 Left Stability Analysis



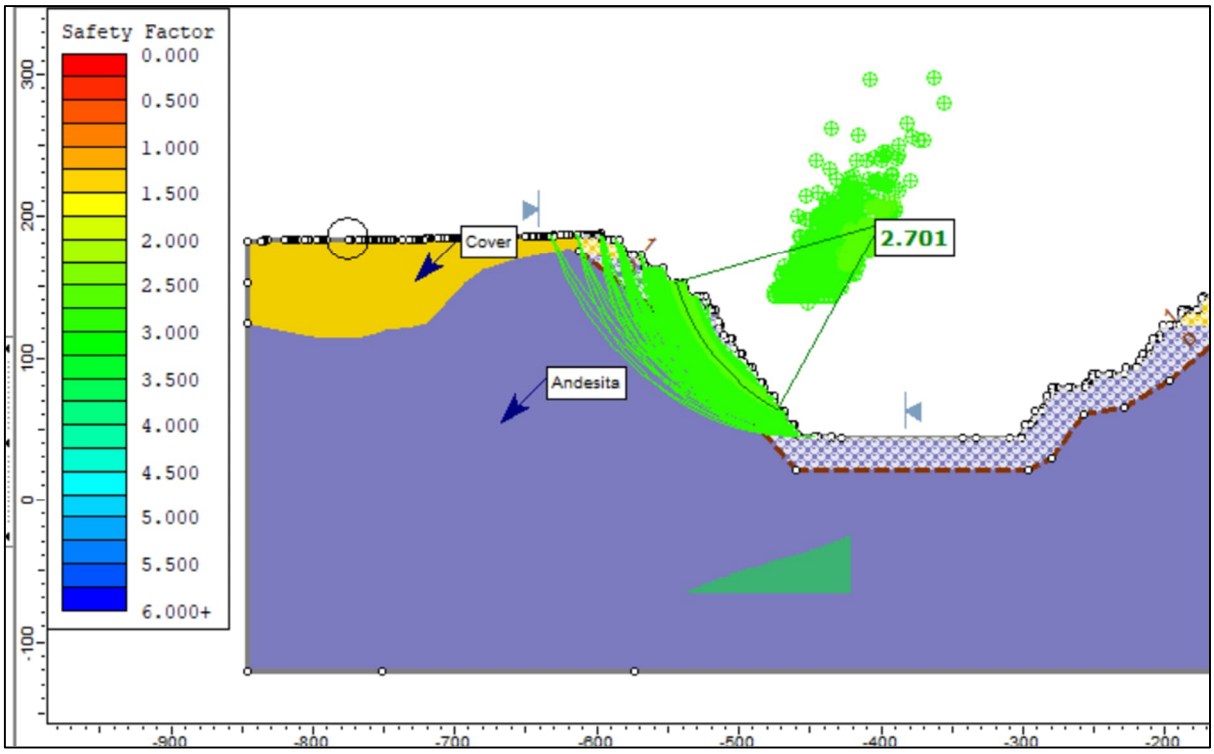
Source: AbraSilver Resource Corp., 2024

Figure 16-28: Section F1 Right Stability Analysis



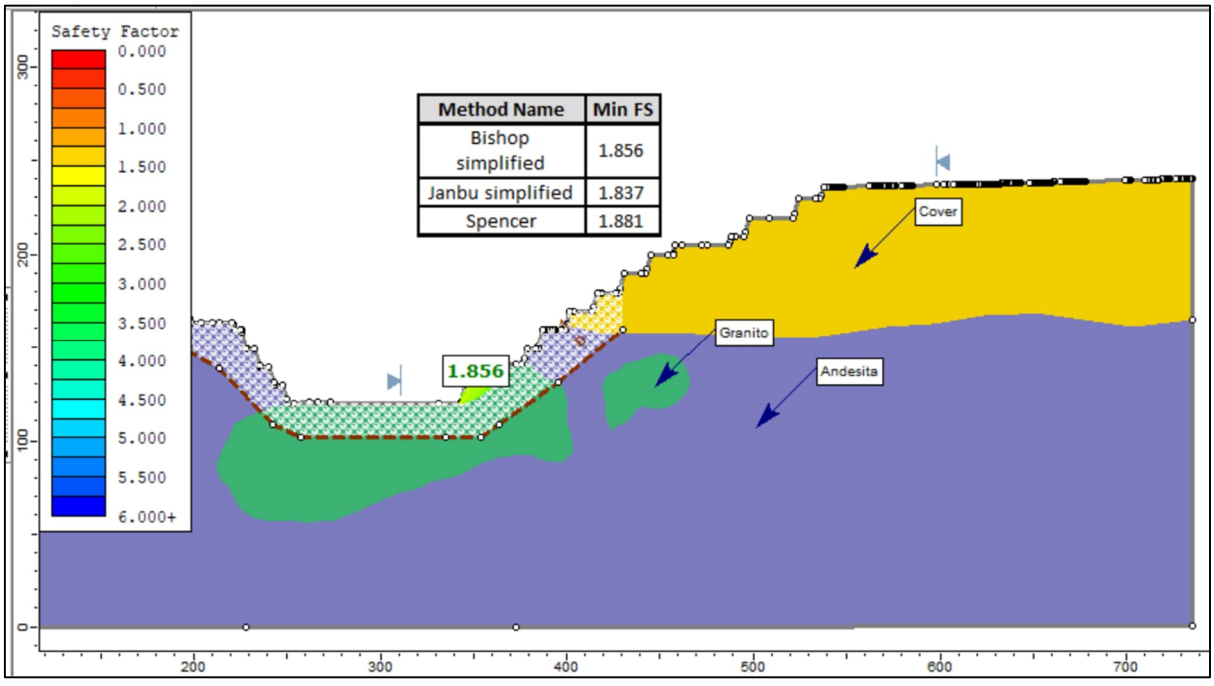
Source: AbraSilver Resource Corp., 2024

Figure 16-29: Section F2 Left Stability Analysis



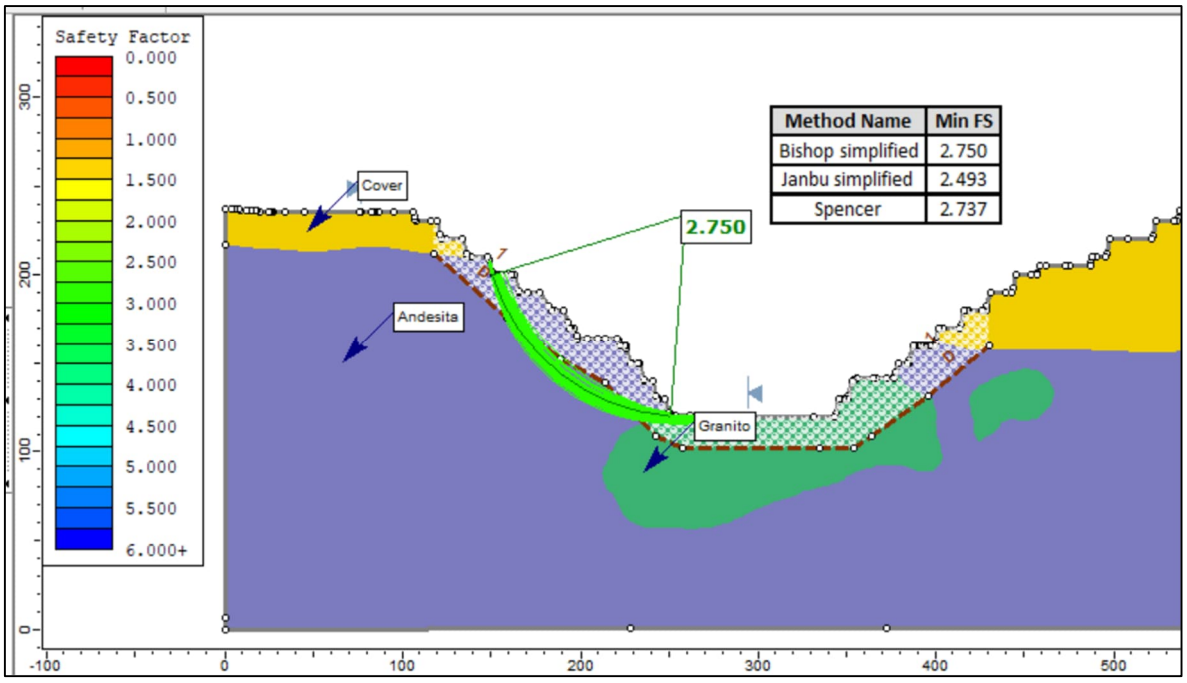
Source: AbraSilver Resource Corp., 2024

Figure 16-30: Section F2 Right Stability Analysis



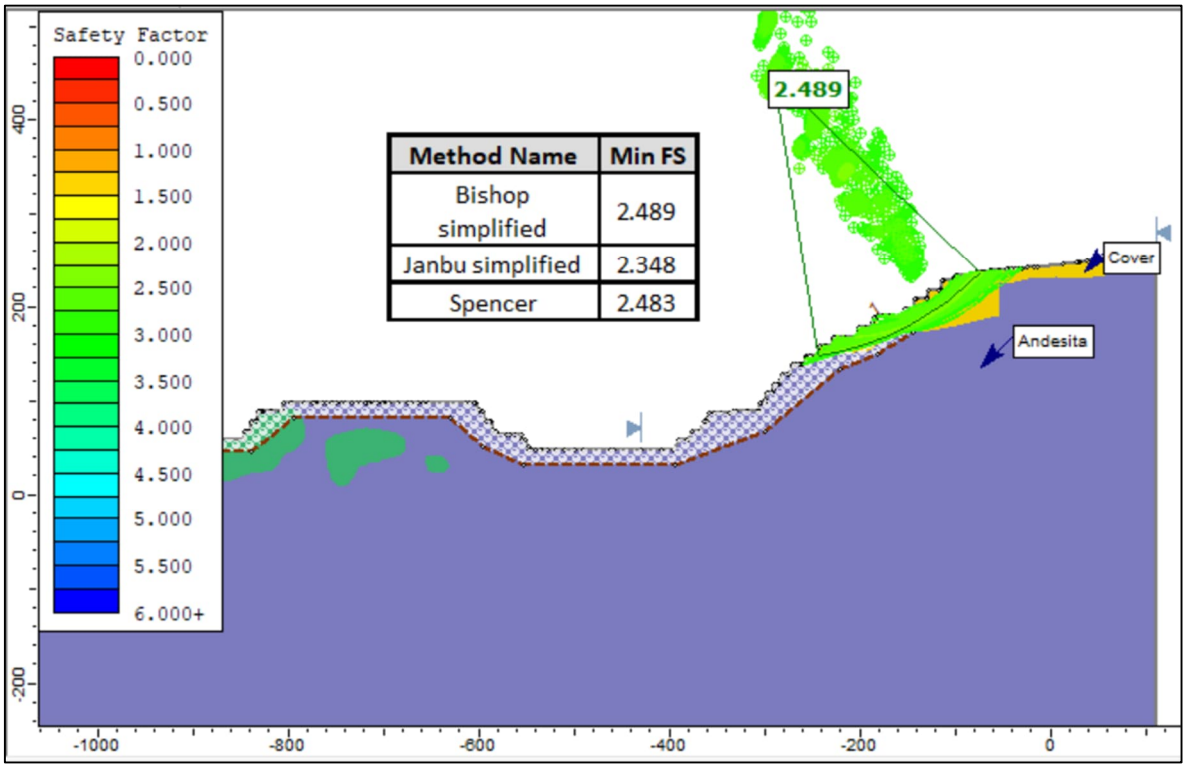
Source: AbraSilver Resource Corp., 2024

Figure 16-31: Section F3 Left Stability Analysis



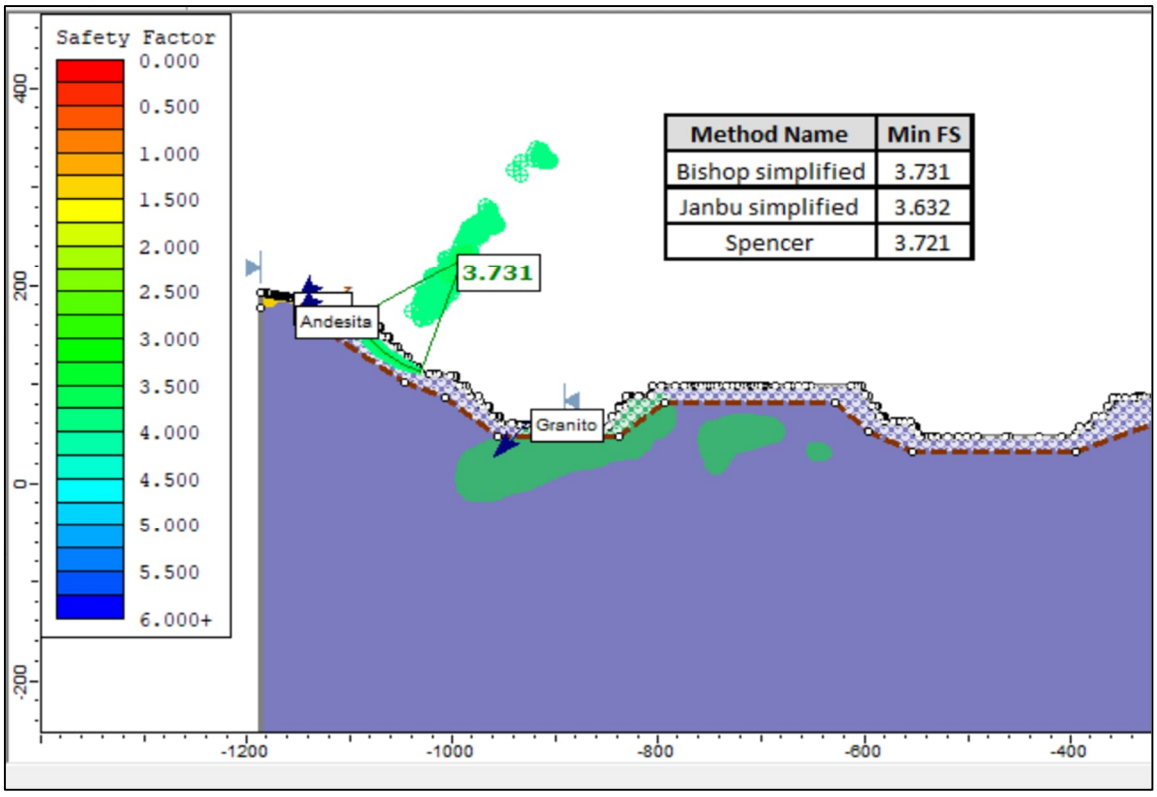
Source: AbraSilver Resource Corp., 2024

Figure 16-32: Section F3 Right Stability Analysis



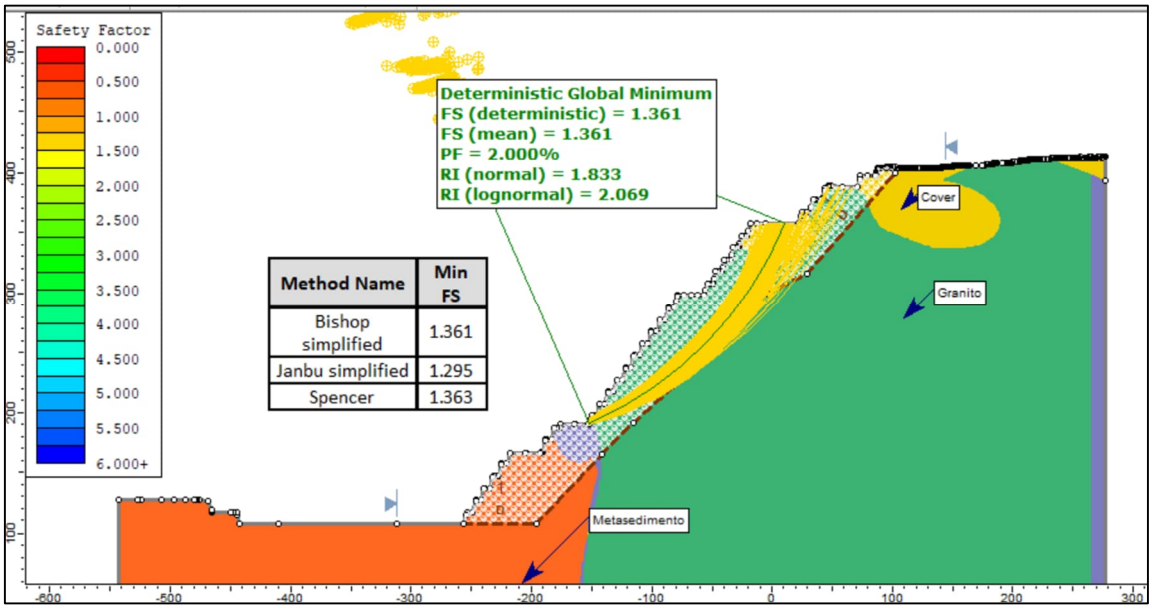
Source: AbraSilver Resource Corp., 2024

Figure 16-33: Section F4 Left Stability Analysis



Source: AbraSilver Resource Corp., 2024

Figure 16-34: Section F4 Right Stability Analysis



Source: AbraSilver Resource Corp., 2024

Figure 16-35: Section F5 Left Stability Analysis

16.9.3 Summary of Results

- There are 2 walls with a factor of safety close to the limit of 1.3.
- All other walls are stable or very stable by mining standards.
- Walls with a safety factor greater than 2 have design parameters that can be optimized.
- In order to reduce the possible overestimation of rock and cover properties consider common operating practices and the inclusion of a damage zone of D=0.8.

16.9.4 Acceptability criterion

Based on the results obtained in the stability analysis performed, the recommended acceptability criterion for the execution of the proposed design in the Diablillos open pit mine is to comply with a Safety Factor equal to 1.3 with a maximum Probability of Failure of 10%.

In addition, in order to ensure the stability of the work, it is necessary to:

- Perform ongoing monitoring, measuring displacements of the mine slopes.
- Ongoing bench-to-bench topographic control. Whenever possible, considering a global monitoring system for the mine under normal operating conditions (robotic system; prism-station-total monitoring software).
- Perform daily geomechanical inspections in the different phases of the mine and generate effective interaction with mine operations personnel.

During operation, it is also recommended to generate the following geotechnical products that are key to good slope control performance:

- Geomechanical risk map.
- Structural prediction map.
- Weekly and monthly recommendations.
- Slope rehabilitation plan.
- Geotechnical reconciliation.
- Recommendation for perimeter blasting.
- Training for mine operations personnel.
- Geotechnical procedures.

16.9.5 General Recommendations

- Since the stability evaluation presented is an ultimate pit evaluation, post-operating degradation of the material must be considered to ensure stability over time.
- Given the level of uncertainty of the geomechanical information used for this study, a safety factor of 1.7 is not recommended. Close to this value the PF value is very close to 0.0%.
- The study of rock and fill qualities should be improved to update models and reduce uncertainty. This allows a lower SF to be used in the design, close to 1.3.

- Previous documents have mentioned the possible presence of water. This should be studied to incorporate it in the design definition and stability determination. Consider the proximity to areas with salt flats.
- It is highly recommended to improve the characterization of the surface called cover in order to obtain reliable results, since this zone is the one with the highest probability of failure.

16.9.6 Design Recommendations

Considering that walls with a low probability of failure have high SFs, we recommend optimizing the design parameters. Mainly seeking to adjust the ramp width to the real needs of the project (22 m).

Considering the above, it is possible to increase the overall angle in the best geomechanical performance zones to about 45°, with an SF of 1.7.

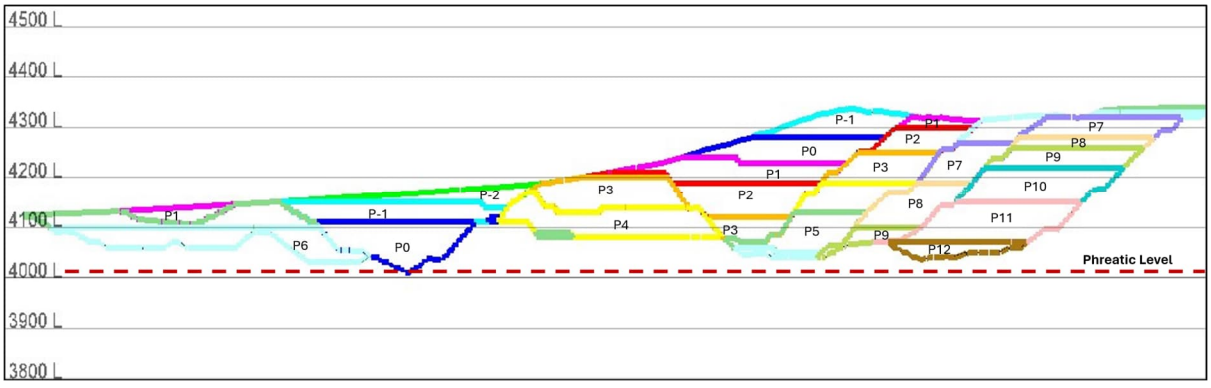
16.10 Water Management

Since most of the drilling was done to define mineralization, a precise definition of the water table was not established. A theoretical logging exercise during the reverse circulation program (dry and wet split) from 1996 to 1997 found the water table to be at level 4005.

Considering this water table and plotting it with the depth for each pit period, a dewatering system should be installed in period 5 and towards the end of the life of mine to avoid water in the mining level.

In the future it is necessary to incorporate some dynamic test drilling to specifically determine the water table in the next drilling campaign; however, the knowledge presently available does not suggest a risk to the current state of reserves.

Figure 16-36 shows the level reached per period with regard to the water table, while Table 16-19 shows the specific level for each period.



Source: AbraSilver Resource Corp., 2024

Figure 16-36: Phreatic Level versus Open Pit Level

Table 16-19: Level of Open Pit Period

Period	Min MASL Level
-2	4,140
-1	4,110
0	4,010
1	4,105
2	4,120
3	4,070
4	4,080
5	4,050
6	4,030
7	4,190
8	4,100
9	4,040
10	4,150
11	4,070
12	4,040
13	4,000

16.11 QP Comments about Section 16

In future drilling campaigns, tests of the cover sector should be carried out and a hydrogeological model built. In this way some slopes of the open pit could be increased.

Strategic planning with variable cut-off grades over time and a marginal phase analysis is additionally recommended for a later engineering stage.

A trade off study should be early 2025 to see different options of size equipment, such as CAT785 vs CAT777, autonomous haulage system.

17. Recovery Methods

The following section has been taken in its entirety from the previous NI 43.101 Technical Report Updated Pre-Feasibility Study, effective December 3rd, 2024, and report date January 17th, 2025.

17.1 General Description

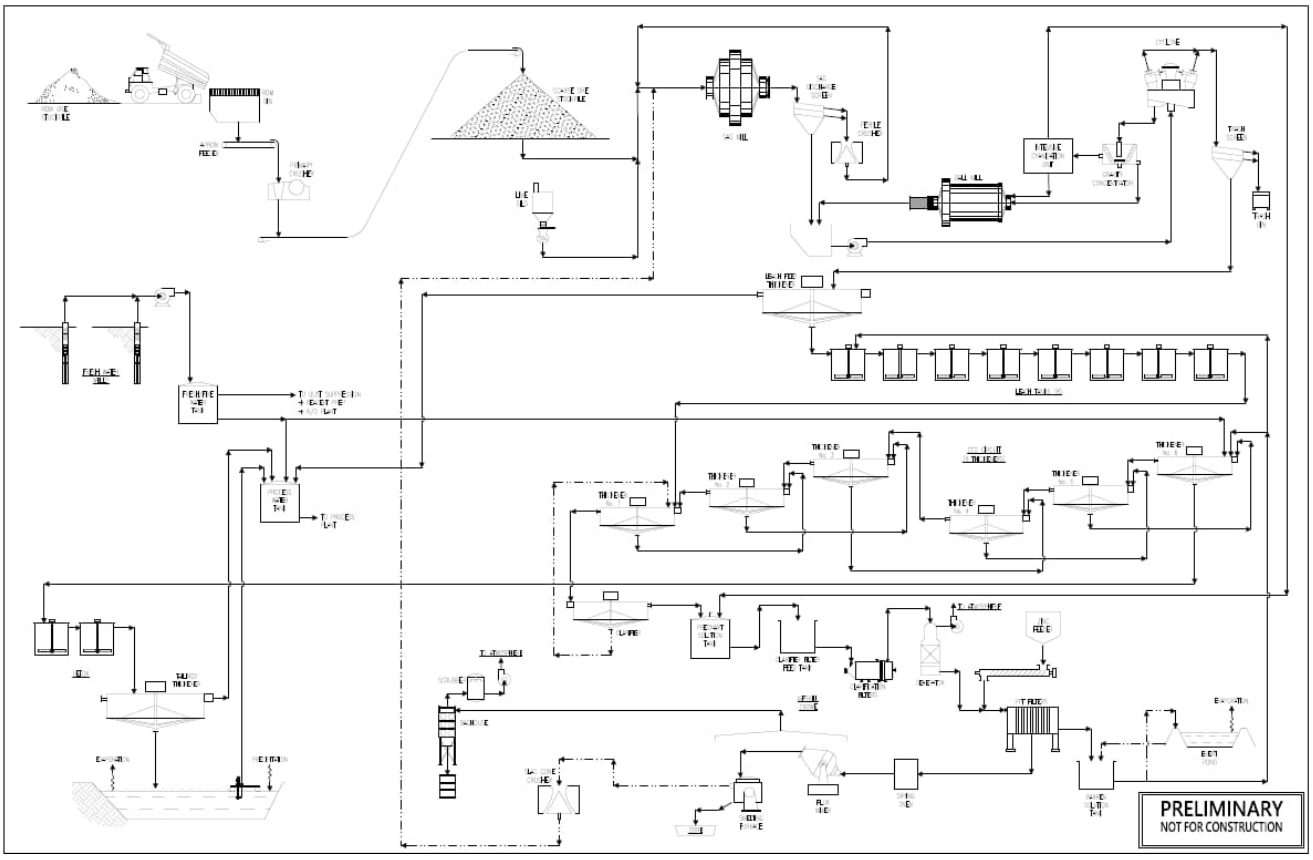
The results of the metallurgical test work described in Section 13, together with industry standard practices, were used to develop process design criteria, which in turn were used to design the process flowsheet, develop the mass balance and equipment design parameters selected for this project. This section describes the process recovery method selected mainly based on the metallurgical test on Oculto material as completed in 2022. The JAC material as tested in 2023, representing approximately 15% of the total resource, has a somewhat different metallurgical performance. Whenever the process design is different between JAC and Oculto material, it will also be mentioned in the section below.

The Diablillos deposit of AbraSilver is a gold and silver mineralized material which from laboratory test work has demonstrated economic metal recovery through cyanidation. Based on the gold and silver head assays of the mineralized material together with their respective laboratory testing recoveries through gravity concentration and cyanidation, a combination of centrifugal treatment and the Merrill-Crowe process have been selected as the most economic process options. Based upon a throughput stipulation by AbraSilver, a processing plant with a daily capacity of 9,000 metric tons of material will be designed. The total nominal operating days is 360 days at mine site, however considering the equipment downtime and required maintenance, 350 plant operating days are assumed per year, which results in a yearly throughput of 3.15 million metric tons. According to the resource model, the average head grade of the Oculto material is 0.88 ppm of gold and 74 ppm of silver; and the JAC material has an average grade of 0.13 ppm of gold and 202 ppm of silver. The plant will consist of the following unit operations:

- Crushing – The primary jaw crushing station will operate in open circuit producing a final product 80 precent passing size (P80) of 150 mm or below. The crushed ore will be transferred to an enclosed coarse ore stockpile and then further conveyed to the grinding circuit.
- Grinding – Grinding size reduction is accomplished in a typical Semi Autogenous Grinding (SAG) and ball mill circuit with SAG oversize pebble crushing. The final grinding product after cyclone classifying has a (P80) size of 150 µm.
- Gravity Concentration and Intensive Leaching - Gravity concentration will treat 50 weight percent of the cyclone underflow or more, to produce a gold rich gravity concentrate. The gravity concentrates will be dissolved with an Intensive Leach reactor for gold and silver recovery. The pregnant solution will report to the pregnant solution tank. The Intensive Leach residue will be fed by gravity to the ball mill feed.
- Tank Leach – The slurry after gravity concentration will report to a tank leach at 45% solids density, facilitated by air injection. The total retention time will be 36 hours and will be followed by the counter current decantation (CCD) circuit.

- Counter Current Decantation – The slurry after cyanide leach will be washed by the CCD circuit, which has a total of six stages and has a wash ratio of 3. The pregnant solution will report to the Merrill-Crowe plant and the washed slurry will report to the cyanide destruction circuit. If the plant will process JAC material only, a total of seven stages of CCD will be employed with a wash ratio of 3.0.
- Merrill-Crowe circuit – The pregnant solution recovered from the CCD circuit will be further purified by pressure filtration to remove the Total Suspended Solids (TSS), de-aerated, and then the gold and silver will be precipitated by zinc dust. The barren solution will be recycled as either the CCD wash water or the process makeup solution.
- Foundry – the precipitate produced from the Merrill-Crowe plant will be transferred to the refinery to produce the final Doré bullion. The Doré production process includes retorting to remove mercury, flux mixing, and smelting. The slag will be crushed and recycled back to the grinding circuit for re-processing.
- Cyanide Destruction and Tailings Handling – The washed slurry from CCD will report to cyanide destruction tanks to reduce the Weak Acid Dissociable (WAD) cyanide to the required level. The process will use the industrially proven INCO process. The detoxed tailings will be further thickened by the tailings thickener and then report to the tailings pond.

The overall flowsheet is depicted in Figure 17-1 below.



Source: AbraSilver Resource Corp., 2023

Figure 17-1: Overall Flowsheet

17.2 Process Design Criteria

The following Process Design Criteria (PDC) and mass balance details the annual mineralized material and product quantities, major mass flows and capacities, major equipment information and reagent consumption for both Oculto and JAC material (Table 17-1). The reagent consumptions and consumables can be found in the operating cost estimate in Section 21.

Table 17-1: Process Design Criteria

Area	Criteria	Unit	Oculto	JAC
General	Daily Throughput	mtpd	9,000	
	Operating days per year	days/year	350	
	Annual Throughput	mtpy	3,150,000	
	Oxide head grade, Au, ppm	ppm	0.88	0.13
	Oxide head grade, Ag, ppm	ppm	74	202
	JK Axb value		56.2	126.8
	Bond Grinding Work Index	kwh/mt	15.8	15.1
	Bond Abrasion Index	g	0.66	0.1
	Crushing Circuit Utilization and Availability	%	75%	
	Grinding and Downstream Circuit Utilization and Availability	%	92%	
	Design factor for Equipment Sizing		1.15	
	Design Factor for Conveyors, Pumps		1.25	
Comminution	Crushing Nominal Feed Rate	mt/hr	479	
	Type of Primary Crusher		Jaw	
	Crusher Open Side Setting	mm	150	
	Coarse Ore Stockpile Type		Enclosed, Conical	
	Stockpile Live Capacity	mt	13,000	
	Stockpile, Total Capacity,	mt	52,000	
	Grinding Circuit and Downstream Nominal Rate	mtph	391	
	SAG Size (Diameter x EGL)	m	7.92 x 3.66	
	Mill Slurry Solids Concentration	%, w/w	75%	
	Pebble Screen Aperture	mm	10.0	
	Pebble Recycle Rate	%	15-35%	
	Pebble Crusher Type		Cone Crusher HP-100 or equivalent	
	Ball Mill Size	m	5.49 x 8.53	
	Ball Mill Circulating Load	%	250%	
	Final Grind Size	micron	150.0	
	Cyclone Feed Slurry Solid Concentration	%, w/w	55%	
Tank Leach and Gravity	Leaching Tanks Feed Slurry Solids Concentration	%, w/w	45%	
	Leaching Retention Time	hr	36.0	

Area	Criteria	Unit	Oculto	JAC
Concentration	Leaching pH		10.5	
	Gold Recovery	%	83.0%	83.8%
	Silver Recovery	%	79.8%	84.9%
	Gravity Concentrator feed		Cyclone underflow	
	Percent of Cyclone Underflow treated	%	50% or above	
	Gravity Concentrator		Two of Falcon SB5200 or equivalent	
	Intense Cyanidation Unit		One of Falcon SLR6000	
	Gold Recovery	%	10.1%	17.3%
	Silver Recovery	%	7.8%	9.1%
CCD	CCD Stage		six	seven
	CCD Thickener Underflow Solids Concentration	%, w/w	55%	50%
	CCD Wash ratio		3.0	
	Final CCD Overflow Total Suspended Solids (TSS)	ppm	No more than 100	
Merrill-Crowe	Merrill-Crowe Nominal Solution Feed Rate	m^3/hr	1125	1,273
	Clarified Solution TSS	ppm	1 or below	
	Dissolved Oxygen (DO) after De-aeration	ppm	0.5 or below	
Cyanide Destruction and Tailings Handling	Cyanide Destruction Method		INCO process	
	Total Residence Time	hr	1.0	
	Detox Slurry Solids Concentration	%, w/w	55%	45%
	Tailings Thickener Underflow Solid Concentration	%, w/w	60%	55%
Major Reagent Consumption	Sodium Cyanide Consumption	kg/mt	0.83	1.79
	Lime Consumption	kg/mt	0.93	2.19
	Diatomaceous Earth Consumption	mt/day	1.4	2.7
	Flocculant Consumption	g/mt	45	95
	Zinc Dust Consumption	kg/hr	34	102
	Sodium Metabisulfite, g/g of WAD CN	g/g WAD CN	4.7	

The above process design Criteria is based on the plant feed of Oculto or JAC respectively, which will lead the equipment size difference in the metal recovery circuit, particularly Merrill-Crowe plant, CCD, and tailings handling circuit. However, considering the potential ore blend in the future plant feed, the requirement of equipment size change will be minimal, which in turn implies a minimal impact on the process CAPEX. However, process OPEX for these ore types should be considered in ore blending for cash flow analysis.

17.3 Process Description

17.3.1 Crushing Circuit

The ore will be trucked from the mine to the ROM stockpile near the jaw crusher station. A front-end loader will convey the material from the ROM stockpile to the jaw crusher dump pocket. The dump pocket will be covered by a stationary grizzly to retain any oversize boulders from entering the jaw crusher. The dump pocket has a live capacity of 200 metric tons of material, which is twice the material the recommended mining truck can contain per individual load. A hydraulic rock breaker is installed near the dump pocket to break the larger oversize boulders retained by the grizzly.

The material is conveyed to the jaw crusher through an apron feeder from the dump pocket. The crusher product reports directly to the jaw crusher discharge conveyor, which further transfers the material to the coarse ore stockpile feed conveyor. A magnet will be installed at the discharge end of jaw crusher discharge conveyor to remove any tramp metal. A metal detector is also installed to alarm the presence of any metals on the conveyor. The jaw crusher has an open side setting (OSS) of 152 mm (equals 6 inches). The jaw crusher product is expected to have a P80 around 152 mm and top size of 254 mm. The crushing production rate will be monitored by a belt scale mounted on the conveyor. A lime silo will be installed at the stockpile feed conveyor and will meter the pebble lime onto the conveyed material to establish an alkaline environment in the grinding circuit.

Due to the high wind velocity at the mine site, the coarse ore stockpile will be enclosed. The coarse material stockpile is designed to have a live volume of 13,000 metric tons of the plant feed. When required, the material will be moved from the “dead” storage zone to the “live” storage zone by a front-end loader. The total stockpile capacity is estimated around 52,000 metric tons. This translates a stockpile of 67 m in diameter and 25.2 m in height. The material will be reclaimed by two variable speed belt feeders, one operating and one standby. The feeders transfer the material to a SAG mill feed conveyor reporting to the grinding circuit. The reclaim rate will be monitored by a belt scale mounted on the SAG mill feed conveyor.

Dust control will be controlled in the dump pocket, transfer conveyors, and coarse ore stockpile. Dust control in the stockpile area will be achieved using two wet type dust collector systems. One of the two dust collector systems will be installed to control dust at the discharge end of the stockpile feed conveyor, and the other one will be installed to control the dust in the reclaim tunnel.

17.3.2 Grinding Circuit

The material reclaimed from the coarse ore stockpile will be initially ground in a SAG mill. Dilution water will be added into the SAG mill feed chute to maintain the slurry density inside the SAG mill at approximately 75% solids by weight. The SAG mill has a diameter of 7.92 m (26 ft) and effective grinding length of 3.66 m (12 ft) and is equipped with a 4,500-kW variable speed drive. The ball charge is typically controlled between 8% to 15% by volume. The SAG mill discharge will

be screened by a vibrating screen (pebble screen) installed with spray bars to rinse the fines from the oversize pebble material. The screen aperture will be 10 mm. The screen oversize will be transferred to a cone crusher through two transfer conveyors, to further reduce the pebble size down to P80 of 12 mm (½ inch) or below. The crushed pebble material will be recycled back to the SAG mill feed conveyor and then to the SAG mill for further grinding. The pebble recycle rate is typically 15% to 35% of SAG fresh feed rate, a HP 100 short head cone crusher or equivalent will be sufficient for this duty. The close side setting (CSS) is around 9 mm or smaller. When the pebble recycle rate falls below 10%, the pebble crusher can be stopped.

The pebble screen undersize will be discharged to a pump box, which also receives the ball mill discharge. The ball mill will have a diameter of 5.49 m (18 ft) and effective grinding length of 8.53 m (28 ft), with an installed power of 4,500 kW. The ball mill will be rubber lined and is operated at 76% of the mill critical speed. The maximum ball charge is assumed to be 33 volume percent. Dilution water will be added into the pump box to maintain the cyclone feed slurry density around 55% solids by weight. The slurry in the pump box will be pumped to a cyclone cluster. Two cyclone feed pumps will be installed, one operating and one standby. The cyclone overflow will be the final grinding product with a particle size of approximately 150 microns. The ball mill circulating load is assumed to be 250%. A portion of the cyclone underflow will be fed to the gravity gold recovery circuit to recover the coarse gold, while the other portion of the underflow reports directly to the ball mill. The cyclone overflow will be sampled by primary samplers and analysed for metallurgical control prior to cyanidation. An online particle size analyser is installed to monitor the particle size distribution of the grinding circuit product.

Grinding balls will be added to the SAG mill and ball mill by ball bins. Air compressors and instrument air will provide the required service for the mill operation and for maintenance. Both SAG mill and ball mill will have their dedicated mill liner handling machines. An overhead crane will be installed for the maintenance of the grinding circuit.

