Filo del Sol Project

NI 43-101 Technical Report, Updated Prefeasibility Study

Argentina and Chile

Effective Date: February 28, 2023

Prepared for: Filo Mining Corp.

Suite 2000 – 885 West Georgia Street
Vancouver, BC, Canada V6C 3E8

Prepared by: Ausenco Engineering Canada Inc.

1050 West Pender, Suite 1200
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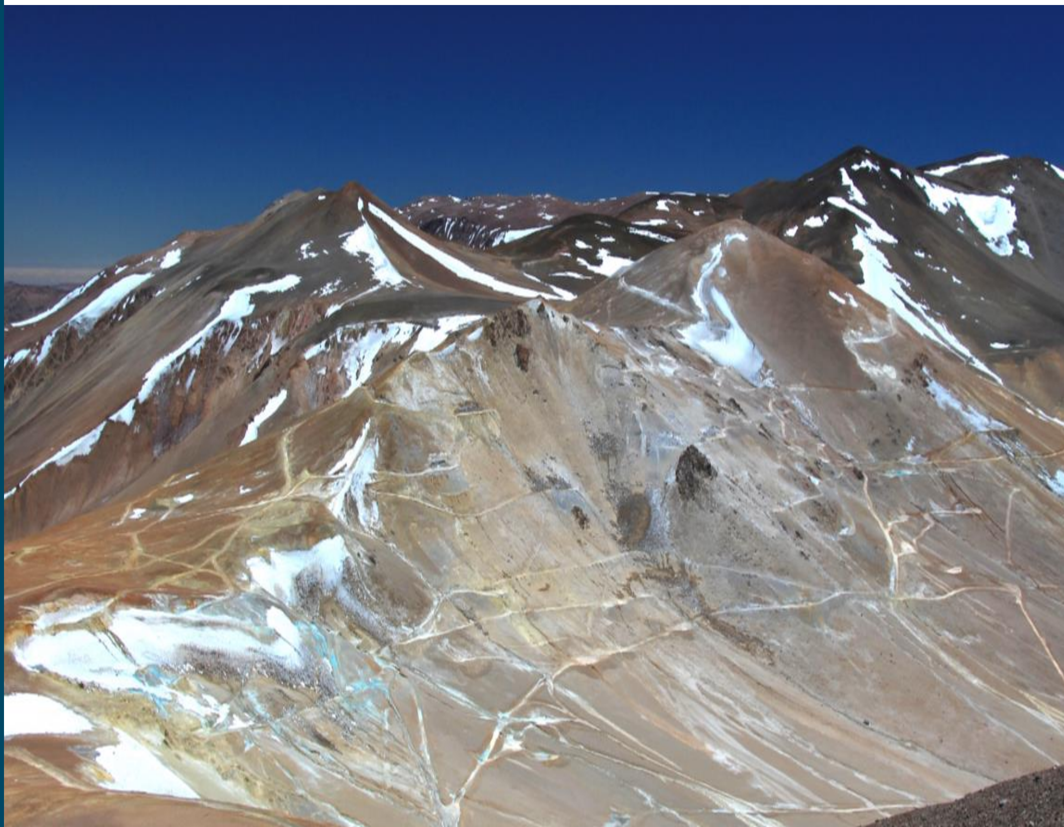
List of Qualified Persons: Scott C. Elfen, P.Eng., Ausenco Engineering Canada Inc. • Kevin Murray, P. Eng., Ausenco Engineering Canada Inc. • Bruno Borntraeger, P.Eng., Knight Piésold • Fionnuala A.M. Devine, P.Geo., Merlin Geosciences Inc. • Neil M. Winkelmann, FAusIMM, SRK Consulting (Canada) Inc. • James N. Gray, P.Geo., Advantage Geoservices Limited • Ryan P. Brown, P.Eng., AGP Mining Consultants • Gordon R. Zurowski, P.Eng., AGP Mining Consultants



CERTIFICATE OF QUALIFIED PERSON

Scott C. Elfen, P.E.

I, Scott C. Elfen, P. E., certify that I am a professional Engineer and employed as VP Global Lead for Geotechnical and Civil with Ausenco Engineering Canada Inc. ("Ausenco") with an office address of 1050 West Pender Street, Suite 1200, Vancouver, BC, V6E 3S7 Canada.

1. This certificate applies to the technical report titled, "Filo del Sol Project NI 43-101 Technical Report, Updated Prefeasibility Study" (the "Technical Report"), prepared for Filo Mining Corp. (the "Company"), with an effective date of February 28, 2023 (the "Effective Date").
2. I graduated from the University of California, Davis with a Bachelor of Science degree in Civil Engineering (Geotechnical) in 1991.
3. I am a Registered Civil Engineer in the State of California (No. C56527) by exam since 1996 and am also a member in good standing of American Society of Civil Engineers (ASCE) Society for Mining, Metallurgy & Exploration (SME) that are all in good standing.
4. I have practiced my profession continuously for 28 years, with experience in the development, design, construction, and operations of mine waste storage facilities, such as waste rock storage facilities and tailings storage facilities, and heap leach facilities focusing on precious and base metals, both domestic and international. In addition, I have developed geotechnical and civil design parameters for pit slope design, plant foundation design, heap leach facilities, and other supporting infrastructure. Examples of projects I have worked on: Barrick Gold's Pierina Gold Mine (Peru) detail design of phases 1 through 7 heap leach pad, Filo Mining's Filo Copper-Gold-Silver Project PFS on-off and static leach pads, Project BHP's Escondida Copper Mine (Chile) detail design of the sulfide leach pads, and Charaat's Tulkubash Gold Project FS design of leach pad.
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 fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101 in connection with those sections of the Technical Report that I am responsible for preparing.
6. I visited the Project on January 14, 2023 for one day.
7. I am a co-author of the Technical Report, responsible for sections 1.16, 18.2, 18.8, 18.9, 18.10, 25.1.10, and 27 of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. As a qualified person, I am independent of the Company as defined in Section 1.5 of NI 43-101.
9. I have been involved with the Filo del Sol Project through my involvement in the previous technical report titled "Filo del Sol NI 43-101 Technical Report, Prefeasibility Study, February 22, 2019," and I was responsible for sections 1.16, 18.2, 18.8, 18.9 and 18.10 of such report.
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 that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated March 17, 2023

"Signed and sealed"

Scott C. Elfen, P.E.

Ausenco

CERTIFICATE OF QUALIFIED PERSON

Kevin Murray, P.Eng.

I, Kevin Murray, P. Eng., do hereby certify that I am a Professional Engineer, currently employed as a Manager Process Engineering with Ausenco Engineering Canada Inc. ("Ausenco"), with an office address of 1050 West Pender Street, Suite 1200, Vancouver, BC, V6E 3S7 Canada

1. This certificate applies to the technical report titled, "Filo del Sol Project, NI 43-101 Technical Report, Updated Prefeasibility Study" (the "Technical Report"), prepared for Filo Mining. (the "Company"), with an effective date of February 28, 2023 (the "Effective Date").
2. I graduated from the University of New Brunswick, Fredericton NB, in 1995 with a Bachelor of Science in Chemical Engineering.
3. I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia, Registration number# 32350 and the Northwest Territories Association of Professional Engineers and Geoscientists' Registration# L4940.
4. I have practiced my profession continuously for 22 years. I have been directly involved in all levels of engineering studies from preliminary economic assessments (PEAs) to feasibility studies including being a Qualified Person for flotation projects such as Ero Copper Corp.'s Boa Esperança Feasibility Study and NorZinc Ltd.'s Prairie Creek PEA. I have been directly involved with test work and flowsheet development from preliminary testing through to detailed design and construction including my direct experience at Red Lake Gold Mine, Porcupine Gold Mine located in Ontario and Éléonore Gold Mine, located in Quebec, while working for Goldcorp Inc./Newmont Corporation.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 - Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101 in connection with those sections of the Technical Report that I am responsible for preparing.
6. I have not visited the Filo del Sol Project.
7. I am responsible for Sections 1.1, 1.11, 1.15, 1.16, 1.19, 1.20, 1.23, 2, 13, 17, 18.1, 18.3, 18.4, 18.5, 18.6, 18.7, 21 excluding 21.4.3, 25.1.9, 25.1.10, 25.1.13, 25.2.2, 25.2.5, 25.3.3, 25.3.4, 25.3.5, 25.3.7, and 27 of the Technical Report.
8. I am independent of the Company as independence is described by Section 1.5 of NI 43101 and in Section 1.5 of the Companion Policy to NI 43-101.
9. I have had no previous involvement with the Filo del Sol Project.
10. I have read NI 43-101, Form 43-101F1 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101 and Form 43-101F1. 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 that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: March 17, 2023

"*Signed and Sealed*"

Kevin Murray, P. Eng.

CERTIFICATE OF QUALIFIED PERSON
Fionnuala A. M. Devine, P.Geo.

I, Fionnuala A. M. Devine, P.Geo., certify that I am employed as a geologist with Merlin Geosciences Inc. ("Merlin"), with an office address of 2946 Mabel Lake Road, Enderby, BC, Canada.

1. This certificate applies to the technical report titled "Filo del Sol Project NI 43-101 Technical Report, Updated Prefeasibility Study," (the "Technical Report"), prepared for Filo Mining. (the "Company"), with an effective date of February 28, 2023 (the "Effective Date").
2. I graduated from The University of British Columbia with a Bachelor of Science degree in 2002 and completed a Master of Science degree from Carleton University in 2005.
3. I am a Professional Geoscientist registered with Engineers & Geoscientists British Columbia, membership number 40876.
4. I have practiced my profession for 18 years. I have been directly involved in mineral exploration for base and precious metals in a variety of deposit types in North and South America during that time. I was QP for the 2018 prefeasibility and 2020 feasibility studies on the Josemaria porphyry Cu-Au project now owned by Lundin Mining.
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 fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101 in connection with those sections of the Technical Report that I am responsible for preparing.
6. My most recent visit was to the Filo del Sol project core logging facility in Rodeo, Argentina between October 9 – 11, 2022 for a visit duration of 3 days.
7. I am responsible for sections 1.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 1.9, 1.10,1.22,1.23,1.24,1.25, 3.1, 4, 5, 6, 7, 8, 9, 10, 11, 12, 23, 24, 26, 25.1.1, 25.1.2, 25.1.3, 25.1.4, 25.1.5, 25.1.6, and 27 of the Technical Report.
8. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
9. I have been involved with the Filo del Sol project since 2014. I first visited the project site in January 2014, then the following year from January 15th to February 21st, 2015 worked on geological mapping for a one month period at the project site and core logging/sampling facility in Copiapó, Chile. I visited again from February 6th to 24th, 2016 to continue with detailed geological mapping in the Filo area, including a focus on the Tamberías area. In February, 2017 I visited the site again for 3 days of geological review, revision mapping and sample collecting. In 2018 and 2019 I made two trips (April 18-21, 2018 and September 5 - 10, 2019) to the core logging facility in Copiapó, Chile to review recent core and participate in core logging review and cross section development. I visited the project site again from December 3 – 6, 2019 for a review of available core and updates from the geology team. My most recent trip was from October 9-11, 2022 to the core logging facility in Rodeo, Argentina to review the past three years of drilling and an update to the geological model for the project.

   During this period I have co-authored several technical reports on the project:
   - Devine, Charchaflié and Grey, 2015
   - Devine, Charchaflié and Grey, 2016
   - Devine, Charchaflié, DiPrisco, Grey, 2017
   - Devine et al., 2017
   - Devine et al., 2019

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 that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: March 17, 2023

"Signed and sealed"

Fionnuala A. M. Devine, P.Geo.

CERTIFICATE OF QUALIFIED PERSON
Neil M. Winkelmann, FAusIMM

I, Neil M. Winkelmann, FAusIMM, do hereby certify that I am a Principal Consultant with SRK Consulting (Canada) Inc. ("SRK"), with an office at 320 Granville St #2600, Vancouver, BC V6C 1S9, Canada.

1. This certificate applies to the technical report titled "Filo del Sol Project NI 43-101 Technical Report, Updated Prefeasibility Study," (the "Technical Report"), prepared for Filo Mining. (the "Company"), with an effective date of February 28, 2023 (the "Effective Date").

2. I am a graduate of the University of New South Wales, Australia with a Bachelor of Engineering in Mining, in 1984. I graduated from the University of Oxford with an MBA in 2005.

3. I am registered as a Fellow of The Australasian Institute of Mining and Metallurgy (AusIMM, #323673).

4. I have practiced my profession continuously since 1984 and I have 39 years of experience in mining. I have significant experience in the valuation of minerals-industry projects accrued over the past 18 years. I have done financial modelling on projects for 18 years for producing mines and projects in development and contributed to the financial model on numerous technical reports for mining companies on precious and base metal projects throughout the Americas during this time. My most recent financial analysis includes work on the previous Pre-Feasibility Study for the Filo Del Sol Project as well as having worked on technical reports including the Josemaria Project NI43-101 in 2020.

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 fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.

6. I visited the property on 21st and 22nd of February 2017 for a period of 2 days.

7. I am a co-author of the Technical Report, responsible for Section 1.17, 1.21, 3.3, 3.4, 19, 22, 25.1.11, 25.1.14, 25.2.4, 25.3.2, and 27 of the Technical Report and I accept professional responsibility for those sections of the Technical Report.

8. As a qualified person, I am independent of the Company as defined in Section 1.5 of NI 43-101.

9. I have had prior involvement with the subject property through my involvement in the previous technical report titled Filo del Sol NI43-101 Technical report, Pre-feasibility Study, February 22, 2019.

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 that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated this March 17, 2023 in Vancouver, B.C., Canada.

"Signed and sealed"

Neil M. Winkelmann, FAusIMM

CERTIFICATE OF QUALIFIED PERSON

James N. Gray, P.Geo.