17.3.3 Gravity Concentration and Intensive Cyanidation Circuit

Two gravity concentrators, Falcon SB5200 or equivalent, will be installed receiving a portion of the cyclone underflow. Fluidization water will be injected into the two gravity concentrators keeping the solids in the slurry better fluidized. The mass pull from these two gravity concentrators will be approximately 190 kg per hour. Each gravity concentrator has an operating cycle of approximately one hour. At the end of each operating cycle, the concentrate from the gravity concentrator will be flushed out to an intensive cyanidation circuit (alternative name Inline Leach Reactor or ILR). An intensive cyanidation system (SLR6000 unit manufactured by Sepro Mineral Systems or equivalent) will be installed to treat the gravity concentrate. The gravity tails will continuously report back to the ball mill feed during the operating cycle.

Based on the laboratory gravity testing, it is estimated that 10.1% gold and 7.8% silver will be recovered to the gravity concentrate for Oculto material, and 17.3% gold and 9.1% silver can be recovered to the concentrate for JAC material. The gravity concentrate will be leached in the ILR circuit with sodium cyanide solution and air to recover the gold and silver from the solids to the

pregnant solution. The intensive cyanidation typically has a very high metal recovery, it is assumed that 99% gold and 97% silver will be recovered to the pregnant solution. The pregnant solution from the ILR circuit will be periodically pumped to the pregnant solution tank at the Merrill-Crowe circuit. The solid tails from the ILR circuit will be flushed back to the ball mill circuit. The gravity gold recovery circuit will be isolated and monitored for security purposes within the grinding circuit area.

17.3.4 Cyanide Leach Circuit

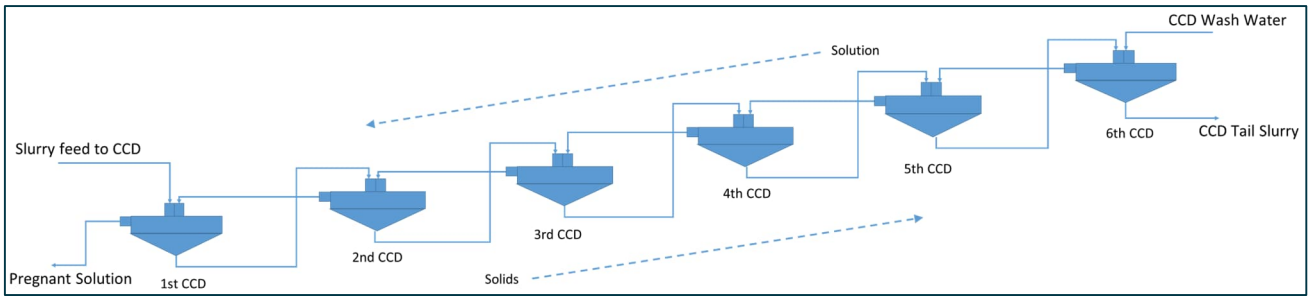
The cyclone overflow will report to a 34 m diameter pre-leach thickener, for the purpose of increasing the leach tank slurry density so that the tank volume required can be optimized. The overflow from the pre-leach thickener will report to the process solution tank. The thickener underflow will be approximately 60% solids by weight and will be pumped to the leaching tanks. If the plant will process the JAC material only, the required pre-leach thickener size will be 40 m in diameter and its underflow density will be controlled around 50% by weight.

Based on the cyanidation test work, the leach retention time is specified as 36 hours. Additional barren solution will be added to dilute the leach slurry density to 45% solids by weight. Milk of lime, if needed, will be added to the leaching tank to control the slurry pH at 10.5 for both optimization of leaching kinetics and conservation of the free cyanide in the process. There will be a total of eight leach tanks (diameter of 15 m and height of 20 m) in series, with each tank equipped with an agitator and an air injection mechanism. The slurry will flow through leach tanks by gravity and then will be pumped to the downstream Counter Current Decantation (CCD) circuit. Sodium cyanide solution will be dosed at the leach tanks to maintain the free cyanide concentration of approximately 1.5 grams per liter and the cyanide concentration will be allowed to decay in the last two leach tanks.

Based on the laboratory testing, the expected gold recovery will be approximately 83% and silver recovery will be approximately 80%. For JAC material, the gold recovery will be around 84% and silver recovery will be around 85%. The leaching tanks will be open to the air and will be equipped with an overhead crane for maintenance.

17.3.5 Counter Current Decantation Circuit

The leached slurry will report to a CCD circuit to recover the pregnant solution. There will be a total of six stages of CCD thickeners (all high-rate thickeners with added flocculant) which will recover most of dissolved gold and silver to the pregnant solution phase feeding the Merrill-Crowe circuit. The CCD wash ratio, defined as the ratio of wash water to the aqueous phase of CCD thickener underflow, will be controlled at 3.0. The CCD thickener underflow density will be approximately 55% solids by weight. The stage mixing efficiency of each CCD thickener is assumed to be 85%. The Figure 17-2 below illustrates the CCD circuit flow streams.



Source: AbraSilver Resource Corp., 2023

Figure 17-2: CCD Circuit Flow Streams

Due to a higher volumetric feed to the first CCD thickener compared to following thickeners, the first CCD thickener will have a diameter of 32 meters, while all the rest of the CCD thickeners will have a diameter of 30 meters. Based on the METSIM simulation, around 99.6% of gold and silver in the solution can be recovered to the pregnant solution for downstream Merrill-Crowe processing.

The overflow from the last CCD stage will be fed to an additional thickener for clarification, which will be installed to minimize the Total Suspended Solids (TSS) in the pregnant solution. The TSS in the pregnant solution is directly linked to the Diatomaceous Earth (DE) consumption and the duration of the downstream DE clarifier operation cycle. The clarifier underflow will be periodically recycled back to the CCD thickener. To minimize the solution TSS, the additional thickener for clarification will have diameter of 32 m.

For JAC material, the slurry will be more viscous and more difficult to settle based on the sedimentation and rheology tests. In addition, the contained value in the pregnant solution will be higher for JAC material compared to the Oculto material. Consequently, if the plant will process JAC material only for a certain period, a total of seven stages of CCD thickening will be required to optimize the project economics. The required CCD thickeners size will be 40 m in diameter for the first one and 38 m in diameter for the rest of the six CCD thickeners. Approximately 99.8% of gold and silver in the CCD feed can be recovered to the Merrill-Crowe plant.

17.3.6 Merrill-Crowe Circuit

The gold and silver in the pregnant solution will be recovered by the Merrill-Crowe process, which includes DE coated clarifiers, two de-aeration towers, and zinc cementation filter presses. This equipment in the Merrill-Crowe circuit will be supplied by a vendor as a package.

The pregnant solution recovered from the CCD circuit will be stored in a pregnant solution tank (diameter of 15 m and height of 20 m), which is to provide a buffer capacity of over two hours for the cyclic operation of the Merrill-Crowe circuit. The nominal solution flowrate through the circuit is 1,145 metric tons per hour. The maximum solution TSS is recommended to be below 100 ppm. The pregnant solution will first be pumped through a DE coated clarifier. There will be a total of four units of DE coated clarifiers, with three units operating and one unit on standby.

Each DE coated clarifier typically has an operating cycle of approximately twelve hours. Before each operating cycle, the DE slurry will be prepared at 2% solids by mixing the dry DE powder and raw water. The DE slurry will be pumped through the DE clarifier for precoating. The water recycled from DE precoating returns to the DE precoat preparation tank. Typically, the thickness of the DE precoat on the filter cloth needs to be 3 mm (1/8 inch) for efficient clarifier operation. Once the precoat application operation is complete, the pregnant solution can be filtered through the clarifier. During solution filtration, additional DE slurry will be metered into the clarifier which is proportional to the pregnant solution flow rate and contained TSS level. This process is called DE body feed. The DE slurry used for body feed is typically 7% solids by weight. Once the pressure drop through the clarifier builds up and the flow rate declines to a certain level, the clarifier will be taken offline for back wash and the spent DE will be sluiced off. The spent DE will be collected in the area sump and then pumped to the CCD circuit. The time duration offline including backwash, sluice, and filter cloth cleaning should be no more than two hours.

The clarified solution produced by the DE clarifier typically has TSS level less than 1 ppm, which is a clear solution. This solution will flow through two deaeration towers (diameter of 3.2 m and height of 6.3 m) to remove dissolved oxygen (DO) to below 0.5 ppm. Both deaeration towers will be operating units. Three vacuum pumps will be installed, with two operating and one on standby, which will provide sufficient vacuum to reduce the DO to the target level.

The solution from the deaeration towers will be pumped to the cementation filter presses through inline booster pumps. A total of four units of filter presses are included, with three operating units and one standby unit. Before bringing each individual filter press online, the cementation filter also needs to be precoated with DE, similar to the operation of DE clarifier. After DE precoating, the cementation filter can be brought online, and the pregnant solution will be pumped through the cementation filter.

Zinc dust will be added to the pregnant solution at the filter feed pipe through a zinc feed cone covered with water to eliminate the entrainment of any oxygen into the solution. Zinc metal in the form of zinc dust has a higher affinity for cyanide and causes the precipitation of gold and silver as metals. The precipitation of gold and silver using zinc dust is almost instantaneous and by the time when the pregnant solution reaches the cementation filter chamber virtually all precious metal precipitate has already formed and will be retained inside the filter chamber. Depending on the cyanide concentration in the pregnant solution, additional cyanide may be required to be added to the pregnant solution to induce more efficient precipitation. In the event of low ambient temperature which may lead to slower precipitation kinetics, lead nitrate can be added to expedite the precipitation and potentially improve the precious metal recovery.

Both the pregnant solution before the cementation filter and the barren solution after cementation will be sampled. There will be solution samplers installed both at the cementation filter feed pipe and discharge pipe which will continuously collect the solution sample, typically one composite sample for every shift. These samples will be used for metallurgical accounting purposes. Spot solution samples will also be collected periodically to check the performance of the Merrill-Crowe unit operation. If needed, required operational adjustments will be implemented.

Once the pressure drop through a cementation filter increases to a certain level or flow rate declines to a certain level, the cementation filter needs to be taken offline and precipitate will be recovered. The typical operation cycle of the cementation filter usually requires several days depending on the precoat efficiency and filterability of the precipitate cake. The cake coming out of the filter press typically includes gold and silver, impurity metals, DE, and moisture. The cake will be collected in a transfer bin which will be manually transferred to the foundry for further processing.

Both the DE precoat tank and the DE body feed tank will be installed inside the Merrill-Crowe plant. The DE precoat is a batch process in which the DE clarifier or cementation filter is coated with a required amount of DE before each operation cycle. DE body feed is a continuous process which is employed to capture the TSS in the pregnant solution and improve the performance of the DE clarifier.

The Merrill-Crowe Process typically has a very high precious metal recovery. Based on the test work performed, it is expected that 99.5% gold and 98.5% silver recoveries can be achieved through the Merrill-Crowe Process. With most of the barren solution recycled back to either CCD or the leach tanks, the actual gold and silver recovery through the circuit is usually slightly higher than the test work indications.

For JAC material, the solution flow to the Merrill-Crowe plant will be slightly higher, which will require a slighter larger capacity of Merrill-Crowe plant, however this does not change the number of clarifiers or filter presses, only a slightly larger unit will be required.

17.3.7 Foundry Circuit

Preliminary assay results indicated that the plant feed contains mercury. A portion of the mercury is expected to be dissolved during the cyanide leaching unit operation and eventually will report to the Merrill-Crowe Process precipitate. The retort unit will remove both mercury and moisture from the wet precipitate.

The mercury will be collected by the gas chiller of the retort unit and the cooled mercury liquid flows for collection to a flask. The mercury collected in the flasks will be periodically shipped out to an appropriate external sales or disposal facility. Both wet precipitate and dry precipitate will be sampled for the purpose of generating a metallurgical balance and developing performance statistics.

The dry precipitate will be mixed with fluxes in a flux mixer based on an appropriate recipe. The mixed charge will be placed into a fuel fired smelting furnace, and the smelting furnace will separate the doré bullion from the slag. The solidified slag will still contain a minor amount of entrained gold and silver, and it will be crushed by a small crusher in the foundry and then recycled back to the grinding mill for further processing.

The off gases from the retort unit and smelting furnace will be collected in their respective gas scrubber units to capture any fugitive dust. The gas scrubbers are highly efficient and off gas emission will conform to the industry standards.

17.3.8 Tailings Detoxification and Handling

The washed slurry from the last stage CCD thickener will have a slurry density of approximately 55% solids by weight. This slurry will report to two cyanide destruction tanks arranged in series to eliminate the remaining Weak Acid Dissociable (WAD) cyanide in the slurry. Both tanks will be equipped with agitators which induce strong agitation of the tailing's slurry and the detox reagents. Air will be injected in both cyanide destruction tanks to facilitate the cyanide decomposition reactions.

Sodium metabisulfite and copper sulphate will be dosed into the two detox tanks based on a certain ratio related to the WAD cyanide content of the slurry feed. Air will also be injected to the detox tank to expedite the oxidation of WAD cyanide. The total required retention time is one hour. The slurry after detox will contain less than 1 ppm WAD cyanide, which will meet the environmental requirement of the mining industry.

Due to the scarcity of water resources at the mine site, it is preferred to conserve as much raw water as possible. Therefore, a larger thickener (diameter of 40 m) is designed so the tailings thickener underflow can achieve at least 60% solids by weight. The tailings thickener overflow will be recovered to the process solution tank, and the underflow will flow by gravity to the tailings pond. The supernatant from the tailings pond, if any, will be recovered back to the process solution tank for re-use.

As JAC material will produce a slurry with a much higher viscosity, a slurry density of 45% solids by weight is recommended for cyanide destruction process. Therefore, larger detox tanks with more air flow will be required for JAC material. The final tailings thickener will still be 40 m in diameter, but the underflow density will be controlled around 55% solids by weight.

17.3.9 Reagents

The reagents used in the process plant include:

- Pebble lime and milk of lime.
- Sodium cyanide.
- Sodium hydroxide.
- Diatomaceous earth.
- Zinc dust.
- Flocculant.
- Fluxes.
- Sodium metabisulfite.

- Copper sulphate.
- Lead nitrate (optional).

17.3.10 Pebble Lime and Milk of Lime

As the cyanide leach needs to be conducted in an alkaline environment with pH of 10.5 or higher, lime addition will be required for the process. A lime silo installed at the SAG mill feed conveyor will provide most of the lime required for the process. The lime silo will be sized to hold 100 metric tons of pebble lime. The pebble lime will be metered onto the SAG mill feed conveyor in a rate proportional to the ore feed rate, which can be read from the weight meter installed on the conveyor.

Milk of Lime (MOL) will be required to dose the leaching tanks and maintain the pH at or above 10.5, which is taken as the optimum pH for the cyanide leach. The cyanide destruction circuit also requires milk of lime (MOL) to destroy the remaining WAD cyanide in the tailings before discharging the tailings slurry to the tailings pond. A lime slaker will be installed onsite, converting the pebble lime to the MOL. The combined lime consumption as MOL is estimated at 3.4 mtpd, including the expected dosing at the leach tanks and the cyanide destruction tanks. With the additional pebble lime used at the SAG mill feed, the total lime consumption is estimated at 10 mtpd.

MOL is assumed to have a concentration of 15% calcium hydroxide and will be stored in an agitated tank (diameter of 3.5 m and height of 5.0 m), which will provide a retention time in excess of one day. The MOL will be delivered through a pump system and a recycled MOL pipeline loop. The lime consumption for JAC material will be much higher, roughly 20 mtpd.

17.3.11 Sodium Cyanide

Due to the remote location of the mine site, sodium cyanide in the form of dry briquettes is recommended for the convenience of transportation and storage. The whole ore cyanidation, based on the laboratory testing program conducted on the gravity tailings for the 150 µm grind size, requires on average 0.71 kg of sodium cyanide per metric ton of solid feed. The intensive cyanidation unit operation is estimated to consume 0.13 kg of cyanide per metric ton of plant feed. Based on the plant throughput, the daily sodium cyanide consumption is taken as 7.8 mt. Assuming 7 days of onsite storage is required, approximately 58 metric tons of sodium cyanide will be stored onsite, assuming 95% purity of the sodium cyanide briquette.

The sodium cyanide solution is recommended to be diluted to 25% concentration by weight. The sodium cyanide briquettes will be delivered onsite in bags (Super Sacks). The sodium cyanide briquettes will be dissolved with barren solution in an agitated mixing tank (diameter of 5 m and height of 8 m), together with sodium hydroxide solution to provide sufficient alkalinity for the sodium cyanide solution, both to conserve the sodium cyanide and prevent cyanide evolution for operation safety considerations. The mixed cyanide solution will be pumped to the sodium cyanide day tank (diameter of 5 m and height of 8 m), which will provide a retention time close to five

days. The cyanide solution will be delivered to the leaching tanks, the intensive cyanidation circuit, and the Merrill-Crowe pregnant solution tank or the ball mill by a pumping system.

As JAC material has much higher silver content, the sodium cyanide consumption is also significantly higher. On average 1.52 kg of sodium cyanide will be required for tank leach and 0.27 kg of sodium cyanide will be required for intensive cyanidation for every metric ton of plant feed. This corresponds to daily sodium cyanide consumption of around 16.8 mt.

17.3.12 Sodium Hydroxide

To eliminate the cyanide volatilization that could potentially occur when mixing sodium cyanide briquettes directly with barren solution, sodium hydroxide solution will be added into the cyanide mixing tank during cyanide solution preparation. Sodium hydroxide will also be delivered onsite in the form of bags of dry briquettes. The dry sodium hydroxide briquettes will be dissolved in a mixing tank (diameter of 3 m and height of 4 m), and then pumped to a sodium hydroxide solution day tank (diameter of 3 m and height of 4 m). This tank will provide a retention time around twelve days.

Seven days of onsite storage is recommended, which equals around 3.5 metric tons of sodium hydroxide dry briquettes on site, assuming 95% sodium hydroxide purity.

17.3.13 Diatomaceous Earth

Diatomaceous earth (DE) is required for the Merrill-Crowe process and will be used at the following three process operations:

- Precoating of DE Clarifiers.
- Body feed of DE Clarifiers.
- Precoating of Precipitate Filter Press.

For the DE precoat for both clarifier and cementation filter press, the DE thickness required to be coated on the filter cloth is around 3 mm or 1/8 inch. The only difference is that the operation cycle for the clarifier is typically about 12 hours, while the operation cycle for the precipitate filter is typically several days. For the cementation filter, a 7-day operation cycle is assumed. For both DE clarifier and cementation filter, DE precoat applications can be performed using one agitated DE precoat tank. Bags of DE will be dumped into the DE precoat tank until a certain amount of DE is accumulated for either the DE clarifier or the precipitate filter. The DE slurry density for precoating is typically controlled around 2% solids for optimum precoat performance. The DE slurry is pumped through the clarifier or filter and the water in the DE slurry returns to the DE precoat tank. The DE slurry density can be calibrated with a Marcy density scale.

During the operation cycle of the DE clarifier, the DE slurry from the DE body feed tank will be metered into the clarifier together with the pregnant solution. The amount of DE required will be proportional to the TSS in the pregnant solution. The TSS in the pregnant solution can be sampled

periodically and measured by a turbidity meter calibrated with TSS level, so a proper amount of DE slurry can be applied to the clarifier. The purpose of DE body feed to the clarifier is to efficiently remove the TSS in the pregnant solution. The typical DE slurry density for body feed is 7% by weight.

The DE in the clarifier will eventually be sluiced from the clarifier to the area sump and eventually will be pumped to the CCD circuit. The DE precoated at the precipitate filter will report to the precipitate cake and will eventually report to the slag produced from the refinery process.

The total consumption of DE for all above functions is around 1.4 metric tons per day. Seven days of DE storage onsite is recommended, which corresponds to ten metric tons of DE onsite. For JAC material, due to a higher TSS in the solution feed and also a larger solution flow to the Merrill-Crowe plant, the total DE consumption is around 2.7 metric tons per day.

17.3.14 Zinc Dust

Zinc dust is employed to precipitate the gold and silver in the Merrill-Crowe process. The zinc dust utilized always has a high purity of 99% zinc and the particle size typically varies between 8 to 12 microns insuring a fast precipitation reaction. The zinc dust will be added into the zinc cone which is covered with water to avoid the entrainment of any oxygen into the solution.

From the laboratory test results and industry standards, the achievement of an elevated gold and silver recovery at the Merrill-Crowe process requires 1.5 kg of zinc dust for every kg of gold and silver to be precipitated. This unit value corresponds to approximately 810 kg of zinc dust consumption per day.

The zinc dust is delivered in steel kegs which hold 50 kg of dry zinc dust. Based on a seven-day onsite storage requirement, it is recommended that 6 metric tons of zinc dust be stored onsite.

17.3.15 Flocculant

Flocculant will be used at the following process locations. Recommended dosages are indicated.

- Pre-leach thickener, 15 g/t of feed solids (30 g/t for JAC material).
- CCD thickener, 12 g/t for the first CCD thickener and 1.2 g/t for the remaining 5 CCD thickeners (total 25 g/t for JAC material).
- Tailings thickener, 16 g/t of feed solids (40 g/t for JAC material).

Based upon the nominal plant feed, the daily consumption of flocculant is approximately 0.4 metric tons. Considering a seven-day onsite storage specification for flocculant, a total of 2.8 metric tons of flocculant in bags will be stored onsite. For JAC material, the daily consumption is approximately 0.86 metric tons.

The solid flocculant needs to be diluted with water to a low concentration for optimum flocculation performance. A design flocculant solution content of 0.1% by weight is assumed for the project

design. The dry flocculant will be dumped into a feeder and then metered into a mixing tank and the dilution water will be injected based on the flowrate of dry flocculant. There will be one flocculant mixing tank and one flocculant day tank. Based on the daily flocculant requirement, approximately one or two batches of mixing are required per day.

The flocculant day tank is also an agitated tank with a diameter of 11 m and height of 15 m. The capacity of the flocculant day tank will provide flocculant solution storage for the operation for approximately three days.

17.3.16 Fluxes

After the retorting process, the dry cake will be mixed with fluxes based on a certain recipe then placed into the smelting furnace to produce the final Doré bullion. A typical flux recipe for 100 kg of Merrill-Crowe precipitate is presented below:

- 3 to 5 kg of soda ash.
- 5 to 10 kg of borax.
- 3 to 5 kg of silica sand.

The flux recipe can be optimized during future operations based on the exact composition of the dry cake. The preceding flux recipe is for reference only. The typical weight ratio between total fluxes and dry cake is 0.2. Based on the estimated dry cake weight per day (estimated around 0.67 mtpd, including gold, silver, assumed percentage of impurity metals, and estimated DE), the daily flux consumption is around 134 kg. For 14 days of onsite storage, the required total fluxes will be around 2 metric tons. JAC material will require approximately 404 kg of flux per day.

17.3.17 Sodium Metabisulfite

The CCD tails slurry will contain WAD cyanide, which needs to be reduced to a certain level before being discharged to the tailings pond. The cyanide destruction operation will use the industry standard INCO process, which utilizes sulphur dioxide (or sodium metabisulfite), copper sulphate, and air to oxide the WAD cyanide. Based on the laboratory test results and the industry standard, 4.7 grams of sulphur dioxide and 0.22 grams of copper will be required for one gram of WAD cyanide.

Based on the total sodium cyanide consumption of 0.83 kg per metric ton of plant feed, the total sodium cyanide consumption will be 0.33 mt/h. Laboratory testing indicated that when leach feed has a sodium cyanide concentration of 1500 mg/L in the aqueous phase, the leached tail has total sodium cyanide concentration of 1300 mg/L in the aqueous phase of which approximately 700 mg/L is in the form of WAD cyanide. Based on this ratio, the WAD cyanide in the form of CN^- is around 0.08 mt/h in the feed to the cyanide destruction circuit.

Based on the required weight ratio of sulphur dioxide to WAD cyanide, the consumption of sodium metabisulfite is about 0.56 mtph. Assuming a 20% concentration in the sodium metabisulfite

solution, the solution flow will be around 2.8 m^3/hr. Sodium metabisulfite will be delivered onsite in the form of dry solids, it will be dissolved in an agitated tank and then pumped to a day tank. The day tank will provide reagent solution storage capacity to sustain the cyanide detox operation for approximately five days. Based on seven days of onsite storage, a total of 95 metric tons of sodium metabisulfite will be stored onsite.

17.3.18 Copper Sulphate

The lab test did not indicate the requirement of copper sulphate for cyanide destruction due to the presence of a minor amount of copper in the test sample. However, copper sulphate is still recommended onsite for provision, and a reduced dosage from typical cyanide destruction is assumed at 0.11 grams of copper per gram of WAD cyanide, the copper sulphate consumption will be 0.02 mtph. Copper sulphate will be delivered onsite in the form of dry solids and then dissolved in the mixing tank to make copper sulphate solution with a 20% concentration. The copper sulphate day tank will provide the solution storage capacity to sustain the detox operation for around four days. Due to the elevated copper content in the pregnant solution from JAC material, no copper sulphate will be needed for the detox.

Based on the onsite storage for seven days operation, a total of four metric tons of copper sulphate needs to be stored onsite. The copper sulphate will be delivered onsite in the form of bags of dry solids, either in the form of CuSO4 or CuSO4.5H2O.

17.3.19 Lead Nitrate

Lead nitrate can potentially improve recovery of the Merrill-Crowe process in certain circumstances. Therefore, lead nitrate is recommended as an optional reagent prepared onsite. Based on industry practice, the pregnant solution flow rate, and zinc addition rate, the lead nitrate consumption is estimated at 8 kg/hr if needed. The lead nitrate will be delivered onsite in the form of dry solids. The mixing tank will provide the solution storage for the operation need of approximately one week.

17.3.20 Antiscalant

Solutions with high pH or high calcium concentration tend to form scale in the pipes and other associated equipment. To minimize the potential scaling, antiscalant is recommended onsite as a provision, added wherever necessary.

Assuming a typical 5 ppm concentration in the solution will be sufficient, the annual consumption of the antiscalant is estimated to be around 45 metric tons. The antiscalant will be delivered onsite in totes.

17.4 Process Solution and Raw Water

The total raw water requirement is 184 mtph, which includes:

- Raw water for CCD wash, 150 mtph.
- Raw water for reagents preparation, 22 mtph.
- Raw water for dust suppression, 12 mtph.

This amount of raw water will all report to the process and eventually be recovered to the process solution. The raw water will be supplied from water wells located approximately 13 km away from the mine site. The total amount of raw water that is allowed to be withdrawn is 220 mtph, which includes the water for camp, mining operation, and office use. The water will be pumped from the wells to the raw water tank onsite, which also serves as fire water storage. The freshwater tank has a diameter of 20 m and height of 24 m. This volume will hold the water required for 24 hours of mine operation with an additional fire water reserve of 1,760 m^3, which is sufficient for anticipated fire suppression requirements.

The raw water tank will be installed at an elevation higher than the mine site, so the water can gravity flow to the various users including the process solution tank and fire hydrants. The total process solution flow rate is 594 mtph. A process solution tank with a diameter of 11 m and a height of 15 m will be installed in the plant to provide a retention time of over two hours.

17.5 Process Air and Instrument Air

Process air will be required at the cyanide leaching and cyanide destruction processes as given below:

- Cyanide leach tanks, compressed air of 3,138 Nm^3/h.
- Cyanide destruction tanks, compressed air of 2,713 Nm^3/h.

The air pressure requirement is estimated to be between 150 kPa to 300 kPa for the above operations. A total of three air compressors will be installed, two operating and one standby, to provide the process air. A dedicated compressor will provide conditioned air, which will be used for various instruments in the plant. The required instrument air is assumed to be around 600 Nm^3/hr.

17.6 Power

The total power requirement from the mine site is estimated approximately at 20 MW, with the majority requirement for the process area. The power will be provided by a hybrid power generation facility composed of a photovoltaic plant and dual fuel diesel/natural gas gensets onsite. The diesel/natural gas gensets will be designed to provide the full power requirement of the mine site, with nine gensets operating most of the time and another two units as an installed spare.

The photovoltaic plant will be composed of two solar power arrays which, one of 3 MW with batteries power storage for camp and service hub operations during construction and operations, while a second one of 17 MW without batteries will provide the plant power during daytime operations and storing the excess energy into the 3 MW batteries storage system (BESS).

Power will be distributed to all process and ancillary facilities at 6.6kV via underground conduit duct-banks to avoid costly overhead power lines maintenance and improving site safety as well.

Off-site facilities, as the raw water well field and pipeline booster station, the tailings water recovery system and the nitrate preparation pad will be supplied by on-site diesel generators of the size according to power requirements, including one main generator and a second back-up one at each location.

The site distribution voltage will be transformed to low voltage, rated at 380/220V, for power supply to motor control centres (MCC) and power distribution centres (PDC) for motors, lighting, instrumentation, etc. All control systems will have an uninterruptible power supply (UPS) and emergency lighting will have battery back-ups to keep critical processes operational and provide a safe shut-down.

18. Project Infrastructure

The following section has been taken in its entirety from the previous NI 43.101 Technical Report Updated Pre-Feasibility Study, effective December 3rd, 2024, and report date January 17th, 2025.

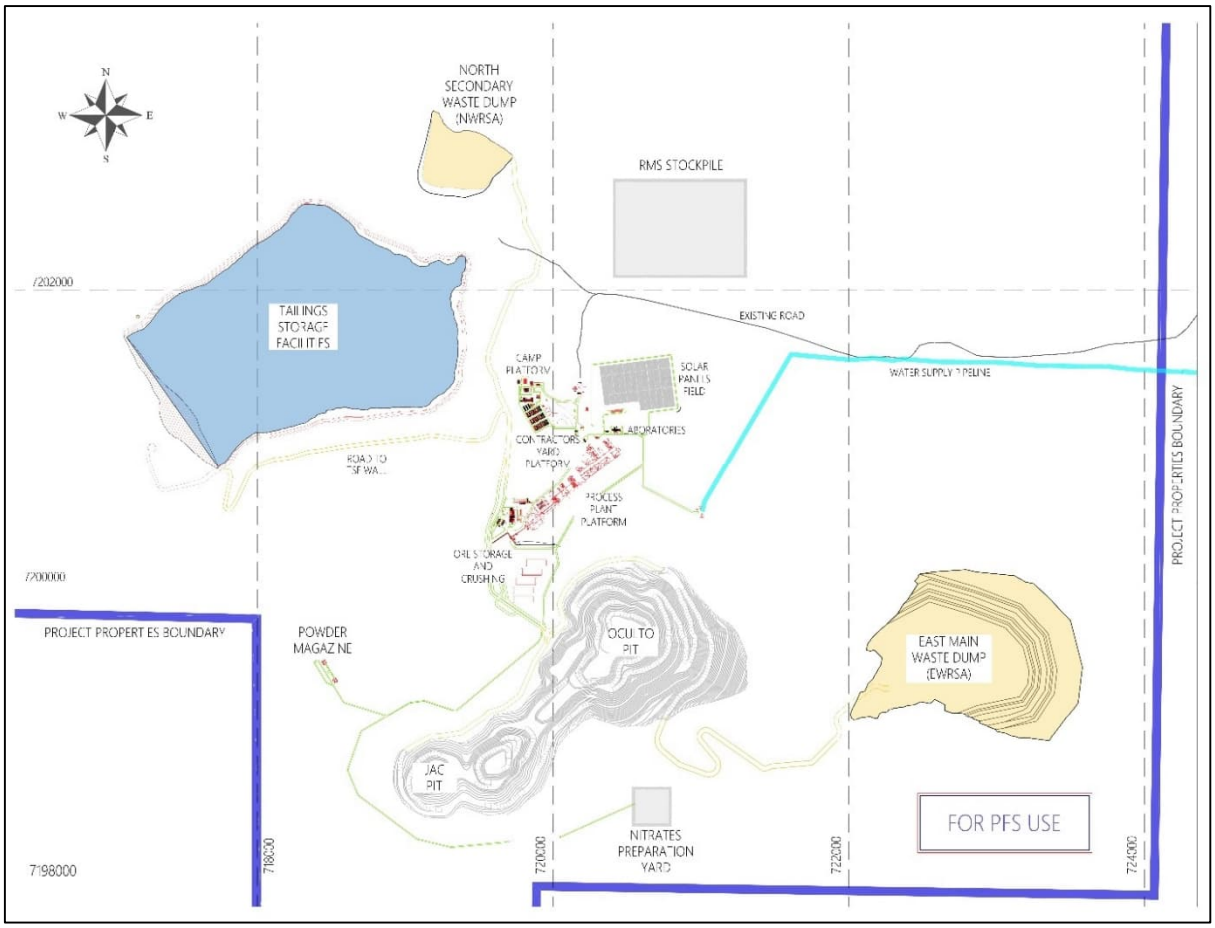
The mine, ore processing plant and mine site related infrastructure will all be located within the Property in Salta and Catamarca provinces. The Project infrastructure is designed to support the operation of two (2) combined open pits feeding a 9,000 tonne per day processing plant, on a 350 day per year operating basis. The overall site layout showing the location of the open pits, processing plant, tailings storage facility (TSF), waste dumps, camp and service hub is provided in Figure 18-1.

18.1 Summary

The infrastructure required for the Diablillos Project will include:

- Site development and access roads.
- Raw Water Supply System.
- Raw water storage and treatment facility, including a firefighting system network.
- A hybrid power generation plant (Diesel + PV) and distribution network.
- An ore processing plant.
- A tailings storage facility (TSF) and associated water recovery management structures.
- Waste dumps with associated water management systems.
- Service hub, truck-shop, warehouse, offices, laboratories, and other service facilities.
- Operations and construction camp and contractors' yard.

The proposed layout of the Diablillos Project is shown in Figure 18-1.



Source: AbraSilver Resource Corp., 2023

Figure 18-1: Diablillos Project Site Layout

18.2 Diablillos General Site Plan

The general site plan details the location of the different project infrastructure, including the camp, service hub, truck shop, plant warehouse, laboratories, ore processing plant, open pit, haul roads, TSF waste rock dumps, raw water wellfield and supply pipeline.

The mill complex site is centrally located, approximately 4.5 km north of Oculto section of the open pit, 4.6 km from JAC and 2.1 km northeast of the TSF. It will contain the service hub, plant offices, laboratories, truck-shop, plant warehouse, maintenance, and the process plant itself.

The layout of facilities has been optimized to take advantage of the topography. The JAC pit colluvial cover will be a quarry for tailings storage facility and road construction material. The complex will include the primary crusher, grinding via milling, gravity separation, leaching, counter current decantation, Merril Crowe process, detoxification, corresponding safety vault, reagents preparation, storage, hybrid power generation plant, and a fuel tank farm.

18.3 Access Roads, Main Site Access, and Internal Roads

The project is approximately 327 km from the capital city of Salta, 340 km from the Jujuy province capital San Salvador de Jujuy and approximately 673 km from Catamarca City Centre.

The three main access routes to the project in Argentina are listed below:

18.3.1 From Salta City

By paved National Highway N° 51, approximately 150 km to the main regional town of San Antonio de los Cobres, then a 15 km stretch of gravel road along the same highway, turning into Provincial Highway N° 129 for another 48 km to reach the small town of Santa Rosa de los Pastos Grandes. The route continues for another 95 km along gravel Highway N° 129, then another 20 km along an internal project road, totalling 327 km. Driving time is five to six hours depending on traffic and road conditions. This route is not suitable for oversized goods transportation.

Alternatively, the same route to San Antonio de los Cobres (150 km), then following the unpaved National Highway N° 51 for 67 km until Provincial Highway N° 27 then another 40 km until the small town of Salar de Pocitos, continuing 3 km to Provincial Highway N° 129 for another 30 km to join the previous route at Santa Rosa de los Pastos Grandes. Total driving time for light vehicles is seven to eight hours for a total distance of 405 km. This route is the most suitable for oversized goods transportation from Salta.

18.3.2 From Jujuy City

By National Highway N° 9 for 60 km until reaching the town of Purmamarca, then turning into National Highway N° 52 for another 100 km of paved road, passing the town of Susques after 5 km onto gravelled National Highway N° 40 for 110 km to National Highway N° 51. Then the route will follow one of the two previous alternatives, totalling either 445 km for the first option or 540 km for the second. Driving time for light vehicles is either eight or nine hours depending on traffic and road conditions.

18.3.3 From Catamarca City

Starting from San Fernando del Valle de Catamarca city, a 200 km paved highway leads southwards along National Highway N° 38 for 75 Km to the town of Chumbicha, where it joins National Highway N° 60. After 136 km the road joins National Highway N° 40 at Cerro Negro, turning northwards for another 135 Km until joining Provincial Route N° 36 at the town of El Eje, passing Londres and Belén. The road continues for another 12 Km to Provincial Route N° 43 and continues for 200 Km towards Antofagasta de la Sierra, one of the main towns in the Catamarca highlands.

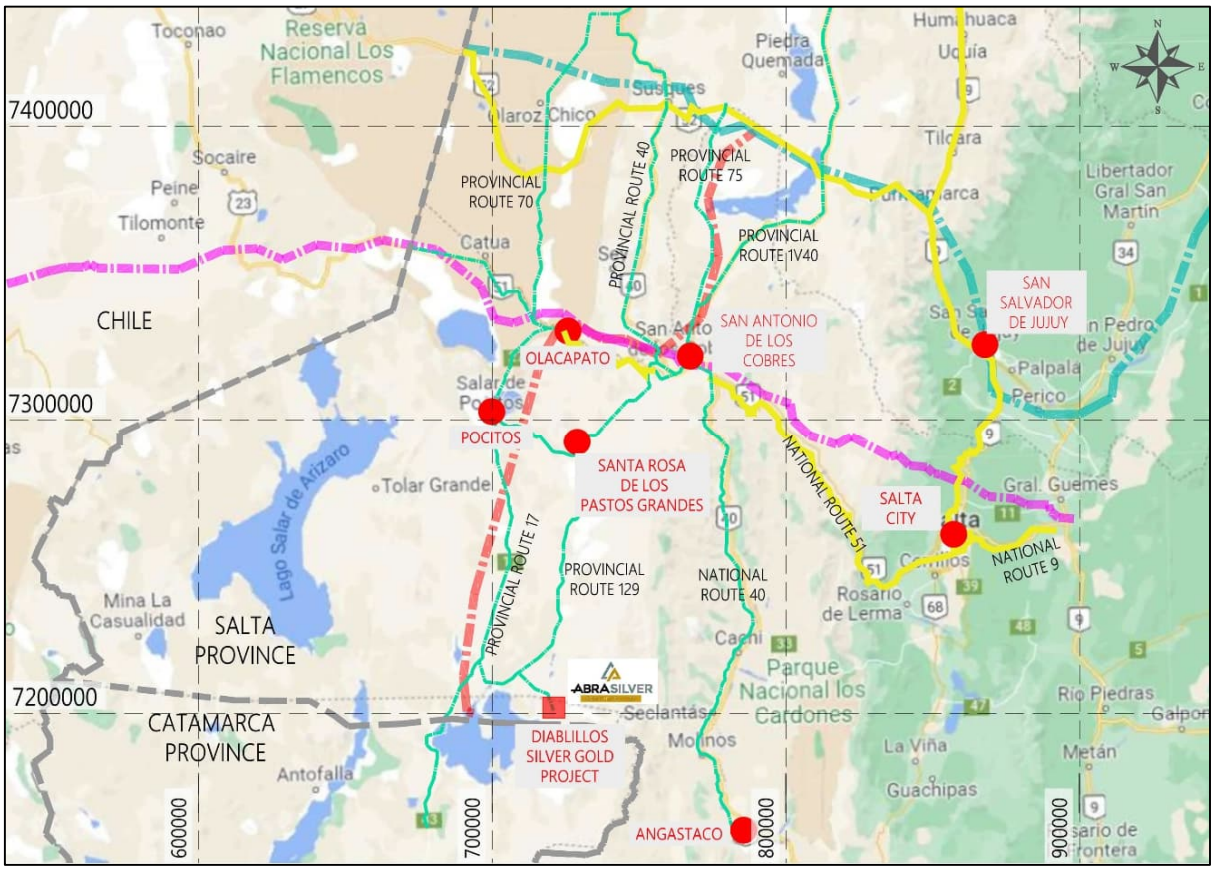
From there a gravel road continues northwards to Salar del Hombre Muerto for another 90 Km, reaching the Livent Lithium Project. On reaching Cerro Gordo dwelling the road winds for 40 Km until it reaches the local community of La Redonda 7 Km from the Diablillos property. This route totals almost 700 Km.

All main roads to the project are well-constructed gravel district roads which are regularly maintained by the provincial highway's authorities of Salta and Catamarca. (Dirección de Vialidad de Salta - Catamarca).

The regional site map is presented in Figure 18-2 and Figure 18-3.

The final access road to the project from Route 129 will be improved and upgraded along its entire 20 km length according to provincial highway standards. Converting it into a two-lane gravel road until it reaches the main access gate which will be manned 24/7. It does not cross any significant water courses and will be fully maintained by AbraSilver to allow two-way traffic including oversize cargo transport.

The main access road from Route 129 is shown in Figure 18-4.



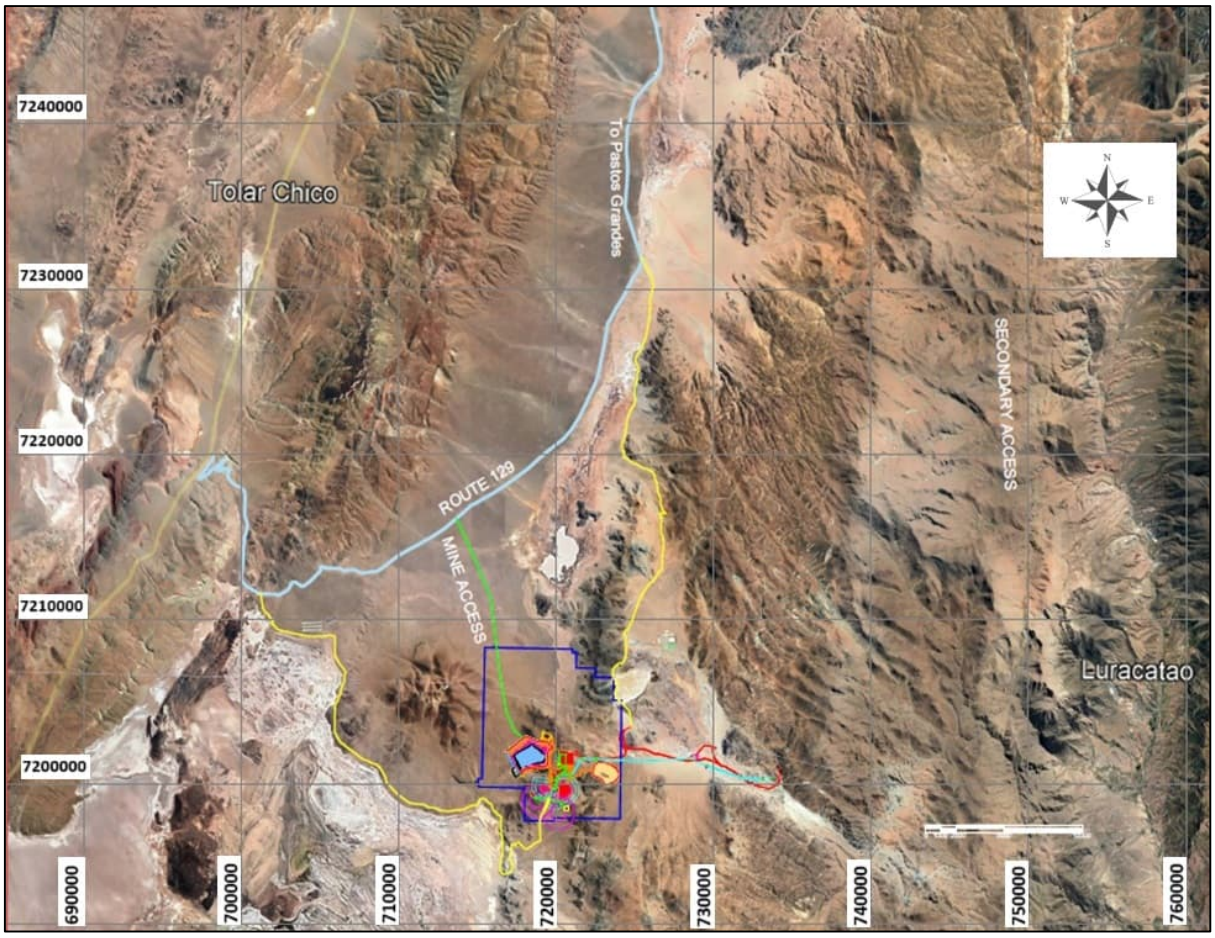
Source: AbraSilver Resource Corp., 2023

Figure 18-2: Regional Site Map – Salta



Source: AbraSilver Resource Corp., 2023

Figure 18-3: Regional Site Map – Catamarca



Source: AbraSilver Resource Corp., 2023

Figure 18-4: Mine Access Road

Due to the size of heavy mine equipment, safety measures require that normal and mine hauling roads are fully segregated from other mine vehicles. The mine haul road will be constructed to support 100 tonne trucks carrying ore and waste to the plant or dumps.

The project internal service roads will be designed according to provincial highway authority standards to provide access to all other facilities, including the raw water wellfield, TSF, powder magazine, camp and service hub.

18.4 Raw Water Supply

The raw water supply is expected to come from several different sources, as 5 basins have been identified in the area next to the project. However, the main raw water supply will come from the Barranquilla basin aquifer, a wellfield of at least 4 wells which is under development. The wellfield is located approximately 17 km from the plant site.

The Diablillos project requires water for the following uses:

- Dust suppression during mining operations.
- Dust suppression in the dry plant area (Crusher and stockpiles).
- Process plant and preparation.
- Service hub, laboratories, truck shop and truck washing station.
- Construction and operations camp.
- Construction activities including concrete preparation.
- Fire water reserve.

The water demand for construction averages 34.15 m^3/h, while operation demands an average of 183.29 m^3/h.

18.4.1 Water Source- Piezometer / Wells

Drilling by AbraSilver has identified an aquifer in the upper part of the Barranquillas valley. Two broad diameter holes drilled by Conhidro have encountered substantial aquifers which are extensions of ones previously discovered by exploration drill holes St-DBL-Ag4 and St-DBL- Ag5. Holes St-DBL-Ag6 and St-DBL-Ag7 are 12-inch diameter rotary holes and hole St-DBL-Ag7 intersected a sequence of gravels with abundant fresh water of more than 50 m3/hr. The hole was drilled in a water easement currently held by AbraSilver.

Pump testing on holes St-DBL-Ag6 and St-DBL-Ag7 demonstrated the potential of the aquifer to host adequate water for the project. Hole St-DBL-Ag7 produced 120 m^3/hr (2,880 m^3/day) of low salinity water, which is a bit more than half of the project requirement. An additional two holes will be drilled to secure the rest of the raw water supply. The recharge of the Barranquillas basin was estimated to be 3,1000,000 m^3/year (354 m^3/h). More than project requirements for make-up water and camp consumption. This also does not consider additional sources of reclaimed water from the tailings dam and open pits.

It is believed this aquifer holds water sufficient for the life of the project, permission has additionally been granted to use this water.

Four wells within the Barranquillas basin wellfield will supply raw water via an aqueduct, with a corresponding booster station, to the main raw water storage and fire water tanks for further treatment and distribution.

Raw water will be used for dust suppression, reagent mixing, gland water, processing requirements, workshop and laboratory requirements. Additional water will be reclaimed from the TSF using a reclaim water barge pumping directly to the cyanide recovery thickener and the process water tank.

18.4.2 Raw Water and Fire Water Distribution

Raw water storage tanks will feed the water distribution system to provide water for processing, reagent mixing and gland seal. Booster pumps will be installed to provide high pressure water to systems that require it, including pump gland seal water. Fire pressure pumps and control systems will be installed at the raw water storage tank to provide pressure to the fire suppression system.

There will be two main reservoir tanks for raw water storage fed by the water pipeline. These tanks will have a capacity of approximately 7,226 m^3 with a fire water reserve capacity of 1,760 m^3, located next to the RO plant. The tank will supply all water for road and crusher dust control.

The secondary tank will be located within the process plant area, to allow gravity flow from the primary tank and supply process water with a capacity of 1,378 m^3.

18.4.3 Potable Water System

A potable water system will be installed to supply water to the camp area, laboratory, office, water plant hose stations and safety showers with a total requirement of 6.8 m^3/hr. The potable water system will consist of a pre-filtration unit coupled to a reverse osmosis unit and associated piping, tankage, and controls.

18.4.4 Sewage Treatment System

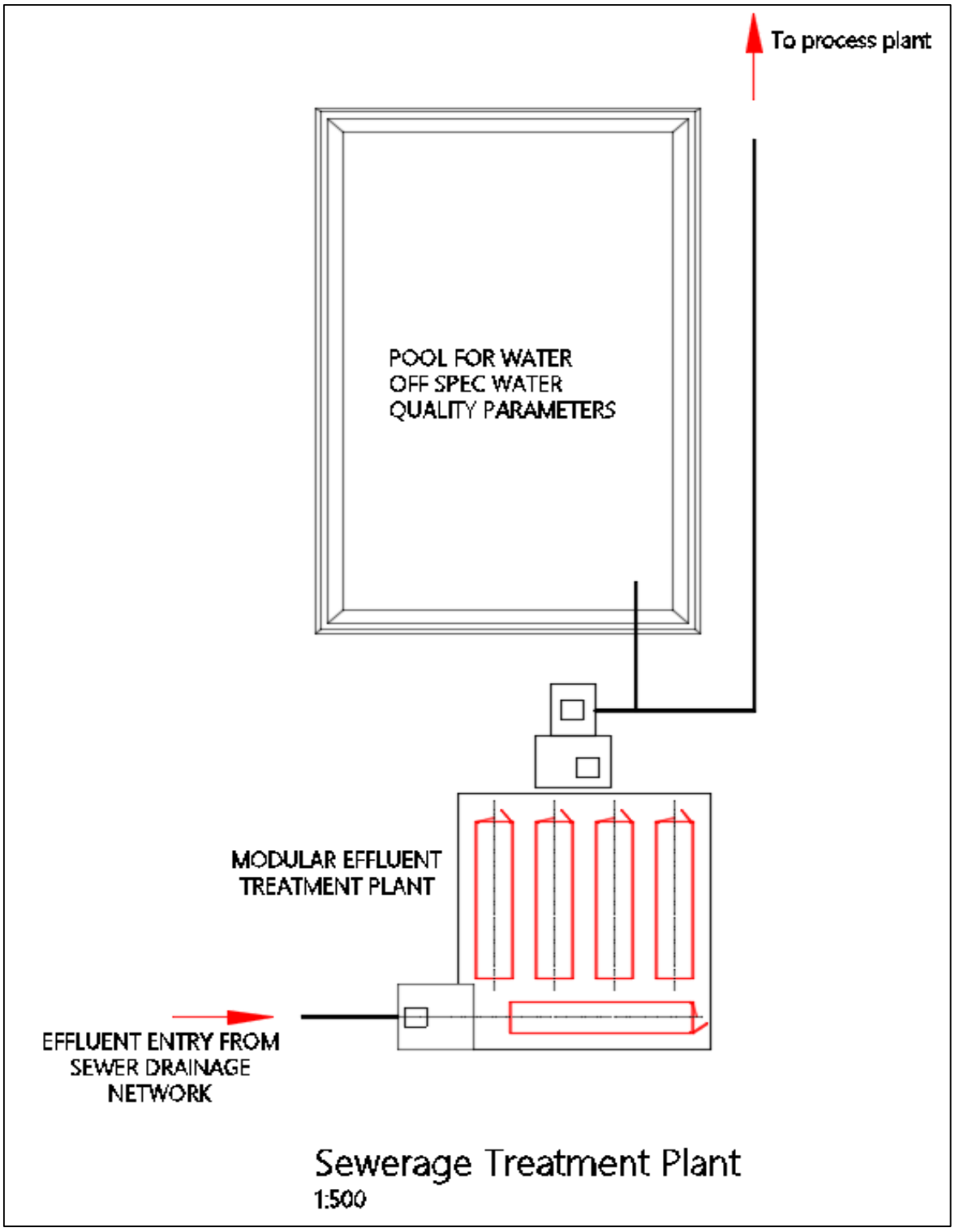
The sewage treatment plant, shown in Figure 18-5 will treat sewage coming from the camps and service hub network. A modular effluent's treatment plant will be constructed, capable of processing effluents for up to 1,200 people during the construction phase and will be resized for 400 people during the operations phase. The wastewater treatment plant will also be managed from the fuel station and truck scale office container.

Plant design will be based on a daily water consumption of 150 l/person and an organic load of 66 g DBO5/(day-person). It will be able to process up to 180 m^3/day and 80 Kg of BDO5 during construction and 45 m^3/d and 20 Kg of BDO5 once the project is in operation. Considering 250 mg/l at the inlet and less than 50 mg/l at the outlet. Sludge will be recovered and shipped to Salta for final disposal and treatment. Treated fluids will be infiltrated into the ground.

As the Project is in an area of low temperature, the active process will be enclosed in an outdoors container to keep the temperature in the range of 10 to 15°C.

The sewage treatment plant will be composed of the following:

- A modular plant to treat a total flow of 200 m3/day.
- Elevator station (Pumping chamber).



Source: AbraSilver Resource Corp., 2023

Figure 18-5: Sewage Treatment Plant

18.5 Power Supply

18.5.1 Electrical Load

The predicted electrical demand is approximately 20MW during operation.

This estimated load is based on the current mechanical and electrical equipment list, including the process plant, reagents, camp, service hub facilities and nominal growth / changes of auxiliary loads over time.

18.5.2 Power Generation

As there is presently no power grid available in the nearby area, power will be generated off-grid with a hybrid power generation plant, composed of solar panels arrays and diesel generators as the primary power source during construction and operations. Off-site facilities, not supplied from the main power plant due to distance, are to be supplied by separate power generation systems. Off-site facilities are the raw water wellfield, the water pipeline booster station, the tailings water recovery system, and the nitrate pad.

The total power generation capacity has been established at 20 MW, considering performance losses of almost 35% due to high altitude and a working capacity of 80%. Reserve capacity will be maintained to absorb peak loads and/or cover possible genset failure.

The diesel power plant is designed to cover the full site power demand, with 10 operating generators and 2 back-up of 2.8 MW each. One of the spare generators will be mobile and available for off-site emergency generation.

15% of the on-site PV power requirements (~ 3 MW) will be covered by an initial photovoltaic system (PV) which will be coupled with a battery energy storage system (BESS) to guarantee continuous power supply to camp, service hub and outdoor lighting, to secure constant power availability in case of fuel supply disruptions due to weather or road blockages.

The remaining part will provide daytime power to charge the BESS and support process operations by synchronizing with the diesel generators through the corresponding energy management system (EMS), thereby reducing emissions and providing fuel cost savings. The EMS will ensure that if the BESS is fully charged, the PV-generated power will be utilized as the primary power source to run the process plant and its site ancillaries.

The EMS will manage the whole power generation, giving priority to PV energy consumption during daytime operation and BESS loading and finally starting the diesel generators when PV power harvest becomes insufficient or inexistent at nighttime.

The main substation, located near the gensets will include an electrical room that houses the system controller and the main power distribution switchgear to provide output power at 6.6 kV. The diesel unit's system controller will also communicate with the main substations by an underground 6.6 or 13.2 kV power distribution network, installed within duct banks to reduce overhead power line maintenance, improve safety, and avoiding interference with cranes or mobile equipment.

The PV system will incorporate PV arrays, inverters, transformers, and controllers, while the BESS will be sized for 3 MW / 20 MWh capacity and include battery modules, inverter, and transformer. An energy management system will be used to charge / discharge battery power. If the PV system or BESS cannot provide power, the diesel gensets will back-feed power to the PV-powered facilities.

The energy management system EMS will communicate with the diesel gensets controller, distribution switchgears, the PV system controller and BESS to operate diesel generators, synchronize and manage PV generated power based on load demands, available PV power and BESS capacity.

For off-site power demands, diesel generators with back-up will be installed at the raw water wells, water pipeline booster station, at the TSF water recovery system and at the nitrate pad.

18.5.3 Power Distribution

The power plant will generate at 6.6 KV, 50 Hz, 3 phase and 3 circuits. The site distribution voltage will be transformed to low voltage, rated at 380/220V, 50 Hz, 3 phases and 4 circuits (3 phases + neutral) for power supply to motor control centres (MCC) and power distribution centres (PDC). All control systems will have an uninterruptible power supply (UPS) and emergency lighting will have battery back-up to keep critical processes operational.

Medium voltage equipment (Ball Mill, SAG mill, etc.) will be operated at 6.6kV. For transformation to low voltage, the transformers will be liquid-filled, pad mounted and rated for outdoor installation. The MCCs and PDCs will be housed indoors.

The estimated electrical load requirement for the mine site is summarized in Table 18-1.

Table 18-1: Estimated Electrical Load Requirement

Electrical Power Load Estimate	
Area	Electrical Load, MW
Process Facilities	
Crushing and conveying	0.6
Grinding	10.0
Leaching	1.4
CCD	1.1
Detox, Tailings handling	1.0
Merrill-Crowe	1.5
Reagents	0.5
Water, Air	0.5
Non-Process Facilities	
Water Supply	0.5
Camp	0.7
Ancillary (Offices, Sub-Station, Truck-shop)	1.0
Mine	0.5
Total	19.3

The total diesel fuel consumption for power generation has been estimated at 100,000 litres per day.

All installations, including electrical equipment, instruments, cables, lighting, and grounding will be installed per national, provincial, and local standards. The lighting will be of the LED type. All areas will be equipped with emergency light fixtures utilizing battery packs which will provide a minimum of 90 minutes of illumination.

18.6 Waste Dumps

Two (2) waste rock dump facilities will be constructed. One East of the Oculto pit (East Main Dump-EWRSA) with a capacity of ~ 302 MM tonnes and one north of the TSF (North Secondary Waste Dump-NWRSA) with a capacity of ~9 MM tonnes.

The main dump location has been selected due to the existence of a granitoid basement. providing geotechnical stability alongside convenient topography. It is not near the property's borders and its distance to the pit exit is acceptable. The secondary dump location has been selected to provide uninterruptible access to a dump to avoid stoppage in case the main waste dump access road is blocked. This also guarantees sufficient waste rock to cover the TSF during remediation works.

The waste rock piles will be built in 30 m lifts to an overall slope of 35 degrees. The inter-bench slopes will have the corresponding rock repose angle of 27 degrees. Details of both WRSA specifications are found in Section 16.5 of this report.

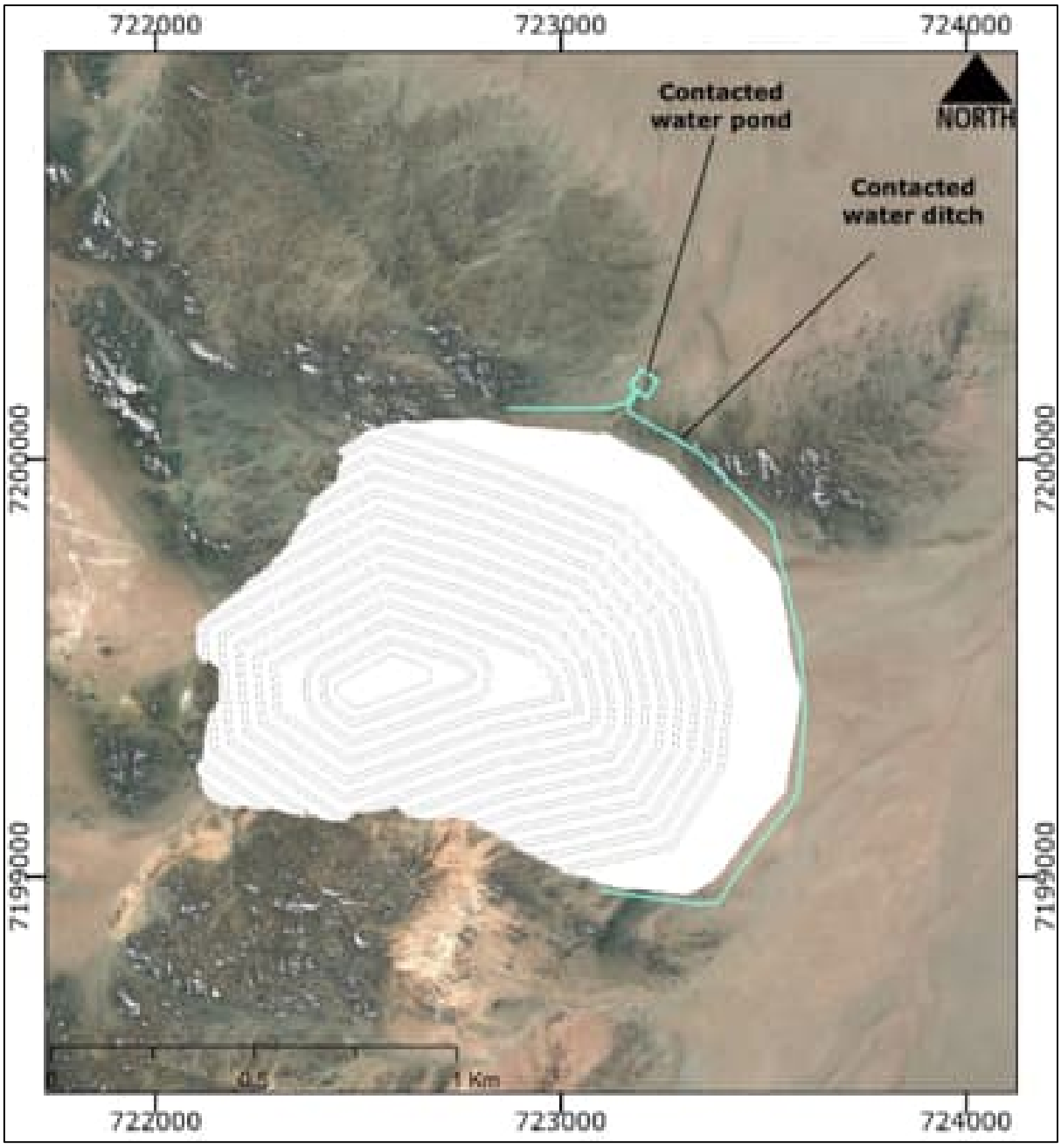
Waste rock will be placed in subaerial storage which will result in exposure of material to atmospheric conditions. Fine-grained portions of exposed material are susceptible to weathering processes that can lead to the mobilization of constituents through oxidation and dissolution. Water that infiltrates into the waste rock pile and/or runoff that flows along the surface of the waste rock dump is referred to as “contact” water.

Contact water, generated from water-rock interactions, will contain soluble constituents (i.e., major ions, metals, and nitrogen species) from mineral weathering by-products and from residual explosives from blasting. This can persist in the waste rock, is water soluble and a source of ammonia and nitrate.

Uncontacted floodwater recollection ditches will be located around both waste rock dumps. The contact water collection ponds/sumps will be located at topographical low points to collect runoff and control physical and chemical parameters. Contact water will be transported to the TSF, or reutilized. The control parameters of contact water are determined according to Provincial Law No. 7017, Water Code of Salta province.

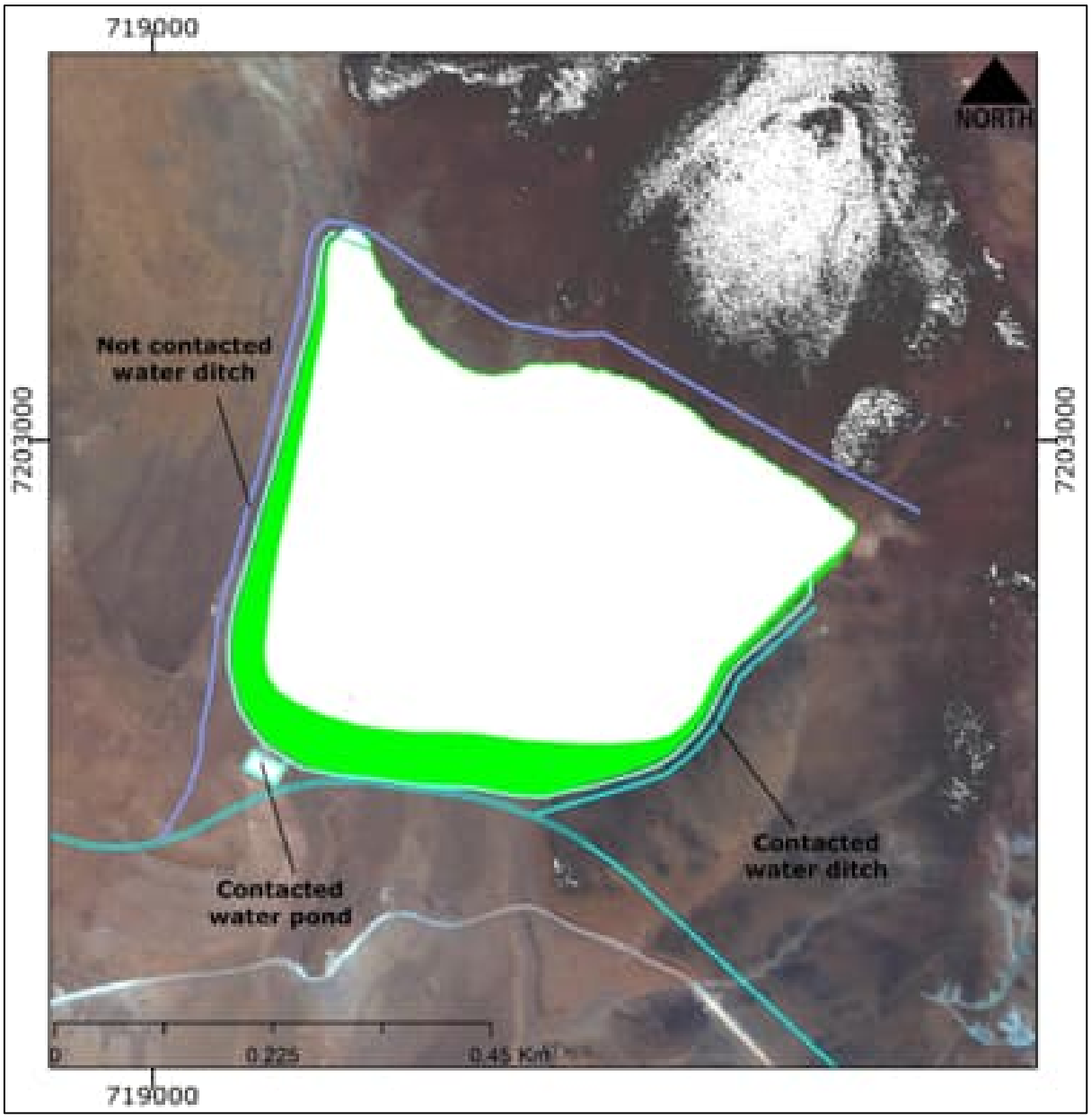
Figure 18-6 shows the main waste dump located on the East side of the project facilities, in its final stage.

Figure 18-7 shows the secondary waste dump located north of the TSF, in its final stage.



Source: AbraSilver Resource Corp., 2023

Figure 18-6: East Waste Dump Contact Water Management



Source: AbraSilver Resource Corp., 2023

Figure 18-7: North Waste Dump Contact Water Management

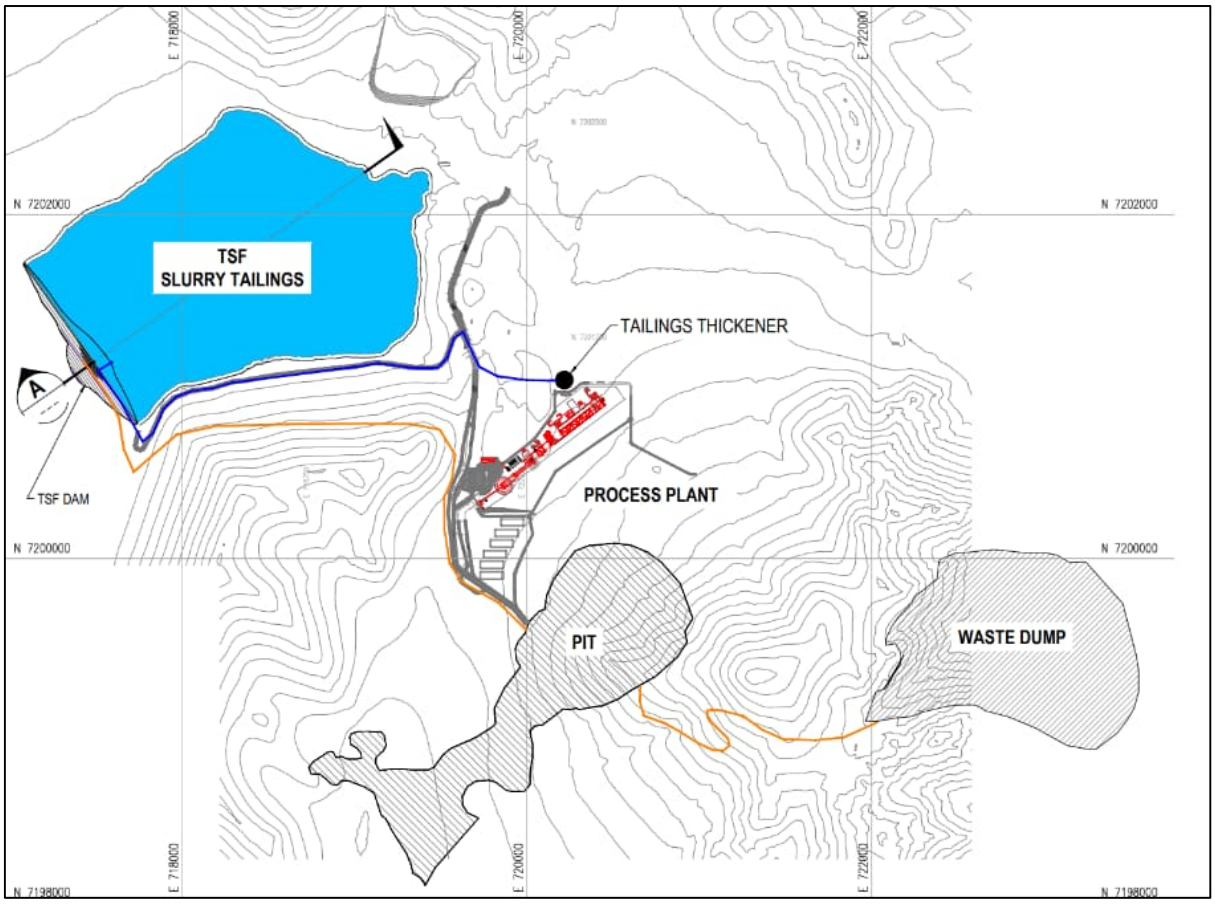
All control ponds will be lined with PEAD membrane and leak control tests will be periodically performed to avoid filtration to natural ground.

Site test work results indicate that only deep gold waste produces Acid Rock Drainage. The waste coming from this domain will be embedded into the main waste dump, with a PEAD membrane installed before dumping.

18.7 Tailings Storage Facility (TSF)

18.7.1 Tailings and Disposal

The TSF for Diablillos has been designed as a conventional dam-and-slurry containment, within a closed valley approximately 2km to the NW of the proposed Process Plant and is depicted in Figure 18-8. It has been designed in general accordance with the Global Industry Standard for Tailings Management, and other, similar best-practice guidelines and documentation.



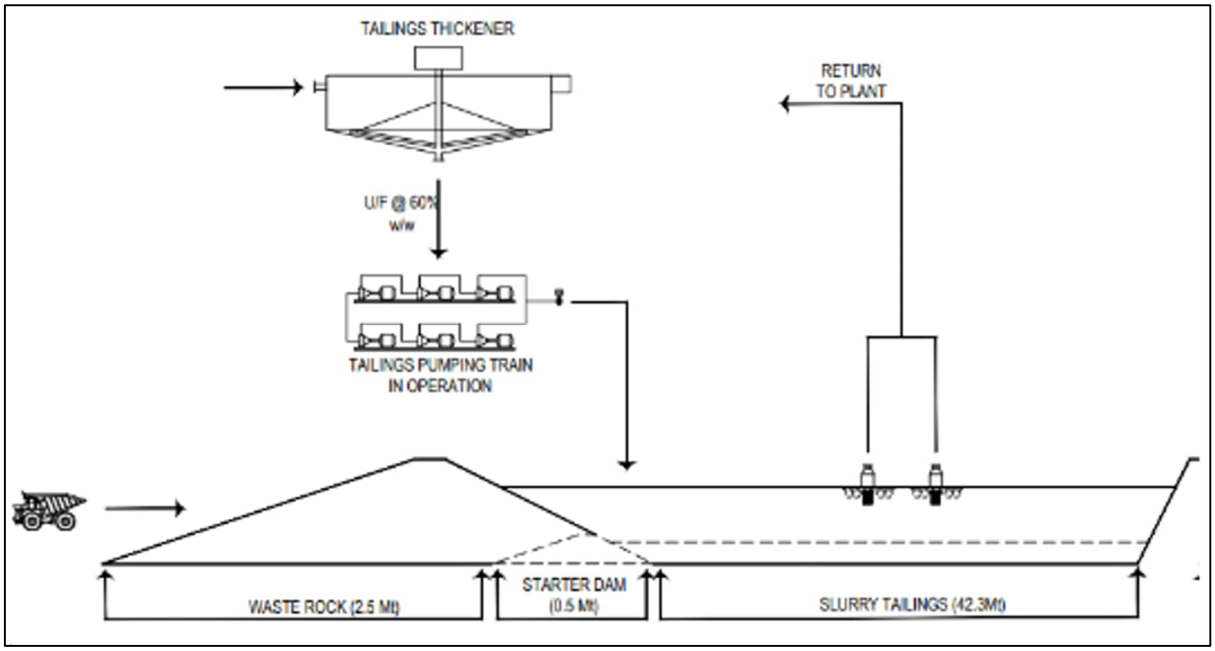
Source: ENIVS, 2025

Figure 18-8: Diablillos TSF

The proposed TSF will receive tailings thickened to approximately 60% solids content by weight, after processing and thickening at the process plant. The thickened tailings will be pumped via pipeline the impoundment and discharged over the area of the impoundment through spigots distributed around the perimeter of the facility.

The dam will be constructed in five phases over the life of the operation, using selected and processed non-PAG waste rock materials from the mining operations. The dam raises will be developed using the downstream raising method. The TSF has been designed for a total storage of 42.3 Mt, which includes all the tailings produced by the plant during the LOM. The valley in which the TSF is located has significant additional capacity readily available, through raises beyond the currently designed height.

A schematic of the tailings management is shown in Figure 18-9.



Source: ENVIS, 2025

Figure 18-9: Schematic of Tailings Management

As depicted in Figure 18-9, supernatant water from the consolidated tailings, as well as capture surface runoff, will be collected and returned to the Process Plant via the barge pump system. The Diablillos site is subject to very high annual evaporation and, as such, water losses from the pond and the wet tailings beach are anticipated to be relatively high.

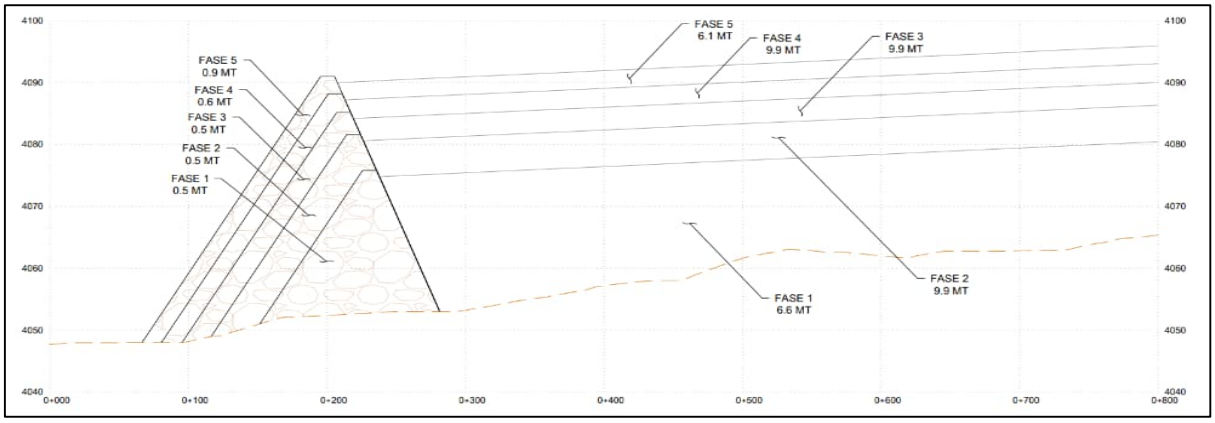
The design currently includes diversion channels to be constructed around the perimeter of the TSF, in order to minimize the quantity of contact water. The diversion channels will route surface runoff around the TSF, and discharge to the water course downstream of the dam and seepage collection pond.