I, James N. Gray, P.Geo., certify that I am employed as President with Advantage Geoservices Limited (Advantage Geoservices), with an office address of 46717 Sylvan Drive, Chilliwack, BC, Canada.

1. This certificate applies to the technical report titled "Filo del Sol Project NI 43-101 Technical Report, Updated Prefeasibility Study," (the "Technical Report"), prepared for Filo Mining. (the "Company"), with an effective date of February 28, 2023 (the "Effective Date").

2. I graduated from the University of Waterloo in 1985 with a B.Sc. in geology.

3. I am a Professional Geoscientist registered with the Engineers and Geoscientists British Columbia, license # 27022.

4. I have practiced my profession for 37 years. I have been directly involved in resource estimation work at operating mines as well as base and precious metal projects in North and South America, Europe, Asia and Africa. Recently I have been the QP responsible for polymetallic mineral resource estimation at HighGold Mining's Johnson Tract in Alaska, Surge Copper's Ootsa Project in British Columbia as well as the Josemaria Deposit (Lundin Mining Corp., Argentina).

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 fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101 in connection with those sections of the Technical Report that I am responsible for preparing.

6. I have not personally visited the Filo del Sol Project.

7. I am responsible for subsections 1.12, 14, 25.1.7, and 27 of the Technical Report.

8. I am independent of the Company. as independence is defined in Section 1.5 of NI 43-101.

9. I have previous involvement with the Filo del Sol Project, having completed the initial mineral resource estimate which had an effective date of November 25, 2014, and three mineral resource estimate updates with effective dates of August 26, 2015, September 27, 2017 and August 8, 2018, as well as a Pre-feasibility Study dated February 22, 2019.

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 that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: March 17, 2023

"Signed and sealed"

James N. Gray, P. Geo.

CERTIFICATE OF QUALIFIED PERSON

Ryan P. Brown, P.Eng.

I, Ryan P. Brown, P.Eng, certify that I am employed as a Principal Mining Engineer with AGP Mining Consultants Inc., with an office address of 132 Commerce Park Drive, Unit K #246 Barrie, Canada L4N 0Z7.

1. This certificate applies to the technical report titled "Filo del Sol Project NI 43-101 Technical Report, Updated Prefeasibility Study," (the "Technical Report"), prepared for Filo Mining. (the "Company"), with an effective date of February 28, 2023 (the "Effective Date").

2. I graduated from Queens University, Kingston with a Bachelor of Applied Science in Mining Engineering in 2009.

3. I am a Professional Engineer of Engineers and Geoscientists British Columbia (No. 174600).

4. I have practiced my profession continuously for fourteen years. I have been directly involved in mine operations and mine design and evaluation across numerous mines including both open pit and underground copper and gold mines. I have been responsible for mine planning at mines of similar scope to Filo del Sol such as Kinross' Round Mountain and Fort Knox mines and have served as the QP for Mineral Reserves and Mine Planning for the Imperial Metals Mount Polley 2014 updated Technical Report.

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 fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101 in connection with those sections of the Technical Report that I am responsible for preparing.

6. I have not visited the Filo del Sol site.

7. I am responsible for Sections 1.14, 16, 21.4.3, 25.2.3, 25.2.8, 25.3.6, and 27 relating to mining cost estimates for the Technical Report.

8. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.

9. I have had no previous involvement with the Filo del Sol project.

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 that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: March 17, 2023

"Signed and sealed"

Ryan P. Brown, P.Eng.

Knight Piésold CONSULTING

CERTIFICATE OF QUALIFIED PERSON

Bruno Borntraeger, P.Eng.

I, Bruno Borntraeger, P.Eng., certify that I am employed as a Specialist Geotechnical Engineer | Associate with Knight Piésold Ltd (Vancouver), with an office address of. 1400-750 West Pender St., Vancouver, BC Canada.

1. This certificate applies to the technical report titled "Filo del Sol Project NI 43-101 Technical Report, Updated Prefeasibility Study," (the "Technical Report"), prepared for Filo Mining. (the "Company"), with an effective date of February 28, 2023 (the "Effective Date").

2. I graduated from the University of British Columbia in Vancouver, Canada with a Bachelor of Applied Science in Geological Engineering, 1990.

3. I am a member in good standing of the Engineers and Geoscientists of British Columbia (License #20926).

4. I have practiced my profession continuously for 32 years. I have been directly involved in geotechnical engineering, mine waste and water management, heap leaching, environmental compliance, mine development with practical experience in feasibility studies, detailed engineering, permitting, construction, operations and closure. I have most recently been a QP for environmental and permitting aspects for the Josemaria Project PFS. and also the Filo del Sol PFS in 2019.

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 fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101 in connection with those sections of the Technical Report that I am responsible for preparing.

6. I visited the property on March 22, 2018 for one day.

7. I am responsible for sections 1.18, 3.2, 20, 25.1.12, 25.2.1, 25.2.6, and 27 of the Technical Report.

8. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.

9. I have been involved with the Filo del Sol project through my involvement in the previous technical report entitled: "NI 43-101 Technical Report, Pre-feasibility Study for the Filo del Sol Project" prepared for Filo Mining Corp. (the "Issuer") dated February 22, 2019, with an effective date January 13, 2019.

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 that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: March 17 2023

"Signed and sealed"

Bruno Borntraeger, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Gordon R. Zurowski,

I, Gordon R. Zurowski, P.Eng, certify that I am employed as a Principal Mine Engineer with AGP Mining Consultants Inc. with an office address of 132 Commerce Park Drive, Unit K #246 Barrie, Canada L4N 0Z7.

1. This certificate applies to the technical report titled "Filo del Sol Project NI 43-101 Technical Report, Updated Prefeasibility Study," (the "Technical Report"), prepared for Filo Mining. (the "Company"), with an effective date of February 28, 2023 (the "Effective Date").

2. I graduated from the University of Saskatchewan with a Bachelor of Applied Science in Geological Engineering in 1989.

3. I am a Professional Engineer of the Professional Engineers of Ontario (No. 100077750).

4. I have practiced my profession for 30 years and have been directly involved in open pit mining including operating, design and evaluation in Canada and worldwide. I have been the QP for statement of reserves for Equinox Gold's Aurizona Mine PFS and the Mesquite mine updated reserve statement. Most recently I have been the reserves QP for the Magna Mining FS study.

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 fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101 in connection with those sections of the Technical Report that I am responsible for preparing.

6. I visited the Filo del Sol Project site on January 14th, 2023 for a visit duration of 1 day.

7. I am responsible for Sections 1.13, 15, 25.1.8, 25.2.7, 25.3.1, and 27 of the Technical Report.

8. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.

9. I have had no previous involvement with the Filo del Sol Project.

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 that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: March 17, 2023

"Signed and sealed"

Gordon R. Zurowski, P.Eng.

Important Notice

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1 SUMMARY

1.1 Introduction

Filo del Sol is an advanced stage copper-gold-silver exploration project that straddles the border between Argentina and Chile (see Figure 1-1). In October 2022, Filo Mining Corp. (Filo Mining) contracted Ausenco Engineering Canada Inc. (Ausenco), Merlin Geosciences Inc. (Merlin), Advantage Geoservices Ltd. (Advantage Geoservices), AGP Mining Consultants Inc. (AGP), SRK Consulting (SRK), and Knight Piésold Ltd. (KP) to conduct an updated prefeasibility study on the project. The report was prepared by Ausenco 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.

The responsibilities of the engineering companies involved in the preparation of the technical report are as follows:

- Ausenco managed and coordinated the work related to the report and developed PFS-level design and cost estimate for the process plant and general site infrastructure. Ausenco completed geotechnical studies and developed the PFS-level design and cost estimate of the heap leach.
- AGP Mining Consultants Inc. (AGP) designed the open pit mine, mine production schedule, and estimated mine capital and operating costs.
- SRK conducted the economic analysis.
- KP conducted a review of the environmental studies, permitting, and conducted site-wide water management.
- Merlin completed the work related to property description, accessibility, local resources, geological setting, deposit type, exploration work, drilling, exploration works, sample preparation and analysis, and data verification.
- Advantage Geoservices developed the mineral resource estimate for the project.

The technical report, with an effective date of February 28, 2023, discloses the results of exploration work completed since the previous technical report was issued in 2019, and provides an update to costs and economic analysis of the 2019 Prefeasibility Study Report.

1.2 Property Description and Location

The Filo del Sol Project is located in the Atacama Region of Northern Chile and adjacent San Juan province of Argentina. The project is 140 km southeast of the city of Copiapó, Chile and straddles the border between Argentina and Chile. The centre of the main deposit area is located at 28.49° S and 69.66° W (decimal degrees, WGS84 datum).

Figure 1-1: Filo del Sol Copper-Gold-Silver Project Map

Source: Filo Mining, 2023

The Filo del Sol property has mineral titles in Chile and Argentina. Those in Argentina are controlled by Filo del Sol Exploración S.A. and are referred to as the “Filo del Sol property,” while those in Chile are controlled by Frontera Chile Limitada and are referred to as the “Tamberías Property.” Both Filo del Sol Exploración S.A. and Frontera Chile Limitada are wholly-owned subsidiaries of Filo Mining Corp. For the purposes of this report, Filo Mining Corp. and its subsidiary companies are referred to interchangeably as “Filo Mining.”

Filo del Sol Exploración S.A. owns eight exploration permits (manifestaciones) in Argentina. In Chile, Frontera Chile Limitada owns 12 exploration concessions and is in the process of obtaining 4 more. They also own three exploitation mining concessions (mensuras) and one unilateral and irrevocable option agreement to purchase 17 additional exploitation concessions.

The total combined area of the project is approximately 13,575 hectares (ha). The project is included within the “Vicuña Additional Protocol” under the Mining Integration and Complementation Treaty between Chile and Argentina. The main benefit during the exploration stage of the Vicuña Additional Protocol is the authorization that allows people and equipment to freely cross the border of both countries in support of exploration and prospecting activities within an area defined as an “operational area.” The development of transboundary mining projects is contemplated under the Treaty.

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

The project is accessible by road from either Copiapó, Chile or San Juan, Argentina. The climate is cold and windy, which is typical of the high Andes. The exploration field season can run year-round; however, winter operations have to contend with severe weather conditions. Field work is based out of the Batidero camp approximately 20 km from Filo del Sol in Argentina. The Batidero camp can accommodate over 1,000 people with approximately 250 to 300 Filo Mining staff based there during the active field season.

The project is in the Andes Mountains with elevations ranging from 4,500 m to 5,500 m above mean sea level (amsl). The mountains are generally not rugged and vehicle access to most of the property is possible. Vegetation is almost entirely absent within the area.

1.4 History

Cyprus-Amax was the first company to conduct significant exploration work in the area, beginning in 1997 and based on recognition of auriferous silica and a Cu-Au porphyry occurrence on the Chilean side of the border. Cyprus–Amax’s work during the 1998-1999 season consisted of 1:10,000 geologic mapping, talus fine sampling, rock chip sampling, road construction to the project site, and a drill program of 2,519 m in 16 reverse circulation (RC) drill holes. Filo Mining became involved in the project through its predecessor company, Tenke Mining Corp., which negotiated purchase arrangements with Cyprus-Amax in August 1999.

1.5 Geological Setting and Mineralization

Filo del Sol is a high-sulphidation epithermal copper-gold-silver deposit associated with a large porphyry copper-gold system. It is located in the Andean Frontal Mountain Range within the Vicuña belt, between the gold and copper-gold porphyry deposits of the Maricunga belt to the north and the high-sulphidation epithermal deposits of the El Indio belt to the south. Mineralization is hosted in Late Cretaceous clastic rocks, mafic dykes and sills, as well as in underlying rhyolitic volcanic rocks that are part of the Permo-Triassic basement.

Overlapping mineralizing events and a high degree of telescoping, combined with weathering effects including supergene enrichment, have created several different types and styles of mineralization. The uppermost part of the deposit includes structurally-controlled gold, tabular high-grade silver (± copper) and high-grade supergene-enriched copper all within a leached and oxidized domain that formed over high-sulphidation Cu-Au-Ag epithermal mineralization and disseminated porphyry Cu-Au mineralization. Within the hypogene domain, there are two distinct types of mineralization: deeper porphyry Cu-Au mineralization in potassic alteration is overprinted and reconstituted by high-sulphidation epithermal Cu-Au-Ag mineralization associated with advanced argillic alteration. The boundary between the two types is sharp and well-defined geochemically.

The Filo del Sol mineral resource described in this report includes predominantly the upper, oxidized, and supergene-enriched portion of the overall deposit. In addition to the Filo del Sol deposit, several other exploration targets occur on the property.

1.6 Deposit Types

The Filo del Sol deposit includes both porphyry Cu-Au and high-sulphidation epithermal Cu-Au-Ag mineralization. The mineralized system in its entirety represents a telescoped porphyry–epithermal system with multiple intrusive and breccia centres, and so combines aspects of both deposit types. The deeper porphyry mineralization contains both disseminated sulphides and various veinlet and stockwork systems that also host sulphides. The upper-level epithermal style mineralization includes siliceous vein fillings, irregular branching fissures, stockworks, breccia pipes, vesicle fillings and disseminations. The currently defined mineral resource presented in this report is best classified as the upper oxidized part of the high-sulphidation epithermal Cu-Au-Ag part of the deposit.

1.7 Exploration

Filo Mining or its predecessor companies have been exploring at Filo del Sol since the 1999/2000 field season. A total of 20 work programs have been completed over these years, and there have been four seasons (2001-2002, 2002-2003, 2008-2009, 2009-2010) where no work was done. Exploration has been limited to the summer season (until 2021-2022), typically between November and April, so exploration seasons are described by the years which they bridge.