The total amount of fill placed in the dam will be approximately 1.6 Mm^3 (3.0Mt); this will include primarily a structural zone comprising coarse rockfill and also filter and transition material zones at the upstream face.

The current design includes lining the impoundment and dam face with an HDPE geomembrane as a low-permeability barrier to potential seepage from the tailings. As a contingency, a seepage collection pond would be constructed downstream of the tailings dam, to capture residual seepage, and this would be returned via pumping as contact water, ultimately to the process plant.

Preliminary site investigations have been carried out at the TSF and indicate that the dam and the impoundment foundations consist primarily of a consolidated sand-gravel-silt material, with 10-50 cm of organic soil materials on surface. The dam and impoundment preparation will include removal of organic materials, and any loose soil, to construct the facility on well-compacted ground.

A representative section of the TSF dam, including the 5 staged raises, is shown in Figure 18-10. The stage has been planned such that sufficient capacity is always available during operation, including allowances for appropriate freeboard. Additionally, preliminary slope stability analyses indicate that the section will be stable under both static and dynamic (earthquake-induced) conditions.



Source: ENVIS, 2025

Figure 18-10: Diablillos TSF Representative Section

18.8 Topsoil and Overburden Stockpiles

The first 50 cm of topsoil, according to local standards, will be stored in the reclamation material stockpile (RMS). Sedimentation ponds will be built to settle out solids before release to the environment. A perimeter ditch will be constructed at the toe of the topsoil stockpile to keep material intact. The stockpiled topsoil will be used for site rehabilitation upon mine closure.

Overburden material from the JAC open pit will be used as construction material for platforms, ROM pad and road. Remaining unused overburden coming from the Oculto open pit will either be used for further construction or be dumped on one of the two waste stockpiles.

18.9 Steel Buildings

All main steel buildings for the camp, service hub, process plant and off-site will be constructed as a pre-engineered steel frame and assembled on site.

Within the foundry, a reinforced concrete block safety vault will be installed to ensure proper security.

18.9.1 Camp Maintenance Warehouse & Workshop

Steel buildings are considered for the camp warehouse, maintenance, instrumentation, and light electrical workshop. The camp warehouse will be used to store all kind of camp related consumables and for the storage of empty core boxes and exploration equipment.

The camp workshop will be used for minor repairs and for specific electrical and instrumentation repairs. Major maintenance and repairs will be performed within the ancillary services hub, within the truck-shop building.

18.9.2 Waste Management Building

A separate steel building for domestic waste classification and compaction has been included, to handle all generated waste until disposal.

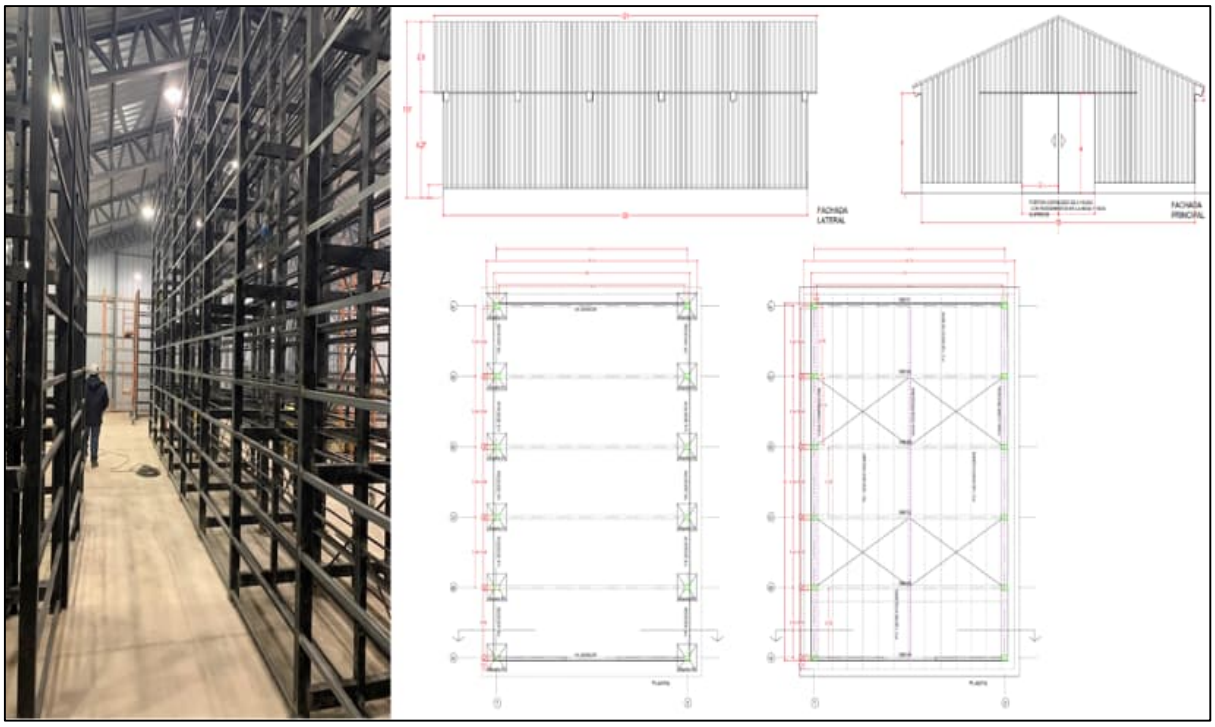
18.9.3 Drilling core Box Storage Building

Exploration facilities have been set aside for core preparation, analysis, and final storage. Geological exploration is expected to proceed independently of the mine operations.

A steel building for core storage has already been installed, with a capacity of approximately 30,000 m of core where all historical and current Phase I and II core boxes are stored.

Figure 18-11 shows the building and steel racks for core storge.

A second core storage building space has been reserved in case future expansions of core storage capacity is required, i.e., Phase III and subsequent drilling campaigns.



Source: AbraSilver Resource Corp., 2023

Figure 18-11: Core Storage Building and Racks for Approx. 30,000 m of Cores

18.9.4 Truck Shop

The truck shop is the main building and centre of activities of the services hub (Figure 18 30). It will host the maintenance area for the off-road mine fleet, the mining equipment, standard trucks, earthworks equipment and the light vehicle fleet including forklifts, scissor platforms, and pick-ups.

The truck-shop will be composed of eight major bays, four for major off-road truck maintenance, two for major crawling equipment, one for lubrication and another for tire repair. Additionally, there will be a fully enclosed truck washing cabin/bay. Two of the main bays will have steel rail flooring so crawlers' can be maintained.

18.9.5 Plant & Mine Warehouse

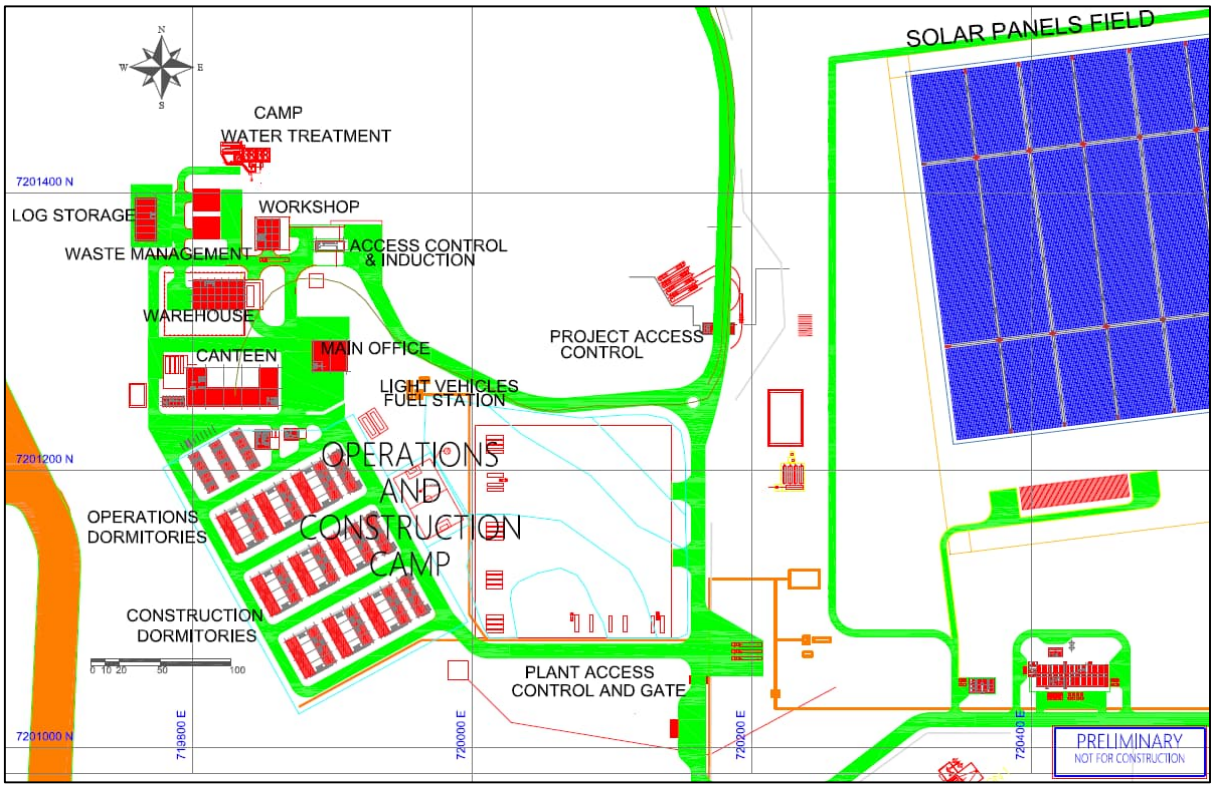
A steel building for hosting the plant, mine and truck-shop warehouse and its fenced external lay-down area will be located opposite the truck-shop.

This warehouse will store all major equipment and spare parts. Afterwards it will serve as storage for spare parts, tools, consumables, and reagents.

18.10 Modular Buildings - Construction and Operations Camp, Service Hub

18.10.1 Camp General Arrangement

The following (Figure 18-12) shows the main construction and operations camp facilities.



Source: AbraSilver Resource Corp., 2023

Figure 18-12: Camp Facilities

18.10.2 Modular Buildings

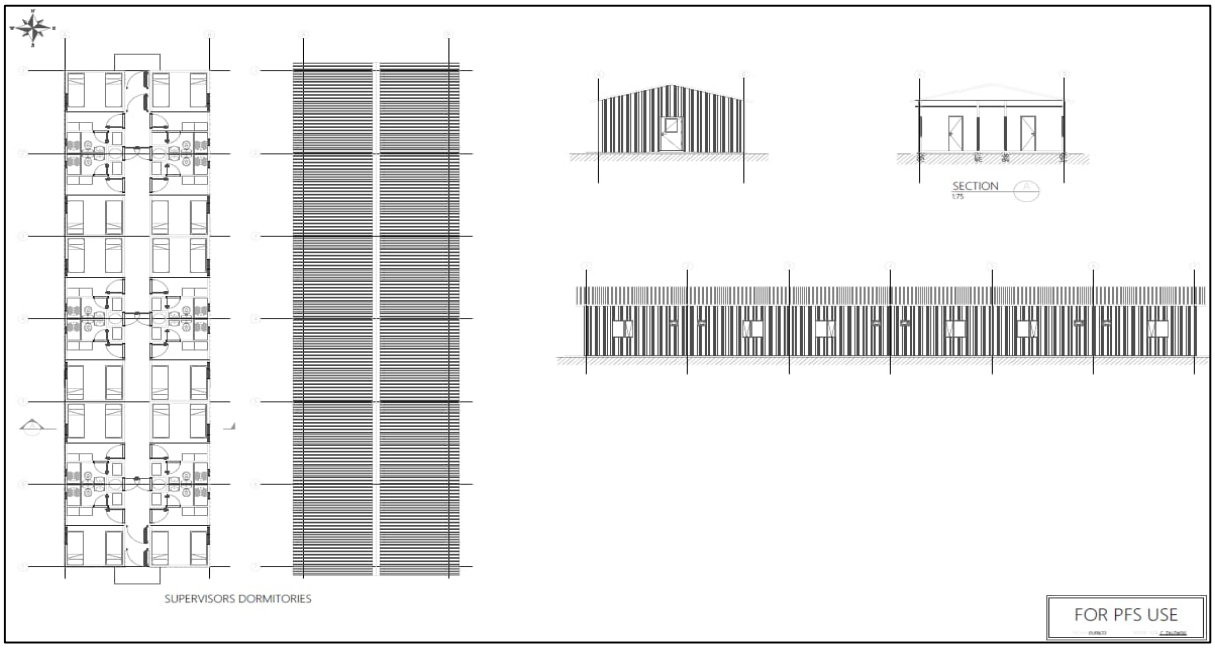
The following facilities were considered within the camp and service hub.

- Accommodation and hospitality areas, including dormitories, canteen, recreational, laundry and medical emergency station.
- Administrative areas, offices, access control, hospitality reception and induction classroom.
- Geology, exploration, and core storage.
- Camp warehouse maintenance, camp maintenance workshop including light electrical and I&C.
- Water, waste treatment, temporary storage, utilities networks.

18.10.3 Dormitories

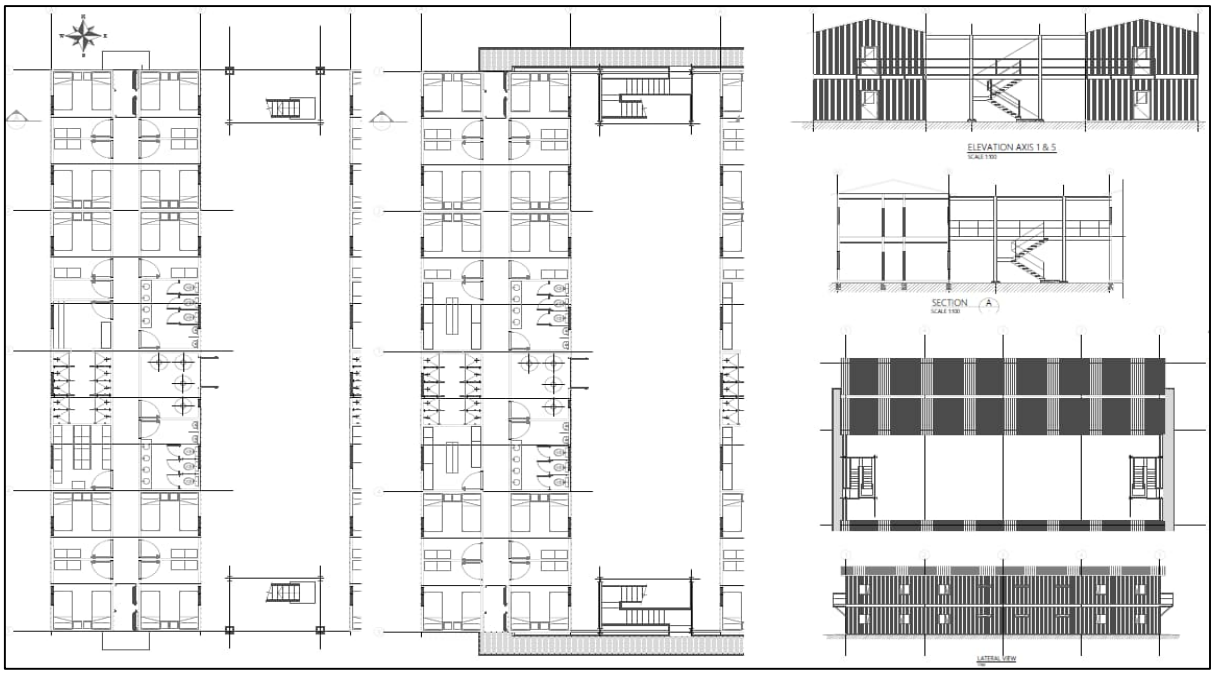
The capacity of the dormitories is 890 people, considering 840 construction workers, 40 supervisors and 10 managers. This accommodation will be distributed in three types of dormitory modules in containers:

- One single level container module for management, with individual bedrooms and bathrooms, hosting up to 10 people (managers + high level visitors).
- Two single level modules for supervisors, with sleeping accommodation and private bathrooms, with capacity for 40 people (supervisors and visitors). These modules follow the same configuration as the management ones.
- Twenty-one double level container modules for operators and construction staff, with one common shared sanitary facility and 20 beds on each floor level, totalling 40 persons to be accommodated per module.
- Each dormitory will have a minimum surface of 9 m2 to comply with local regulations and union standards. Typical dormitories module configurations are shown in Figure 18-13 and Figure 18-14.



Source: AbraSilver Resource Corp., 2023

Figure 18-13: Typical Supervisor's Dormitories Lay-Out

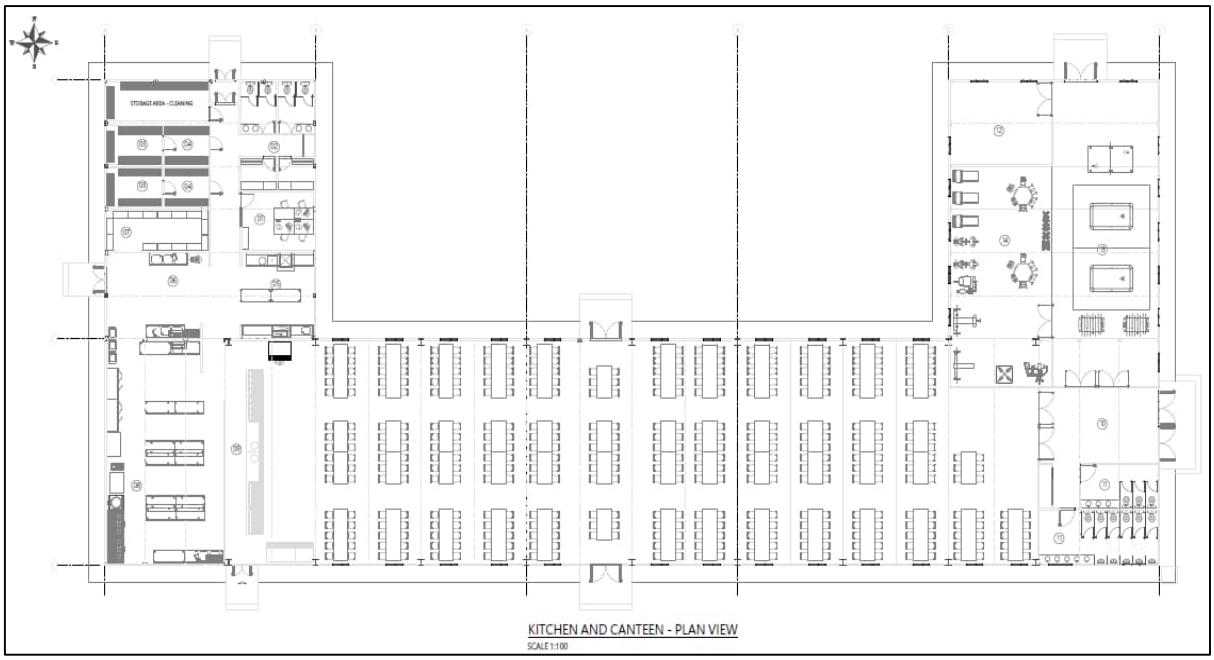


Source: AbraSilver Resource Corp., 2023

Figure 18-14: Typical Operations and Workers Dormitories

18.10.4 Kitchen and Canteen

Kitchen and canteen will be designed to accommodate up to 560 guests during construction phase in two shifts during the day (breakfast, lunch, and dinner). For the operations phase the interior will be redesigned to offer a single meal shift for up to 402 persons according to the provision indicated in the Figure 18-15.

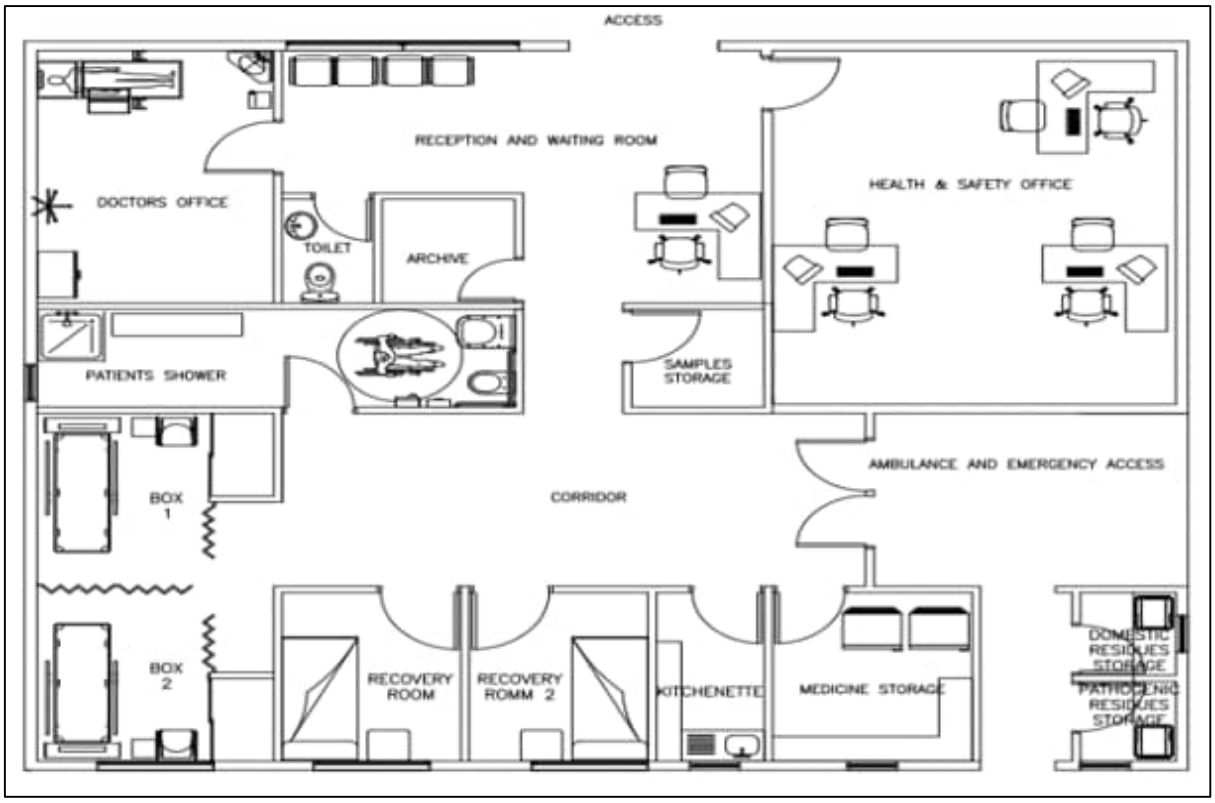


Source: AbraSilver Resource Corp., 2023

Figure 18-15: Typical Lay-Out of Kitchen, Canteen & Recreational Facility

18.10.5 First Aid and Medical Emergency Station

A first aid and medical emergency station is shown in Figure 18-16 and will be installed within the camp. Emergency stations will also be distributed throughout the project facilities, in line with the Health, Safety & environment Design Criteria.

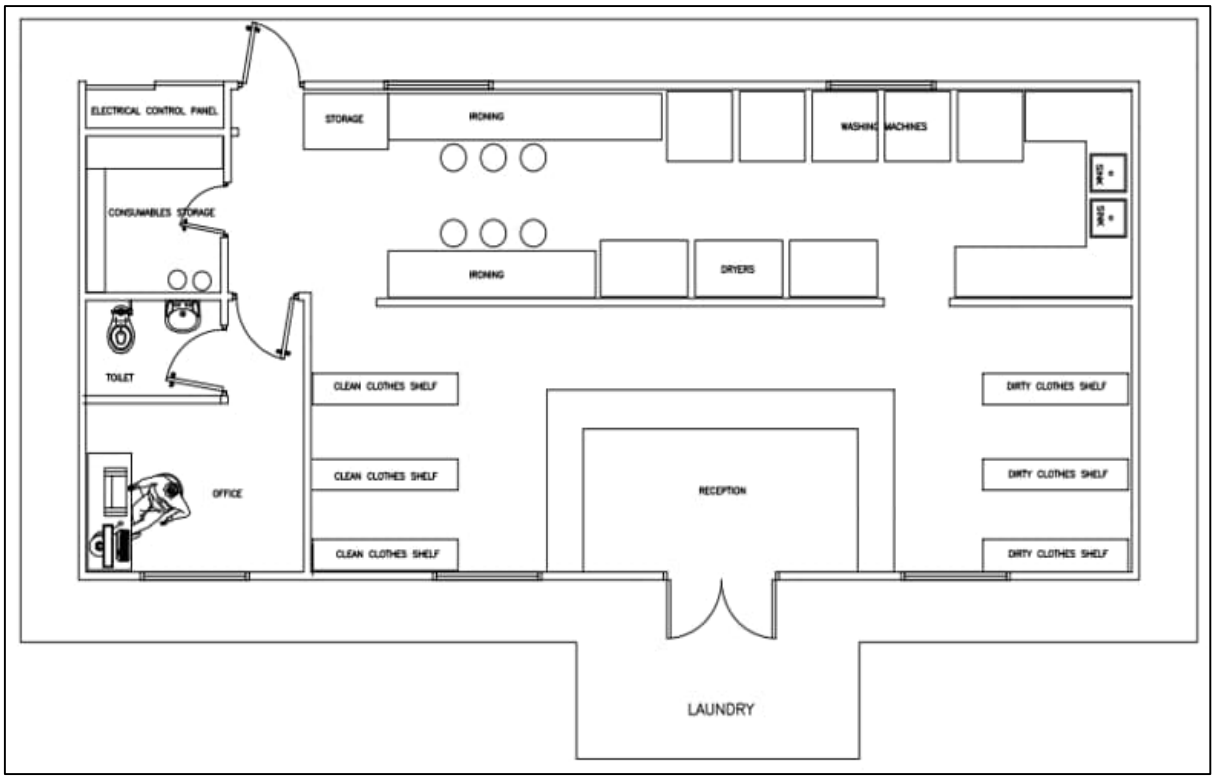


Source: AbraSilver Resource Corp., 2023

Figure 18-16: Typical Lay-Out of Emergency Medical Station

18.10.6 Laundry

A separate building for the camp laundry and staff clothes washing and ironing be installed near the canteen as shown in Figure 18-17. It will host six laundry machines and four driers. The driers will be fed with GLP gas from the gas storage tanks. As it is considered a hazardous area, safety distances and effluents are to be treated accordingly before being released into the sewage network.



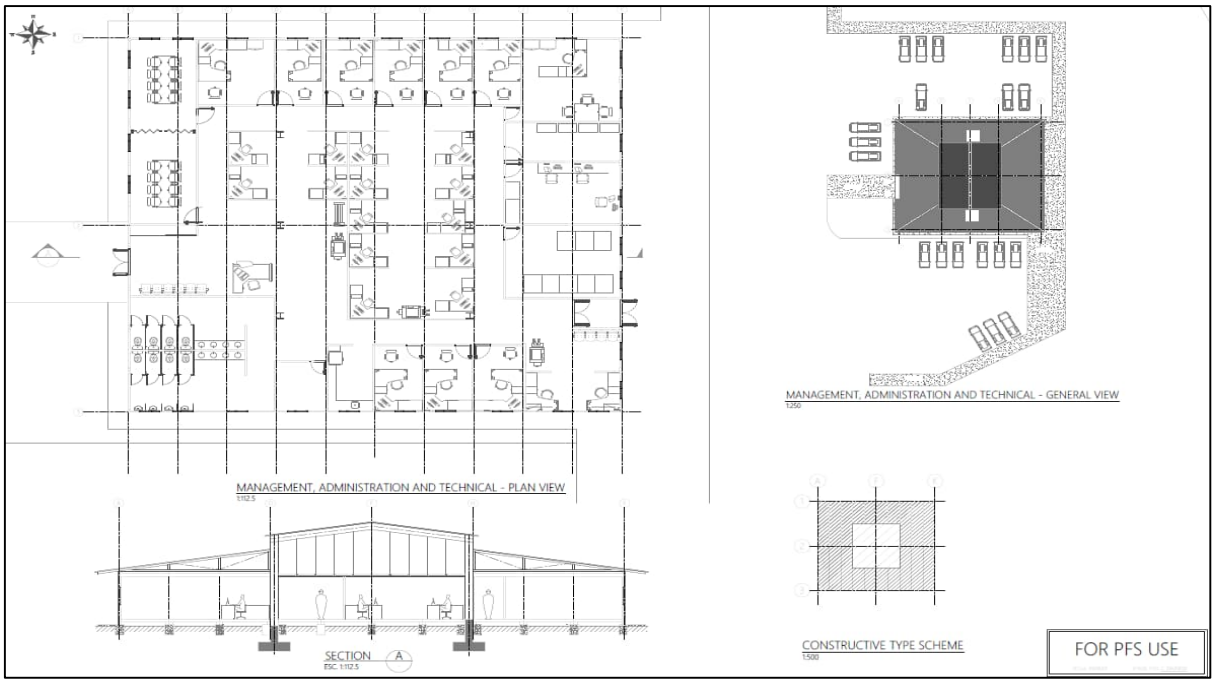
Source: AbraSilver Resource Corp., 2023

Figure 18-17: Typical Lay-Out of Laundry

18.10.7 Administration Offices, Access Control, & Access Barriers

An access control booth will be located at the project facilities main entrance, with a parking space for trucks and light vehicles. Bus drivers, visitors and minor supply trucks will be instructed to proceed to the main camp gate. This will be used for document control, hospitality, health, safety, and environment inductions. Truck drivers, suppliers and service contractors will be diverted to the mine and service hub and its corresponding warehouse or reagent storage.

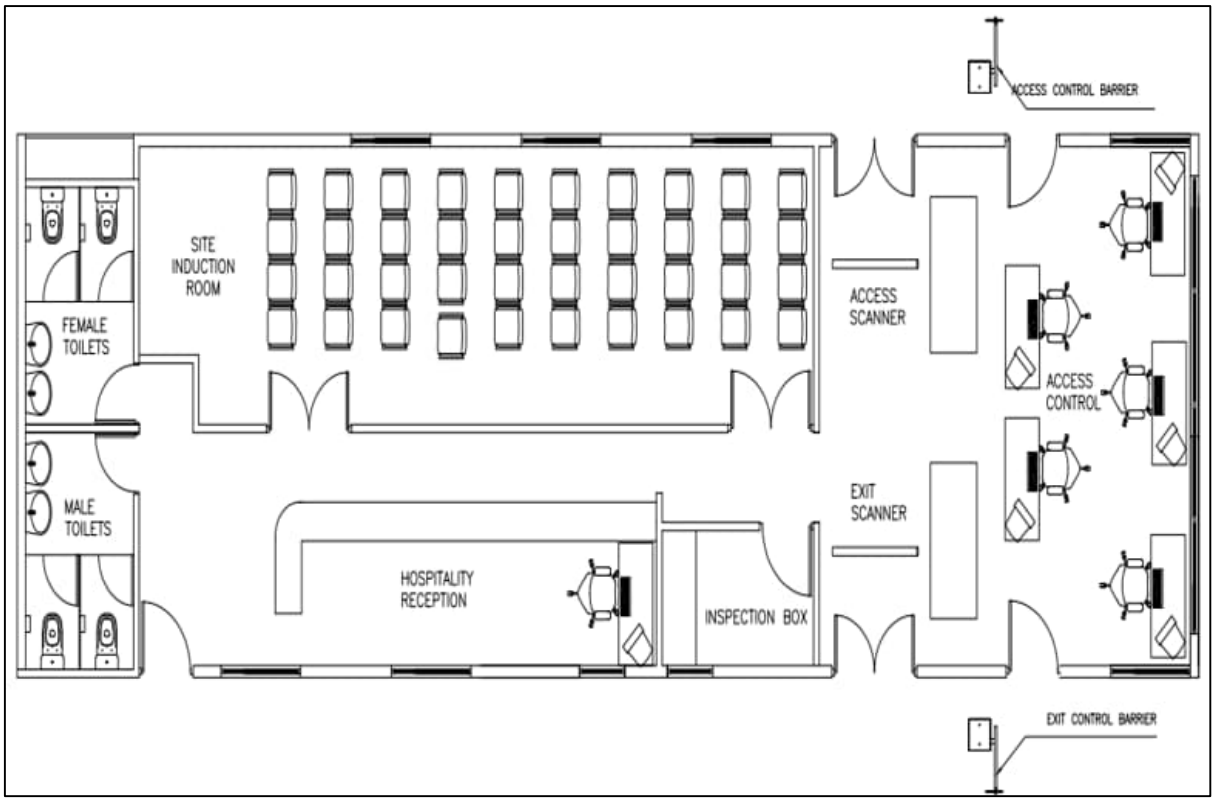
The administration buildings (Figure 18-18), comprise the project site offices, the hospitality management offices, access control and project access barriers.



Source: AbraSilver Resource Corp., 2023

Figure 18-18: Typical Lay-Out of Administrative Offices

Figure 18-19 shows the main camp access control barrier, induction rooms and hospitality reception desk.



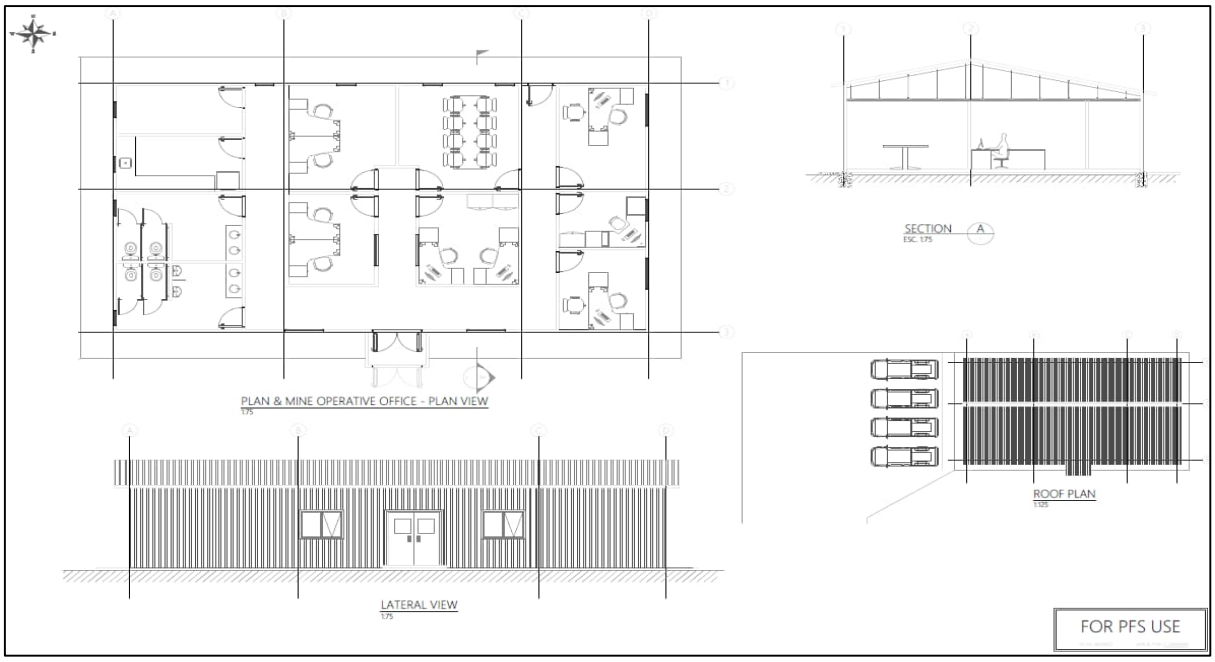
Source: AbraSilver Resource Corp., 2023

Figure 18-19: Typical Lay-Out of Access Control, Hospitality Reception and Site Induction

18.10.8 Service Hub, Plant & Mine Operations Offices

An office building as shown in Figure 18-20, will be located close to the process plant entrance with adequate parking. These offices will be used jointly by plant and mining operations staff. They will accommodate the plant superintendent office and supporting technical team.

Each area will include briefing / meeting rooms for daily briefing activities.

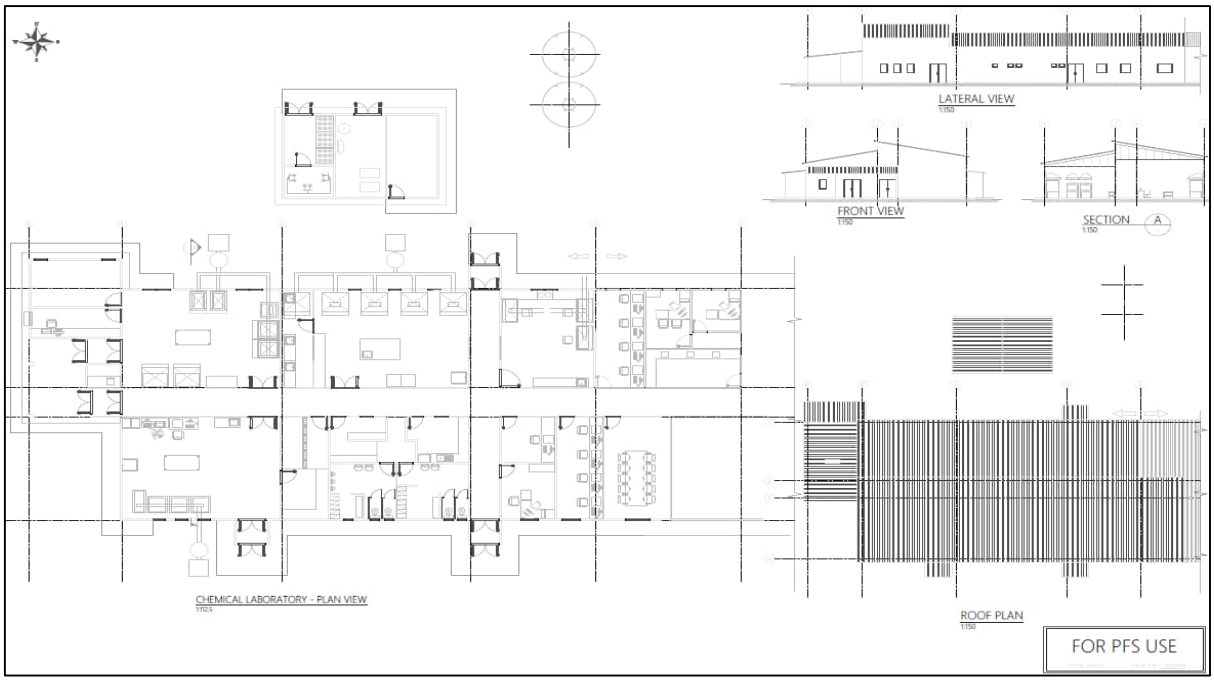


Source: AbraSilver Resource Corp., 2023

Figure 18-20: Typical Lay-Out of Work Front Offices for Mine and Process Plant Staff

18.10.9 Laboratories

The central laboratory will be next to the plant operations office facilities, with easy access for light vehicles coming from the mine and processing plant. Figure 18-21 shows the typical combined chemical and metallurgical laboratory.



Source: AbraSilver Resource Corp., 2023

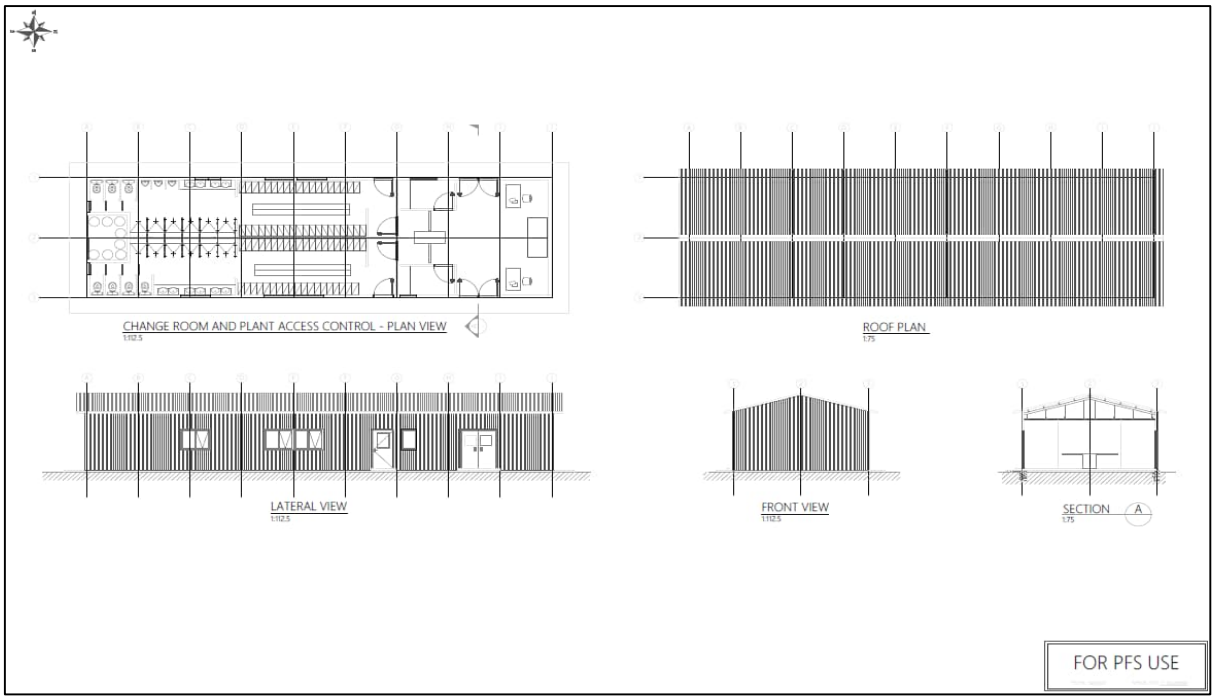
Figure 18-21: Typical Lay-Out of the Chemical & Metallurgical Laboratory

The chemical laboratory (Figure 18-21) will perform all necessary chemical test work related to the plant operation, including Atomic Absorption, fire assays, etc., to determine mineral contents of the different solutions and optimize process variables.

18.10.10 Service Hub, Plant Access Control and Change Room

As the process plant will be located within an enclosed compound, control room, laboratory and work front operators will have to undergo scrutiny and change clothes to access the plant.

A change house with lockers and an access control will be installed next to the process plant access gates as shown in Figure 18-22.



Source: AbraSilver Resource Corp., 2023

Figure 18-22: Typical Lay-Out of Plant Access Control and Change and Locker Rooms

18.10.11 Powder Magazine

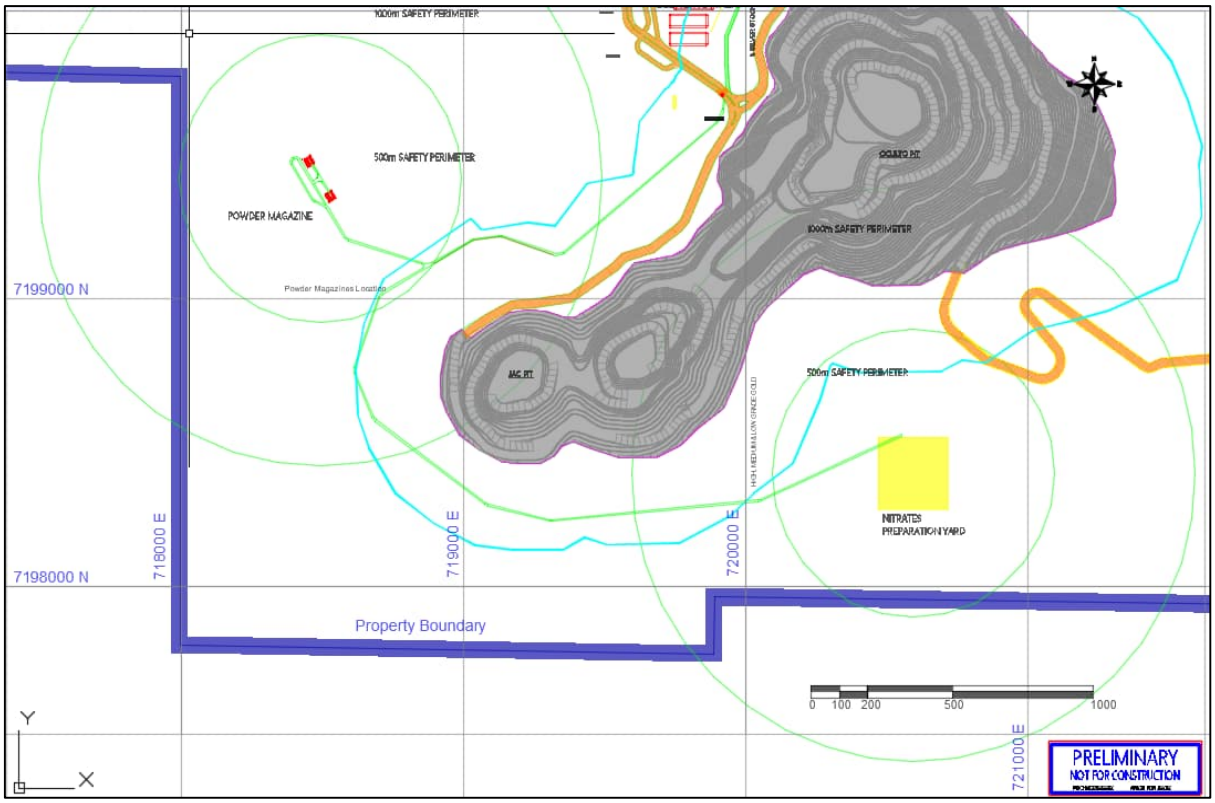
The Powder magazine and nitrates preparation yard are shown in Figure 18-23.

The nearest facility to the powder magazine is 1.5 km aerial distance to the crusher area.

The bulk explosives facilities are in two areas, with the areas being more than 2,350 m apart. These areas are composed of primary explosives and detonation and nitrates preparation yard.

All the containers are situated within a secure area, surrounded by fencing and managed with regular security patrols.

Raw materials, such as emulsion explosives, fuel oil, primary explosives (dynamite and booster) and blasting accessories are brought to site by road and stored at the explosives facility site until required.



Source: AbraSilver Resource Corp., 2023

Figure 18-23: Explosive Magazines Plot Plan

18.11 Site Services

18.11.1 Diesel Storage

The mine requires diesel to be stored within a diesel fuel tank farm covering a nine-day reserve, totalling 1,500,000 litres. The diesel will be stored in 6 vertical tanks north of the warehouse.

The fuel dispensing station and truck scale instruments and control hardware will share the same modular building.

18.11.2 Site Wide Communications

The mine site will employ a site-wide communications system based on a single mode fibre optic backbone. VOIP telephones, intranet/internet access, and control system network connectivity will be integrated into this fibre backbone, so these systems are accessible anywhere on site. Broadband internet access will be purchased from a satellite internet service provider. The corporate network (intranet) will be isolated from the control system network via a firewalled DMZ (de-militarized zone) network.

Industrial communications from the process plant or water well field will be routed to the central control room. The control room hardware and off-camp offices will be connected to an underground fibreoptics network through adequate polypropylene pipes buried at least 50 cm deep.

The remaining site communications will either be handled with VHF radios/handheld and wi-fi internet with separate band-width capacity for operations and camp leisure use. A steel tower for a VHF communication antenna will be installed at the camp.

18.11.3 Site Fencing

Fencing will be installed only around off-site facilities including raw water wells, booster station, powder magazine and reclaim water gensets at the TSF. At site, only warehouse lay-down area, foundry and gravity concentrator will be fenced to secure high value material. The remaining facilities will be chain linked wherever the H&S staff consider it necessary for safety reasons.

18.12 Utilities Network and Water Distribution

The following utilities networks will be installed within the camp, service hub, laboratories, and specific consumption points.

- Fresh water distribution network.
- Firefighting distribution network.
- Sewage recollection network.
- Gas distribution network (Camp, laboratories, and foundry).
- High (HV) and Low Voltage (LV) electrical distribution network.
- Industrial communications network (fibreoptics).

18.12.1 Firefighting Water Distribution Network

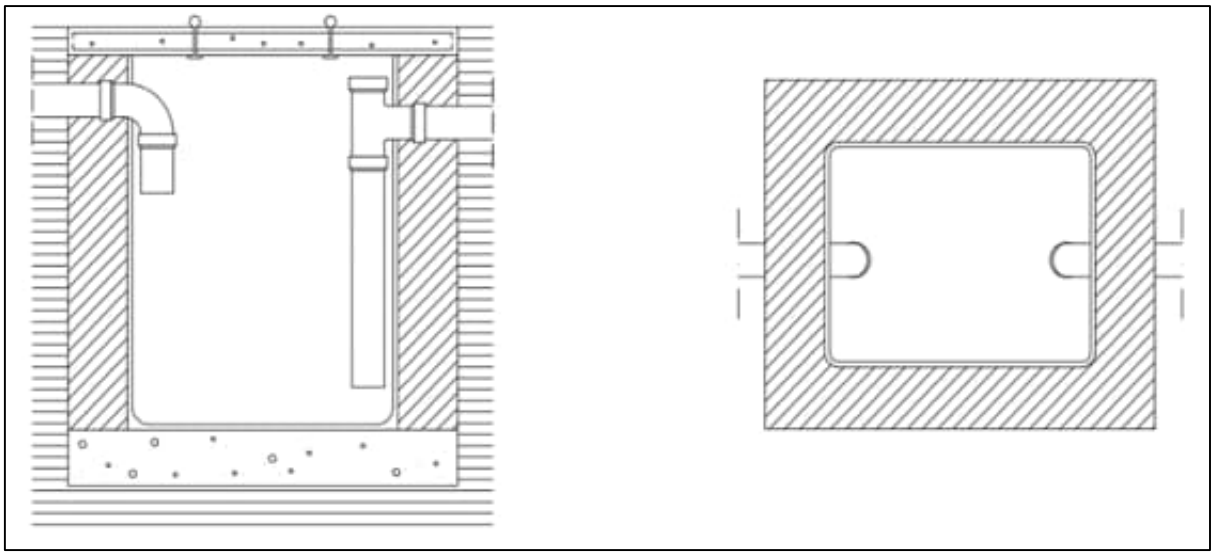
An underground firefighting water network will be installed throughout the camp facilities. A separate one will be considered for the process plant and industrial hub.

The corresponding jockey pump and emergency power system for firefighting will be installed next to the freshwater storage and treatment plant.

The network will be composed of 1 m buried carbon steel pipes which will distribute water to fire hydrants and hose cabinets installed sitewide according to the safety standards.

18.12.2 Kitchen Grease Separation Chamber

Wastewater coming from the kitchen and canteen will undergo grease separation before entering the sewage network. The chamber will be emptied on a weekly basis and grease stored for final disposal at the waste dump (Figure 18-24).



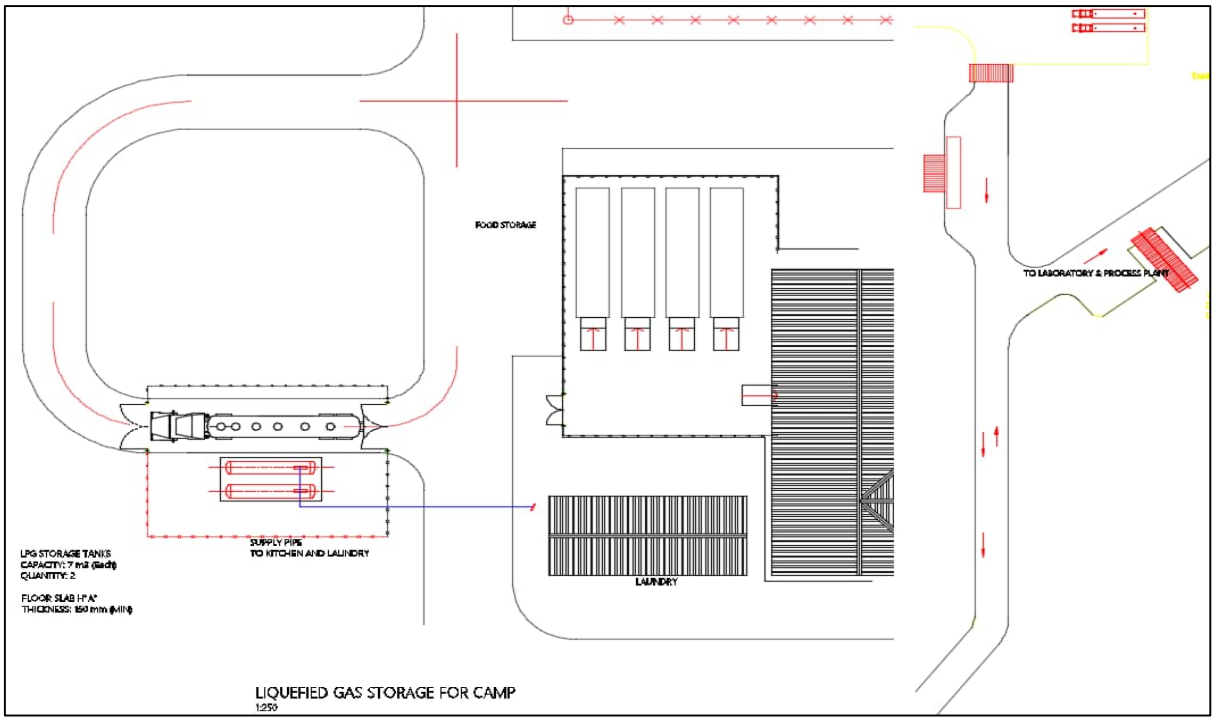
Source: AbraSilver Resource Corp., 2023

Figure 18-24: Typical Lay-Out of a Concrete Grease Separation Chamber

18.12.3 Gas Distribution Network

Bottled petroleum liquified gas will feed kitchen ovens, and eventually laundry driers will be installed. Gas will be distributed through buried epoxy paint coated carbon steel pipes to the kitchen and laundry buildings as shown in Figure 18-25.

According to gas consumption of laboratories and foundry equipment, another petroleum liquified gas tank farm of 7 m^3 will be installed close to both facilities.



Source: AbraSilver Resource Corp., 2023

Figure 18-25: Typical Lay-Out of a Gas Distribution Network

19. Market Studies and Contracts

The following section has been taken in its entirety from the previous NI 43-101 Technical Report Updated Pre-Feasibility Study, effective December 3rd, 2024, and report date January 17th, 2025.

No specific marketing study was completed for the potential sale of gold/silver Doré from the Diablillos Project. Silver and gold refining terms used in the analysis are based on company discussions with refiners, industry standards, and are considered reasonable when compared to other published studies.

Historic silver and gold prices (www.kitco.com) were compared against other online sources for accuracy. The twenty-year period from 2004 to 2024 is shown in Figure 19-1 and Figure 19-2.

This study recommends that, as the Project advances towards development, a detailed marketing report and logistics study for end-product shipment should be undertaken to ensure the accuracy of the precise terms. Table 19-1 outlines the terms used in the economic analysis.

Table 19-1: Sales Terms Assumptions

Parameter	Unit	Value
Provincial Royalties	%	3.0
Gold (Au) Payable	%	99.8
Silver (Ag) Payable	%	99.0
Gold (Au) Refining Charge	USD/pay oz Au	4.00
Silver (Ag) Refining Charge	USD$/pay oz Ag	0.70

At this time, no contractual arrangements for shipping or refining exist; nor are there any contractual arrangements made for the sale of the doré. Historical prices for silver and gold are shown in Figure 19-1 and Figure 19-2 respectively, averages can be found in Table 19-2.



Source: AbraSilver Resource Corp., 2023

Figure 19-1: Silver Prices per Troy Ounce 2004 – 2024



Source: AbraSilver Resource Corp., 2023

Figure 19-2: Gold Prices per Troy Ounce 2004 – 2024

Table 19-2: Metal Prices for Silver and Gold

Commodity	Spot Price April 11, 2024	36 Month Trailing Average April 11, 2024	60 Month Trailing Average April 11, 2024	20 Year Historic High	20 Year Historic Low
Silver	$22.66	$23.16	$21.76	$48.41	$5.54
Gold	$2,044.13	$1,858.72	$1,765.67	$2,077.16	$374.55

As of April 11th, 2024, the spot price of silver was trading below the 36-month trailing average, but above the 60-month trailing average price.

The spot price of gold was trading above both the 36-month and 60-month trailing averages.

Industry accepted practices indicate using a 36-month trailing average for Mineral Resources and Mineral Reserves estimation, thus, a silver price per ounce of $23.50 and a gold price of $1,850 per ounce is recommended for the Mineral Resource estimate in Chapter 14 and Mineral Reserve estimate in Chapter 15. The economic analyses in Section 22 of this report is presented using a silver price of $25.50 per once and gold price of $2,050 per ounce to reflect recent market conditions.

19.1 Royalties

The Doré produced and sold from the Diablillos project is subject to a provincial 3.0% NSR royalty and a 1.0% NSR agreed to be paid to EMX.

19.2 Contracts

There are currently no signed contracts for transport and sale of the doré production. However, as there is a ready and well-established market for doré refining, the sale of this product and the revenue from it is well understood.

There are also no current signed contracts in place for the development or operation of the Diablillos project. Those that exist are for the current exploration and camp support activities.

20. Environmental Studies, Permitting and Social or Community Impact

The following section has been taken in its entirety from the previous NI 43-101 Technical Report Updated Pre-Feasibility Study, effective December 3rd, 2024, and report date January 17th, 2025.

20.1 Regulations and Permitting

20.1.1 Environment Requirements

The environmental legislation relevant to the Diablillos Project is as follows:

- National Law No. 24,585, of the Environmental Protection for Mining Activity Act, incorporated into the Mining Code. Salta Province stipulates that the Mining and Energy Secretary is the implementing authority, while in Catamarca it is the Ministry of Mining. The submission of environmental impact reports and the corresponding monitoring is the responsibility of this government authority.
- Provincial Environmental Protection Law No. 7,070, Salta Province. This law declares that it is a matter of Provincial public order that all necessary actions, activities, programs, and projects must be conducted to preserve, protect, defend, improve, and restore the environment and natural resources within a framework of sustainable development. The Provincial government is tasked to function as above except in matters governed by special laws such as Law No. 24,585.
- Civil Code, Art 14 and 240, fall under the category of “collective rights” as recognized by the National Constitution (also called rights of the third generation, with its peculiarities such as rights that do not belong to an individual, but everyone). Article 14 states, “the law does not protect abuse of individual rights where they may affect the environment and collective rights in general.” Article 240 regulates the limits on the exercise of individual property rights including environmental rights.

Article 41 of the National Constitution refers to the rights of individuals residing in an area of productive activity to enjoy a healthy environment without compromising future generations. It also indicates that the Argentine State and Provincial governments will issue rules as appropriate to their jurisdictions. Other related legislation includes the Mining Code of the Argentine Republic.

20.1.2 Environmental Obligations

According to Provincial Law 7070, Article 41, Decrees 3097/00 and 1587/03, the technical manager of any environmental work must be registered as an Environmental Impact Evaluator in the Ministry of Environment and Sustainable Development of Salta Province. Maintenance of this registration requires biannual re-enrolment and the updating of the technical manager’s curriculum vitae. An EIA must be signed as an affidavit, and the person signing the EIA is liable under civil and criminal law for the content of the report.

20.1.3 Permitting

Mining in Argentina is governed primarily by Federal legislation, Provincial laws, and decrees, and by Municipality regulations and controls. The principal permitting process, as well as regulatory activities during operations, are managed by Salta Province through the Salta Mining Court and Salta Mining and Energy Secretariat, in conjunction with other public offices. In the province of Catamarca, this is managed by the Mining Ministry.

20.1.4 Sectorial Permits

Following approval of the EIA, the following sectorial permit applications will be filed:

- Work authorization for foreign professionals.
- On-site bulk fuel storage.
- Domestic and industrial effluents.
- Authorization for explosives use and storage.
- Authorization for chemical precursors use and storage.
- Water management.
- Mining operations.
- Communications.

20.1.5 Permitting Agencies

Permitting government agencies are listed in Table 20-1 and Table 20-2Table 20-2, including a summary of the most important required approvals and permits.

Table 20-1: Required Key Permits and Authorizations

Area	Key permits and Authorizations	Jurisdiction
Mining Law 24, 196	Certificate of Registration under number 242	Secretaría de Minería de la Nación
Import and export	Certificate of Import & Export	Customs
Mining Property	Granting of Mining Property	Salta Mining Court House
Surface Rights	Camp Rights	Salta Mining Court House and the Catamarca Mining Ministry
	Right of way	
	Surface water right and pipeline	
	Camp rights	
	Plant and process easement	
	Waste dump easement	
	Open Pit easement	
Water Resources	Water Concession for Mining Activity Use	Secretaría de Recursos Hídricos de Salta, Secretaria de Agua y Ambiente de la Provincia de Catamarca.
	Report on all drill water supply holes	

Area	Key permits and Authorizations	Jurisdiction
Waste Management	Pathogenic Waste Provincial Registration as Generator	Secretaría de Ambiente y Desarrollo Sustentable. Secretaría de Minería y Energía. Salta Secretaría de Agua y el Ambiente Catamarca.
	Operator and transporter of Hazardous Wastes	
Environmental Management	Environmental Impact Declaration (Approval of Environmental Impact	Secretaría de Minería y Energía de Salta. Ministerio de Minería de Catamarca
	Environmental Impact Statement, (art.250 Mining Code, art.34 of Provincial Law 7,141)	
	Comply with the DIA requirements	
Camp qualification	Camp municipal qualification	Municipio San Antonio de los Cobres y Antofagasta de la Sierra.
Explosives	Registration for Explosives Users	Ministerio del Interior Registro Nacional de Armas (RENAR)
	Registration for Users and Vendors of Explosive Services (vendor)	
	Certification of Powder Magazines	
	Storage of Ammonium Nitrate and Controlled Products	
Contract foreign professionals	Hire foreign professionals	RENURE (Migrations)
Fuel	Registration in the Liquid Fuel Dispensing Registry or National	Secretaría de Energía
	Liquid Petroleum Gas Registry (contractor)	
Chemicals	Storage and use of chemical products	SEDRONAR – R.N.P.Q (Registro Nacional de Precursores Químicos)
Communications	Use of satellite telephone and internet	Private contracts with vendors. Commission Nacional de Communications
	Use of VHF handheld radios	
Cultural and Natural Heritage	Notification of Accidental Discovery of Artifacts Potentially Relevant to	Secretaría de Cultura de la Provincia de Salta, Dirección de Antropología de Ministerio de Educación, Ciencia y Tecnología de Catamarca
	Cultural or Natural Heritage of the Province.	
	Request for Liberation of an Area (i.e., free of culturally significant artifacts)	

Table 20-2: Required Key Permits and Authorizations

Area	Key permits and Authorizations	Jurisdiction
Health	Authorization for Installation and Operation for Food Preparation and Operation of Dining Area	Municipio de San Antonio de los Cobres Municipio Antofagasta de la Sierra. AOMA (Mining Union)
	Authorization to Operate Water	
	Potabilization Plant	
	Potable Water Certificate	
	Medical Post, Doctors, and Ambulances	
	Emergency Plan for Contingencies Health Security in Mining Activities	
Transit and transportation	Transit of special machinery and gasoline	National roads: Vialidad Nacional
		Provincial roads: Vialidad de la Provincia de Salta
Use of Soil	Quarry Concession in fiscal provincial areas	Juzgado de Minas de la Provincia de Salta
Water Management Works and Structures	Authorization for construction a water management structure	Secretaría de Recursos Hídricos de la Provincia de Salta
	Water management works Approval and authorization to operate.	
	Environmental impact report for the water pipes and construction	Secretaría de Minería y Energía
Mining Operations	Mining producer registration	Secretaría de Minería y Energía de Salta, Ministerio de Minería Catamarca
	Ore transport guidelines	
	Notice of start-up of mining activity	
	Notice of suspension of operations or abandonment	
	Qualification of plant and facilities	Municipio San Antonio de los Cobres
Closure Plan	Mining Closure Assessment	Secretaría de Minería y Energía de Salta

20.1.6 Permits

20.1.6.1.1 Existing Permits

AbraSilver has obtained permit approvals from both the provinces of Salta and Catamarca for exploration activities. In Salta, an Environmental Impact Declaration (DIA) was obtained, while in Catamarca the company received a provincial resolution.

In addition to the Project's current Environmental Impact Report (EIR) for the Exploration Stage, the Project holds easements outlined in Table 20-3.

Currently, there is a permit to use well water from St.DBL.Ag 7, called SRH 1515, with Resolution number 009/23.

Table 20-3: Existing easements

Item	File Number	Area (ha)
Water Easement	19332-2008	1
Water Easement	19333-2008	1
Water Easement	19334-2008	6
Camp & Road Easement	16225-1997	25
Road Easement	18927-07	36

20.1.6.1.2 Future Permits

The company has submitted applications for two new water easements of 4 ha each. While the expectation is that these will be granted, the process has only recently begun so they have not been included above.

Additionally, four new easements were requested in 2023.

An Environmental Impact Report is required as part of the application for environmental approval. The EIR includes the following:

- General Project Information
- General Description of the Environment
- Description of the Project
- Description of the Environmental Impacts
- Description of the Environmental Management Plan

On September 30, 2024, AbraSilver submitted an Environmental Impact Assessment, EIA, to both the Salta and Catamarca provincial authorities for implementation of the Diablillos project. The EIA included results of baseline environmental studies and social engagement dating to 2021. A local consultant with extensive experience in the mining and energy sectors was used to develop the EIA application.

20.2 Environmental Baseline

Within the framework of the elaboration of the environmental baseline of the Diablillos Project, three survey works were carried out at different times of the year, to characterize the flora, fauna, and limnology communities of the project's area of influence and its immediate environment, to understand their dynamics and to establish the biodiversity baseline in the area under mining concession of the Diablillos metalliferous project. Sampling the biotic indicators was duly selected.

On June 8, 2022, AbraSilver announced that it has completed a comprehensive Environmental Baseline Study (the “Study”) at its Diablillos Project in Salta, Argentina. Following this initial

report, routine field data collection and interpretation continue to generate a comprehensive record of the different seasons and project areas that will be influenced by the future mine.

The purpose of the Study and its updates is to collect and analyse environmental baseline parameters relevant to the area of a potential future mine prior to significant project activities taking place on and around the property's concessions.

The study conducted will help identify and mitigate potential environmental and social impacts that may arise with future project development. The study covered several important areas, including hydrology, flora, fauna and limnology.

From 2021 to the present, AbraSilver has been working with a local consulting firm to develop and update the project's environmental baseline.

20.2.1 Location and Access Routes

The Diablillos property is located approximately 160 km southwest of the city of Salta as the crow flies and 360 km northwest of the capital of Catamarca as the crow flies, located along the border between the provinces of Salta and Catamarca, Argentina. The property encompasses a concession area in both provinces of approximately 11,680 ha in the high Puna and Altiplano region of northwestern Argentina, with geographic coordinates at the centre of the property of 25°18' south latitude by 66°50' west longitude.

20.2.2 Climatology

The characterization of the climate is referred to in Section 5.3, see details in this section.

20.2.3 Air Quality

The climatic conditions of the area contribute to the transfer of dust by winds. Depending on the direction of the wind, the dust load can come from the edge of the salt flat where the concentrations of silt and clay are higher, which causes an increase in the levels of particles naturally suspended in the atmosphere. The removal of material by mining activity will cause an increase in the number of fine particles available in the operating areas.

The EIA report analysed the air quality, origin and vectors of natural transfer. Frequent monitoring will be implemented where the results will be reported indicating the concentrations of selected pollutants that are within the guide levels established by current legislation. The results correspond to the natural levels of each site, with the levels of pollutants from the activities carried out in the area at the time of the evaluation corresponding to the levels of particulate matter transported by the wind. Air sampling will be carried out periodically in accordance with resolution 004/18. This process will involve community members, government authorities and other agencies in the sampling process and the results will be discussed.

20.2.4 Water Quality

The Barranquillas Aquifer System (SAB) is a unit of fluvial origin. At the site where the St.DBL.Ag7 Well is located, the area is proposed for the priority supply of the mine. The geology of the basin is made up of a growing system of medium sands to coarse gravels up to approximately 140m depth. The recharge basin of this system is 23,000 hectares with a 150mm isohyet, north-south parallel to the major axis of the basin.

During 2023 and 2024, systematic environmental chemical studies were carried out on samples taken from the St.DBL.Ag7 well. It was concluded that the conductivity and pH values between the field and the laboratory are consistent, reflecting consistency in the results despite variations related to temperature in the field. An average annual conductivity of 2516uS/cm and an average pH of 5.6 were obtained. The water is classified as sodium chlorinated and has demonstrated stability in its composition over time.

Comparisons between the results and the guide levels show that the metals that exceed the tolerances specified in Law No. 24585, Table No. 1 “Human Drink”, are arsenic and aluminum. In Table No. 6 “Livestock Drink”, the metals out of tolerance are boron, zinc and arsenic.

Although it is not classified as suitable for human or animal consumption, with proper treatment it can be used for domestic purposes in camping.

The morphological characteristics of the reservoir seem to indicate that it is a free to semi-confined aquifer, with edges and base formed by fractured Palaeozoic rocks (metamorphites, granites and granitoids), which determine the thickness and extent of the sediments of river fill.

20.2.5 Hydrology

From a hydrological point of view, the morphological factor directly affects the configuration of a centripetal or endorheic drainage that converges at points of lower heights, forming interior lagoons and salt flats. With reduced rainfall characteristics (precipitation less than 200 mm on the eastern edge, and less than 100 mm annually in the Puna in general) and predominantly summer, the courses have almost entirely a temporary regime, with permanent courses being rare.

This low rainfall means that the recharge of the aquifers is very poor in quantity but effective by rapid infiltration due to the transmissivity of the sediments.

20.2.6 Soil Science

The area has poorly developed and very fragile soils, due to the climatic and relief conditions, which condition and limit their formation.

In general, the taxa are called aridisols, which occur in areas with an arid climate. It is characteristic that these soils have a pale surface horizon (composed of salts and other minerals and poor in organic matter), entisols (formed on recent sediments such as alluvium, dunes or are found on very steep slopes where erosion predominates).

In this area, the prevailing conditions restrict the formation of soils, so there are large areas of rocky outcrops of varied composition and origin (mainly igneous, volcanic and metamorphic) and/or surfaces covered by salt flats.

20.2.7 Flora and Fauna

The phytogeographic provinces of the Puna and the High Andes are characterized by a very arid climate, with low temperatures, a wide daily thermal amplitude, intense solar radiation and almost constant exposure to strong winds. Studies and surveys of vegetation at a local level, which provide basic information on the functioning and dynamics of these ecosystems, are of fundamental importance for their sustainable management. Mitigation of the impacts on aquifer systems, areas of low vegetation cover, endemic and/or threatened species, which could result from human activity, must be based on knowledge of their ecological foundations, and these are updated and detailed in the environmental impact studies for the exploitation of the mine.

In the area of influence of the Diablillos Project, the vegetation unit with the greatest representativeness was the shrub steppe, followed by the grassland with shrubs. Both vegetation units are encompassed within the so-called high-Andean steppe, but while the former has a predominance of low shrubs, the latter is characterized by the predominance of grasses and grasslands. Of the totality of plots and observation points surveyed in this work, fourteen of them are framed in one of these two plant formations. This corresponds to the general physiognomy of the region, which shows not very steep slopes and small, mostly temporary watercourses.

Regarding flora, a total of 29 species were recognized throughout the project's area of influence, which mainly includes shrubs and grasses. To. Horrida and F. argentinensis, were the two most common and recurrent species in all vegetation units analysed. At the other extreme, species that were identified as exclusive in their environment or with very low density were: And. Tweedian and F. Bryoids; associated with wetter areas and to. Minor, B. Tola, C. Erinacea and F. Punensis associated with grassland areas.

From the estimation of the percentage of soil covered by vegetation we can divide the study area into two regions with distinctive characteristics. The first of these includes the area around the Co. Oculto and the current mining camp; the second is located within the vicinity of the Barranquilla River, at the height of the camp's current water intake. The surroundings of the Hidden Zone have relatively low vegetation densities (about 40% of the ground area covered). On the other hand, in the vicinity of the Barranquilla River water intake, the coverage values exceed 54% of the soil surface.

Regarding fauna, a total richness of 55 species of native vertebrates could be identified, considering the interest groups proposed in this case, corresponding to 1 species of amphibian, 2 reptiles, 42 species of birds and 11 species of mammals, one of which is domestic. In addition to two exotic species.

20.2.8 Ecosystem Characterization

Ecosystem characterization allows a more complete understanding of the existing interactions between a given project and the environment, and vice versa.

Eco-regions, or biomes present in the Diablillos Mine area are the Puna eco-region and the high Andean eco-region. These eco-regions were divided into ecological units summarized by descriptions of the soil, flora, and fauna, and their particularities and interactions (shelters, niches, eco-tones, barriers, and corridors comprising areas of frequent use). The degree of disturbance that these communities are experiencing due to human activity is currently being evaluated.

20.2.9 Local Ecosystem Characterization

To characterize the local ecosystem to be presented in the EIA, a summary of environmental conditions of the area was formulated, based on the data from baseline studies, carried out by the consultant Elisa Cozzi and Associates since 2021

20.2.10 Identification of Protected Area

The Diablillos Project is not located within any area figure of conservation, considering the current system of protected areas of the Provinces from Salta and Catamarca.

For reference, protected areas (from both provinces) are listed below relevant at a regional level.

- Los Andes Multiple Use Natural Reserve (Salta Province).

*The project is located outside this protected area.

- La Vicuña Natural Reserve Area (Salta Province).

*The final surface of the area protected is not defined.

- Laguna Blanca Biosphere Reserve Area (Catamarca Province).

*The project is located outside this protected area.

20.2.11 Archaeology

This section was carried out with the report "Archaeological Baseline Study (ELBArq) of the "Diablillos" Project, located on the N sector of the Salar del Hombre Muerto, department of Los Andes, province of Salta". It was developed as an integral part of the Environmental Baseline (LBA) prepared by the consulting firm EC & ASOCIADOS S.R.L. (EC)1, with the company ABRASILVER RESOURCE CORP. (ARC)2 being the operator of the project in question.

The field survey tasks were carried out by ARQUEOAMBIENTAL Archaeological Consultants during 2021 and updated in 2024.

The objective of this study is to evaluate the archaeological situation of the study area, so that the results obtained are used as base information for the development of future work.

The field survey carried out resulted in the creation of an archaeological record that includes a total of 17 (seventeen) archaeological findings, corresponding to sets of archaeological material, isolated simple structures and sets of low-sensitivity structures.

Of these findings, only 2 are located within the direct area of influence of the Hidden Mine, the rest are far away or outside the limits of the mining concessions.

20.2.12 Environmental Risks

20.2.12.1 Risk Assessment

The assessments and weighting of environmental risks are evaluated in the Environmental Impact Report submitted to the control authorities every 2 years, where the main risks, if any, for the environment and communities are evaluated and determined.

As a result of the environmental risk assessment carried out in the EIA, there are modifications to the landscape and the generation of uncontrolled dust. On the other hand, as positive findings, the generation of employment and improvements in local infrastructure such as access are considered to have a high positive impact.

20.2.12.2 Operations

The topography of the land will be altered by mineral extraction, especially around the pits and by construction of the waste dumps, stockpiles and TSF. The modifications made by the waste dumps do not represent geomorphological risks because they will be installed on sites that have no relevant natural hazards and will be designed to withstand earthquakes and extreme rainfall events.

20.2.12.3 Dump or Sterile Rock Deposit

While it is predicted that the waste dump area will have adequate long-term stability, monitoring of slope stability and deformation of the dump must be carried out on a continuous basis during its construction. In the case of the waste dump, its development has been executed in accordance with its design with no observations of subsidence as of the effective date of this Report. Any cracks that may be identified are surveyed, monitored, and subsequently remediated to enable the sectors for loading. The waste dump is located in a dry environment with no adverse issues related to wind erosion identified as of the effective date of this Report. The granulometry of the material varies, forming a 37° slope, that includes slight localized bulging, but nothing that interferes with the footing design. No instabilities have been identified.

20.2.12.4 Water

The risk of water contamination will always be an issue to consider and control. Monitoring and sampling will be constant, and the results will be reported to the relevant authorities. A comprehensive water quality monitoring program has been implemented downstream at various points to eliminate any uncertainty about possible contamination. Normally in this type of mine there are concerns about the risks of acid drainage and the presence of cyanide from the leaching circuit, although this is minimized by several factors such as water recirculation, monitoring stations, lack of meteoric water supply and climatic conditions that slow down the reaction speed of chemical weathering.

20.2.12.5 Air Quality

The operation generates emissions of particulate matter (dust) in the mine and plant area, mainly due to mining operations such as drilling and blasting, movement of vehicles on dirt roads, loading and unloading of extracted materials; and mineral processing operations such as crushing and agglomeration.

All these parameters were considered in the design of the mine, location of the facilities and methodologies to be implemented to mitigate or eliminate some risks.

The details of mitigation and risk assessments are detailed in the EIA presented by the company for the exploitation stage.