Surface work completed to date has included talus fine sampling, rock chip sampling, geological mapping, and induced polarization (IP) and magnetic geophysical surveys.

1.8 Drilling

Drilling at Filo del Sol was initiated by Cyprus in 1998-1999, and until the end of 2022, 44,950 m of reverse circulation (RC) drilling in 185 holes and 52,064 m of diamond drilling (DD) in 106 holes have been completed on the property.

1.9 Sampling Preparation, Analysis, and Security

Sampling from drilling, sample preparation, analysis and sample security has been conducted at or above recognized industry standards applicable at the time samples were taken at Filo del Sol. More than 83% of the current RC and DDH dataset had a rigorous QA/QC protocol with blanks, standards, and laboratory duplicates. Around 5% have been checked at a second laboratory but at the time, did not have blank and standard controls. The remaining 12% of the dataset has been satisfactorily verified with duplicates. No sample appears to be misplaced or intentionally deleted from the database. The current drillhole dataset for the Filo del Sol project is consistent and has adequate quality to be used for resource estimation.

1.10 Data Verification

As verification of information provided by the company, F. Devine (from Merlin) was directly involved in updating the geological model for the project in 2015-2019. This included completing extensive surface geological mapping and core logging, data and interpretation review and discussion with company personnel. She visited the project again from October 9-11, 2022, to review the most recent drilling and geological model updates. Ten samples of quartered core were taken from drill holes drilled over the past three years from a range of Cu, Au, Ag grade domains, and the results correlate well with original values.

A visit to the Copiapó office and support facilities was carried out by J. Gray, between June 16-21, 2014. Six samples were taken from a variety of geological settings. Samples were coarse rejects from RC drill cuttings and were approximately 5 kg. Results of these independent samples agreed closely with the original values.

Independent assaying of individual samples used to create metallurgical test composites was carried out by SGS Lakefield. These results compare well with the original sample analyses. The results of these checks are considered a satisfactory confirmation of the results reported by Filo Mining.

1.11 Mineral Processing and Metallurgical Testing

Four phases of comprehensive metallurgical test programs between 2001 and 2018 focused on assessing the feasibility of using heap leaching to recover copper, gold, and silver from the various mineralization types identified. The first phase was conducted in 2001 by Novatech S.A. of Santiago, Chile on various samples of the oxide and mixed zones. The 2001 testwork was preliminary in nature and consisted of bottle rolls and diagnostic leaches on 20 samples of RC chips. The second phase was conducted by SGS Minerals (Lakefield) in 2016 on one sample of each of the oxide gold, oxide copper and mixed silver mineralization. The third phase was conducted at SGS Minerals (Lakefield) in 2017 on samples from several different zones of mineralization within the deposit. The fourth, more comprehensive, phase was conducted at SGS Minerals (Lakefield) in 2018 on various samples from the four main zones (Tamberías gold oxide (TMB AuOx), Filo del Sol gold oxide (FDS AuOx), Tamberías copper-gold oxide (TMB CuAuOx) and Filo del Sol copper-gold oxide (FDS CuAuOx) + M-Zone (M-Ag)).

To confirm and improve the 2016 and 2017 results, a fourth phase of work was carried out in early 2018 using surface samples, RC chips, and diamond drill core samples. In total, 14 surface trench samples, 32 RC chips samples and 20 diamond drill hole intervals were collected and sent to SGS (Lakefield) for various test programs. More than 3,500 kg of sample was shipped to the SGS facility in where it was subjected to various physical, chemical, and detailed mineralogical characterization tests.

Most of the phase four metallurgical program was devoted to heap leaching, which was simulated by completing column leaching tests on material ranging from 12.5 mm to 63.5 mm crush size and using approximately 50 to 250 kg of sample per column test. Cyanide column leaching was tested for the gold oxide ore types (11 column tests), while sequential column leaching (acid leaching followed by washing/neutralization and cyanide leaching) was used for the copper-gold oxide ore types (18 sequential column tests).

Variability and process optimization testing were carried out using bottle roll tests on minus 10 mesh material. Both cyanide leaching (21 bottle roll tests) and sequential leaching (72 sequential leach bottle roll tests) were conducted during the 2018 program.

The results of the test program were used to determine the preferred leach configuration together with expected leach recoveries for copper, gold, and silver. Deductions to the testwork extractions were applied to expected copper, gold, and silver recoveries to simulate scale-up to a commercial production facility. Metal recovery equations for Cu, Au, and Ag were determined and applied to each ore type in the production schedule and financial model. The equations are detailed in Section 13.4. and result in estimated life-of-mine metal recoveries of 78%, 70% and 83% for Cu, Au and Ag, respectively, with the current mine plan.

Beginning in 2020, initial testwork to evaluate flotation characteristics of the deeper sulphide mineralization discovered by drilling subsequent to the 2019 PFS was initiated.

Preliminary sulphide metallurgical testwork was conducted on three composite samples of sulphide material from drill core originating from the 2018-2019 and 2019-2020 drilling campaigns. This material was intended to represent mineralization that is not included in the resource model. The metallurgical testwork was also completed at SGS Minerals (Lakefield) during 2020, 2021 and 2022. The focus of the preliminary testing was to provide insight and direction for future testing requirements for the hypogene sulphide portion of the deposit.

The samples varied from 0.33% to 0.57% Cu, 0.38 to 0.41 g/t Au and 1.3 to 10.3 g/t Ag. Two samples had a low arsenic content (“HiRes” material) of ≤10 g/t and one sample (“HiCN” material) had a high arsenic content of 1,400 g/t or 0.14%. The flowsheet and reagent scheme were not fully optimized for this testing program but for the HiRes sample, a concentrate containing 22% Cu, 18 g/t Au, 37 g/t Ag and 880 g/t As was produced, while the HiCN sample produced a concentrate containing 26% Cu, 14 g/t Au, 106 g/t Ag and 52,400 g/t (5.24%) As. Additional post concentrate treatment testwork was completed on the HiCN sample which successfully evaluated a number of potential options for As reduction using commercially available technology.

The flotation cleaner tailings were subjected to intensive cyanide leaching tests. The results indicated that an additional 10% to 16% of the gold and 10% to 26% of the silver could potentially be recovered. Using a concentrator and tailings leach process, approximately 88% of the copper and 80% of the gold was recovered from the HiRes sample, and 90% of the copper and 75% of the gold from the HiCN sample.

Preliminary comminution testing indicated that the composite samples reflected a moderate hardness, with an indicated Bond ball mill work index of 14 to 15 kWh/t.

1.12 Mineral Resource Estimates

The Filo del Sol updated mineral resource estimate replaces that released in February 2019. Although this update considers the results of 60 new holes completed since the previous mineral resource estimate, it should be noted that the block model limits were not changed from the 2019 model and the new resource does not include the deeper, high-grade mineralization of the Aurora Zone.

This resource update is based on a total of 61,800 metres of drilling in 247 holes including an additional 1,156 metres of reverse circulation drilling in six new holes and 18,725 metres of diamond drilling in 54 new holes from drilling completed in since the 2017-2018 field season. The resource estimate presented below is the total indicated and inferred resource, divided between oxide and sulphide mineralization.

The mineral resource estimate shown in the Table 1-1 has an effective date of January 18, 2023. The mineral resources are inclusive of mineral reserves. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There are no known legal, political, environmental, or other risks that could materially affect the potential development of the mineral resource.

Table 1-1: Mineral Resource Estimate (Effective January 18, 2023)

Zone	Cutoff	Category	Tonnage (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Au (koz)	Ag (koz)
Oxide	See notes	Indicated Inferred	362.2 132.7	0.34 0.25	0.33 0.30	13.3 9.9	2,683 725	3,839 1,284	154,670 42,370
Sulphide	0.30%	Indicated Inferred	70.4 78.9	0.31 0.31	0.35 0.33	2.5 3.1	473 542	790 834	5,710 7,960
Total		Indicated Inferred	432.6 211.6	0.33 0.27	0.33 0.31	11.5 7.4	3,156 1,267	4,629 2,118	160,380 50,330

Notes: **1.** The qualified person for the resource estimate is James N. Gray, P Geo. of Advantage Geoservices Ltd. **2.** The mineral resources were estimated in accordance with the CIM Definition Standards for Mineral Resources and Reserves. **3.** Sulphide copper equivalent (CuEq) assumes metallurgical recoveries of 84% for copper, 70% for gold and 77% for silver based on similar deposits, as no metallurgical testwork has been done on the sulphide mineralization, and metal prices of $4/lb copper, $1800/oz gold, $23/oz silver. The CuEq formula is: CuEq=Cu+Ag*0.0077+Au*0.5469. **4.** All figures are rounded to reflect the relative accuracy of the estimate. **5.** Mineral resources are not mineral reserves and do not have demonstrated economic viability. **6.** The resource was constrained by a optimised pit shell using the following parameters: Cu $4/lb, Ag $23/oz, Au $1800/oz, slope of 29° to 45°, a mining cost of $2.72/t and an average process cost of $9.86/t. **7.** Cutoff grades are 0.2 g/t Au for the AuOx material, 0.15% CuEq for the CuAuOx material and 20 g/t Ag for the Ag material. These three mineralization types have been amalgamated in the oxide total above. CuAuOx copper equivalent (CuEq) assumes average metallurgical recoveries of 77% for copper, 72% for gold and 71% for silver based on preliminary metallurgical testwork, and metal prices of $4/lb copper, $1800/oz gold, $23/oz silver. The CuEq formula is: CuEq=Cu+Ag*0.0077+Au*0.6136. **8.** Mineral resources are inclusive of mineral reserves.

1.13 Mineral Reserve Estimates

The initial mineral reserve estimate for Filo del Sol shown in Table 1-2 has an effective date of February 28, 2023 and is based on the mineral resource statement with an effective date of January 18, 2023.

Table 1-2: Filo del Sol Mineral Reserve Estimate @$0.01/t NVPT Cutoff (Effective February 28, 2023)

Category (All Domains)	Tonnage (Mt)	Grade				Contained Metal		
		Cu (%)	Au (g/t)	Ag (g/t)	NVPT ($/t)	Cu (Mlb)	Au (koz)	Ag (koz)
Proven	-	-	-	-	-	-	-	-
Probable	259.6	0.39	0.34	16.0	32.5	2,220	2,867	133,334
Total Proven and Probable	259.6	0.39	0.34	16.0	32.5	2,220	2,867	133,334

Notes: **1.** The qualified person for the estimate is Mr. Gordon Zurowski, P.Eng. of AGP Mining Consultants, Inc. **2.** The mineral reserves were estimated in accordance with the CIM Definition Standards for Mineral Resources and Reserves. **3.** The mineral reserves are supported by a mine plan, based on a pit design, guided by a Lerchs-Grossmann (LG) pit shell. Inputs to that process are metal prices of Cu $3.50/lb, Ag $20/oz, Au $1600/oz; mining cost average of $2.72/t; an average processing cost of $9.65/t; general and administration cost of $1.46/t processed; pit slope angles varying from 29 to 45 degrees, inclusive of geotechnical berms and ramp allowances; process recoveries were based on rock type. The average recoveries applied were 83% for Cu, 73% for Au and 80% for Ag, which exclude the adjustments for operational efficiency and copper recovered as precipitate which were included in the financial evaluation. **4.** Dilution and mining loss adjustments were applied at ore/waste contacts using a mixing zone approach. The volumes of dilution gain and ore loss were equal, resulting reductions in grades of 1.0%, 1.3% and 1.0% for Cu, Au and Ag, respectively. **5.** Ore/waste delineation was based on a net value per tonne (NVPT) cutoff of $4.5/t considering metal prices, recoveries, royalties, process and G&A costs as per LG shell parameters stated above, elevated above break-even cutoff to satisfy processing capacity constraints. **6.** The life-of-mine stripping ratio in tonnes is 1.57:1. **7.** All figures are rounded to reflect the relative accuracy of the estimate. Totals may not sum due to rounding as required by reporting guidelines.

1.14 Mining Methods

The Filo del Sol deposit is a large near surface, bulk mineable deposit that is well suited for extraction by conventional open pit methods. Ore and waste will be drilled, blasted, and loaded by diesel hydraulic face shovels and front-end loaders from 12-metre benches. Haul trucks will haul the material to the ore crusher, a short-term stockpile, or the waste dump as required. Autonomous haulage was incorporated to take advantage of the technology's proven productivity improvements and operating cost savings.

The open pit will have a mine life of 13 years, including pre-stripping, with a life-of-mine strip ratio of 1.57:1. A maximum mining rate of approximately 68 Mt/a (including waste but not rehandle) is required to provide the nominal 60,000 t/d of ore to the process facility. A total of 260 Mt of ore is expected to be processed over the life of the mine.

1.15 Recovery Methods

Ore will be trucked from the mine and either stockpiled or direct tipped into the primary crusher at a nominal throughout of 60,000 t/d or 21.9 Mt/a. The ore will be further crushed through a closed-circuit secondary crushing system to a stockpile.

Crushed ore will be processed at an on/off heap leach pad where the copper will be leached in sulphuric acid and then recovered from the leach solution by solvent extraction and electrowinning to produce London Metal Exchange (LME) grade copper cathodes. Metal leaching is expected to occur over 13 years.

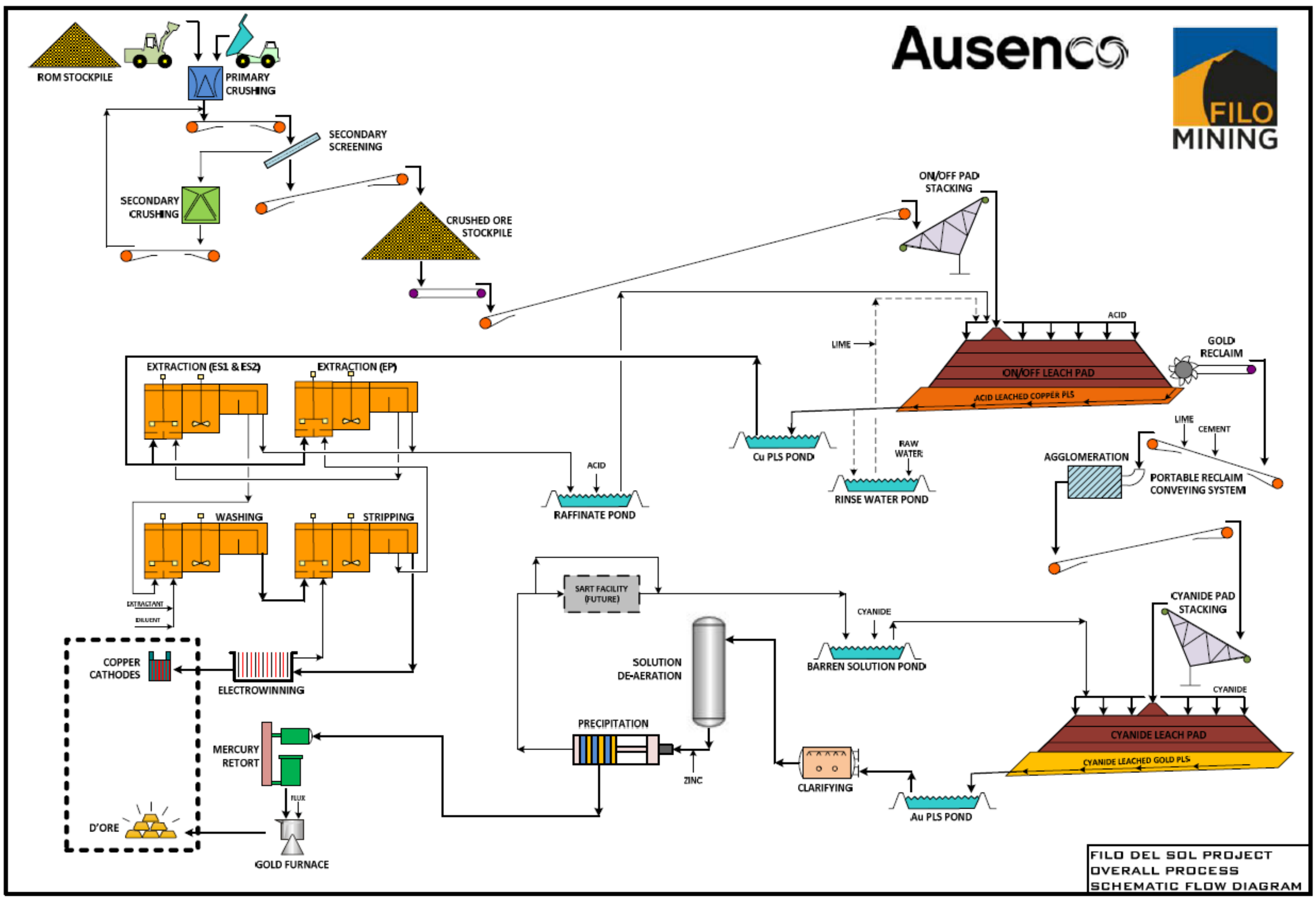
Once the copper is leached, the ore will be rinsed, neutralized, and removed from the on/off leach pad by a bucket wheel reclaimer. The material will then be agglomerated using cement, and subsequently stacked on a permanent heap leach pad where gold and silver will be leached in a cyanide solution. Gold and silver will be recovered from the pregnant gold leach solution by a Merrill-Crowe zinc precipitation process and then smelted to produce doré.

A sulphidization, acidification, recycle and thickening process (SART) will be installed in the second year of operation. The SART unit operation will treat a portion of the barren gold leach solution before it is recycled to the permanent cyanide leach pad. The SART process will reduce the copper load in the leach solution and regenerate cyanide, which is bound to the dissolved copper thus reducing overall cyanide consumption and providing revenue from the corresponding copper sulphide precipitate.

A process flow diagram is shown in Figure 1-2. The process plant includes the following facilities:

- two-stage crushing of run-of-mine material;
- copper on/off leach pad;
- copper solvent extraction with two stages of extraction, stripping and washing followed by electrowinning; and
- cyanide leach pad followed by Merrill-Crowe circuit and gold refinery.

Figure 1-2: Overall Process Schematic Flow Diagram



Source: Ausenco, 2019

1.16 Infrastructure

Infrastructure to support the Filo del Sol project will consist of site civil work site facilities/buildings, on-site roads, a water management system, and site electrical power. Site facilities will include both mine facilities and process facilities, as follows:

- mine administration offices, truckshop, explosives storage, fuel storage and distribution, ore stockpiles, waste stockpiles, and truck wash;
- process facilities including the crushing facilities, leach pad, on/off pad, process plant, process plant workshop, assay laboratory, freshwater infrastructure;
- general facilities include a gatehouse, administration building, communications, and switchyard; and
- catchments, ponds, and other site water management infrastructure.

An overall site layout is provided in Figure 1-3.

A geotechnical program was carried out as part of the design of the heap leach facilities, primary crusher, waste dump facility, and stockpiles. The field program included surface mapping and a test pit program to take samples of soil and rock from plant site, primary crusher site, waste dump facility, stockpiles, and leach pads site along with a corresponding laboratory testing program to understand the foundation conditions for these site facilities and material properties of borrow sources.

The Filo del Sol project infrastructure is situated on alluvium and colluvium that is underlain by weathered bedrock. Most of the mine site has permafrost 0.5 to 1.0 metres below the surface. The design of structures took this into account.

The major infrastructure items are listed below.

1.16.1 Access Road

Approximately 48 km of light vehicle road will require upgrading to a 9-m-wide, two-lane, dirt road to connect the Filo del Sol mine site to the national highway system at Iglesia Colorada. Roads will connect various mine facilities, including the camp, open pit, truckshop, crushers, process plants, heap leaches, electrical substations, and administrative buildings.

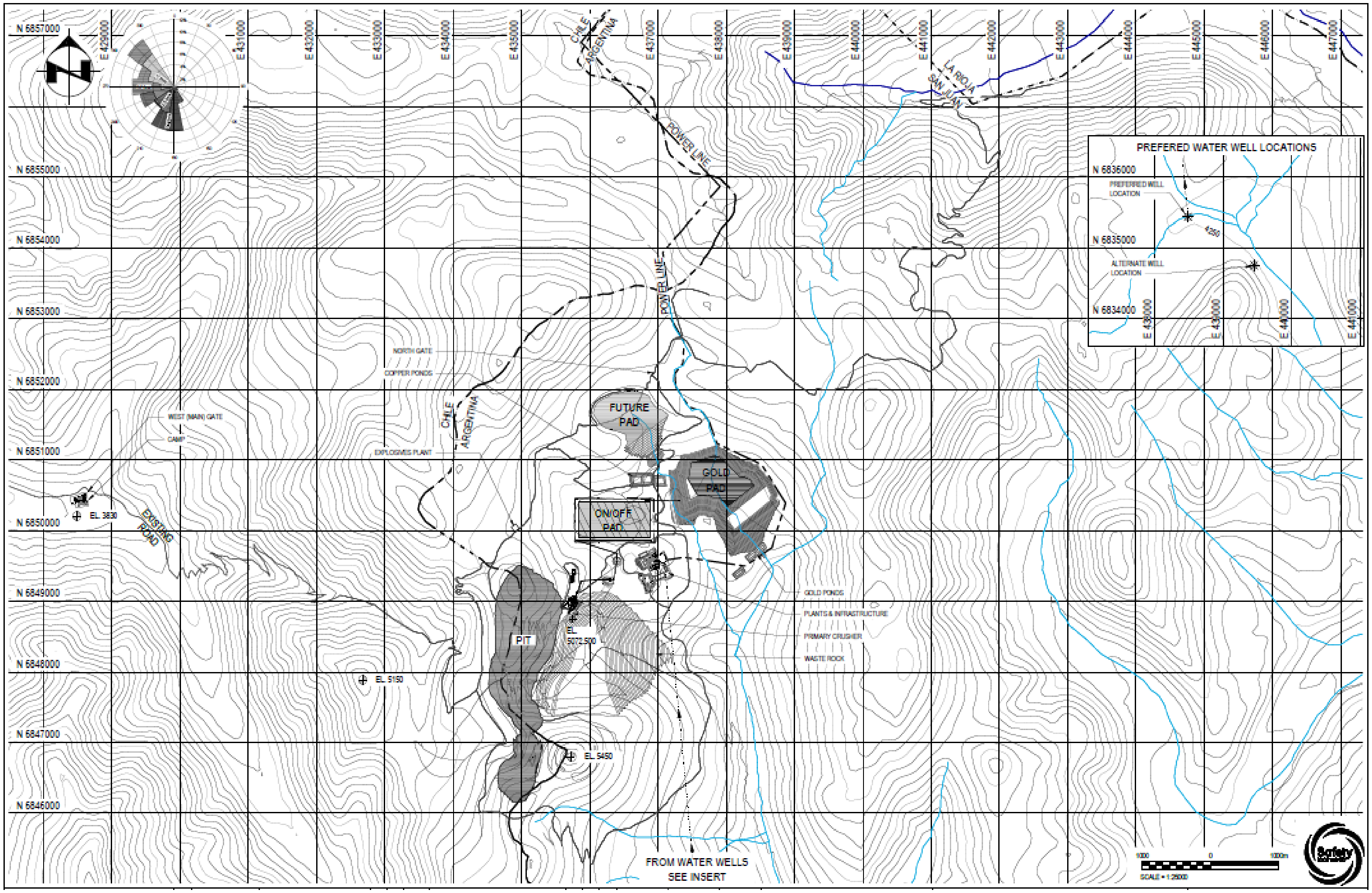
1.16.2 Water Supply

Water will be supplied from local aquifers in Argentina, located near the proposed plant site. The water makeup requirement is estimated to be 75 L/s based on a 60,000 t/d nominal feed rate.

1.16.3 Power Supply

The site will be supplied with electricity through a 127 km long, 110 kV, single circuit power transmission line connected to the Los Loros substation in Chile. Average electrical demand is estimated to be 56 MW.

Figure 1-3: Infrastructure Layout Plan



Source: Ausenco, 2019

1.16.4 Product Transport

Copper cathode will be transported by truck to Puerto Caldera, a port near the city of Caldera located 77 km by road northwest of Copiapó. The approximate trucking distance from the plant site is 245 km; approximately 48 km of existing road will require upgrades to accommodate the truck traffic. Doré will be transported approximately 175 km to Aeropuerto Desierto de Atacama for ongoing airfreight.

1.16.5 Waste dump

During mining operations, waste rock generated during the extraction of ore from open pit operations will be permanently stored immediately east of the Filo del Sol pit. Due to the presence of near-surface permafrost throughout the facility's upper end of its footprint, "bottom up" construction and the excavation of keyway in the toe area are required to provide good contact and stability of the ultimate facility.

1.16.6 Heap Leach Facilities

The heap leach facilities include two leach pads: an on/off copper pad and a permanent gold pad. The on/off heap leach facility is located approximately 1 km northeast of the open pit and consists of 580,000 m^2 dynamic leach pad, operation ponds and process plant. The permanent gold heap leach facility is located immediately east of the on/off pad and consists of 1.6 Mm^2 permanent gold heap leach pad, operation ponds. The process plant is located next to the on/off pad process plant.

1.17 Market Studies and Contracts

The principal planned products are copper cathode and gold/silver doré.

No specific marketing study was conducted for the study. Copper cathode and gold/silver doré are readily traded commodities. Accordingly, it is appropriate to assume that the products can be sold freely and at standard market rates.

The company has no contracts in place.

1.18 Environmental, Permitting, and Social Licence

KP completed environmental baseline work for the project in 2017 and 2018 and reviewed the historical work from other independent consultants who assisted in the environmental work. This work will be used to support the preparation of an Environmental Impact Assessment (EIA).

An EIA and its subsequent Declaración de Impacto Ambiental (DIA) are required for the exploration phases of mineral development. The Filo del Sol project has maintained all previous exploration activity permits in good standing, which required the submission of an EIA and receipt of a DIA. The most recent DIA was issued on March 23, 2022 and is valid for two years, whereupon it can be renewed.

Baseline studies to date have been carried out on geosciences, air and water, terrestrial biota, the human environment, and natural and cultural heritage. The list of environmental components to be studied was derived from the Chilean national environmental assessment regulations, the Argentine national mining environmental law and from the

International Finance Corporation's Sustainability Performance Standards (IFC 2012). Baseline studies are ongoing and will continue into the upcoming field season.

Communication with the local community, private landowners, and other interested parties is ongoing.

1.19 Capital Cost Estimate

The capital cost estimate was developed in Q1 2023 US dollars based on budgetary quotations for equipment and construction contracts, as well as Ausenco's in-house database of projects and studies including experience from similar operations. The estimate conforms to Class 4 guidelines for a prefeasibility study level estimate with a +30/-20% accuracy, according to the Association of the Advancement of Cost Engineering International (AACE International).

Table 1-3: Capital Cost Estimate

Description	Initial (US$M)	Sustaining (US$M)	Closure (US$M)	Life of Mine (US$M)
Direct Costs				
Mine	230	9	-	239
Processing	610	131	-	741
On-Site Infrastructure	117	-	-	117
Off-Site Infrastructure	188	-	-	188
Subtotal Direct Costs	**1,145**	**140**	**-**	**1,285**
Indirect Costs	185	-	-	185
EPCM Services	149	-	-	149
Owner's Costs	50	-	-	50
Provisions	275	-	-	275
Subtotal Indirect Costs	**660**	**-**	**-**	**660**
Closure	-	-	69	69
Total	**1,805**	**140**	**69**	**2,013**

*Numbers above are rounded to the nearest integer; therefore, some subtotals may not balance due to rounding.