20.2.12.6 Soil

The process plant was designed as closed circuits and therefore no soil contamination is expected. There is, however, always a potential risk of failure in the leach circuit, in the transfer of solutions to different facilities, from drainage of solutions to the tailings pond, and from heavy precipitation that may cause solution in the circuit to exceed levels. A contingency settling pond has been constructed to provide protection against such events.

Within the plant, there will be a central drainage system to collect any spillage and send it to the containment ponds.

During operations, closure, and post-closure, events, or situations of low probability of occurrence may arise involving the contamination of the land, such as an oil spill, chemical spill, or accidental discharge of process solutions. Each of these events will have health and safety action plans and environmental remediation plans as necessary.

20.2.12.7 Flora and Fauna

The necessary monitoring was carried out to record and consider revegetation, growth periods, native species, among others, to guarantee the safekeeping of useful material to re-cover the activity areas during the useful life of the project.

Regarding fauna, the records indicate that the impact on its habits in indirect areas is minimal and in direct areas it can be restricted by civil works that do not allow the circulation of fauna on roads and in the mine pit.

20.2.12.8 Sites of Archaeological Interest

No sites of archaeological significance were found in the area that are directly affected by mining activity.

20.2.12.9 Noise

Due to the location of the mine, the environmental noise impact will be virtually nil for nearby residents and local wildlife. Noise and vibration controls will be installed where necessary.

20.2.12.10 Closure and Post Closure Phase

Environmental risks during the closure stage will be reduced by remediation and monitoring work. At the closure stage, soil will be contoured by heavy machinery to minimize the long-term impact of mining activity and return the topography of the land to resemble prior conditions. However, the movement of soil, and thus the risk, will be significantly less than in the mining operations stage.

20.2.12.11 Environmental Management Plan (EMP)

One of the priorities of AbraSilver is the care and protection of the environment. During the exploration stage, the company exercised any extent possible for the potential environmental impacts on the area. The same effort will be made during operational stage and in the closure stages of the mine. AbraSilver will define environmental principles to enable the development of mining operations efficiently from a productivity standpoint and from an environmental perspective, which include:

- Comply with existing environmental laws and regulations.
- Establish and maintain an environmental management program to guide operations.
- Involve the entire staff of AbraSilver and contractors in the Environmental Management Plan.
- AbraSilver promote environmental awareness among employees and the communities were operations occurrence.
- Mitigate the potential environmental impacts that do occur and support environmental improvement programs for common benefit.

The EMP defines the criteria, the design of specifications, and management practices that are applied to the Diablillos mine to mitigate, control, and monitor changes in the baseline conditions during operations, closure, and post-closure of the mine. Corresponding prevention measures, mitigation of potential environmental impacts, as well as rehabilitation measures, are outlined as appropriate.

Prevention measures will avoid potential environmental impacts, while mitigation actions are intended to minimize, correct, or compensate for environmental impacts of the proposed Diablillos mine at different stages. Measures to increase, improve, and enhance the positive environmental impacts caused by the Mine will also be implemented.

An environmental contingency plan will be implemented to predict potential incidents.

20.2.12.12 Issues of Environmental Concern

The following list specifies the most important issues to be considered from an environmental perspective that must be detailed in the EMP.

- Surveys of residents living in communities near the mine.
- Interests of those agencies evaluating the EIAs.
- Special requests by the Mining and Energy Secretary of the Production Ministry of Salta province (Secretaria de Minería y Energía, dependiente del Ministerio de Producción de Salta (DIA, Environmental Impact Declaration).
- Publications by Argentine media.
- Mining activity of concern to Argentine citizens.
- Statements by non-governmental organizations (NGOs), politicians, and others.

20.3 Community Relations

20.3.1 Social-Economic and Cultural Aspects

For the identification of the area of direct influence (AID), geographical and linkage criteria were considered, that is, the proximity of the project to the community, and the possibility of establishing ties there through the purchase of products and the contracting of services, as well as those that arise from interactions with the inhabitants. According to these criteria, the community may be impacted by the development of the project, and conversely, it may be the community that impacts the project.

The AID of the Diablillos project are the communities of Santa Rosa de los Pastos Grandes, located in the Department of Los Andes, Province of Salta; and the community of Ciénaga La Redonda, belonging to Villa Antofagasta de la Sierra, in Catamarca. These communities are the closest to the location of Diablillos. Scattered posts near the project were also included.

The AII of the Diablillos Project is the municipality of San Antonio de los Cobres, in Salta; while the most relevant institutions of the department of Los Andes are established there, and because it is the center that provides commercial support to the town of Santa Rosa de los Pastos Grandes. The locality Estación de Pocitos is also included as an AII, due to its proximity to the last-mentioned locality.

The most relevant information to take from the perception study, in relation to mining, is that, in general, they agree that the activity should be carried out in the area as long as certain requirements are met, related, in order of importance, to the collaboration with the town and its inhabitants, the hiring of local labour and services, and care for the environment.

Likewise, they expressed having a good image of the CSR area based on the support they receive. Given that the stallholders do not have daily contact due to the distance they are from AbraSilver's mining properties, no mentions were obtained regarding the identification of direct negative impacts on the quality of life or the surrounding environment. In this sense, we can infer a positive assessment of the actions that the CSR area has been carrying out through the permanent visit of the area of direct influence, by providing health care and in the contribution of merchandise and basic supplies.

20.3.2 Stakeholder Engagement

The Social Communication Program (SCP) is a fundamental instrument in the socio-environmental management of the projects and operations by AbraSilver, fulfilling the function of qualifying audiences and content, as well as mentioning communication and dialogue channels with populations, social organizations, and public institutions in their areas of direct influence. This program is included in the Environmental Management Plan within the environmental licensing process.

The communication actions proposed in this document have the guiding principle to provide society with information about the project, its licensing phases, possible impacts, and control/minimization actions, thus, aiming to contribute to the involvement and participation of interested parties, showing transparency in the execution of the company's business.

20.4 Closure

The rehabilitation and closure plan consist of three main stages:

1. Decommissioning planning
2. Execution of decommissioning
3. Implementation of the socio-environmental and geotechnical follow-up and monitoring actions of the post-closing. Waste piles will be graded as needed, capped with a vegetation suppression layer, and revegetated with herbaceous-shrub species. A final protective cover can be placed over the pile to facilitate revegetation and minimize erosion, at which point the sedimentation pond may be decommissioned. A cap layer of soil will be placed and seeded on the open pit berm areas. A fence will be built around the open pits, and all mine haul roads will be blocked off.

The closure plan is based on assessments of available technical information and local conditions throughout the life of the venture.

The recovery of degraded areas is a complex process that encompasses many interrelated activities and must be carefully monitored to avoid possible deviations, such as delays in soil correction or significant erosion.

During the recovery process of degraded areas, the aim is to maintain the original structure, whenever possible, prioritizing the use of native species in the areas to be recovered.

The pits and piles must have a final configuration with properly inclined slopes, an adequate drainage network to direct surface water, and the application of replanting techniques on the slope faces for efficient execution of a Mine Closure Plan. Replanting reduces the action of erosive agents, in addition to mitigating the negative visual effect from mineral extraction.

The equipment and civil structures demobilization that could be used in the project will not be carried out yet, even if there is the existence of a deposit and a mining concession that justifies the continuation and processing operations is proven.

If there is no technical-economic feasibility to continue the project, the company will remove the civil, metallic, and other structures related to the processing plant and of other supporting areas. The pit and piles geotechnical stability and the revegetation will also be part of the schedule.

Further details of the Mine Closure Plan will be presented below if the activity cessation is considered after experimental mining. A schedule of activities planned for the mine closure can be seen in Table 20-4.

Table 20-4: Schedule of Mine Closure Activities

Activities	Year 0	Year 1	Year 2	Year 3	Year 4+
Excavation and backfill					
Surface drainage					
Geotechnical monitoring					
Required earthworks					
Surface regrading					
Biomat and Hydroseeding application					
Facilities dismantling, demolition and demobilization					
Preparation of periodic inspection reports					

20.5 Corporate Social Responsibility

As part of a social responsibility program, AbraSilver maintains regular contact with local authorities and community members. Periodic meetings are held for information, contact, and to aid. AbraSilver assists in the development of community service providers, highlighting opportunities for local entrepreneurs that will be brought by the Diablillos Project.

AbraSilver is additionally part of a "social commission" along with other companies active in the area. This commission was created by the provincial government to facilitate cooperation between the government, communities, and companies in the region.

In meetings to date, the communities have expressed their support for the development of mining projects. The general feedback is that the local communities are excited at the prospect to develop the region and the personal opportunities that development will bring.

21. Capital and Operating Costs

The following section has been taken in its entirety from the previous NI 43-101 Technical Report Updated Pre-Feasibility Study, effective December 3rd, 2024, and report date January 17th, 2025.

21.1 Introduction

Capital and operating costs for the Diablillos Project were developed based on the investment required to achieve and sustain the mine plan and production schedule. The capital cost (CAPEX) was estimated based on process plant, infrastructure and TSF designs, including equipment, materials, labour, services required, indirect costs, owner's costs and contingency. Operating costs (OPEX) were estimated for equipment, labour, materials, power, supplies, fuel, explosives and other consumables. The capital and operating cost estimates were prepared by Bmining Chile, INSA Consultora, Envis Consulting and AbraSilver.

Capital and operating costs are quoted as United Stated Dollars ($).

21.2 Capital Costs (CAPEX)

CAPEX for the Diablillos Project is for a silver/gold mine capable of producing and processing an average of 9,000 tpd of ore.

The initial capital investment was estimated in accordance with the guidelines from the Association for the Advancement of Cost Engineering (AACE) and international Class 4 standards at an expected accuracy range of ±25%.

This summary describes the approach, methodology, format, and scope used for estimating CAPEX. This includes guidance notes and key assumptions.

The summary of CAPEX by discipline is presented in Table 21-1.

Table 21-1: Summary of Initial CAPEX by Discipline

Description	Initial Capital (MUSD)	Sustaining Capital (MUSD)	Total (MUSD)
Mining	128.6	27.1	155.7
Processing plant	111.7	-	111.7
Infrastructure	166.7	49.1	215.8
Owner and Indirect cost	110.2	0.3	110.6
Contingency	26.3	-	26.3
Total capital	543.5	76.5	620.0

*Note: Values may not sum due to rounding.

21.2.1 Material Take-Offs

Quantities were developed using preliminary Material Take-Offs (MTOs), complemented by estimates and allowances derived from the following engineering documents:

- General arrangements and site plot plans,
- Architectural drawings,
- Layouts drawings
- Process flow diagrams,
- Preliminary Piping & Instrumentation Diagrams (P&IDs)
- Mechanical and Electrical Equipment Lists,
- Electrical Single Line Diagrams.

Relevant quantities obtained from the MTOs are summarized in Table 21-2.

Table 21-2: Relevant MTOs

Discipline	Description	Unit	Quantity
Civil	TSF Liners (Phase 1)	m^2	886,142
Concrete works	Lean Concrete	m^3	875
	Structural Concrete	m^3	24,975
	Reinforcement steel	tn	2,019
Steel structure works	Steel Buildings	tn	1,953
	Steel structures	tn	1,286
	Cladding	m^2	49,416
Piping works	Carbon Steel	m	24,643
	Carbon Steel - Rubber Lined	m	2,694
	Stainless Steel	m	455
	HDPE	m	37,271
	PVC	m	5,626
Electrical works	Cable Trays	m	10,410
	Grounding cables	m	13,380
	Lighting protection cables	m	2,905
	Low voltage cables	m	33,472
	Medium voltage cables	m	1,750

21.2.2 Labour and Site Conditions

Most construction packages included man-hour estimates, and therefore minimal additional estimation was required for this report. Where necessary, Argentine standard construction productivity factors were applied. This included installation of tank, reactor, mechanical

equipment, electrical and instrumentation. These factors were adjusted to account for high-altitude productivity losses, worker quality, and absenteeism. However, productivity impacts related to labour market conditions, availability or scheduling constraints, were not factored in.

A total of 6,130,500 man-hours is estimated for the construction and commissioning period. To improve working conditions, a 15-day on/15-day off roster with 10 effective working hours per day was adopted, offering better terms than the construction union's standard 21-day on/7-day off schedule.

Hospitality and emergency medical care will be managed by the Owner's camp administration. Ensuring consistent quality in accommodation, meals, and emergency services. Transportation will also be handled by the Owner's team, providing safe and reliable travel between Salta City and site.

21.2.3 Data Sources

Data for the estimates were obtained from:

- Mine schedules,
- PFS-level engineering design for process plant, ancillaries, and off-site facilities from previous technical report, camp by owner's engineering team, TSF by Envis Consulting.,
- Topographical information from site surveys and preliminary geotechnical investigations,
- Budgetary equipment quotes from suppliers based in Argentina and overseas,
- Budgetary unit costs from local contractors for different disciplines,
- Data from similar recently completed studies and projects.

The following parameters and qualifications were considered:

- No allowance has been made for exchange rate fluctuations.
- Quotes in Argentine Pesos were converted to USD at the exchange rate corresponding to the date of quotation.
- No escalation is included in the estimate.

21.2.4 Mining CAPEX

Mining CAPEX was estimated by Bmining Chile. The breakdown of this cost includes 48% for the acquisition of the mining fleet, 35% for pre-stripping activities, 17% for sustaining CAPEX and 1% for explosive storage. The sustaining CAPEX includes acquisition/renewal of mining equipment and mine dewatering as the operation progresses to greater depths. Table 21-3 summarizes the capital requirements for the mining area.

Table 21-3: Mining CAPEX

Description	Mining Capital (MUSD)
Mine fleet	74.1
Mine infrastructure	0.1
Pre-stripping	54.4
Sustaining CAPEX	27.1
Total	155.7

21.2.5 Processing CAPEX

The definition of process equipment requirements was based on process flowsheets and process design criteria, defined in Section 17. All major equipment was sized based on the process design criteria and a mechanical equipment list. Mechanical scopes of work were developed and sent for budgetary pricing to equipment suppliers. The components costed for the process plant include:

- ROM pad and retention wall
- Process plant - ore stockpile – dome
- Process plant - steel buildings
- Process plant - modular buildings, mainly electrical rooms
- Process plant – concrete, steel, mechanical, piping, electrical and I&C
- Crushing
- Grinding
- Gravity concentration
- Tank leaching
- Counter current decantation (CCD)
- Merril-Crowe zinc precipitation
- Foundry
- Cyanide recovery
- Cyanide detox.

The costed components under Reagents Storage and Preparation include:

- Sodium cyanide storage and preparation
- Milk of lime storage and preparation
- Flocculants - storage and preparation
- Merril-Crowe reagents - storage and preparation
- Detox reagents - storage and preparation

Other ancillaries considered were:

- Process water treatment, storage and distribution
- Compressed air plant
- Petroleum liquified gas tank farm
- Conventional power plant
- Photovoltaic power plant

94% of the processing capital cost (104.6 MUSD) corresponds to plant construction, the remaining 6% (7 MUSD) is allocated to reagent storage and preparation (See Table 21-4).

Table 21-4: Processing CAPEX

Description	Processing Capital (MUSD)
Process plant	104.6
Reagents storage and preparation	7.0
Total	111.7

21.2.6 Infrastructure CAPEX

Table 21-5 summarizes the infrastructure CAPEX. Key components include:

- Permanent Site Facilities: This includes the construction of platforms, haul roads, operations camp, service hub, and site-wide utilities.
- Ancillary Services: Covers essential services required throughout the mine's life, such as the process water treatment system, storage, distribution, compressed air plant, fuel storage facilities, conventional and photovoltaic power plants.
- Site-wide piping: Includes process plant piping as well as piping and pipelines to support ancillary systems, such as workshops and infrastructure.
- Site-wide Electrical: The power distribution network will deliver electricity from onsite generators and a photovoltaic (renewable) system. All distribution networks will be installed underground in duct banks. The network includes switchgear, electrical rooms, transformers, and lighting protection. Diesel generators will be installed at off-site locations. Within buildings, power cabling will be routed through cable trays.
- Site-wide Instrumentation and Control (I&C): The scope covers the central control room, the industrial communications network, IT, remote-control units, and fibre optics.
- The TSF will be constructed in five stages over the mine's life. Stage 1 (22.7 MUSD) will be built during the pre-production years, while stages 2 through 5 will be constructed as sustaining capital during production years 2, 5, 8, and 11, respectively (48 MUSD).
- Off-site infrastructure is mainly related to the water supply system and the aggregates quarry. The raw water supply system is composed of raw water wellfield, raw water pipeline and booster pumping station.
- 2 MUSD has been allocated as capitalized operating costs, covering utilities, off-site staff and labor expenses during the construction phase.

Table 21-5: Infrastructure CAPEX

Description	Infrastructure Capital Cost (MUSD)
Permanent site facilities	29.5
Ancillary Services	49.5
Sitewide piping	28.3
Sitewide electrical	25.9
Sitewide I&C	5.2
TSF - Phase 1	22.7
Waste dumps	0.0
Off-site facilities / infrastructure	3.4
Capitalized OPEX	2.0
Sustaining CAPEX/TSF - Tailings Storage Facility phase 2-5 and waste dump	48.0
Sustaining CAPEX/ waste dump	1.1
Total	215.8

21.2.7 Indirect and Owner's Cost

Table 21-6 details the costs associated with indirect and owner-related expenditure.

Table 21-6: Indirect and Owner's Cost

Description	Indirect and Owner's Cost (MUSD)
Project preparation	8.0
Construction site services and costs	83.1
Engineering and Construction Management	5.1
Owner's costs during construction	9.1
Camp and service	4.8
Sustaining CAPEX/survey permits	0.3
Total	110.6

21.2.8 Construction Support and Site Costs supplied by the Owner

Construction Site costs include the following components:

- Construction fuel
- Catering and hospitality
- Personnel transport to site
- Access control and site security
- Medical service

- Chemical toilets
- Hazardous and general waste management
- Equipment, tools and supplies to be provided by the owner – construction
- Concrete plant (aggregates + concrete plant)
- Mobile equipment rentals
- Testing equipment / instruments
- Camp maintenance and other consumables
- Spare parts and lubricants for vehicles
- Health and safety signage
- Internal material handling, unloading and site distribution
- Overland bulk transportation costs
- Maritime bulk and air bulk transportation costs (depending on schedule requirements).

Contractor's indirect costs include all overheads such as contractual requirements (safety, insurance, etc.), mobilization and overheads as included into their budgetary quotations. Unit prices submitted by contractors are "all-in" rates and are totally considered as direct costs. The listed site services to be supplied by the owner, were excluded from contractors' budgetary quotations.

Transportation costs were obtained from a specialized logistics consultant. Providing overseas and inland freight figures from different shipment ports and a road study for all types of cargo, including insurance and demurrage.

The freight and logistics costs include brokerage, agent fees, warehouse services, and import charges. It is assumed that there will be no requirement for major ongoing air freight to site.

21.2.9 Engineering and Construction Management Support

Engineering and Construction Management includes:

- Detailed engineering for execution
- Vendor representatives
- Commissioning and start-up assistance.

21.2.10 Owner's costs

Owners Costs during construction include the Owner's project execution team, construction insurance policies, initial reagents and diesel.

No allowances for spare parts have been considered, as spares for start-up and the first two years of operations was included in vendor quotes.

First fills include the start-up fill of the reagents, lime, fuel, gas, and lubricants.

21.2.11 Contingency

Contingency accounts for the difference in costs between the estimated and actual costs of materials and equipment. A comprehensive list of quotations for equipment and services of approximately 80% of total costs were obtained from budgetary quotations, the following contingencies have been considered.

- 2% on major equipment
- 7% on minor equipment and all remaining supplies and services packages.

The resulting contingency included in the initial CAPEX is 26.3 MUSD.

21.2.12 Assumptions, Restrictions and Exclusions

The capital cost estimate fulfils all local requirements regarding labour, construction, services contracts, materials and equipment supplies.

The estimate assumes no access restrictions to site at any time, nor delays in execution due to Owner's financing, permitting or other delays from labour disputes.

21.2.13 Mine Remediation and Closure

Mine Closure costs relate to ongoing environmental monitoring, facilities demolition, removal, camp and off-site facility dismantling, TSF cover, facility area rehabilitation, and EPCM costs. Costs associated to mine remediation and closure were detailed separately in this estimate. The remediation scope is detailed in Chapter 20.

The following items are specifically excluded from the Capital Estimate:

- Escalation and interest during construction
- Schedule delays and associated costs or schedule recovery or acceleration
- Scope changes
- Unidentified ground conditions
- Extraordinary climatic events, labour disputes and force majeure,
- Research and exploration drilling and other sunk costs
- Salvage values.

21.3 Operating Costs (OPEX)

Operational costs are considered following commissioning when ore production starts. Costs estimated in previous periods are considered as CAPEX.

Table 21-7 presents the estimated total operating costs for the LOM and unit costs per ton milled.

Table 21-7: Summary of OPEX

Description	LOM Cost (MUSD)	Unit cost (US$/t milled)
Mining	613.8	14.50
Processing, utilities and maintenance	961.3	22.71
G&A and logistics	165.6	3.91
Camp and service	181.8	4.29
Total OPEX	1,922.5	45.42

21.3.1 Mine OPEX

The mine OPEX was estimated by Bmining Chile based on the mine plan and the equipment operating cycles. OPEX was supported by vendor information and/or benchmark database for ownership costs.

The mine fleet will be diesel powered with a diesel price of USD 0.95/l is considered in the OPEX.

Mine dewatering is not expected to start until year 10.

Blasting is assumed to be performed by a local contractor providing all necessary explosives and manpower.

Mine OPEX include laboratory assay costs for mining, process, and environmental test work. Required assays and quantities for normal operations were estimated by the owner and benchmarked against other projects.

Table 21-8 shows the mine OPEX, where the total cost of mining is 614 MUSD and the unit cost is 1.75 US$/t moved.

Table 21-8: Mine OPEX

Mining Operating Cost	LOM Cost (MUSD)	Unit Cost (US$/t milled)	Unit Cost (US$/t moved)
Mine staff and labour	140.7	3.32	0.40
Drilling & Blasting	58.9	1.39	0.17
Excavation & Hauling	285.5	6.74	0.81
Ancillary & support	59.0	1.39	0.17
Mine road maintenance & dust control	57.8	1.36	0.16
Mine dewatering	0.6	0.01	0.00
Metallurgical lab	10.8	0.26	0.03
HS&E and Training	0.7	0.02	0.00
Total	613.8	14.50	1.75

21.3.2 Process OPEX

Process OPEX was estimated considering cost related to the plant, utilities and maintenance costs as summary in Table 21-9.

Table 21-9: Summary of Process OPEX

Process Operating Cost	LOM Cost (MUSD)	Unit Cost (US$/t milled)
Processing plant	350.1	8.27
Utilities & off-site facilities	453.8	10.72
Maintenance	157.4	3.72
Total	961.3	22.71

The ore processing plant covers crushing and conveying, grinding, tank leaching, CCD and Merril-Crowe, cyanide detoxification, reagents storage and handling. The plant will process 9,000 tpd with ramp-up periods during the first two years.

Consumption of reagents and plant consumables were specified within the process design Criteria and quoted from suppliers. Main reagents include sodium cyanide, quicklime, sodium hydroxide, sodium metabisulphite, zinc dust, diatomaceous earths, flocculants, fluxes and anti-scalant.

Table 21-10 summarizes the operating cost related to the processing plant with a total of 350 MUSD or 8.27 US$/t milled.

Table 21-10: Processing plant OPEX

Processing Plant Operating Costs	LOM Cost (MUSD)	Unit Cost (US$/t milled)
Plant staff and labour	80.8	1.91
Crushing	5.5	0.13
Grinding	52.5	1.24
Leaching and CCD	1.4	0.03
Merril-Crowe and foundry	3.2	0.08
Cyanide recovery and detoxification	1.5	0.04
Reagents	200.8	4.74
HS&E and Training	4.5	0.11
Total	350.1	8.27

Utilities and off-site operating costs consider cost of power generation, raw water wellfield, water treatment plant, fuel station, gas tank farms and TSF operation.

Power generation is one of the most significant utilities costs accounting for 75% of the total. A hybrid power plant has been designed, composed by a 3 MW photovoltaic plant with a battery

storage system (BESS) for the camp and service hub operation and a 17 MW photovoltaic plant without batteries for generating the plant power consumption during daytime. The photovoltaic plant will be complemented with a dual fuel power plant (diesel – natural gas) depending on the future supply availability of natural gas, for a total of 20 MW installed power. The cost of operating the dual fuel power plant considers diesel fuel at $0.95/l.

Table 21-11 summarizes operating costs related to utilities and off-site facilities with a total of 453.8 MUSD. 75% belongs to the cost of power generation and 22% to the cost of the fuel storage facility.

Table 21-11: Utilities & Off-Site Facilities OPEX

Utilities & Off-Site Facilities Cost	LOM Cost (MUSD)	Unit Cost (US$/t milled)
Utilities and off-site staff and labour	10.1	0.24
TSF	0.1	0.00
Water production & treatment	0.3	0.01
Power generation	341.5	8.07
Fuel station	0.4	0.01
Fuel storage facility	101.3	2.39
Total	453.8	10.72

Maintenance costs include estimated maintenance consumables for the different project facilities, including major spare parts for the mine fleet and process plant (i.e. liners and screen replacements). Wear rates for liner and screen replacement frequencies were estimated based on vendor-supplied data.

Underground power lines, raw water wellfield pumps and water pipeline booster station maintenance and TSF facilities were also included.

Table 21-12 presents the cost of maintenance with a total of 157.5 MUSD and a unit cost of 3.72 US$/t milled.

Table 21-12: Maintenance OPEX

Maintenance operating cost	LOM Cost (MUSD)	Unit cost (US$/t milled)
Maintenance staff & labour	35.3	0.83
Camp and service hub maintenance	8.6	0.20
Plant mobile fleet maintenance	0.4	0.01
Power generation maintenance	5.7	0.13
General support fleet maintenance (truck shop, staff, camp, etc.)	2.2	0.05
Plant maintenance	98.8	2.33

Maintenance operating cost	LOM Cost (MUSD)	Unit cost (US$/t milled)
Off-site maintenance - access & internal service roads maintenance	0.8	0.02
Off-site maintenance - overhead power lines	3.0	0.07
Off-site maintenance - water wellfields and water treatment plant	1.7	0.04
Off-site maintenance - TSF	0.9	0.02
Total	157.4	3.72

21.3.3 G&A and Logistics

The G&A costs for the operations stage were compiled using current internal data as shown in Table 21-13.

Table 21-13: General and Administrative Cost

General and Administrative Cost	LOM Cost (MUSD)	Unit Cost (US$/t milled)
General administration and office staff, site management	20.1	0.48
Pacific Rim Argentina SA Office Expenses	1.2	0.03
Pacific Rim Argentina SA Office External Support	2.9	0.07
Pacific Rim Argentina SA Office Travel Expenses	0.4	0.01
Mining tenements and other fees	0.9	0.02
Pacific Rim Argentina SA ESG	2.6	0.06
Total	28.2	0.67

Logistic costs were split into upstream and downstream logistics. The upstream logistics consider the transportation of supplies, consumables, and reagents to site. In general terms and according to the Process Design Criteria, most of the reagents and consumables storage capacities have been designed for a seven day supply margin.

Downstream logistics consider transportation of doré bars from site to a refinery located in the United States. Transportation, security, and insurance for doré transportation to the refinery were quoted by a specialized contractor.

The production of mercury, which must be treated and disposed of according to international standards, was considered and estimated for final disposal at facilities in Europe. The resulting costs can be found in Table 21-14.

Table 21-14: Logistic Cost

Logistic cost	LOM Cost (MUSD)	Unit cost (US$/t milled)
Logistics staff and labour	14.6	0.34
Upstream logistics - Inland overland transportation - Bs. As. to Salta	65.6	1.55
Upstream logistics - Overseas maritime shipping	27.3	0.65
Downstream logistics - Doree transportation from mine to refinery	29.9	0.71
Downstream logistics - Special Hg transport to Antwerpen - 40' cont.	0.0	0.00
Total	137.4	3.25

21.3.4 Camp and Service

OPEX for the camp consider staff and consumables to operate and maintain the camp and provide the necessary services. The estimate includes staff accommodation, meals, room cleaning, recreation, medical and emergencies. The camp includes a workshop to perform all camp, minor civil, electrical, and I&C maintenance, and the workshop consumables are included in the estimate.

The service hub includes all costs related to the truck-shop and plant maintenance workshop, including the plant and mine operating offices as shown in Table 21-15.

Table 21-15: Camp and Service Cost

Camp and service cost	LOM Cost (MUSD)	Unit cost (US$/t milled)
Camp staff and labour	31.2	0.74
Hospitality	86.0	2.03
Staff travel and site transportation	51.0	1.21
HS&E	4.3	0.10
Medical	2.3	0.05
Water consumption	0.0	0.00
Miscellaneous consumables	4.2	0.10
Waste management	2.0	0.05
Effluent management	0.7	0.02
Total	181.8	4.29

21.4 Comparison to the Previous Technical Report

A comparison of the CAPEX previously reported is reported in Table 21-16.

Table 21-16: Comparison of CAPEX estimate to the previous Technical Report

Description	Updated PFS Study	Prior PFS (Mar. 25, 2024)	Change	
			Updated PFS vs. Prior PFS	
	$ millions	$ millions	% Change	$ Change
Mining	128.6	39.3	227%	89.3
Processing	111.7	96.9	15%	14.8
Site Infrastructure	166.7	152.0	10%	14.7
Owner and Indirect Costs	110.2	64.9	70%	45.3
Initial Capital Costs (excl. contingency)	517.2	353.2	46%	164
Contingency & Other Provisions	26.3	20.3	30%	6
Initial Capital Costs	543.5	373.5	46%	170

The main changes to the CAPEX estimate are as follows:

- Quotations were updated for the new exchange rates applied to imported equipment. This overall increase amounts to 65 MUSD.
- The mine plan now includes capitalized waste of 50 MUSD associated with the new mine plan. The previous mine plan did not include any capitalized waste as the overburden from JAC is used for construction materials. The 50 MUSD was previously reported as OPEX and is associated with waste movement to access the Shallow Gold zone.
- Owner and Indirect costs were updated to reflect a re-estimation of camp, bussing and catering costs required to support the new project execution plan.

Contingency increase as a result of the increased direct costs for plant and equipment.

22. Economic Analysis

The following section has been taken in its entirety from the previous NI 43-101 Technical Report Updated Pre-Feasibility Study, effective December 3rd, 2024, and report date January 17th, 2025. Only the description of RIGI has been updated to reflect the RIGI investment commitment requirements that were not available at the original time of writing. AbraSilver received official RIGI approval for the Diablillos project on May 7th, 2026.

22.1 Introduction

AbraSilver and Mining Plus developed the economic model using CAPEX and OPEX cost inputs from INSA Consultora, Bmining Chile, Envis Consulting and AbraSilver. The model was prepared following accepted engineering and financial principles. All dollar amounts in this analysis are expressed in US dollars, unless otherwise specified.

The economic analysis assumes the project will be 100% equity financed and considers incentives introduced under the new Large Investment Incentive Regime (*Regimen de Incentivo para Grandes Inversiones* -RIGI), enacted by the Argentinean Congress in July 2024 and implemented across most provinces, including Catamarca and Salta (See Chapter 22.14.1).

The economic analysis includes construction of the pre-operational stage (3 first year), and the subsequent production periods (14 years). The net present value (NPV) at a 5% discount rate is calculated from the start of project construction.

The financial evaluation estimates NPV, payback period (time in years to recapture the initial capital investment), and the internal rate of return (IRR). Annual cash flow projections were estimated over the life of the mine based on CAPEX, OPEX and sales revenue. Revenues are based on silver and gold production. The CAPEX and site OPEX estimates were developed specifically for this project and have been presented in earlier sections of this report.

Sensitivity analysis was done to key changes, like metal prices, metal recoveries, capital and operating costs.

22.2 Forward-Looking Information Cautionary Statements

Certain information and statements contained in this section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented herein. Information that is forward-looking includes the following:

- Mineral Resource and Reserve estimates.
- Assumptions about commodity prices and exchange rates.
- Proposed mine production plan.
- Projected mining and process recovery rates.
- Sustaining costs and proposed operating costs.
- Assumptions as to closure costs and closure requirements, as well as salvage value of assets at end of production.
- Assumptions as to environmental, permitting, and social risks.

Additional risks to the forward-looking information include:

- Production cost changes from what is assumed.
- Unrecognized environmental risks.
- Unanticipated reclamation expenses.
- Unexpected variations in quantity of mineralized material, grade, or recovery rates.
- Accidents, labour disputes, and other risks of the mining industry.
- Geotechnical or hydrogeological conditions during mining being different from what was assumed.
- Failure of mining methods to operate as anticipated.
- Failure of plant, equipment, or processes to operate as anticipated.
- Ability to maintain the social license to operate.
- Changes to interest rates.
- Changes to tax rates.

22.3 After-Tax Financial Result

Based on the Cash Flow Model results, the Diablillos Project has an unlevered after-tax NPV@5% of 747 MUSD, an after-tax IRR of 27.6%, a payback period of 2 years, and an annual average gross operating margin of 181 MUSD from start of operation. This considers a long-term silver price of 25.5 USD$/oz and gold of 2,050 USD$/oz. The key financial metrics of the Project are summarized in Table 22-1.

The All-In Sustaining Cost (AISC), which averages USD$12.67/oz AgEq over the life of mine, includes royalties, selling cost, mining cost, processing cost, administrative costs and sustaining capital.

Table 22-1: Summary of After-Tax Financial Results

Item	Unit	Value
Gold price	$/Oz	2,050
Silver price	$/Oz	25.5
After-Tax NPV @5%	M$	747
After-Tax IRR	%	27.60%
Pre-Tax NPV @5%	M$	1,114
Pre-Tax IRR	%	35.60%
Life of Mine (LOM)	years	14
Average Annual Production - Au (LOM)	oz	71,852
Average Annual Production - Ag (LOM)	Koz	7,613
Average Annual Production - AgEq (LOM)	Koz	13,362
Average Annual Production - Au (Year 1 to 5)	oz	58,740
Average Annual Production - Ag (Year 1 to 5)	Koz	11,694
Average Annual Production - AgEq (Years 1 to 5)	Koz	16,384
Total Production - Au	Koz	962
Total Production - Ag	Koz	103,193
All-In Sustaining Cost (LOM)	$/oz AgEq	12.67
All-In Sustaining Cost (Year 1 to 5)	$/oz AgEq	11.23
All-In Sustaining Cost (LOM)	$/oz AuEq	1,019
All-In Sustaining Cost (Year 1 to 5)	$/oz AuEq	903
Head Grade - Au (LOM)	g/t	0.81
Head Grade - Ag (LOM)	g/t	91
Head Grade - AgEq (LOM)	g/t	159
Recovery Au (LOM)	%	86.80%
Recovery Ag (LOM)	%	83.60%
Head Grade - Au (Year 1 to 5)	g/t	0.71
Head Grade - Ag (Year 1 to 5)	g/t	143
Head Grade - AgEq (Year 1 to 5)	g/t	201
Recovery Au (Year 1 to 5)	%	85.20%
Recovery Ag (Year 1 to 5)	%	83.50%
Initial Capital Expenditure (Contingency included)	M$	544
Sustaining Capital Cost	M$	77
Mine Life Operating Costs	$/t processed	45.4
Payback	years	2

22.4 Methods, Assumptions and Basis

The economic analysis was performed on the following assumptions and basis:

- The financial analysis was performed on Proven and Probable Mineral Reserves.
- The LOM NPV was determined on a pre-tax and after-tax basis.
- Annual cash flows used for NPV calculations are assumed to be realized at year-end.
- All costs and sales do not consider inflation or escalation factors.
- All gold and silver sales are assumed to occur in the same period as produced.
- Details of CAPEX and OPEX costs are provided in Chapter 21 of this report.
- Cash flows shown include royalty payments.
- Final closure costs are included in the period incurred.
- The financial analysis includes working capital.
- Taxes considers Large Investment Incentive Regime (RIGI).
- After-tax results and royalty payments were provided by Abra Silver.
- Mining Plus has not verified taxes.

22.5 Silver and Gold Price

The silver and gold prices used for the economic evaluation are 25.50 and 2,050 US$/oz respectively as defined by AbraSilver. These metal prices are guided by 3-year trailing averages, comparisons with other recently published studies and recent market conditions. The forecasts used are meant to reflect the average metals price expectation over the life of the project. No price inflation or escalation factors are considered.

The economic sensitivity of gold and silver prices is shown in the in Section 22.15.

22.6 Mine Production

Mine production was developed by Bmining, Chile, and it is reported direct ore, ore rehandled from stockpile and waste. The annual production figures were obtained from the mine plan as reported in Section 16. A total of 42.3 Mt of ore is mined at an average grade of 0.81 g/t Au and 90.70 g/t Ag. During the construction phase 35.8Mt is pre-stripping and during the operation phase a total of 262.7 Mt of waste is mined for a waste-ore stripping ratio of 6.2.

22.7 Plant Production

The silver and gold production plan was estimated by Bmining Chile considering the recoveries described in Chapters 13 and 17. The process plant is designed based on the following criteria:

- 9,000 tpd ore production.
- 83.6% average silver recovery.

- 86.8 % average gold recovery.
- Estimated life of mine silver production of 103.2 million ounces and gold production of 0.96 million ounces.

22.8 Revenue

Annual revenue is determined by applying estimated silver and gold prices to annual payable metal estimated for each operating year (Table 22-2). Sale prices for the entire life of mine production are not escalated or hedged. Revenue is the gross value of payable silver and gold and the NSR (Net Smelter Return) revenue accounts for royalty and selling costs.

Table 22-2: LOM Revenue summary

Description	LOM (MUSD)
Gold gross revenue	1,967.7
Silver gross revenue	2,626.2
Smelting and refining - Au	3.8
Smelting and refining - Ag	72.1
EMX Royalty Corp - Au	19.7
EMX Royalty Corp - Ag	26.3
Provincial government Royalty - Au	58.9
Provincial government Royalty - Ag	76.6
Total NSR revenues	4,336.4

22.9 Total Operating Cost

The average total unit operating cost over the life of the mine is estimated at $45.4 per tonne of ore processed. A detail of the operating cost is shown in Chapter 21 and summarized in Table 22-3.