The following parameters and qualifications were considered:

- The estimate was developed in Q1 2023 US dollars.
- Metric units of measure are used throughout the estimate.
- Actual estimate accuracy is defined by the stated maturity of the information available.
- No allowance has been made for exchange rate fluctuations.
- In addition to contingency a growth allowance was included.
- There is no escalation added to the estimate.
- Data for the estimates have been obtained from numerous sources, including the following:
  - mine schedules

- pre-feasibility level engineering design
- topographical information obtained from the site survey
- geotechnical investigations
- vendor equipment and material supply costs
- budgetary unit costs from contractors for civil, concrete, steel, electrical, piping and mechanical works
- data from similar recently completed studies and projects.

1.20 Operating Cost Estimate

The average yearly operating cost for the project, cost per tonne of processed material and life of mine costs is summarized in the Table 1-4 for mining, processing, and site G&A.

Table 1-4: Operating Cost Estimate

Operating Costs	$/t Processed	US$M/a	Life of Mine (US$M)
Mining	6.63	132	1,720
Processing	9.72	213	2,523
Site G&A	1.67	37	434
Total	**18.01**	**382**	**4,677**

1.21 Economic Analysis

Analysis of the project demonstrates that the mine plan has positive economics under the assumptions used. The project post-tax NPV at an 8% discount rate is estimated to be $1.31 billion, with an IRR of 20%. The project financial summary is shown in Table 1-5.

Note: Cash flows have been discounted to the start of construction, assuming that the project execution decision will be taken, and major project financing will be carried out at this time. Schedule and expenditure for the feasibility study, including technical and economic studies, engineering studies, cost estimating, resource delineation and infill drilling, pit slope geotechnical characterization, metallurgical sampling and test-work, associated exploration, strategic optimization, mine, plant and infrastructure design, permitting and other pre-construction activities were not modelled.

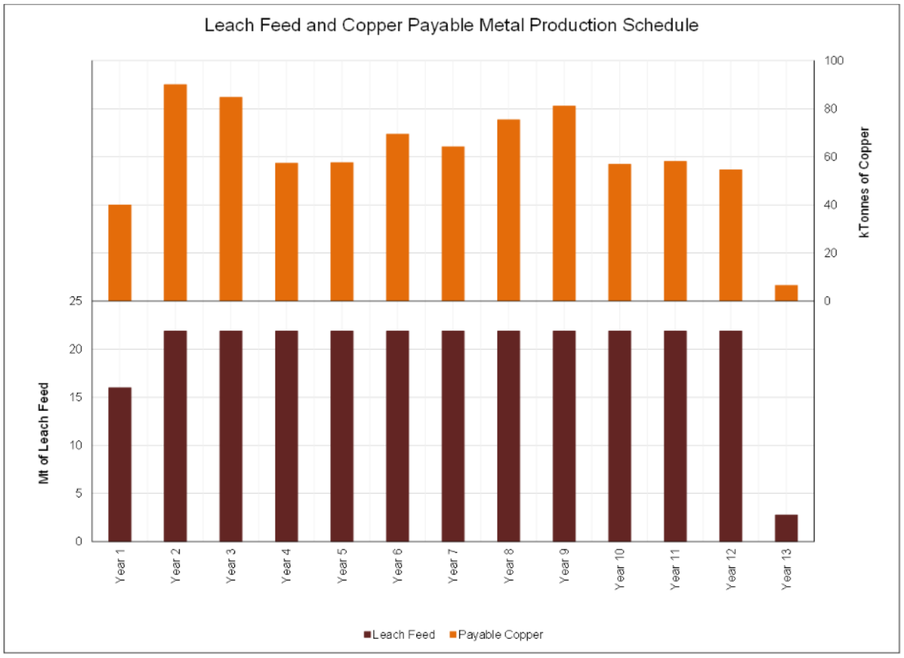
Table 1-5: Project Financial Summary

Project Metric	Units	Value
Pre-Tax NPV (8%)	US$M	2,040
Pre-tax IRR	%	24%
After-Tax NPV (8%)	US$M	1,310
After-Tax IRR	%	20%
Undiscounted After-Tax Cash Flow (Life of Mine)	US$M	3,560
Average Operating Margin*	%	60%
Payback Period from Start of Processing (Undiscounted, After-Tax Cash Flow)	years	3.4
Initial Capital Expenditures	US$M	1,805
LOM Sustaining Capital Expenditure (Excluding Closure)	US$M	140
LOM C-1 Cash Costs (Co-Product)	$/lb CuEq	1.54
Nominal Process Capacity	t/d	60,000
Mine Life (including pre-stripping)	years	13
Average Annual Copper Production**	tonnes	66,000
Average Annual Gold Production**	oz	168,000
Average Annual Silver Production**	oz	9,256,000
LOM Recovery – Copper***	%	78%
LOM Recovery – Gold	%	70%
LOM Recovery – Silver	%	83%

Notes: * Operating Margin = Operating Cashflow/Net Revenue. ** Rounded and excluding final year of minimal leach operation. *** Excluding 1% Cu recovery to concentrate for SART process.

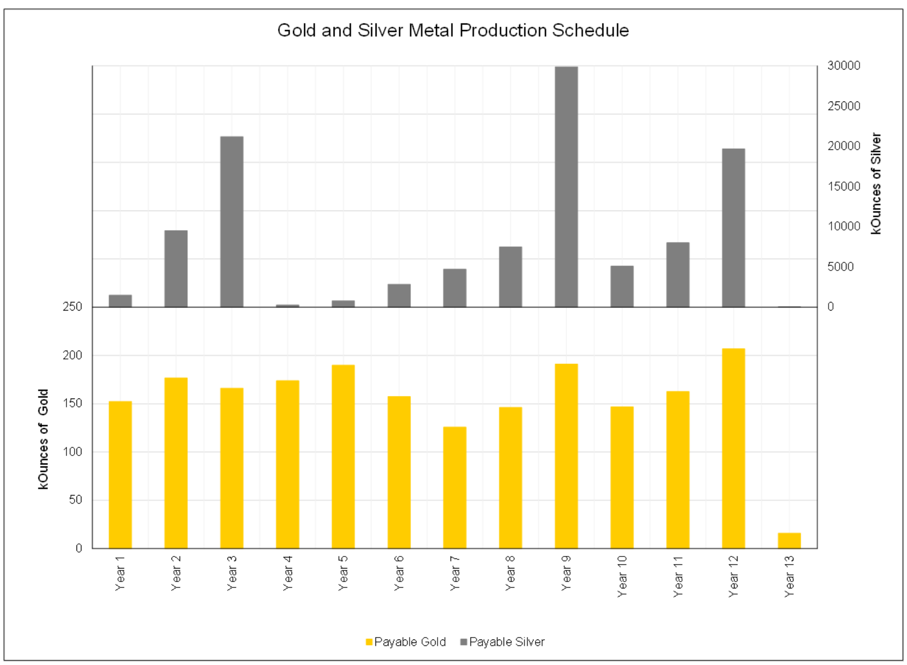
The proposed production schedule and metal production profile are shown in Figure 1-4 and Figure 1-5.

Figure 1-4: Leach Feed and Copper Metal Production Schedule



Source: SRK, 2023

Figure 1-5: Gold and Silver Metal Production Schedule

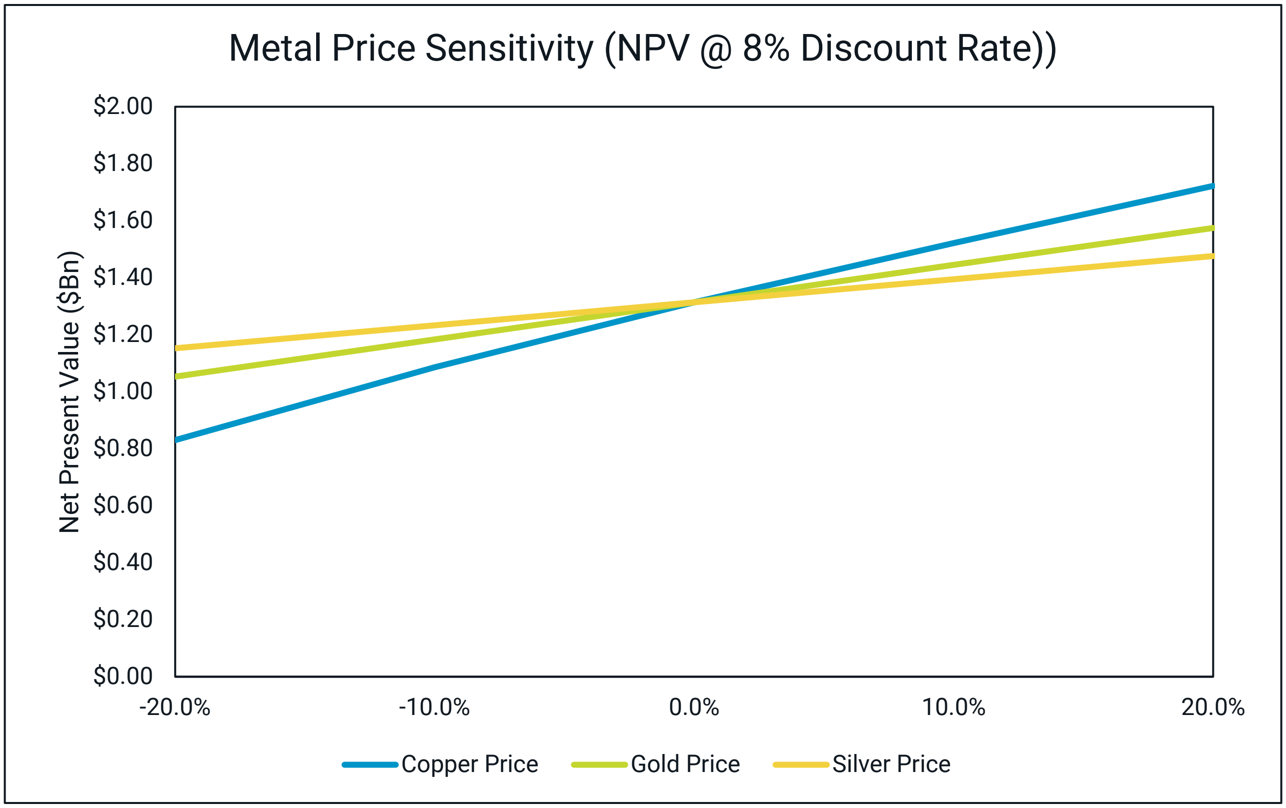


Source: SRK, 2023

A cash flow valuation model for the Project has been developed using a long-term copper price of $3.65/lb, a gold price of $1,700/oz, and a silver price of $21/oz. Figure 1-6 shows the sensitivity of estimated NPV for the project's cash flow at various changes to metal prices at 8% discount rate. Figure 1-7 shows the sensitivity of estimated NPV for the project's cash flow at various changes to operating costs, capital costs and metal prices at 8% discount rate.

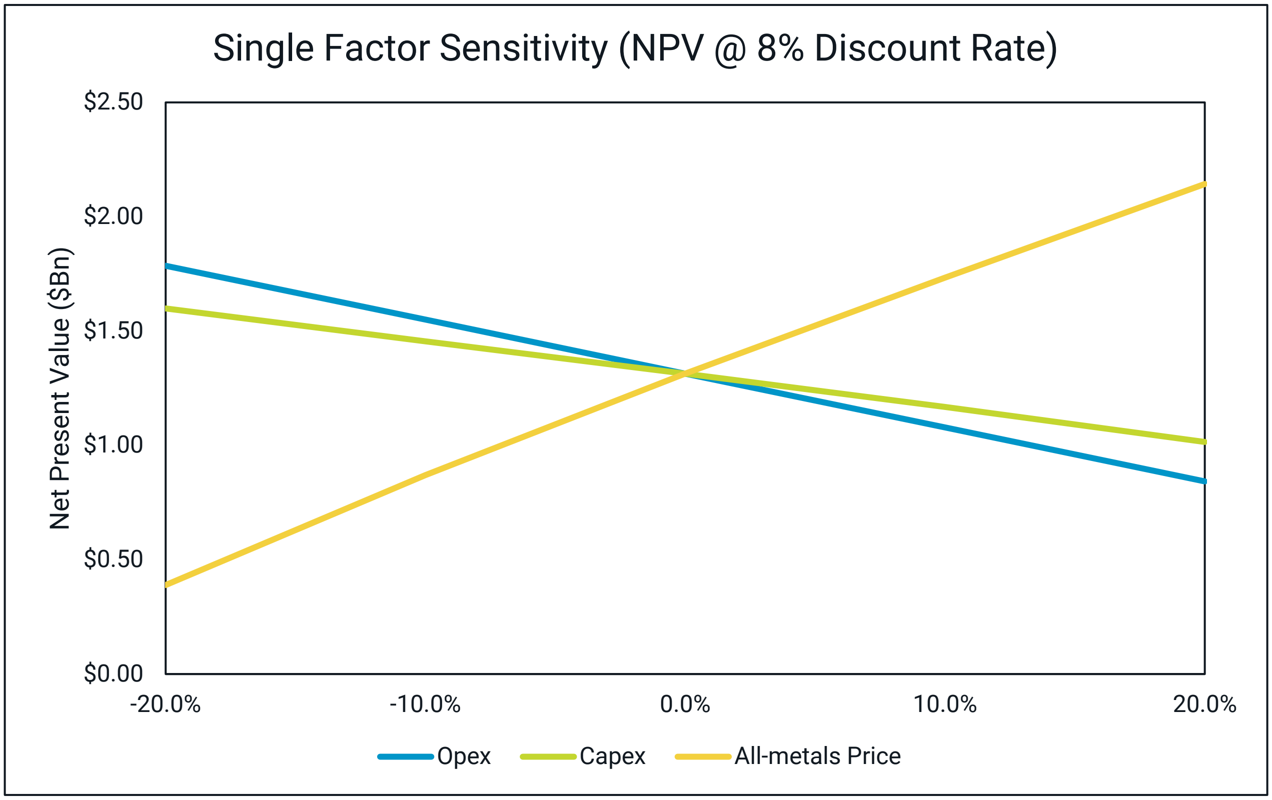
A summary of the post-tax project economics is listed in Table 1-5.

Figure 1-6: Metal Price Sensitivity



Source: SRK, 2023

Figure 1-7: Single-Factor Sensitivity Chart



Source: SRK, 2023

1.22 Adjacent Properties

There are no relevant adjacent properties for the purposes of this report.

1.23 Other Relevant Data and Information

Drilling since 2019 has established Filo del Sol as a major deposit of copper, gold, and silver in a unique highly telescoped high-sulphidation epithermal/porphyry deposit. This style of deposit forms some of the largest copper-gold deposits known and will require a significant effort in order to fully delineate. Several high-potential target areas exist for the discovery of new mineralized centres, and it remains to be determined if these will prove to be separate deposits themselves, or different parts of one very large deposit contiguous with what has already been discovered.

1.24 Interpretations and Conclusions

The work completed during this prefeasibility study has indicated that the Filo del Sol Project has potential economic merit. The financial analysis has shown a positive net present value and internal rate of return.

The Filo del Sol Project encompasses a very large zone of alteration and a mineral deposit that remains open in several directions within a prolific mineral district. Both high-sulphidation epithermal gold-silver-copper and porphyry copper-gold mineralization have been discovered and both styles of mineralization are compelling exploration targets.

The Filo del Sol Project is amenable to development by open pit mining methods. AGP considers that there are no technical incumbrances to mining using standard mining equipment. In addition, AGP assessed and included autonomous haulage as part of the overall mine plan.

Metallurgical testing results indicate that Filo del Sol mineralized material is amenable to the application of conventional crushing, sequential acid and cyanide heap leaching, solvent extraction-electrowinning for recovery of copper (as cathodes) and Merrill-Crowe processing for recovery of gold and silver (as gold/silver doré).

The design of the project infrastructure is reasonably straightforward, and with significant precedent in the region. No “novel” solutions are proposed. Ausenco considers that there are no fatal flaws with respect to the project infrastructure assumptions and outlook.

The constructability of the envisaged project appears to be viable. No unusual aspects of location, logistics, or availability of resources that may affect the construction have been identified.

1.25 Recommendations

Although this PFS outlines a compelling economic case for additional studies and eventual development of the Filo del Sol oxide resource, the extent and tenor of the significant sulphide mineralization discovered by drilling since 2019 indicates that the focus should continue to be on outlining and defining the full potential of the Filo Del Sol property. Once a more comprehensive understanding of the entirety of the mineralization has been developed, options on how to best progress the development of the deposit will be assessed.

Recent drilling has intersected long intervals (>1km) of high-grade mineralization beneath, and to the north of, the current mineral resource. Although this additional mineralization has not been fully defined and remains open to expansion it is already significant enough to change the entire scope of the project. This zone of mineralization beneath and north of the resource has been named the Aurora Zone.

In addition to the Aurora Zone, more widely spaced drilling has encountered significant mineralization in areas distal to the resource, namely the Bonita zone, the Flamenco Zone and the Gemelos Zone.

To continue to define the mineralized potential of Filo Del Sol, an initial program of 35,000m of diamond drilling is recommended in order to accomplish 3 main objectives:

- Infill and short-range expansion drilling of the Aurora Zone
- Medium-range (1 – 2km) step out drilling to expand the Bonita Zone and determine if it, and other apparently satellite zones, are contiguous with the Aurora Zone, and
- Long-range (>2km) exploration drilling to test new target areas indicated by geology and surface sampling, primarily the Gemelos and Flamenco Zones
- This work is not contingent on any other work programs.

Data collected from this drilling should be used to create a comprehensive geological model incorporating lithology, alteration and mineral zonation which can be used to develop an updated mineral resource estimation with a goal of adding the sulphide material to the current oxide resource.

One of the key discoveries since 2019 is a zone of very high-grade material which occurs between 700 m and 1,000 m below surface. Grade variability within this zone indicates that it will likely need to be drilled at close spaced centres in order to be fully delineated and defined.

Given the technical challenges with completing this drilling from surface, an assessment of the viability of an underground drill drift should be completed which would allow this, and other areas of the Aurora zone, to be drilled from underground. As the project advances, underground access would also facilitate the recovery of bulk samples for metallurgical testwork.

The mineralization discovered by drilling since 2019 is primarily hypogene sulphide mineralization and will require processing by a crush/grind/float process rather than a leach process as described in the current study. Additional geometallurgical studies and metallurgical testwork are recommended in order to better understand the mineralogical distribution of ore minerals and develop a better understanding of the number, size and distribution of geometallurgical zones within the deposit.

Environmental base line studies and data collection should also continue to ensure a comprehensive and continuous record of data collection.

A summary of all major recommended works proposed to be completed along with the recommended budget totals is provided in Table 1-6, which totals $84.6M for all works as outlined in Section 26.

Depending on the results from this initial diamond drill program, subsequent drill programs may be required to achieve the level of understanding of the entirety of the mineralization required for evaluation of future development options.

Table 1-6: Filo del Sol Recommended Work Program Cost Estimate

Program Component	Cost Estimate (US$M)
Environment, Social and Governance	3.8
Land Holding Cost	1.2
Resource Drilling and Support	69.0
Project support logistics	7.1
Metallurgical and Engineering Studies	3.5
Total	**84.6**

2 INTRODUCTION

2.1 Terms of Reference

The Filo del Sol Project is an advanced stage polymetallic exploration project, which spans the border of Argentina and Chile, with mineral titles in both countries.

This Technical Report was prepared in order to summarize the results of the technical work that has been completed since publication of the previous technical report in 2019. Subsequent to that report, the Company has completed approximately 45,000m of additional drilling resulting in the discovery of a large zone of sulphide mineralization below and adjacent to the mineral resource. This work has enabled a much better understanding of the project geology and has highlighted the potential for a sulphide resource in addition to the currently defined oxide resource described in this report.

Geological modelling and data density are deemed to be insufficient to allow for the estimation of a resource, which incorporates the sulphide mineralization at this time; however, the 2019 oxide resource was updated with data from portions of the new drilling which intersected the block model limits. These limits remain unchanged from 2019.

All technical aspects of the 2019 Prefeasibility Study (PFS), which contemplates the mining and heap leach processing of the Filo del Sol deposit, remain unchanged. Cost estimates were updated in order to bring the economic analysis of the project to current status. Major cost categories (permanent equipment, material purchase, installation, subcontracts, indirect costs and owner's costs) were identified and analysed.

This report follows the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1. This Technical Report provides a mineral resource and classification of the mineral resource prepared in accordance with the CIM, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines, May 19, 2014 (CIM, 2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, November 19, 2019 (CIM, 2019).

2.2 Qualified Persons

The following Qualified Persons (QP) co-authored this technical report which is based on the PFS. These QPs have approved the information in this report that pertains to the sections of the PFS technical report that they are responsible for, summarized in Table 2-1.

Table 2-1: List of QPs and Areas of Responsibilities

Qualified Person	Company	Area(s) of Responsibility
Scott C. Elfen	Ausenco	1.16, 18.2, 18.8, 18.9, 18.10, 25.1.10, and 27
Kevin Murray	Ausenco	1.1, 1.11, 1.15, 1.16, 1.19, 1.20, 1.23, 2, 13, 17, 18.1, 18.3, 18.4, 18.5, 18.6, 18.7, 21 excluding 21.4.3, 25.1.9, 25.1.10, 25.1.13, 25.2.2, 25.2.5, 25.3.3, 25.3.4, 25.3.5, 25.3.7, and 27
Bruno Borntraeger	Knight Piésold	1.18, 3.2, 20, 25.1.12, 25.2.1, 25.2.6, and 27
Fionnuala A.M. Devine	Merlin Geosciences	1.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 1.9, 1.10,1.22,1.23,1.24,1.25, 3.1, 4, 5, 6, 7, 8, 9, 10, 11, 12, 23, 24, 26, 25.1.1, 25.1.2, 25.1.3, 25.1.4, 25.1.5, 25.1.6, and 27
Neil M. Winkelmann	SRK	1.17, 1.21, 3.3, 3.4, 19, 22, 25.1.11, 25.2.4, 25.3.2, and 27
James N. Gray	Advantage Geoservices Limited	1.12, 14, 25.1.7, 25.1.14, and 27
Ryan P. Brown	AGP Mining Consultants	1.14, 16, 21.4.3, 25.2.3, 25.2.8, 25.3.6, and 27
Gordon R. Zurowski	AGP Mining Consultants	1.13, 15, 25.1.8, 25.2.7, 25.3.1, and 27

Each of the individuals above are independent QP's for the purposes of NI 43-101. All scientific and technical information in this report in respect of the Filo del Sol project or the PFS is based on information prepared by or under the supervision of those individuals. The PFS Qualified Persons (QP), as defined by CIM, and their areas of responsibilities are summarized in Table 2-1.

2.3 Site Visits and Scope of Personal Inspection

2.3.1 Resource Statement Site Visits

For the purposes of the Resource Statement and in accordance with NI 43-101 guidelines, the following site visits were made:

Fionnuala Devine first visited the property in January 2014 as part of a two-day tour focused on the geology and 2013/2014 exploration program of the Filo del Sol system. Ms. Devine returned in January 2015 to lead the geological mapping program of the Filo del Sol area, which included 27 days on site, 17 days of which were spent on field traverses in the Filo del Sol area. She returned in February 2016 for several days of visits to review the ongoing surface geology work and again in February 2017 for three days to visit the Tamberías area, conduct revision mapping, and visit other exploration targets within the immediate Filo del Sol area. During the visits by Ms. Devine, attention was given to the treatment and validation of historical drilling data and included tours and description of sampling procedures. Additional visits to review core in April and December 2018 included work with the geology team to develop cross sections through the resource area. Her most recent visit was October 9-11, 2022, to review the most recent drilling from 2019 - 2022 and geological model updates.

James Gray visited the Copiapó office and core storage facility between 16th June 2014 and 21st June 2014. The project site was not visited by Mr. Gray.

2.3.2 2017 Site Visits

In accordance with NI 43-101 guidelines, the following site visits were made:

Neil Winkelmann visited the property in February 2017. A preliminary assessment of the overall regional and site logistics and suitability for the conceptual mining and processing plan was made. Two access routes were travelled, and the overall site topography was assessed. Some inspection of surface geotechnical conditions was available in road cuts. Exploration camps in Chile (Los Helados camp - currently idle) and Argentina (current Filo del Sol exploration camp) were visited. Some drill sample handling was witnessed, but not inspected nor audited in detail.

2.3.3 2018 Site Visits

Scott C. Elfen visited the site on February 3, 2018. Mr. Elfen visited to look at potential sites for the leach pads along with looking at the general site-wide geotechnical and geohazards conditions for the various mine facilities, except the pit.

Mr. Bruno Borntraeger, P.Eng., visited the site on March 22, 2018.

2.3.4 2022 Site Visits

Fionnuala Devine visited the project from October 9-11, 2022. Time was spent at the geology and core logging facility in Rodeo, Argentina to engage with company personnel and review the most recent drilling from 2019 – 2022 as well as the most recent geological model updates. Verification samples were collected, and results are reported in Section 12.

2.3.5 2023 Site Visits

Mr. Zurowski visited the site in January 2023 to inspect the selected drill core from the ongoing drill program and the terrain in the vicinity of the proposed pits and possible waste dump locations.

Mr. Elfen visited the site in January 2023 to look at potential sites for the leach pads along with looking at the general site- wide geotechnical and geohazards conditions for the various mine facilities, except the pit.

Table 2-2: QP Site Visits

Qualified Person	Company	Date(s) of Site Visit
Scott C. Elfen	Ausenco	February 2018, January 2023
Kevin Murray	Ausenco	Did not visit the property.
Bruno Borntraeger	Knight Piésold	March 2018
Fionnuala A.M. Devine	Merlin Geosciences	January 2014, January 2015, February 2016, February 2017, April 2018, September 2019, December 2019, October 2022
Neil M. Winkelmann	SRK Consulting Inc.	February 2017
James N. Gray	Advantage Geoservices Limited	Did not visit the property
Ryan P. Brown	AGP Mining Consultants Inc.	Did not visit the property
Gordon R. Zurowski	AGP Mining Consultants Inc.	January 2023

2.4 Effective Dates

Mineral Resources have an effective date of January 18, 2023.

Mineral Reserves have an effective date of February 28, 2023.

The overall effective date of this PFS report is taken to be February 28, 2023.

2.5 Information Sources and References

The key information sources for the Report included previous technical reports and documents as listed in Section 2.6 (Previous Technical Reports) and Section 27 (References).

Additional information was sourced from Filo Mining personnel where required.

2.6 Previous Technical Reports

The Filo del Sol project has been the subject of previous technical reports, as summarized in Table 2-3.

Table 2-3: Summary of Previous Technical Reports

Reference	Company	Name
SRK Consulting (Canada) Inc., December 18, 2017	Filo Mining Corp.	Independent Technical Report for a Preliminary Economic Assessment on the Filo del Sol Project
Ausenco Engineering Canada, Merlin Geosciences Inc, Advantage Geoservices Ltd., and Knight Piésold Ltd., February 22, 2019	Filo Mining Corp.	NI 43-101 Technical Report, Prefeasibility Study for the Filo del Sol Project

2.7 Reporting Standards

All currency is expressed in US dollars unless specifically noted otherwise.