Table 22-3: Operating Cost Summary

Description	LOM (MUSD)	$/t ore processed	$/oz Ageq
Mining	613.8	14.5	3.4
Processing	350.1	8.3	1.9
Utilities & off-site facilities	453.8	10.7	2.5
Maintenance	157.4	3.7	0.9
G&A	28.2	0.7	0.2
Logistics	137.4	3.2	0.8
Camp and service	181.8	4.3	1
Total Operating cost	1,922.5	45.4	10.7

22.10 Selling Cost and Royalties

The refining cost of gold sales used in the economic evaluation is $4/oz Au and $ 0.7/oz for silver. This value was estimated and provided by AbraSilver. Over the life of the project, the total royalty cost amounts to 181.5 MUSD, with an average cost estimated at $1.4 per payable ounce of silver equivalent

The royalties for the Diablillos Project are as follows:

- A 1% NSR royalty payable to EMX Royalty Corporation and
- 3% payable as a provincial mining royalty.

22.11 Capital Expenditure

22.11.1 Initial Capital

The financial indicators for the Diablillos Project were determined under the assumption of 100% equity financing for initial capital. The total initial capital estimate includes pre-stripping, infrastructure construction, owners' costs, contingencies and is 543.6 MUSD. A breakout of the capital cost is shown in Chapter 21.

22.11.2 Remediation and Closure Capital

Remediation costs include environmental monitoring at the end of the mine life, progressive guarantees from year 6 of the operation stage, remediation and dismantling. Dismantling considers the mine, waste dump, tailing storage facilities, plant and camps. This amounts to 26.4 MUSD.

22.11.3 Sustaining Capital

Sustaining capital is estimated at 76.9 MUSD and includes the renewal of the mining fleet, construction of phases 2, 3, 4 and 5 of the TSF, maintenance of the waste dump and import duties.

22.12 Total All-In Sustaining cost (AISC)

The average total All-In Sustaining cost over the life of the mine is estimated at $12.67 per ounce of payable silver equivalent, as shown in Table 22-4.

Table 22-4: All-In Sustaining Cost

Description	$/Oz Ag
Mining	3.4
Processing	1.9
Utilities & off-site facilities	2.5

Description	$/Oz Ag
Maintenance	0.9
G&A	0.2
Logistics	0.8
Camp and service	1.0
Selling cost and royalties	1.4
Sustaining capital and closure capital	0.6
Total cost	12.7

22.12.1 Working Capital

Working capital for accounts receivable and accounts payable will vary over the mine life based on revenue, operating costs, and capital costs. The turnover rate is 30 days for accounts receivable and 30 days for accounts payable. All working capital is assumed to be recaptured by the end of the mine life and the closing value of accounts is zero.

22.13 Depreciation

Depreciation of assets has been estimated based on Argentine regulations considering the RIGI regime. Fiscal reserves for exploration and resource development are not included in the depreciation calculation. The salvage value is not considered for the depreciation value of capital items, as salvage is considered taxable income.

22.14 Royalties and Taxes

22.14.1 Large Investment Incentive Regime – RIGI

The economic model has incorporated the incentives offered under the new large investment regime (RIGI) which was passed into law by the Argentinean congress in July, 2024 and implemented in most Argentinean provinces, including Catamarca and Salta.

Qualifying projects with expenditures above 200 MUSD, through the formation of a Special Purpose Vehicles (VPU, authorized entity type or creation of a dedicated branch), may apply for RIGI before the law expires in July, 2026. Once a project is approved for RIGI, a minimum of 80 MUSD must be invested within two years of approval (by no later than July, 2028), and the additional 120 MUSD to meet the minimum RIGI requirement invested in a reasonable timeframe. The Diablillos project meets all the required qualifications under RIGI. The execution plan considers obtaining RIGI approval by no later than Q2, 2026, giving the project until Q2, 2028 to spend 80 MUSD. According to the construction schedule, the 80 MUSD threshold for committed capital spending will be achieved in approximately five months after the project investment decision. Project execution would therefore be required to commence no later than the end of Q4, 2027 to ensure the project captures the RIGI benefits.

These benefits include:

Corporate income tax

- A reduction of the federal tax rate from 35% to 25%.
- Accelerated tax depreciation of plant and equipment.
- Tax losses will be adjusted for inflation with no time limit for carry-forward losses. If tax losses are not absorbed by taxable income within five years, these losses can be transferred to third parties.
- Dividend and profit distributions are taxable at a rate of 7%, however, after seven years from the registration date, distributions will be taxed at a reduced rate of 3.5%.
- Elimination of restrictions for interest expense and foreign exchange losses derived from foreign and local financing during the first five years from the registration date.

Value added tax (VAT)

- VPUs will be allowed to pay VAT to their suppliers using Fiscal Credit Certificates.

Tax on banking transactions

The Financial Transaction Tax, which generally applies at a 0.6% rate on debits and credits in Argentinian bank accounts, will be fully creditable against income tax payments.

Customs incentives

- Imports of goods, as well as temporary imports of capital goods, spare parts, parts, and components, will be exempt from import duties, statistical rate, and from all federal and/or local tax withholding, collection, advance payment, or retention regimes.
- Exports carried out by the VPU will be exempt from export duties (8% on gold and 4.5% on silver), if applicable, after three years from the approval of the application process.
- Registered VPUs can freely import and export without direct prohibitions, quantitative restrictions, quotas, or economic qualitative restrictions.

Foreign exchange incentives

- Proceeds from exports performed by the VPU will be exempt from the obligation to enter and/or negotiate and liquidate in the foreign exchange market in the following percentages: (1) 20% after two years from the registration date; (2) 40% after three years from the registration date; and (3) 100% after four years from the registration date.
- Further, VPUs also will have free availability of foreign currency from local or external financing disbursed after the enactment of the regime. Additionally, exchange regulations imposing restrictions or prior authorizations for accessing the currency market for repaying loans, interest, accessories, other financial debts, or repatriating direct investments by non-residents, or for paying dividends, profits, or interests to non-residents, will not apply to VPUs provided some requirements are met.

Tax Stability

- RIGI will grant eligible VPUs with regulatory stability in tax, customs, and exchange matters for their projects, ensuring that the incentives awarded will not be affected by their repeal or by the creation of more burdensome regulations. This stability will last for 30 years from the registration date.

22.14.2 Royalties

The economic evaluation of the Diablillos project also considers the following royalties and taxes:

- A municipal tax of 1.2% is applied on revenue net of royalties and export duties.
- Argentina has a bank transaction tax for deposits for bank debits and credits of 0.6%
- in and 0.6% out for a total of 1.2%. This applies to all cash flows into and out of the project.
- Stamp Tax of 1.6 % applied on 80% of operating and capital costs.

22.15 After- tax NPV Sensitivity Analysis

The most important model input is the silver and gold price. To assess sensitivity, six price scenarios were examined as illustrated in Figure 22-1 and Figure 22-2

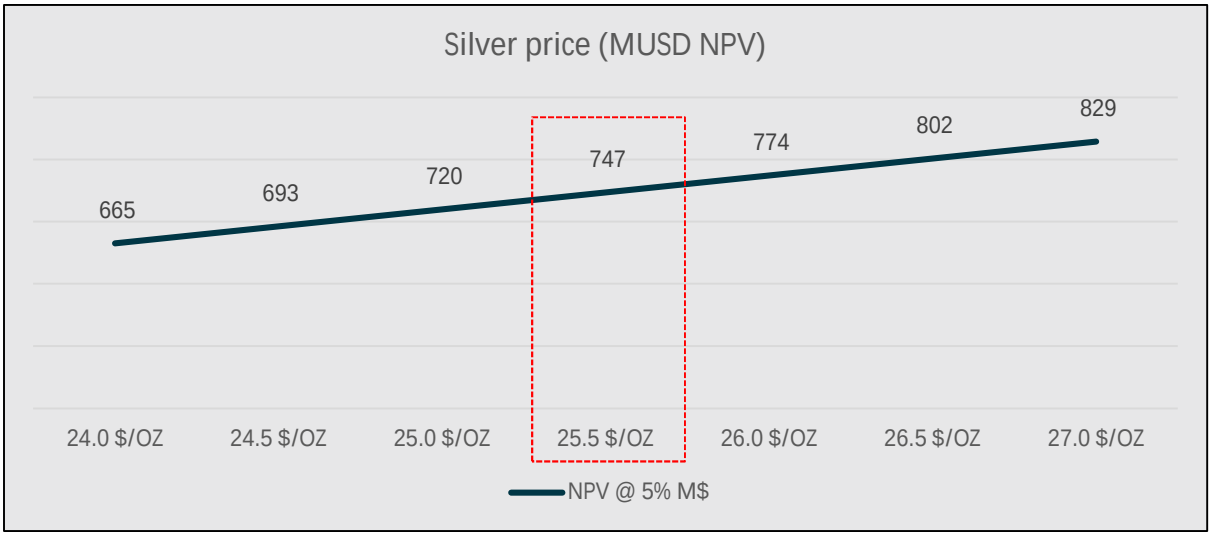


Figure 22-1: Silver Price Sensitivity

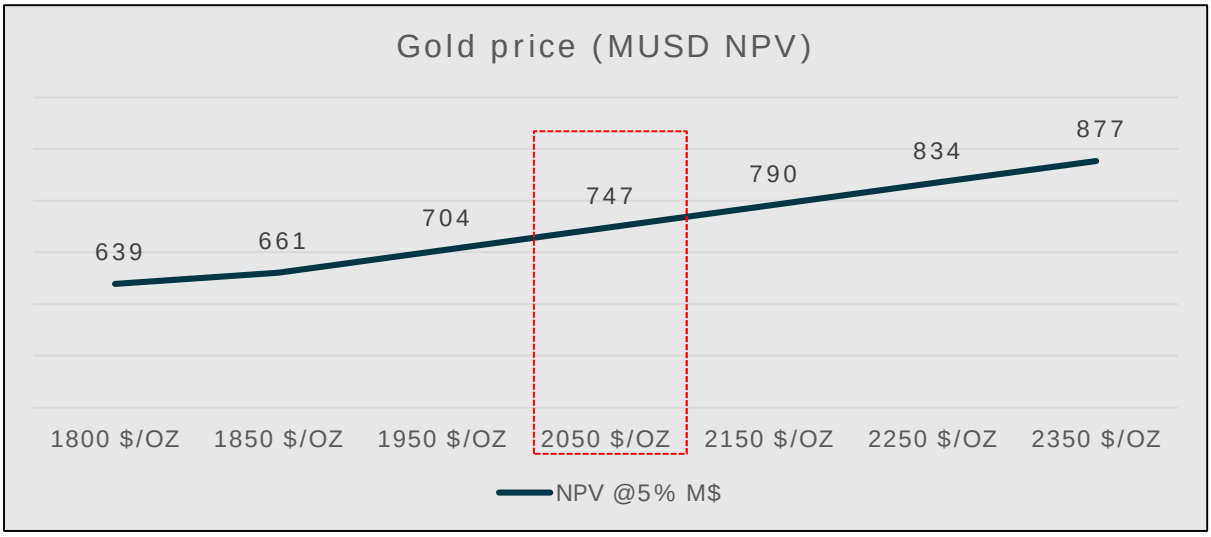


Figure 22-2: Gold Price Sensitivity

The sensitivity analysis also considered: CAPEX, OPEX, silver recovery, gold recovery and the discount rate (Figure 22-3 and Figure 22-4).

As shown in Figure 22-3, the sensitivity analysis shows that the Diablillos Project, apart from the price of silver and gold, is more sensitive to OPEX than CAPEX. If the OPEX is increased by 15%, the NPV would be reduced to $600M, while if the CAPEX increases by the same percentage, the NPV would be reduced to $663M.

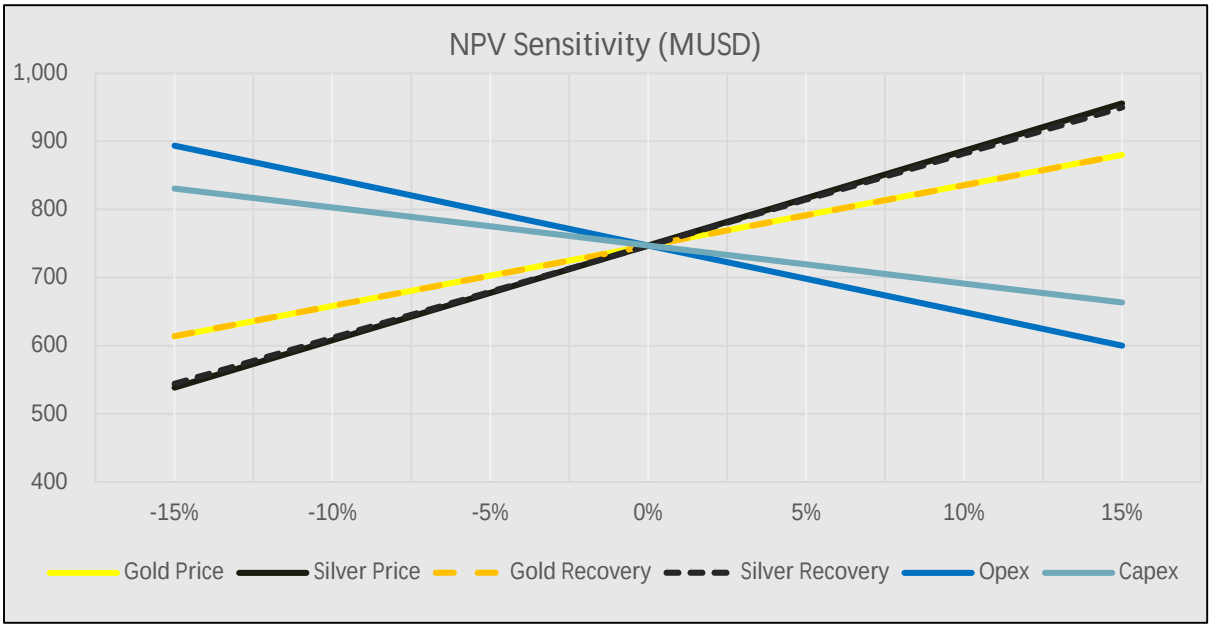


Figure 22-3: Sensitivity Analysis

Figure 22-4 shows the sensitivity of the NPV to the discount rate. It indicates that at a 12% discount rate, the NPV is 361 MUSD, demonstrating the robustness of the project.

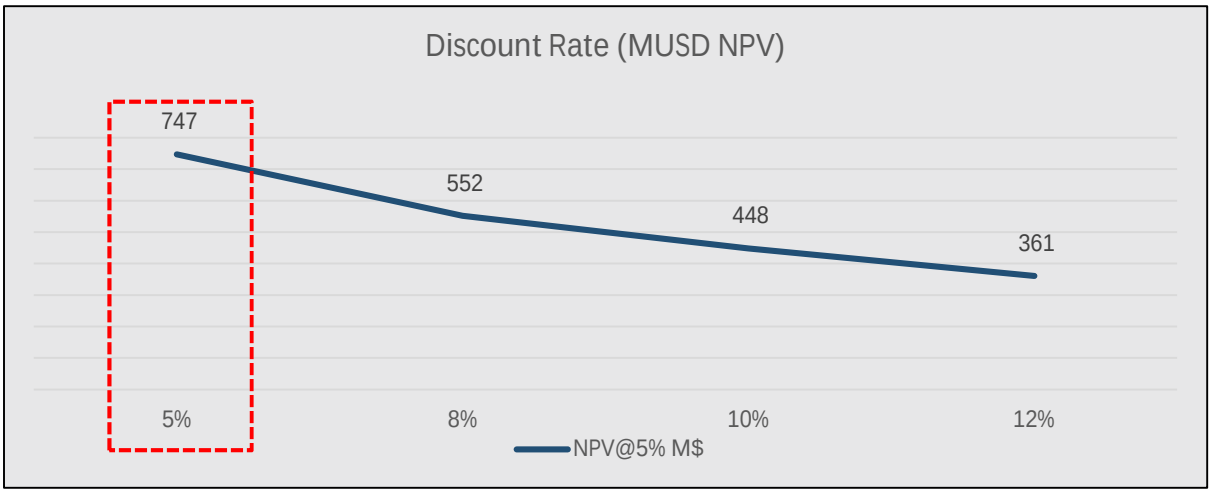


Figure 22-4: Discount Rate Sensitivity

A summary of the annual and cumulative cash flow balance is presented in Figure 22-5. The cumulative cashflow over the 17-year life of the mine amounts to $1,231M.

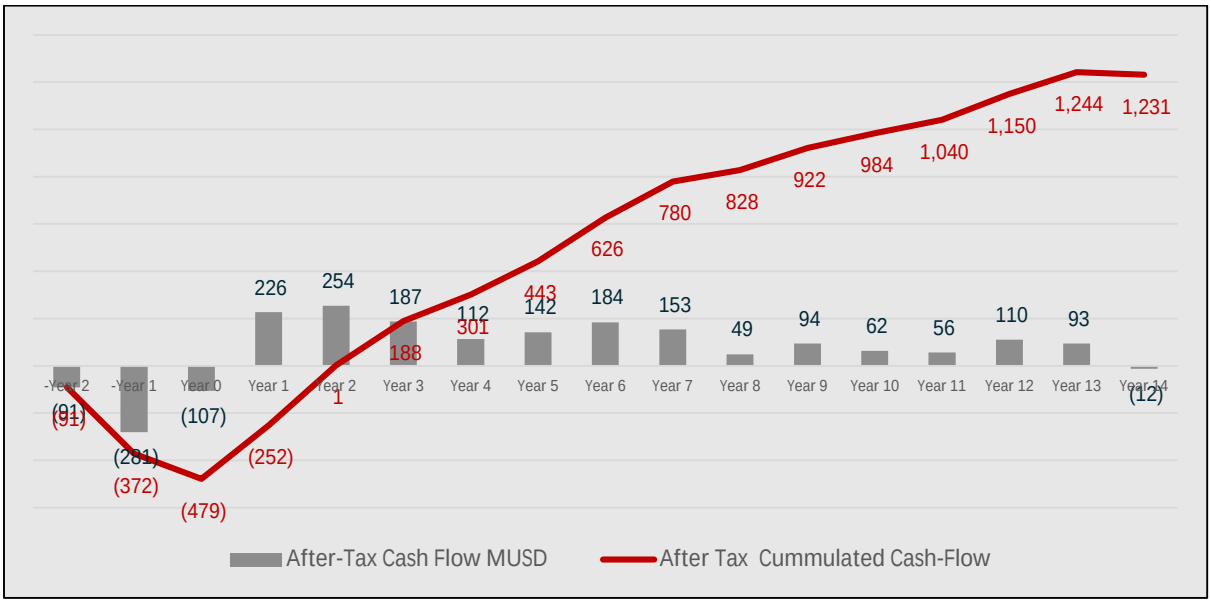


Figure 22-5: Cash Flow Profile

The annual production schedule and the estimated cash flow forecast for the Diablillos Project are presented in Figure 22-5 and in detail in Table 22-5. This table provides a detailed breakdown of the mine production plan, silver and gold production schedules, projected sales, applicable royalties, annual NSR values, OPEX, CAPEX, and taxes. These figures are further explained in

Chapter 22 and preceding chapters, offering a comprehensive overview of the project's economics and operational framework.

Table 22-5: Diablillos Cash Flow by Year

Abrasilver Resource - Diablillos Project Cash Flow		Stage	Construction		Pre-Operation	Operation														
		Total	-Year 2	-Year 1	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14
			2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042
Mining Schedule																				
Ore direct to plant	Kt	32,626	-	-	1,575	592	3,150	3,150	3,150	3,150	1,739	1,194	3,150	3,150	1,003	3,150	3,150	1,323	-
Au grade	g/t	0.84	-	-	0.49	0.68	0.47	0.85	0.58	1.23	0.27	0.45	0.58	1.33	0.48	1.01	1.35	1.06	-
Ag grade	g/t	79.86	-	-	90.3	55.4	203.7	126.3	88.0	81.5	151.0	107.0	53.4	25.7	49.0	28.9	18.4	14.6	-
Ore to stockpile	Kt	9,703	-	298	1,988	-	72	571	3,052	757	-	-	1,546	52	-	1,010	356	-	-
Au grade	g/t	0.72	-	0.7	0.2	-	0.7	0.4	0.7	1.0	-	-	0.9	0.9	-	1.5	1.0	-	-
Ag grade	g/t	127.16	-	77.2	281.0	-	30.6	85.8	147.4	98.9	-	-	32.4	25.0	-	19.7	13.4	-	-
Ore rehandled from stockpile	Kt	9,703	-	-	298	1,988	-	-	-	-	1,411	1,956	-	-	2,147	-	-	1,903	-
Au grade	g/t	0.72	-	-	0.7	0.2	-	-	-	-	0.8	0.7	-	-	0.7	-	-	1.2	-
Ag grade	g/t	127.16	-	-	77.4	281.0	-	-	-	-	155.2	132.8	-	-	59.2	-	-	24.3	-
Waste to dump	Kt	298,357	12,751	23,052	20,081	21,930	21,644	21,998	18,384	26,548	23,874	24,286	20,223	17,284	22,438	16,840	4,594	2,430	-
Cover/overburden material	Kt	63,038	10,632	9,784	2,552	5,685	3,098	2,701	10,580	4,879	1,859	6,357	2,157	2,541	213	-	-	-	-
Waste	Kt	235,319	2,119	13,268	17,529	16,244	18,546	19,298	7,804	21,669	22,015	17,929	18,066	14,743	22,225	16,840	4,594	2,430	-
Total material mined	Kt	340,687	12,751	23,350	23,644	22,522	24,866	25,720	24,586	30,455	25,613	25,480	24,919	20,486	23,441	21,000	8,100	3,753	-
Total material moved	Kt	350,391	12,751	23,350	23,942	24,510	24,866	25,720	24,586	30,455	27,024	27,436	24,919	20,486	25,588	21,000	8,100	5,656	-
Stripping ratio	w:o	6.20	-	77.3	5.6	37.0	6.7	5.9	3.0	6.8	13.7	20.3	4.3	5.4	22.4	4.0	1.3	1.8	-
Production Schedule																				
Total ore to plant	Kt	42,330	-	-	1,873	2,580	3,150	3,150	3,150	3,150	3,150	3,150	3,150	3,150	3,150	3,150	3,150	3,226	-
Au grade	g/t	0.81	-	-	0.5	0.3	0.5	0.9	0.6	1.2	0.5	0.6	0.6	1.3	0.6	1.0	1.4	1.2	-
Ag grade	g/t	90.7	-	-	88.2	229.2	203.7	126.3	88.0	81.5	152.9	123.0	53.4	25.7	55.9	28.9	18.4	20.3	-
Au contained	Koz	1,108	-	-	32	28	48	86	59	125	50	62	59	135	65	102	137	120	-
Ag contained	Koz	123,443	-	-	5,314	19,019	20,630	12,787	8,914	8,257	15,483	12,461	5,411	2,604	5,666	2,929	1,861	2,108	-
Average gold recovery	%	86.8%			87.9%	85.8%	85.5%	84.5%	86.3%	83.7%	83.8%	85.5%	88.1%	86.3%	87.1%	89.1%	89.2%	89.2%	
Average silver recovery	%	83.6%			79.6%	85.3%	85.1%	81.8%	83.1%	82.7%	85.4%	82.9%	82.3%	82.2%	83.0%	81.7%	81.7%	81.8%	
Au recovered	Koz	962	-	-	27.7	24.1	40.9	73.0	51.1	104.6	42.1	52.6	52.2	116.4	56.7	91.3	122.2	106.9	-
Ag recovered	Koz	103,193	-	-	4,227.6	16,226.2	17,546.4	10,461.5	7,410.3	6,826.0	13,225.4	10,332.2	4,455.7	2,139.1	4,704.0	2,393.3	1,521.0	1,724.0	-
Revenues																				
Gross revenue																				
Gold price	USD/oz	2,050.00	2,050	2,050	2,050	2,050	2,050	2,050	2,050	2,050	2,050	2,050	2,050	2,050	2,050	2,050	2,050	2,050	
Silver price	USD/oz	25.50	25.5	25.5	25.5	25.5	25.5	25.5	25.5	25.5	25.5	25.5	25.5	25.5	25.5	25.5	25.5	25.5	
Gold payable	Koz	960	-	-	28	24	41	73	51	104	42	53	52	116	57	91	122	107	-
Silver payable	Koz	102,986	-	-	4,219	16,194	17,511	10,441	7,396	6,812	13,199	10,312	4,447	2,135	4,695	2,389	1,518	1,721	-
Gold gross revenue	MUSD	1,968	-	-	57	49	84	149	105	214	86	108	107	238	116	187	250	219	-
Silver gross revenue	MUSD	2,626	-	-	108	413	447	266	189	174	337	263	113	54	120	61	39	44	-
Total gross revenue	MUSD	4,594	-	-	164	462	530	415	293	388	423	371	220	293	236	248	289	263	-
Sales																				
Smelting and refining - Au	MUSD	4		-	0	0	0	0	0	0	0	0	0	0	0	0	0	0	-
Smelting and refining - Ag	MUSD	72		-	3	11	12	7	5	5	9	7	3	1	3	2	1	1	-
Total smelting and refining	MUSD	76	-	-	3	11	12	8	5	5	9	7	3	2	4	2	2	2	-
Royalties																				
EMX Royalty Corp - Au	MUSD	20	-	-	1	0	1	1	1	2	1	1	1	2	1	2	2	2	-
EMX Royalty Corp - Ag	MUSD	26	-	-	1	4	4	3	2	2	3	3	1	1	1	1	0	0	-
Provincial government Royalty - Au	MUSD	59	-	-	2	1	3	4	3	6	3	3	3	7	3	6	7	7	-
Provincial government Royalty - Ag	MUSD	77	-	-	3	12	13	8	6	5	10	8	3	2	3	2	1	1	-
Total royalties	MUSD	181	-	-	6	18	21	16	12	15	17	15	9	12	9	10	12	10	-
NSR																				
Net revenues for gold	MUSD	1,885	-	-	54	47	80	143	100	205	82	103	102	228	111	179	239	210	-
Net revenues for silver	MUSD	2,451	-	-	100	385	417	248	176	162	314	245	106	51	112	57	36	41	-
Total NSR revenues	MUSD	4,336	-	-	155	433	497	392	276	367	397	349	208	279	223	236	276	251	-

Abrasilver Resource - Diablillos Project
Cash Flow

		Stage	Construction		Pre-Operation	Operation													
		Total	-Year 2	-Year 1	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14
			2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042
Operating costs																			
Mining	MUSD	614	-	-	39	43	44	46	43	53	52	53	49	48	47	45	23	29	-
Processing	MUSD	350	-	-	15	26	26	26	26	26	26	26	26	26	26	26	26	26	-
Utilities & off site facilities	MUSD	454	-	-	17	34	34	34	34	34	34	34	34	34	34	34	34	34	-
Maintenance	MUSD	157	-	-	3	12	12	12	12	12	12	12	12	12	12	12	12	12	-
G&A	MUSD	28	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Logistics	MUSD	137	-	-	5	9	10	10	10	10	10	10	10	10	10	10	10	10	-
Camp and service	MUSD	182	-	-	9	13	13	13	13	13	13	13	13	13	13	13	13	13	-
Total operating cost	MUSD	1,923	-	-	90	138	141	143	140	150	148	150	145	145	144	142	120	126	-
Cash cost per ounces of Ag equivalent	US$/oz Ag eq	12.1		-	15.4	9.3	8.4	10.3	13.7	11.2	10.5	11.9	18.2	13.8	17.0	15.8	11.7	13.4	-
Cash cost per ounces of Au equivalent	US$/oz Au eq	973		-	1,237	745	675	824	1,099	900	846	954	1,465	1,109	1,364	1,272	944	1,076	-
Capital costs																			
Initial capital																			
Mining	MUSD	128.6	20	73	36	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing plant	MUSD	111.7	0.2	102	10	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure	MUSD	166.7	33	63	71	0	-	-	-	-	-	-	-	-	-	-	-	-	-
Indirect cost	MUSD	110.2	37	40	32	2	-	-	-	-	-	-	-	-	-	-	-	-	-
Contingency	MUSD	26.3	-	-	13	13	-	-	-	-	-	-	-	-	-	-	-	-	-
Import duties (guarantees)	MUSD	0.1	0	0	0														
Total inicial capital	MUSD	543.6	90	277	161	15	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining capital																			
Sustaining	MUSD	77	-	-	-	4	21	9	2	16	0	-	10	-	0	12	0	1	0
Contingency	MUSD	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Import duties (guarantees)	MUSD	0				0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total sustaining capital	MUSD	77	-	-	-	4	21	10	2	16	0	0	10	0	0	12	0	1	0
Remediation and closure capital																			
Remediation and closure cost	MUSD	26	-	-	-	-	-	-	-	-	1	1	1	1	1	1	1	1	22
Total capital cost	MUSD	647	90	277	161	19	21	10	2	16	1	1	11	1	1	13	1	2	23

Abrasilver Resource - Diablillos Project
Cash Flow

		Stage	Construction		Pre-Operation	Operation													
		Total	-Year 2	-Year 1	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14
			2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042
Working Capital																			
Working capital																			
Accounts receivable yearly		4,594		-	164	462	530	415	293	388	423	371	220	293	236	248	289	263	-
Accounts receivable adjusted		378		-	14	38	44	34	24	32	35	30	18	24	19	20	24	22	-
Change in accounts receivable				-	14	24	6	(9)	(10)	8	3	(4)	(12)	6	(5)	1	3	(2)	(22)
Accounts payable yearly		2,180		-	99	168	174	167	157	170	174	172	157	158	157	154	133	138	-
Accounts payable adjusted		179		-	8	14	14	14	13	14	14	14	13	13	13	13	11	11	-
Change in accounts payable				-	8	6	1	(0.6)	(0.8)	1	0	(0.2)	(1.2)	0	(0.1)	(0.3)	(1.7)	0	(11.3)
Change in working capital		0		-	(5)	(19)	(5)	9	9	(7)	(3)	4	11	(6)	5	(1)	(5)	3	10
Pre-tax Cash Flow																			
Gross Revenues	MUSD	4,594	-	-	164	462	530	415	293	388	423	371	220	293	236	248	289	263	-
Selling cost	MUSD	(257)	-	-	(10)	(30)	(33)	(24)	(17)	(21)	(26)	(22)	(12)	(14)	(13)	(12)	(13)	(12)	-
Operating cost	MUSD	(1,923)	-	-	(90)	(138)	(141)	(143)	(140)	(150)	(148)	(150)	(145)	(145)	(144)	(142)	(120)	(126)	-
Gross Margin from Operations	MUSD	2,414	-	-	65	294	356	248	136	218	248	198	63	134	79	94	156	125	-
Change in Net Working Capital	MUSD	0	-	-	(5)	(19)	(5)	9	9	(7)	(3)	4	11	(6)	5	(1)	(5)	3	10
Capital expenditures	MUSD	(647)	(90)	(277)	(161)	(19)	(21)	(10)	(2)	(16)	(1)	(1)	(11)	(1)	(1)	(13)	(1)	(2)	(23)
Pre-Tax Cash Flow before depreciation	MUSD	1,767	(90)	(277)	(102)	256	329	248	143	195	245	202	63	128	83	80	150	126	(12)
Taxes																			
Gross Margin from Operations	MUSD	2,414	-	-	65	294	356	248	136	218	248	198	63	134	79	94	156	125	-
Depreciation	MUSD	(491)	-	(21)	(117)	(158)	(70)	(18)	(20)	(18)	(16)	(16)	(10)	(6)	(4)	(4)	(4)	(4)	(4)
Operating Income	MUSD	1,923	-	(21)	(52)	136	286	231	116	200	232	183	53	129	75	90	152	121	(4)
Provincial tax - Gross income	MUSD	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Provincial tax - Stamp tax	MUSD	(33)	(1)	(4)	(3)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(0)
Municipal tax on total revenue	MUSD	(52)	-	-	(2)	(5)	(6)	(5)	(3)	(4)	(5)	(4)	(2)	(3)	(3)	(3)	(3)	(3)	-
Export duty refund Au	MUSD	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Export duty refund Ag	MUSD	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Taxable income	MUSD	1,926	-	-	-	129	278	224	110	193	225	176	48	123	70	85	147	116	-
Federal tax	MUSD	(481)	-	-	-	(32)	(69)	(56)	(28)	(48)	(56)	(44)	(12)	(31)	(18)	(21)	(37)	(29)	-
Transaction tax	MUSD	31	1	3	3	2	2	2	2	2	2	2	2	2	2	2	1	2	0
Effective tax	MUSD	(451)	-	-	-	(23)	(67)	(54)	(26)	(46)	(54)	(42)	(10)	(29)	(16)	(19)	(35)	(28)	0
Total taxes	MUSD	(536)	(1)	(4)	(5)	(30)	(75)	(61)	(31)	(53)	(61)	(48)	(15)	(34)	(20)	(24)	(40)	(32)	(0)
After-tax Cash Flow																			
EBIT	MUSD	1,923	-	(21)	(52)	136	286	231	116	200	232	183	53	129	75	90	152	121	(4)
Depreciation	MUSD	491	-	21	117	158	70	18	20	18	16	16	10	6	4	4	4	4	4
Taxes	MUSD	(536)	(1)	(4)	(5)	(30)	(75)	(61)	(31)	(53)	(61)	(48)	(15)	(34)	(20)	(24)	(40)	(32)	(0)
Change in Net Working Capital	MUSD	0	-	-	(5)	(19)	(5)	9	9	(7)	(3)	4	11	(6)	5	(1)	(5)	3	10
Capital expenditures	MUSD	(647)	(90)	(277)	(161)	(19)	(21)	(10)	(2)	(16)	(1)	(1)	(11)	(1)	(1)	(13)	(1)	(2)	(23)
After-Tax Cash Flow	MUSD	1,231	(91)	(281)	(107)	226	254	187	112	142	184	153	49	94	62	56	110	93	(12)
After Tax Cummulated Cash-Flow	MUSD		(91)	(372)	(479)	(252)	1	188	301	443	626	780	828	922	984	1,040	1,150	1,244	1,231

23. Adjacent Properties

23.1 Metallic Projects

23.1.1 Condor Yacu

The Condor Yacu property adjoins Diablillos on the southern boundary and was once part of the Diablillos claim block. Prior to 1990, the property was explored by various parties, including geological studies by a Dr. O Gonzalez from 1971 to 1973, metallurgical testwork carried out by S. Hochschild S.A. of Copiapo, Chile, on behalf of the Banco Nacional de Desarollo (BND) in 1975, and a magnetic survey and surface sampling program by Pecomrio S.A.M. in 1981.

In 1984, the BND and the Mining Directorate of the Catamarca Province mined approximately 350 tons, which were systematically sampled and analyzed. The University of Jujuy carried out some gravity-concentration testwork in 1985, which was ultimately unsuccessful. Geological mapping at a scale of 1:1000 was conducted by Kleine-Hering in 1987.

Exploration in 1987 and 1988 is not well documented. However, AbraSilver's geologists believe that Ophir drilled 22 RC holes in 1987. During the 1990s, Cavok S.R.L obtained the property and carried out a ground magnetic survey and drilled 15 diamond drill holes in 1999 and 2000. In 2001, Cardero Resource Corp. (Cardero) signed an agreement with Cavok S.R.L. to acquire a 100% share of the project. In the same year, an induced polarization (IP) survey was carried out over the property and 396.24 m were drilled in five diamond drill holes. A further nine holes totaling 842.17 m were completed in 2002.

In 2003, Maximus Ventures Ltd. (Maximus) signed an agreement with Cardero to acquire an 80% interest in the Project. In the same year, Maximus drilled a total of 1,516.10 m in 17 diamond drill holes. Both Cardero and Maximus withdrew from the project in 2004.

The Condor Yacu prospect is located 2.75 km to the southeast of Oculto and is thought by AbraSilver's geologists to be closely associated with the eastern bounding Pedernales graben fault. This zone of mineralization occurs in granitoids of the Oire Formation of the Faja Eruptiva. The main Condor Yacu structure was divided into two zones, termed the Southern Outcrop and the Northern Outcrop.

Most exploration has focused on the Southern Outcrop, consisting of a high sulphidation silicified breccia within granodiorite host rocks. At surface the zone is over 16 m wide, narrowing to less than 2 m at depth. It has been intersected in drill holes over a north-south strike of 90 m and to a vertical depth of 140 m. Drilling has intersected grades of up to 28.35 g/t Au, 147 g/t Ag, and 2.67 % Cu. The Northern Outcrop is also a silicified, brecciated north-south trending structure. It is about 15 m wide on surface, narrowing to 10 m at a depth of 100 m, open-ended along strike. Grades are generally lower than at the Southern Outcrop, with gold less than 2.0 g/t Au.

A third zone is known to exist to the east of the Northern and Southern Outcrop areas. This zone is buried below overburden and has been subject to minimal exploration. Gold values of up to 0.34 g/t Au have been reported.

23.1.2 Rumi Cori

The Rumi Cori property also adjoins Diablillos on the southern boundary. This is an epithermal prospect consisting of several siliceous veins in granite that is lcoated 2 km south of Diablillos. Unconfirmed values of gold (0.50 g/t) and copper (0.69 %) have been reported. Surficial exploration has been carried out on the prospect to date.

23.1.3 Incahuasi

The Incahusi project is located 41 km southeast of Diablillos. The mine was originally exploited by Jesuit missionaries and mining continued until 1954 when it ceased due to flooding. The deposit comprises gold in mesothermal veins of Ordovician age. The veins occur in marine sedimentary rocks of the same age and consist of meta-pelites and greywackes. The veins of have north-south trending strikes of up to a minimum of 700 m with widths varying between 0.5 m and 2.6 m. Underground development has traced the veins for a minimum down dip extension of 130 m. The mineralization occurs as free gold in quartz veins and veinlets with minor associated pyrite, arsenopyrite, and chalcopyrite. ROM gold grades were reportedly 17.6 g/t Au with local bonanza grades of up to 300 g/t Au. Past production is estimated at 2,000 kg Au.

23.1.4 Inca Viejo

The Inca Viejo project is located 16 km north of Diablillos. The area has been worked since Inca times, but the first systematic exploration was carried out in 1994 and 1995 by Grupo Minera Aconcagua S.A. This work consisted of lithological, alteration, structural, and mineralization mapping, surface geochemistry, and 11,500 line-meters of spectral IP on 11 sections.

Host lithologies consist of basement Paleozoic rocks characterized by meta-sedimentary rocks of Ordovician age. Intruded by Silurian granite, granodiorite, and rhyodacite. These basement rocks are in turn intruded by a dacite porphyry with associated breccia pipes and bodies. Mineralization consists of porphyry-style copper and gold within the intrusives and breccias. A later unaltered andesitic porphyry intrudes the dacite porphyry. The dacite displays an altered potassic silicified core with a halo of sericite alteration.