2.8 Definitions

A list of unit abbreviations and acronyms is provided in Table 2-4 and Table 2-5.

Table 2-4: Unit Abbreviations

Abbreviation	Description
%	percent
%w/w	dry weight concentration of a solution
°C	degrees Celsius
μ	micro
μm	micrometre
3D	Three-Dimensional
B	Billion
C$	Canadian dollars

Abbreviation	Description
cm	centimetre
d/a	days/year
dBA	decibels A
g	gram
g/cc	grams per cubic centimetre
g/cm^3	grams per centimetre cubed
g/L	grams per litre
g/t	grams per tonne
ha	hectare
HP	horsepower
hr	hour
kg	kilogram
kg/t	kilograms/tonne
km	kilometre
koz	thousand ounces
kPa	Kilo pascal
kt	kilo tonnes
kt/d	thousand tonnes per day
kV	kilovolt
kWh	Kilowatt hour
kWh/t	kilowatt hour/tonne
L/h/m^2	litres/hours/square metres
L/s	litre per second
lbs	pounds
M	million
m	metre
m^2	square metre
m^3	cubic metre
mamsl	metres above mean sea level
masl	metres above sea level
mg/L	milligrams per litre
mm	millimetres
Mm^2	Million square metres
Mt	million tonnes
Mt/a	million tonnes per annum
mV/V	millivolts per volt
MW	Megawatt
MWh	Megawatt hour
oz	ounce

Abbreviation	Description
P_{80}	Passing grind size
ppb	parts per billion
ppm	parts per million
t	metric tonne
t/d	tonnes per day
$t/m^2/h$	tonnes per metre squared per hour
US$	United States dollars
X	times

Table 2-5: Acronyms and Abbreviations

Abbreviation	Name
AAS	Atomic absorption spectrometry
ABA	Acid base Accounting
ACME	ACME laboratories
AgCl	Silver Chloride
AGP	AGP Mining Consultants Inc.
AGP	Silver
Al	Aluminum
ALS	ALS Limited
AP	Acid potential
AP	Accounts Payable
AR	Accounts Receivable
As	Arsenic
Au	Gold
AuOx	Gold only oxide zone
BGC	BGC Engineering Inc
Bi	Bismuth
BWI	Ball mill work index
CaO	Calcium oxide
CN	Cyanide
CNWAD	Cyanide Weak Acid Dissociable
Cu	Copper
Cu2S	Copper Sulphide
CuAs	acid soluble copper
CuCN	cyanide soluble copper
CuEq	Copper Equivalent
CuRES	residual copper
CWI	Crusher work index
DD	Diamond Drilling

Abbreviation	Name
DGA	Water Management Division (Dirección General de Aguas)
DIA	Environmental Impact Statement (Declaración de Impacto Ambiental)
EIA	Environmental Impact Study (Estudio de Impacto Ambiental)
EIA	Environmental Impact Assessment
ENSO	El Nino-Southern Oscillation
EPCM	Engineering, procurement, construction management
EW	Electrowinning
Fe	Iron
GARD	Global Acid Rock Drainage
H2SO4	Sulphuric Acid
HDPE	high density polyethylene
Hg	Mercury
HLFs	heap leach facilities
HYPO	hypogene zone
ICE	Consolidated Assessment Report (Informe Consolidado de Evaluación)
ICP-AES	Inductively coupled plasma atomic emission spectroscopy
ICSARA	Informe Consolidado de Solicitud de Aclaraciones, Rectificaciones y Ampliaciones
ID	Inverse Distance
IFC	International Finance Corporation’s Sustainability
ILO	International Labour Organization
INTA	Instituto Nacional de Tecnología Agropecuaria
IP	Induced Polarized Survey
IRR	Internal Rate of Return
ISO	International Organization for Standardization
KP	Knight Piésold
LG	low grade
LiDAR	Light Detection and ranging
LLDPE	low liner density polyethylene
LOM	Life of Mine
MAG	Magnetic Geophysical survey
MCC	motor control centres
MMUs	mobile manufacturing units
Mo	Molybdenum
MWMP	Meteoric Water Mobility Procedure
NaCN	Sodium Cyanide
NN	Nearest Neighbour
NNP	Net Neutralizing potential
NPR	Neutralization potential r5atio
NPV	Net Present Value

Abbreviation	Name
NPV	neutralization potential
NR40	National Route 40
NSR	net smelter return
NVPT	net value per tonne
ORP	Oxidation reduction potential
OX	oxide
PAG	potentially acid generation
PCS	Process control system
PEA	Preliminary Economic Assessment
PET	Potential Evapotranspiration
PFS	Prefeasibility Study
PLS	Pregnant leach solution
QA	Quality Assurance
QC	Quality Control
QEMSCAN	Quantitative Evaluation of Minerals by Scanning Electron Microscope
QP	Qualified Persons
RC	Reverse Circulation
RCA	Environmental Qualification Ordinance (Resolución Calificación Ambiental)
RF	Revenue factor
ROM	Run of Mine
RWI	Rod mill work index
SART	Sulphidization, acidification, recycle and thickening process
Sb	Antimony
SCC	Standard Council of Canada
SEA	Environmental Assessment Service (Servicio de Evaluación Ambiental)
SERNAGEOMIN	National Geology and Mining Service (Servicio Nacional de Geología y Minería)
SFE	Shake Flask Extraction
SMN	National Meteorological Service (Servicio Meteorologico Nacional)
SX	Solvent Extractant
TARP	trigger action response plan
TC/RC	Treatment charge/Refining Charge
TIC	total organic carbon

3 RELIANCE ON OTHER EXPERTS

The QPs have relied upon the following other expert reports, which provided information regarding mineral rights, surface rights, property agreements, royalties, and taxation as noted below.

3.1 Ownership, Mineral Tenure, and Surface Rights

The QPs have not independently reviewed ownership of the Project area and the underlying property agreements. The QPs have also not independently reviewed the Project mineral tenure and the overlying surface rights. The QPs have fully relied upon, and disclaim responsibility for, information derived from Filo Mining staff and legal experts retained by Filo Mining for this information through the following documents:

- Title Opinion letter from Bofill Mir Abogados addressed to Filo Mining Corp. February 17, 2023.
- Title Opinion letter from Randle Legal addressed to Filo Mining Corp. February 7, 2023.

This information is used in Section 4 of the Report and in support of the Mineral Resource estimate in Section 14 and the financial analysis in Section 22.

3.2 Environmental, Permitting, and Social

The QPs have reviewed the Project environmental, permitting and social information including, but not limited to, the following:

- BGC Engineering, 2013. Proyectos de Exploraciones Minera Vicuña: Los Helados, Josemaría y Filo del Sol: Estudio Glacial y Periglacial. Informe Final. Report prepared for MFDO y DEPROMINSA, March 2013.
- BGC Engineering, 2015a: Los Helados, Josemaría, and Filo del Sol – Cryology Summary: report prepared for NGEx, October 2015.
- Bethsabe Manzanares, 2015: Resumen Ejecutivo Estudios Para la Linea Base Ambiental Proyecto Josemaría: report prepared for NGEx by Asesoría Ambiental, October 2015.

This information is used in Section 20 of the Report and in support of the Mineral Resource estimate in Section 14 and the financial analysis in Section 22.

3.3 Taxation

The QPs have not independently reviewed the Project taxation position. The QPs have fully relied upon, and disclaim responsibility for, taxation information derived from publicly available information and through the normal course of Business conducted by Filo Mining in the relevant jurisdictions.

This information is used in Section 22 of the Report.

3.4 Markets

The QPs have not independently reviewed the market studies, pricing, or contract information. The QPs have fully relied upon and disclaim responsibility for, commodity price projections derived from publicly available information.

Metal price and exchange rate forecasting is a specialized business requiring knowledge of supply and demand, economic activity and other factors that are highly specialized and requires an extensive global database that is outside of the purview of the QP. The QPs consider it reasonable to rely upon the experts for metal prices and exchange rate forecasts.

The information is used in Section 19 of the Report. The information is also used in support of the Economic Analysis in Section 22.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

The Filo del Sol Project is located 140 km southeast of the city of Copiapó, Chile and straddles the border between Argentina and Chile. The centre of the main deposit area is located at 28.49° S and 69.66° W (decimal degrees, WGS84 datum).

The Filo del Sol property is comprised of mineral titles in both Chile and Argentina. Those in Argentina are controlled by Filo del Sol Exploración S.A. and are referred to as the Filo del Sol Property; those in Chile are controlled by Frontera Chile Limitada and are referred to as the Tamberías Property. Both Filo del Sol Exploración S.A. and Frontera Chile Limitada are wholly-owned subsidiaries of Filo Mining Corp.

The total area of the combined properties is 13,575 ha. This area does not match the sum of the individual claim areas (17,935 ha) for three reasons: i) the border between Chile and Argentina is not completely defined in this area; ii) the border between San Juan Province and La Rioja Province in Argentina is not completely defined; and iii) the Company controls two “pisos,” or layers of claims, which overlap in Chile.

The mineral resource reported here which comprises the Filo del Sol deposit lies within the Caballo I claim in Argentina and the Tronco 1 1/41 and Tronco 2 1/76 claims in Chile.

There are no significant factors or risks that may affect access, title or the right or ability to perform work on the property other than those described below, and all annual property payments are up to date. There are no known environmental liabilities on the property, other than general reclamation considerations in the event that exploration was completed, and the project was abandoned.

4.2 Mining Integration and Complementation Treaty Between Chile and Argentina

On 29th December 1997, Chile and Argentina signed the "Tratado entre la República de Chile y la República Argentina sobre Integración y Complementación Minera" (Mining Integration and Complementation Treaty between Chile and Argentina; or the Treaty), in an effort to strengthen their historic bonds of peace and friendship and intensify the integration of their mining activities.

The treaty provides a legal framework to facilitate the development of mining projects located in the border area of both countries. The treaty objective is to facilitate the exploration and exploitation of mining projects within the area of the treaty.

On 20th August 1999, Chile and Argentina subscribed to the Complementary Protocol and on 18th July 2001, an Administrative Commission was created.

Additional protocols have been signed between Chile and Argentina, which provide more detailed regulations applicable to specific mining projects.

One of these protocols, and the first granted for exploration purposes, is Filo Mining´s “Proyecto de Prospección Minera Vicuña” (Vicuña Mining Prospection Project), dated 6th January 2006. This protocol allows for prospecting and exploration activities in the Filo del Sol Project area. The main benefit of the Vicuña Additional Protocol during the exploration stage is the authorization which allows for people and equipment to freely cross the international border in support of exploration and prospecting activities within an area defined as an “operational area.” Development of transboundary projects is the specific objective of the Treaty.

4.3 Properties in Argentina

In Argentina, mineral rights are acquired by application to the government through a system based entirely on paper staking. A mineral property may go through several stages of classification during its lifetime. This begins with a Cateo (exploration permit). Once an application for a Cateo has been made, any mineral discoveries made by third parties belong to the Cateo applicant. A Cateo consists of one to 20 units, each unit being 500 ha. A fee, calculated per ha, is required within five days of the Cateo’s approval. The term of a Cateo, the length of which varies based on size, begins 30 days after approval. A Cateo of one unit has a duration of 150 days and for each additional unit its duration is increased by an additional 50 days. An additional requirement is that larger Cateos must reduce in size at certain times. At 300 days after approval, half of the area in excess of four units must be relinquished. At 700 days after approval, half of the remaining area must be relinquished.

To move to the next stage the Cateo holder must apply within the term of the Cateo by reporting a mineral discovery. Upon approval, this will result in a Manifestacion de Descubrimiento or mining rights for an area up to 3,000 ha. This area is comprised of mining units, with one mining unit being 100 ha in the case of a disseminated deposit unit and 6 ha in the case of a vein deposit unit. Once this is approved, the holder may conduct a Mensura or legal survey to apply for a Mina or mining lease. The property will generally stay in the Manifestacion stage until a mineral resource has been defined.

An annual exploration fee due to the Province of San Juan is proportional to the mining units covered by each mina. These fees were increased by the Argentine government as of the first semester of 2015. There is no expiry date for a mina if the annual payments are made. All tenures listed in Table 4-1 are currently up to date. Each disseminated deposit mining unit covers 100 ha and costs ARS 3,200 per annum and each vein deposit mining unit covers 6 ha and costs ARS 320 per annum. The total fees are shown in Table 4-1.

Table 4-1: Manifestaciones Owned – Argentina

Concession	File Number	Area (ha)	Mining Units	Annual Fee (ARS)
Caballo I	520-0323-C-99	451*	5	16,000
Caballo II**	520-0324-C-99	76*	13	4,160
Vicuña 1	520-0099-C-98	1,439*	15	48,000
Vicuña 2	520-0100-C-98	1,483*	15	48,000
Vicuña 5	425-247-B-00	1,500	15	48,000
Vicuña 6	414-145-C-04	1,504	15	48,000
Vicuña 7	1124-029-C-09	1,324	15	48,000
Vicuña 8	1124-286-F-14	1,488	15	48,000

Notes:
* Area uncertain due to undefined National or Provincial boundary.
** Caballo II is comprised of vein deposit mining units.

The Argentine Mining Code also requires the presentation of a plan of investment for each Mina. The plan of investment contemplates a minimum expenditure of 300 times the annual fee and should be accomplished within five years following the request from the government. No request from the government has been made with respect to any of the Minas.

4.3.1 Surface Rights

The properties of Filo del Sol Exploración S.A. are located in the Iglesias Department of the Province of San Juan, in the area called “Cerro el Potro” within the “Usos Múltiples” (“Multiple Uses”) Area of the San Guillermo Provincial Reserve, where mining activities are fully authorized. The owner is the Provincial State.