Minera Aconcagua drilled eight widely spaced (between 300 m and 500 m) RC holes. The best copper values were in borehole AR5, which returned an intersection of 0.70% Cu over 30 m. Borehole AR6 had an average of 0.23% Cu over 73.5 m. Surface gold values are up to 1.70 g/t Au with the central part of the system having values greater than 0.2 g/t Au over an area of 300 m by 100 m. The best gold values intersected were in borehole AR1, which returned a value of 0.25 g/t Au over 54 m in the leach cap.

23.1.5 Pistola de Oro

The Pistola de Oro project is 20.5 km north-northeast of Diablillos. It includes the Volcan and Soroche mines, which were worked on a limited scale in the past before the workings collapsed. These mines are located on a polymetallic (Au-Ag-Cu-Zn-Pb) vein system in Precambrian basement rocks consisting of micaceous schists. Vein gangue mineralization is principally quartz, with a minimum strike length of 650 m and a minimum down dip extension of 70 m. A sample taken in 2009 reportedly returned values of 2.21 g/t Au, 165 g/t Ag, 1.13% Cu, 5.18% Pb, and 0.55% Zn.

A second type of mineralization occurs in hydrothermal breccia. It has an ellipsoid shape on surface with dimensions of 600 m by 300 m. Composition is angular clasts of bleached micaceous schists varying in size from millimetre-scale to more than 20 cm in diameter. The matrix is black to dark grey and aphanitic consisting of quartz and tourmaline. The mineralization is fine-grained and consists of malachite and sphalerite. A sample taken in 2009 returned a value of 0.42 g/t Au, 7.9 g/t Ag, 0.86% Cu, 0.16% Pb, and 0.11% Zn. The results of a limited drill program carried out in the late 1990s are unknown.

23.1.6 Vicuña Muerta

The Vicuña Muerta project is located 30 km to the north-northeast of Diablillos. This project consists of an unexplored porphyry complex. Geology consists of a rhyolitic porphyry intruded into Ordovician granites, granodiorites, diorites, and gabbros. Three phases of porphyritic intrusion have been recognized and have been hydrothermally altered consisting of quartz-sericite, argillic alteration and silicification. In the 1990s La Pacha Minera, reported maximum values from surface rock chip and soil sampling of 0.29 g/t Au to 0.38 g/t Au, 145 g/t Ag to 210 g/t Ag, and 0.11% Cu to 0.35% Cu. In addition to the porphyry mineralization, satellite auriferous veins have been sampled with values of up to 7.47 g/t Au. No drilling has been done on this project.

23.2 Lithium and Non-Metallic Projects

There are 23 lithium projects active in the area, with two already producing and expected to expand. Another is under construction, while 16 are at a feasibility stage or advanced exploration. Another 20 are in the early stages of exploration. The following projects are located close to Diablillos:

23.2.1 Fenix

The Fenix project is owned and operated by the Argentine company Minera Altiplano S.A., a subsidiary of Livent Corporation, formerly FMC Corporation. The project is 30 km southwest of Diablillos in the western basin of the Salar de Hombre Muerto. The operation has been producing lithium carbonate and lithium chloride since 1998 and has an estimated life of 2038. An expansion is planned in two consecutive stages from the current 20,000 to 40,000 tons per year. This is based on an off-take agreement with the German car manufacturer BMW, with delivery planned to begin in 2025. Exploitation is through the pumping of brines directly from the Salar (salt pan) to a

fully automated selective absorption plant that extracts lithium and returns the solution to the Salar. The onsite plant derives its energy from a natural gas pipeline, which is used to drive the steam boilers required by the treatment process. Electrical energy is derived from five diesel powered generators. The company notably has an airstrip for transportation of employees and the delivery of consumables.

23.2.2 Kachi

Kachi is located 100 km south of Fenix in the Catamarca province and under advanced exploration. They are currently performing testworks at a pilot scale with their technological partner, Lilac Solutions, to validate their direct extraction process technology.

23.2.3 Sal de Vida

Sal de Vida is in the eastern basin of the Salar de Hombre Muerto and 10 km southwest of Diablillos. Galaxy Resources Ltd merged with lithium producer Orocobre and was renamed Arcadium. The project is set for 32,000 tonnes per year of lithium carbon equivalent production using conventional brine extraction, evaporation, and processing. Pilot ponds and testing are currently underway.

23.2.4 Sal de Oro

Galaxy Resources sold their northern properties called Sal de Oro within the Salar del Hombre Muerto to the Korean POSCO, who are now advancing the project. A construction camp and pilot facilities are currently under construction.

23.2.5 Sal de los Angeles

The Sal de los Angeles project is in the Diablillos Salar located east of Diablillos. The project is currently operated by a Joint Venture conformed by Salta Exploraciones SA and Potasio y Litio Argentina SA, with the former under control. They are currently operating seven evaporation ponds fed with brine from an artesian well. A construction camp is to be completed within the next months, which will compliment the existing camp. The estimated final production rate targets 15,000 tonnes per year of lithium carbon equivalent and 50,000 t/y of potassium chloride.

23.2.6 Centenario – Ratones

Lithium exploration activities have focused on the Centenario and Ratones salars, 25 km north of Diablillos. The property concessions are owned by the local company Eramine Sudamerica S.A., which is wholly owned by the French conglomerate Eramet. The Eramet website reports that the company is producing from the Centenario-Ratones mine as of 2024.

23.2.7 Tincalayu

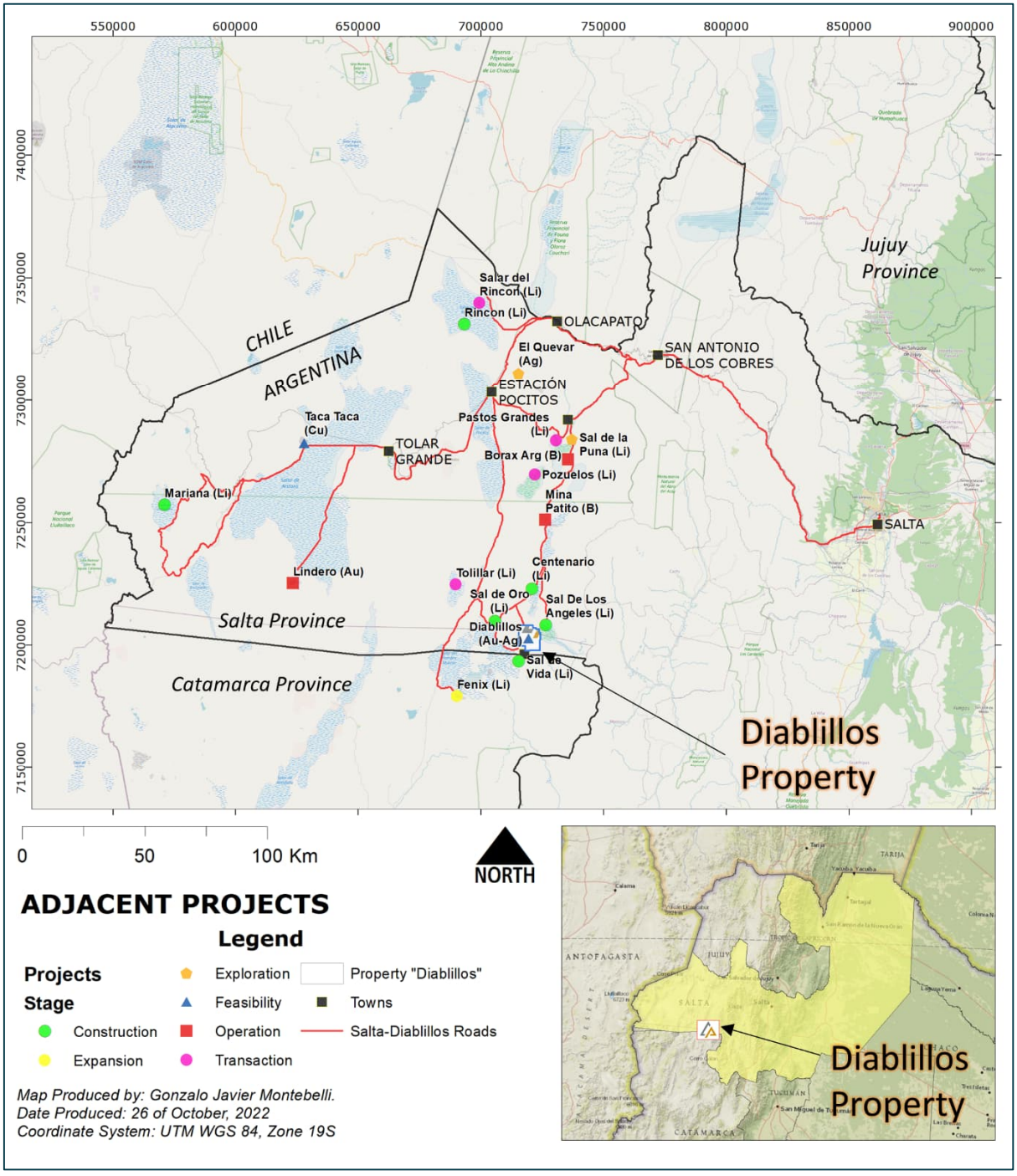
Borax Argentina is the principal producer of borate products in Argentina. The Tincalayu open pit mine and plant are located 26 km west of Diablillos. The borates occur in Tertiary age rocks and are related to paleo-salars.

23.2.8 Pozuelos – Pastos Grandes

The Pozuelos project is on the Salar de Pozuelo and is being operated by Litica, a subsidiary of the Argentine oil and gas company Pluspetrol. They are currently setting up a pilot plant for a DLE process targeting production of 25,000 t/y of lithium carbon equivalent .

23.2.9 Salar de Pastos Grandes

This project is owned by Proyecto Pastos Grandes, a 100% owned local subsidiary of Millennial Lithium Corporation of Canada. The target is to produce 24,000 t/y of lithium carbon equivalent with a LOM of 40 years based on conventional evaporation and processing techniques. The project obtained its EIA approvals and finalized its DFS. It is currently operating evaporation ponds and a liming plant. This project has produced high purity battery grade lithium carbonate from a pilot plant.



Source: AbraSilver Resource Corp., 2023.

Figure 23-1: Metallic and Non-Metallic Adjacent Projects

24. Other Relevant Data and Information

The following section has been taken in its entirety from the previous NI 43-101 Technical Report Updated Pre-Feasibility Study, effective December 3rd, 2024, and report date January 17th, 2025. The description of RIGI and key interim project milestones have been updated to reflect new information.

24.1 Project Execution Schedule

The Project Execution Plan (PEP) includes all the activities for implementing the Diablillos project to first silver and gold production (First Silver), including pre-construction planning required to reach an investment decision for the project. The strategy of the PEP is to deliver the project within the timeframe to capture the economic incentives offered under RIGI, the new congressional law for large investments. The mechanism and benefits of RIGI are described in Chapter 22.

This section describes the basis of the schedule, the overall project execution approach and key milestones.

24.2 Critical Path Summary

The following is a critical path summary to First Silver for the Diablillos project:

- Preparation and completion of the Definitive Feasibility Study (DFS)
- Update of EIA application upon the end of the DFS
- Completion of the RIGI application
- Preparation of a NI 43-101 Technical Report for the DFS
- Investment decision/financing
- Camp construction
- Completion of the bore field installation and piping connection to the camp
- Order and completion of the SAG/Ball mill installation and other long lead items
- Completion of the TSF, including pumping and piping systems
- Pre-commissioning
- Mechanical, electrical and instrumentation completion
- Commissioning completion.

The project schedule includes five months for the preparation of approval documents and finance.

The key milestones leading up to an investment decision are as follows:

- EIA approval
- Completion of the DFS
- RIGI approval.

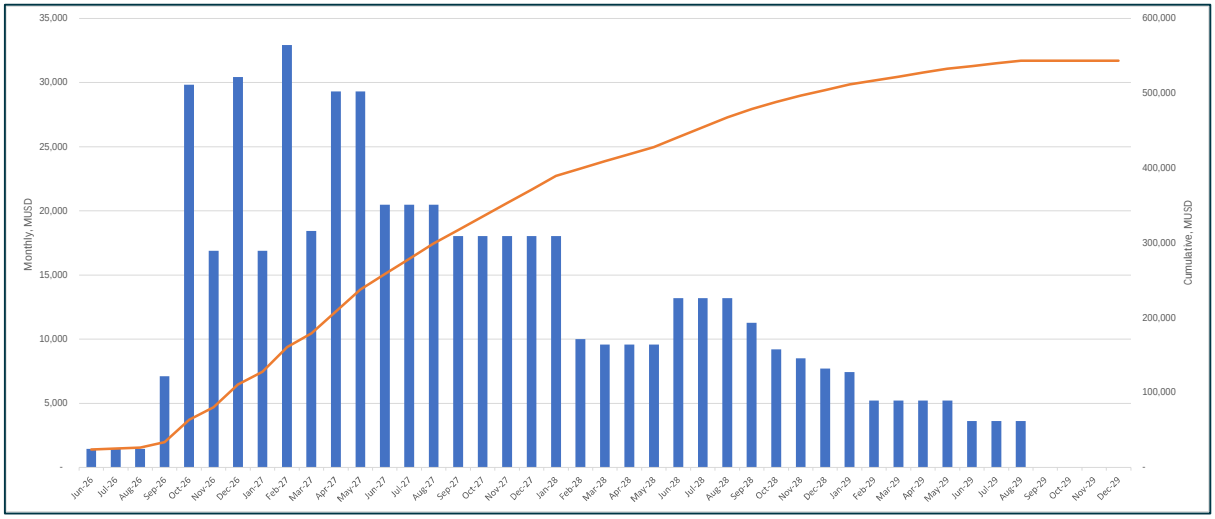
The RIGI milestone places an external time constraint on the project to capture significant economic benefits as described in Chapter 22. The application window for RIGI closes in July, 2026. The PEP is based on achieving key milestone dates as outlined in Table 24-1.

Table 24-1: Key Milestones

Milestone	Finish
EIA Approval	Q4 2025 (achieved – Salta)
RIGI Application	Q4 2025 (achieved)
RIGI Approval	H1 2026 (achieved)
Complete DFS (Design, Estimate and Schedule)	H1 2026 (achieved)
Investment Decision	H2 2026
Mechanical Completion	H1 2029
First Silver	H2 2029

24.3 RIGI Window

As described in Chapter 22, RIGI requires that 80 MUSD of initial CAPEX is invested within 24 months of approval. Based on a RIGI approval in H1 2026, this investment must be completed by H1 2028. The construction cashflow estimate of the PFS update forecasts 80 MUSD of cash commitments will be achieved within six months of the investment decision. This allows nearly 18 months of schedule contingency to comply with the RIGI minimum investment requirements. The cash commitment schedule is shown in Figure 24-1.



Source: AbraSilver Resource Corp., 2024

Figure 24-1: Cash Commitment Schedule

24.4 Construction Approach

Project implementation will start after investment decision. It is estimated that the project will take 36 months to complete. The construction support facilities engineering will already be finalized as part of the early works activities (i.e. exploration camp relocation, bulk earthworks for platforms and camp construction, service hub, etc.) allowing site work to start immediately.

While early works activities progress, the remaining detailed engineering will be finalized, so key facilities can be constructed using infrastructure already in place.

During the DFS phase, key procurement packages will be defined, suppliers and contractors identified. Special care will be taken to comply with the local regulations of Catamarca and Salta provinces. Long lead items will also be identified during that stage.

24.5 Project Type

The Project is considered a greenfield site with no infrastructure and service connections available.

Engineering and Site Supervision will be awarded to a single engineering contractor, ensuring responsibility in terms of quality. The main engineering contractor will be responsible for project technical quality and the plant production capacity post ramp-up.

The Construction Management activities will be either managed by the owner's team or by another contractor and will be responsible for timely and cost-efficient delivery.

24.5.1 Logistics and Sequencing

As part of the mine development, JAC open pit overburden removal will provide material for early work platforms and road backfill. Oculto cover removal may also be used for TSF impoundment construction. As JAC open pit overburden is largely colluvial material, this does not require blasting. Local bulk earthworks contractors can thus readily remove this material. The haul road tasks will be scheduled so the owner crew can assist with construction.

Process facility construction will commence with crushing/screening area before advancing to the covered stockpile and grinding areas. Steel buildings will be equipped with overhead power cranes, so mechanical equipment and piping installation can be completed within an enclosed area, protected from the harsh climate.

The TSF will be constructed early to ensure completion of the first phase prior to process plant start up. Haul roads and waste dumps will be constructed as needed during operation. Water containment embankments and other related earthworks will be completed upfront.

The mining fleet will be purchased early, once the truck shop is available for assembly, allowing for pre-stripping activities to support local contractor activities.

24.6 Permitting

The project requires the approval of certain permits as detailed in Section 20. The Environmental Impact Assessment was submitted to approval agencies in September 2024 and is expected to be approved within 12-15 months. This is a key permit required to start construction. Following initial construction approvals, further construction and operations permits will be required prior to the commencement of operations.

24.7 Project Risk Review

A risk review process was undertaken as part of the PFS. AbraSilver is incorporating action plans to mitigate these risks (and others) as part of the upcoming DFS.

Key environmental and social risks were discussed in Section 20. Other key technical risks and uncertainties were also identified and will be addressed in the next phase of the project. These are shown in the Table 24-2 (list is only an excerpt).

Table 24-2: Risk Matrix Excerpt

Risk	Impact	De-Risking Recommendations
Project delay resulting in failure to meet RIGI benefits	Fail to achieve RIGI benefits included in the economic model. Significant impact to project economics (approx. 30% of NPV). Project more difficult to finance.	Project schedule includes 18 months of contingency for meeting the minimum RIGI spending requirement.
Geotechnical risks	1. Pit slopes are dependent on detailed geotechnical analysis. 2. Facilities foundations for heaviest	Conduct geotechnical site investigation program prior to beginning the DFS.

Risk	Impact	De-Risking Recommendations
	equipment and TSF impoundment must be characterized in more detail. 3. Soil suitability for TSF constructions have to be further test	
Topographical risks	1. Mining and bulk earthworks volumes costs are dependent on correct topography. 2. The use of different coordinate systems are required by authorities and those commonly used as industry practice (Gauss-Kruger vs. UTM).	Final check of topography to be performed and secure that adequate coordinate systems are used for deliverables to be submitted to government agencies and for site use.
Seismic Classification of Facilities	Current design criteria considered in line with official regulations and standards for buildings, but TSF design criteria adopted was conservative for the preliminary dam break study in the absence of site-specific seismic studies.	Conduct site specific seismic studies to improve TSF design parameters and perform a more detailed dam break study under the new design.
Waste Rock Characterization and Treatment	Acid Generating potential and heavy metal leaching characterization performed, but waste facility design for ARD material enclosure has not been sufficiently designed (i.e. lining, treatment, co disposal, etc.).	Complete long-term waste dumps design to complete regulatory compliance framework and to ensure efficient waste management design and operation on site.
Technical professionals' scarcity	Most of engineering discipline professionals are not available or their knowledge does not conform to required standards.	Define detailed staffing plan for early contracting of technical owners' team.
Construction /Execution Risk Availability of Local Contractors	Supplier vetting and development may be required to mitigate execution related risk.	Further outline the procurement packages to identify suitable contractors and vendors including technical and regulatory compliance assessment, through formal vetting.
Site Closure Monitoring	Site monitoring and obligations post closure - unclear legislative requirements or guidelines - long term obligations.	Perform full EIA for baseline monitoring data to be captured. Engage formally with regulators utilizing prepared closure planning.

24.8 Project Opportunities Review

The key opportunities were assessed and will be evaluated as part of the upcoming DFS. These are summarised in Table 24-3.

Table 24-3: Opportunities Matrix Excerpt

Opportunity	Potential Benefit	Action Plan to Capture
Mine plan optimization	Improved open pit design and phases to improve grade profile and reduce waste movement.	Ongoing optimization work with key consultants such as Whittle Consulting.
Power connection to the national grid	Eliminate the CAPEX of the on-site generation facility and reduce the cost of electricity by 50%.	Utilities operating in the region have announced intentions to study and connect the Puna district to the national grid via a high voltage power line. AbraSilver will engage these utilities in

Opportunity	Potential Benefit	Action Plan to Capture
		negotiations for a long term supply contract.
Increase water availability to the project	Additional potential water sources have been identified. Exploiting these would remove the 9ktpd limit on plant throughput.	Continue water exploration activities. Evaluate economic trade-off results of increased plant throughput and re-evaluate plant size.
Improve TSF design	Alternative tailings disposal solutions may reduce cost and environmental footprint.	Trade-offs of alternative TSF designs will be evaluated prior to commencement of the DFS.
Improve waste rock dump design	Reduce waste haulage costs and environmental footprint.	Evaluate co-disposal of waste rock with tailings as part of ongoing trade-off study work with TSF design.
Improve project sustainability metric	Greenhouse gas reduction, water conservation, improved overall environmental footprint of the project	Benchmark project sustainability score to peers prior to commencing the DFS. Sustainability targets to be incorporated into the DFS project charter.

24.8.1 Potential Leachable Material

Preliminary analysis based on historical sampling and metallurgical testing indicate the potential to treat the lower grade Diablillos oxide mineralisation by heap leaching. The Mineral Resource estimate includes 31Mt grading 0.16 g/t Au and 13 g/t Ag, which is below the economic cutoff grade for tank leaching, but potentially economically recoverable with a dedicated heap leaching circuit. The potentially heap leachable mineralisation would be crushed, stacked on a heap leach pad and irrigated with sodium cyanide. Gold and silver would be stripped from the pregnant solution in a separate processing plant, resulting in increased metal production. Although detailed cost estimates have not yet been completed, it is estimated that the CAPEX would be incremental to the existing 9 ktpd tank leaching circuit.

A further benefit of heap leaching marginal mineralisation would be a reduction in the size of the waste rock facilities as some of the waste material in the tank leaching mine plan will be now recoverable by heap leaching.

A PEA for the heap leaching option will be evaluated and presented in parallel to the DFS results in H1 2026.

24.8.2 Mine Plan Optimization

The mine plan has numerous opportunities for further optimisation. Some of the options that will be considered are:

- Improved stripping ratios (following further geotechnical analysis)
- Pushback selection and timing
- Ramp and pit exit optimisation

- Haulage optimization
- Waste dump optimization (including location)
- Blend optimization including “floating” cut-off grade analysis
- Process optimization

24.8.3 Power Connection to the National Grid

Electricity for the project is currently planned to be supplied from an on-site self-generation power facility comprising a combined photovoltaic-diesel plant. The capital cost of this facility is 33 MUSD and the operating cost is calculated a US $0.19/kWh of generation. AbraSilver is currently in discussions with local power generation and distribution companies to supply electricity to Diablillos via a planned high voltage powerline to the Puna district. Power supply from a grid connection would eliminate the CAPEX of the on-site self-generation facility and reduce operating costs significantly. Based on market surveys performed by AbraSilver, long-term power prices can be contracted for $0.07-$0.10/kWh. Electricity supplied at market comparative power prices would reduce OPEX by 15 – 20 MUSD/year.

24.8.4 Increase Water Availability

Throughput for the process plant was capped at 9 ktpd due to the water limitation of the Baranquillas borefield. Additional potential sources of water supply in the region have been identified. Further exploration activities are planned during the DFS to confirm the volume available. If water supply is available, the throughput rate of the process plant will be re-evaluated in trade-off studies as part of the DFS. Additional water supply may also support requirements for a heap leaching circuit as described in 24.8.1.

Increased water recovery from tailings and water efficiency measures may further increase available water.

24.8.5 TSF and Waste Rock Dump Designs

Opportunities to improve the design of the TSF will be evaluated during the DFS. The scope of the improvements encompasses constructability, total cost of ownership (encompassing CAPEX and OPEX trade-offs), operability and safety.

The TSF design review will investigate co-disposal of tailings with waste rock. This would potentially reduce or eliminate the need for separate facilities by constructing the dam wall with the waste rock. The resulting benefits would include lower waste rock disposal costs and a reduced environmental footprint.

24.8.6 Sustainability

During the DFS, opportunities to further improve the sustainability of the Diablillos project will be evaluated. An exercise is planned to evaluate the project’s sustainability. This will include a benchmark against peers, and the evaluation of sustainability goals. Specific opportunities

include (but are not limited to) a reduction of environmental footprint, water consumption and emissions.

25. Interpretations and Conclusions

The Diablillos Project is an advanced-stage silver-gold project located in Salta and Catamarca Provinces, Argentina. The Project hosts a large, near-surface, high-sulphidation epithermal silver-gold system with strong supergene overprinting. The mineralization is hosted principally within the Oculto, JAC, Fantasma, Laderas and Sombra deposits, which together form the basis of the current Mineral Resource Estimate ("MRE").

Based on the site visit and subsequent evaluation, the "QP" Mr. Luis Rodrigo Peralta offers the following conclusions:

- The Mineral Resources conforms to CIM (2014) definitions and complies with all disclosure requirements for Mineral Resources set out in NI 43-101. The gold and silver grade estimation process was consistent with CIM Mineral Resource and Mineral Reserve estimation best practice guidelines.
- The updated MRE incorporates additional drilling completed since the July 2025 Mineral Resource estimate, including Phase V drilling completed up to December 31, 2025. The current drilling database used for the Diablillos MRE comprises 790 drill holes, including both diamond drilling and reverse circulation drilling, for a total of 170,368 m. The QP considers that the drilling database, sampling protocols, analytical procedures and QA/QC results are adequate to support the current Mineral Resource estimate.
- The current MRE incorporates a revised geological, alteration and oxidation model for the Oculto, JAC, Fantasma, Sombra and Laderas zones. The geological interpretation reflects the current understanding that mineralization is controlled by a combination of lithology, alteration, oxidation state and structural features. The principal mineralizing structures are interpreted as steeply dipping feeder zones, with lateral migration of mineralizing fluids along favourable permeable horizons, particularly near the contact between volcanic rocks and the underlying basement. This interpretation supports both steeply dipping and shallowly dipping mineralized controls within the model.
- Gold and silver grades were estimated into a three-dimensional block model using Ordinary Kriging ("OK"), with Inverse Distance squared ("ID2") estimates used as check models to assess potential bias. Drill hole intervals were composited to 1 m length, consistent with the most common sample interval. Grade capping was applied on a domain-by-domain basis to reduce the influence of high-grade outliers. Bulk density values were assigned based on the geological, alteration, and oxidation domains, supported by a large database of specific gravity measurements.
- The Mineral Resource has been classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves. Classification was based on drill hole spacing, geological continuity, grade continuity, estimation confidence, data quality, and model validation results.
- The updated MRE demonstrates a significant increase in the scale of the Diablillos Project. Total Measured and Indicated Mineral Resources, combining tank leach and heap leach material, are estimated at 231 Mt grading 33 g/t Ag and 0.34 g/t Au, containing approximately 248 Moz Ag and 2.5 Moz Au, equivalent to 454 Moz AgEq. Inferred Mineral Resources total 49.347 Mt grading 12 g/t Ag and 0.26 g/t Au, containing approximately 18 Moz Ag and 0.4 Moz Au, equivalent to 50 Moz AgEq.

- The updated tank leach Mineral Resource, which represents the higher-grade portion of the deposit and the primary processing route, totals 102 Mt in the Measured and Indicated categories grading 65 g/t Ag and 0.62 g/t Au, containing approximately 212 Moz Ag and 2.0 Moz Au, equivalent to 389 Moz AgEq. The Inferred tank leach Mineral Resource totals 14 Mt grading 25 g/t Ag and 0.57 g/t Au, containing approximately 11 Moz Ag and 0.2 Moz Au, equivalent to 34 Moz AgEq.
- The updated heap leach Mineral Resource represents a significant addition to the Project. Measured and Indicated heap leach Mineral Resources total 129 Mt grading 8 g/t Ag and 0.12 g/t Au, containing approximately 35 Moz Ag and 0.5 Moz Au, equivalent to 64 Moz AgEq. Inferred heap leach Mineral Resources total 34 Mt grading 6 g/t Ag and 0.14 g/t Au, containing approximately 6 Moz Ag and 0.1 Moz Au, equivalent to 16 Moz AgEq.
- Compared with the July 2025 Mineral Resource estimate, the current MRE represents a material increase in contained metal. Total Measured and Indicated contained silver increased by approximately 25%, contained gold increased by approximately 48%, and contained silver-equivalent ounces increased by approximately 30%. This growth reflects the success of recent exploration and definition drilling, an improved geological model, updated economic assumptions, and the inclusion of a significantly expanded heap leach component.
- The most important contribution to the increase in the tank leach resource comes from the Oculto deposit. At Oculto, Measured and Indicated tank leach tonnage increased materially, with contained silver and contained gold also increasing. This increase was driven primarily by infill drilling that connected gaps in the previous resource model and step-out drilling that expanded the mineralized orebody. The Oculto deposit remains the principal driver of the Project's Mineral Resource base and continues to show potential for further growth, particularly toward Oculto East and along the broader Oculto mineralized corridor.
- The JAC deposit continues to represent a high-grade silver-dominant component of the Project. The updated MRE demonstrates additional growth in tonnage and contained silver relative to the previous estimate. Recent drilling has improved confidence in the continuity of mineralization and has contributed to the expansion of the mineralized orebody. The area between JAC and Oculto remains prospective for additional mineralization and should continue to be evaluated through further drilling.
- The Fantasma, Laderas and Sombra satellite deposits also contribute to the overall Mineral Resource. Although smaller than Oculto and JAC, these zones demonstrate that the Diablillos system extends beyond the principal deposits and retains potential for further resource growth. Sombra is of particular interest due to its location and its potential to extend toward the southeast, possibly linking with the broader Oculto mineralized system.
- The inclusion of a dual-processing approach is one of the most significant outcomes of the updated MRE. The tank leach process remains the primary processing route for higher-grade material, while heap leaching provides a potential secondary processing route for lower-grade oxide mineralization. The heap leach Mineral Resource represents a large volume of lower-grade mineralization that may provide development flexibility, improve the overall utilization of the mineralized orebody, and potentially convert material previously considered waste into payable material within the conceptual open pit.
- The Mineral Resource was constrained within a conceptual open pit shell using a Net Value per Block ("NVB") approach. The tank leach material was reported using an approximate average cut-off grade of 39 g/t AgEq, while the heap leach material was reported using an approximate average cut-off grade of 15 g/t AgEq. The MRE assumes long-term metal prices of US$34.50/oz Ag and US$3,200/oz Au. Metallurgical recoveries used in the MRE are based on available metallurgical test work and geo-metallurgical modeling.

- Mr. Peralta considers that the Mineral Resource has reasonable prospects for eventual economic extraction by open pit mining methods and conventional precious metal processing methods, including tank leaching and potential heap leaching. Mineral Resources are not Mineral Reserves and have not demonstrated economic viability. Mining recovery and dilution factors have not been applied to the Mineral Resource estimate.
- No material environmental, permitting, legal, title, taxation, socio-political, marketing or other relevant issues are known to the QP that could materially affect the potential development of the Mineral Resource. However, as with all advanced-stage mineral projects, the Mineral Resource remains subject to risks related to metal prices, metallurgical performance, mining costs, processing costs, geotechnical assumptions, permitting, environmental requirements and future engineering studies.
- The updated MRE confirms Diablillos Project as a large-scale silver-gold project with substantial Mineral Resources in the Measured and Indicated categories. The significant increase in contained metal relative to the July 2025 MRE demonstrates the effectiveness of recent exploration and resource definition drilling. The addition of a large heap leach Mineral Resource provides an important opportunity to enhance the Project development scenario and should be further assessed through mine planning, metallurgical testwork and economic analysis.
- Mr. Peralta concludes that the Diablillos Project continues to warrant advancement through ongoing Definitive Feasibility Study work, together with additional exploration and resource definition drilling. Particular focus should be placed on Oculto East, the Oculto-JAC connection, Sombra, and other satellite targets where further drilling may support additional Mineral Resource growth. The ongoing Phase VI drilling program is expected to provide further information for future Mineral Resource updates.

26. Recommendations

26.1 Mineral Resource and Exploration

The Qualified Person recommends that AbraSilver continue advancing the Diablillos Project through a combination of engineering studies, metallurgical investigations, resource definition drilling and exploration drilling. The results of the current Mineral Resource Estimate confirm that Diablillos remains a rapidly growing silver-gold project with substantial opportunities for additional Mineral Resource expansion and future Mineral Reserve growth.

The updated Mineral Resource Estimate demonstrates a significant increase in contained silver, gold and silver-equivalent ounces compared to the previous estimate. The increase is principally the result of successful exploration and definition drilling completed during 2025, improvements to the geological interpretation and the incorporation of a dual-processing strategy that includes both tank leaching and heap leaching. These results validate the current exploration model and support continued investment in both resource growth and project development activities.

The highest priority exploration target remains the Oculto East area, where recent drilling has demonstrated that gold-dominant mineralization extends beyond the limits of the current conceptual open pit shell. Additional drilling in this area is expected to further expand the Mineral Resource base and improve the understanding of the geological controls on mineralization.

The corridor between the Oculto and JAC deposits should also continue to be a priority exploration focus. Recent drilling has demonstrated the potential for additional mineralized zones between the two deposits and has highlighted the possibility of further resource growth along the broader mineralized trend. Continued drilling in this area may support future integration of mineralized zones currently modelled separately.

The Sombra deposit also warrants additional exploration and resource definition drilling. Mineralization remains open in several directions and geological evidence suggests potential continuity toward the northeast, where the mineralized corridor may ultimately connect with the broader Oculto system. Additional drilling in this area may contribute both to resource growth and improved confidence in the existing Mineral Resource.

Infill drilling is recommended in selected portions of the current Mineral Resource where drill spacing remains sufficient only for Inferred classification. The objective of this drilling program should be to improve geological confidence and support future conversion of Inferred Mineral Resources into the Indicated category, thereby increasing the potential for future Mineral Reserve growth and improving mine planning flexibility.

The introduction of a heap leach Mineral Resource represents one of the most significant outcomes of the current Mineral Resource update. The heap leach mineral resource contains a substantial volume of lower-grade mineralization that may provide an opportunity to enhance the overall economics of the Project by converting material that would otherwise be considered waste

into potentially economic mineralized material. Consequently, Mr. Peralta recommends that additional engineering studies and metallurgical investigations be completed to further evaluate the technical and economic viability of heap leach processing.

Additional metallurgical testwork should focus on variability testing across the principal lithological, alteration and oxidation domains represented within the heap leach Mineral Resource. These studies should seek to further refine recovery assumptions, reagent consumption estimates, leach kinetics and operating cost parameters, thereby reducing uncertainty associated with future economic evaluations.

Mr. Peralta also recommends that the geological model, geometallurgical model and Mineral Resource estimate continue to be updated as new drilling information becomes available. The ongoing exploration activities and future drilling campaigns are expected to contribute to further growth of the Mineral Resource inventory and may support additional improvements in resource classification and project economics.

Based on the results obtained to date, the QP considers that the Diablillos Project remains highly prospective and fully justifies continued investment in exploration, engineering and metallurgical studies. The combination of a large and growing Mineral Resource base, strong exploration upside, and the potential development of both tank leach and heap leach processing routes provides a compelling basis for continued advancement of the Project.

26.2 Recommended Work Program and Budget

The recommended work program is designed to support continued Project advancement, further evaluate the heap leach opportunity, increase confidence in the existing Mineral Resource and identify opportunities for future resource growth.

The estimated cost of the recommended work program is summarized in Table 26-1.

Table 26-1: Recommended Work Program and Estimated Budget

Recommended Activity	Estimated Cost (US$M)
Additional Engineering Studies for Heap Leaching	3
Infill Drilling (15,000 m @ US$500/m)	7.5
Additional Step-Out Drilling (10,000–15,000 m @ US$500/m)	5.0 - 7.5
Additional Metallurgical test work for Heap Leaching	2.0 – 3.0
Total Estimated Budget	17.5 – 21.0

Mr. Peralta considers that the proposed expenditure of approximately US$17.5 million to US$21.0 million is justified by the scale of the current Mineral Resource, the demonstrated success of recent drilling campaigns, the significant potential for additional resource growth, and the

opportunity to further enhance the project value through the evaluation and optimization of heap leach processing.

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Law No. 25,688: Environmental Water Management

Law No. 25,743: Cultural and Archaeological Heritage

Law No. 25,831: Free Access to Public Environmental Information Provincial Constitution, Articles No. 30, 79, 81, 82

Provincial Law No. 7,017: Water Code of Salta Province

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Provincial Law No 8164: Royalties. Local Supplier Registration of Salta Province

Provincial Resolution 84/22: Registry of mining suppliers

Provincial Resolution 087/2018 Social and economic impact tables

Provincial Resolution 3652/10 Annual analysis of drinking water.

Provincial Resolution 197/2023 Provision of housing modules for control and inspection authorities of Mining Secretary of Salta Province.

Provincial Law No. 4218 Preservation of vestiges and/or remains of anthropological archaeological sites existing in the province of Catamarca.

SAyA Resolution No. 065/05 Regulations and conditions for the discharge of liquid waste in Catamarca.

Resolution S.E.M. No. 146/05 The regulations are approved that determine the procedure and conditions for granting information regarding environmental matters held by this Secretary of State for Mining.

Resolution S.E.M. N° 65/07 Establishes that all mining companies active in the Province that carry out water, soil and air monitoring in Catamarca.

Ley Provincial Law N° 4865/95: Adheres to National Law No. 24051 and establishes the Hazardous Waste Regime. By request of this law, companies must register as Generators of Hazardous Waste in the Provincial Registry of Generators, Operators and Transporters of Hazardous Waste, through Decree 473/01

Resolution 065/05: Discharge of effluents. Establishes the conditions for overturning the discharge of Residual Liquids required for discharge to the Receiving Body. This resolution establishes that all discharge of waste liquids must be declared, and the Enforcement Authority will grant the Exception, or not, of having a conditional dumping permit, depending on the volume of effluents generated.

Law 2257/73: Establishes the water code of the province of Catamarca. Its regulations are outdated, and its application is articulated through provisions of the Secretariat of Water Resources of the Province of Catamarca.

Ley 5682/2020: Establishes the Mining Procedures Code

Disposition 74: Environmental Impact in Mining. Establishes the Presentation of the Environmental and Social Impact Study for any activity that may be susceptible to degrading the Environment. In Catamarca, the Ministry of Mining is the Authority that governs the control of the ESIAs linked to the Mining Activity, through the Provincial Directorate of Mining Environmental Management.

Ley N 27.742: Regimen de Incentivo para Grandes Inversiones (RIGI). Decreto-2024-749-APN-PTE. Titulo VII.

Date and Signature Page

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All data used as source material and text, tables, figures, and attachments of this document have been reviewed and prepared in accordance with generally accepted professional engineering and environmental practices.