On April 15, 2021, four claimants, collectively the Lancaster Group, filed an opposition in certain Filo del Sol Project mining dockets, allegedly based on their capacity as owners of the “Los Tres Mogotes” ranch; however, the Lancaster Group has not registered the surface land on the Real Estate Registry of the Province of San Juan and there is no legal evidence of their ownership. If the Lancaster Group were able to provide evidence of ownership of the land near where the Filo del Sol project is located, it would be likely that the Administrative Court of Mines would uphold their right to compensation for the time not covered by the statute of limitations and to grant the appropriate bond.

4.3.2 Environmental Permits

- Caballo I and Caballo II: approved exploration EIR and evaluation of 4th update.
- Vicuña 1, Vicuña 2, Vicuña 5, Vicuña 6, Vicuña 7 and Vicuña 8: approved exploration EIR and evaluation of 4th update.
- No additional environmental permits are required to carry out the recommended work program, with the exception of permits required for a contingent underground decline.

4.4 Properties in Chile

Chile’s mining policy is based on legal provisions that were enacted as part of the 1980 constitution. According to the law, the state owns all mineral resources, but exploration and exploitation of these resources by private parties is permitted through mining concessions, which are granted by the courts.

4.4.1 Mineral Tenure

The concessions have both rights and obligations as defined by a Constitutional Organic Law (enacted in 1982). Concessions can be mortgaged or transferred, and the holder has full ownership rights and is entitled to obtain the rights of way for exploration (pedimentos) and exploitation (mensuras). In addition, the concession holder has the right to defend ownership of the concession against state and third parties. A concession is obtained by a claims filing and includes all minerals that may exist within its area. Mining rights in Chile are acquired in the following stages.

4.4.1.1 Pedimento

A pedimento is an initial exploration claim whose position is well defined by UTM coordinates, which define north-south and east-west boundaries. The minimum size of a pedimento is 100 ha and the maximum is 5,000 ha with a maximum length-to-width ratio of 5:1.

The duration of validity is for a maximum period of two years; however, at the end of this period, and provided that no overlying claim has been staked, the claim may be reduced in size by at least 50% and renewed for an additional two years. If the yearly claim taxes are not paid on a pedimento, the claim can be restored to good standing by paying double the annual claim tax the following year.

New pedimentos are allowed to overlap with pre-existing ones; however, the underlying (previously staked) claim always takes precedent, providing the claim holder avoids letting the claim lapse due to a lack of required payments, corrects any minor filing errors, and converts the pedimento to a manifestacion within the initial two-year period.

4.4.1.2 Manifestacion

Before a pedimento expires, or at any stage during its two-year life, it may be converted to a manifestacion or exploration concession. Within 220 days of filing a manifestacion, the applicant must file a “Request for Survey” (Solicitud de Mensura) with the court of jurisdiction, including official publication to advise the surrounding claim holders, who may raise objections if they believe their pre-established rights are being encroached upon. A manifestation may also be filed on any open ground without going through the pedimento filing process.

The owner is entitled to explore and to remove materials for study only (i.e., sale of the extracted material is forbidden). If an owner sells material from a manifestation or exploration concession, the concession will be terminated.

4.4.1.3 Mensura

Within nine months of the approval of the “Request for Survey” by the court, a government licensed surveyor must survey the claim. Surrounding claim owners may be present during the survey. Once surveyed, presented to the court, and reviewed by the National Mining Service (Sernageomin), the application is adjudicated by the court as a permanent property right (a mensura), which is equivalent to a “patented claim” or exploitation right. Exploitation concessions are valid indefinitely and are subject to the payment of annual fees. Once an exploitation concession has been granted, the owner can remove materials for sale.

4.4.2 Tamberías Properties

Frontera Chile Limitada is the owner of 12 granted Exploration Mining Concessions, four Exploration Mining Concessions in the process of being granted, three Exploitation Mining Concessions, and one unilateral and irrevocable option agreement to purchase 17 Exploitation Mining Concessions, hereinafter the “Properties” that form the Project. These properties are listed in Table 4-3, Table 4-4, and Table 4-5 and shown in Figure 4-1.

Table 4-2: Exploration Mining Concessions Granted – Chile

Concession's Name	ID Number	Hectares	Expiration Date
Frontera IV 5	03203G199	300	March 27, 2023
Tambería III 1	03203G612	300	August 26, 2023
Tambería III 2	03203G578	300	July 26, 2023
Tambería III 3	03203G587	300	July 30, 2023
Tambería III 4	03203G600	300	August 18, 2023
Tambería III 5	03203G613	300	August 26, 2023
Tambería III 6	03203G579	300	July 26, 2023
Tambería III 7	03203G588	100	July 30, 2023
Tambería III 8	03203G601	300	August 18, 2023
Tambería III 9	03203G619	300	July 30, 2023
Tambería III 10	03203G580	300	January 28, 2024
Tambería III 11	03203G589	300	July 30, 2023

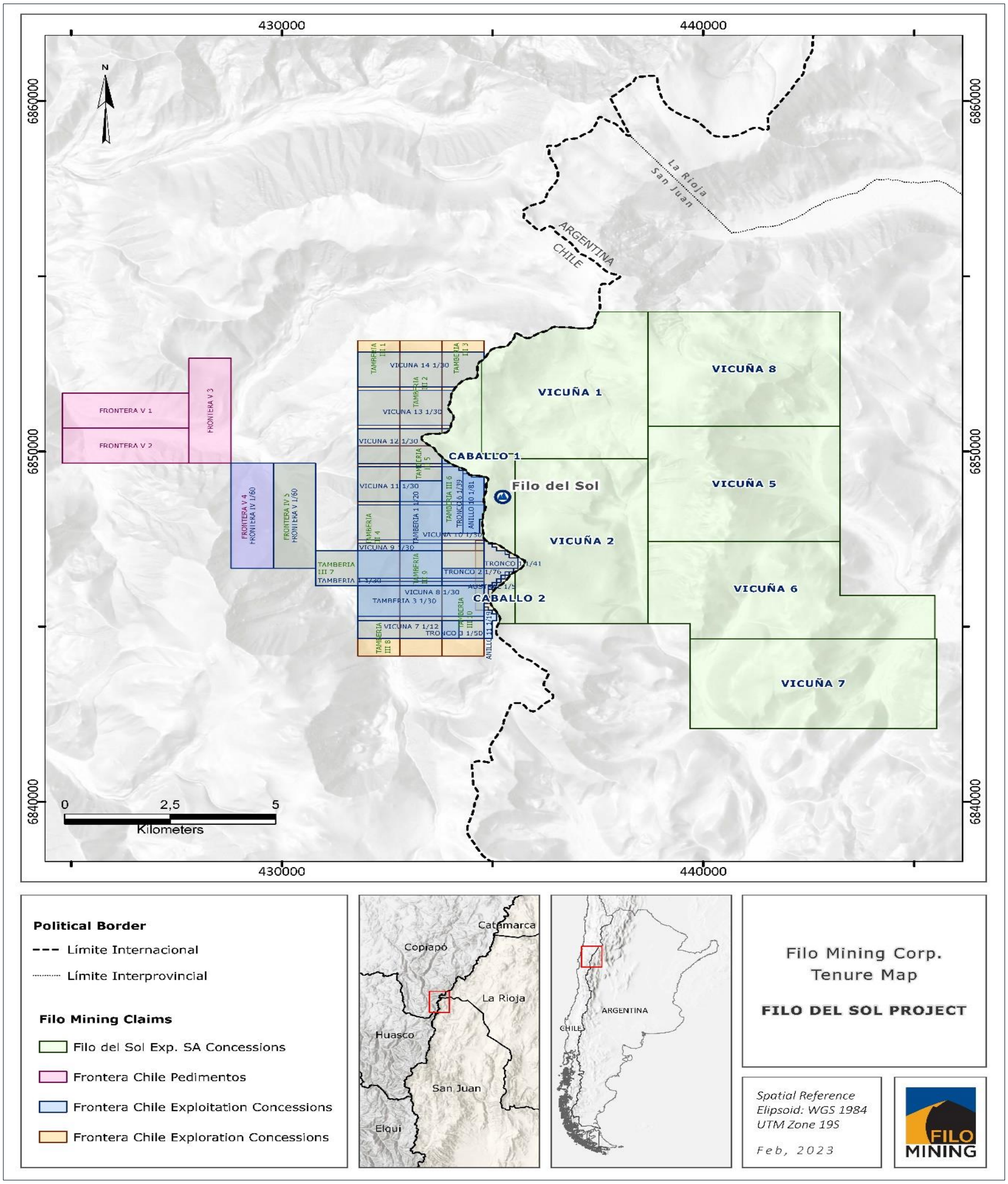
Table 4-3: Exploration Mining Concessions in the Process of Being Granted

Concession's Name	ID Number	Hectares	Expiration Date
Frontera V 1	03203F296	300	N/A
Frontera V 2	03203F287	300	N/A
Frontera V 3	03203F292	300	N/A
Frontera V 4	03203F291	300	N/A

Table 4-4: Exploitation Mining Concession Granted

Concession's Name	ID Number	Hectares	Expiration Date
Frontera IV 1/60	032037278	300	N/A
Frontera V1/60	032037279	300	N/A
Austral 1/5	032034757	5	N/A

Figure 4-1: Mineral Titles



Source: Filo, 2023

4.4.3 Unilateral and Irrevocable Option Agreement

By public deed dated 25th March 2011 before the Santiago Notary Public of Antonieta Mendoza Escalas, Compañía Minera Tamberías SCM granted to Sociedad Contractual Minera Frontera del Oro SpA a unilateral and irrevocable option to purchase the mensuras shown in Table 4-5 (the "Option Agreement").

Table 4-5: Exploitation Mining Concessions (Mensuras) Under Option – Chile

Concession's Name	ID Number	Hectares	Expiration Date
Vicuna 8 1/30	032032884	300	N/A
Vicuna 10 1/30	032032886	300	N/A
Vicuna 11 1/30	032032887	300	N/A
Vicuna 13 1/30	032032888	300	N/A
Vicuna 7 1/12	032032881	120	N/A
Vicuna 9 1/30	032032885	300	N/A
Vicuna 12 1/30	032032882	300	N/A
Vicuna 14 1/30	032032889	300	N/A
Tronco 1 1/41	032034145	41	N/A
Tronco 2 1/76	032034146	76	N/A
Tronco 3 1/50	032034147	50	N/A
Tamberia 3 1/30	032034048	300	N/A
Tamberia 1 1/30	032034047	300	N/A
Tamberia 1 1/20	032034046	200	N/A
Tronco 6 1/39	032034193	178	N/A
Anillo 10 1/81	032034351	81	N/A
Anillo 11 1/19	032034352	19	N/A

By public deed dated 27th July 2012 before the Santiago Notary Public of Antonieta Mendoza Escalas, Minera Frontera del Oro SpA assigned the Option Agreement to Frontera Chile Limitada. Frontera may exercise the Option Agreements within the period that ends on 30th June 2026. The purchase price of the Option Agreement is $20,000,000, to be paid in installments during the term of the Option Agreement, and a royalty of 1.5% of the Net Smelter Return. There are no work commitments. To date, $5,250,000 of the total has been paid.

4.4.4 Surface Rights

The majority of the surface land rights in the area of the Tamberías Property are held by a local community, "Comunidad Civil Ex Estancia Pulido," with one small area owned by a different landowner. Filo Mining has an agreement with both landowners to provide access to the project for a period of four years, beginning on November 30, 2021.

4.4.5 Environmental Permits

By resolution No. 192, dated 2nd September 2013, the Servicio de Evaluación Ambiental of the III Region approved the Environmental Impact Declaration (DIA) presented by Frontera for the exploration of the Tamberías Project. According to this resolution, Frontera is authorized to develop four exploration campaigns including an aggregate number of 200 drill holes. No additional permits are required to carry out the recommended work program.

4.5 Water Rights

Water rights in Argentina are owned by the Province of San Juan. In Chile, water rights are privately held, and Filo has an agreement in place to purchase water from the local owner of the water rights. Filo has permits to use water sufficient to maintain the drilling program described in Section 26 of this report.

4.6 Royalties and Encumbrances

Argentinian royalties were estimated at 3% of “mine head revenue” which is defined as net revenue minus all operating costs other than mining costs. Chilean royalties were estimated based on a private 1.5% NSR royalty applicable after recovery of costs by the owner. This cost recovery was estimated to take 3 years of production (estimated on a whole-of project basis), and the royalty was applied thereafter.

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

5.1 Access

The Project is accessible by road from either Copiapó, Chile (140 km northwest) or San Juan, Argentina (360 km south-southeast); as shown in Figure 5-1. Access from Copiapó (Chile) is via the C-35 sealed road in a southeasterly direction through the towns of Tierra Amarilla and Punta del Cobre, along the Copiapó River valley through the small villages of Pabellon, Los Loros, La Guardia and Iglesia Colorada. Past these villages, the road becomes the C-453 and continues towards the El Potro bridge for approximately 130 km. Crossing the bridge, a gravel road leads the final 50 km to the drill sites. The total driving time from Copiapó to the project site is approximately four hours. Access by this route is generally possible from November to April.

The Project can also be accessed from the City of San Juan, Argentina. This route runs northwards via National Route 40 (NR40), through the town of San Jose de Jachal to Guandacol. From Guandacol, travel is along approximately 240 km of gravel road toward the northwest, across Las Juntas, Zapallar, Las Cuevas, Salina de Leoncito and Cuesta de La Brea, to the Project. Access via this route takes approximately nine to ten hours.

Bi-national access to the Project is provided through the Mining Integration and Complementation Treaty between Chile and Argentina. This treaty allows personnel and equipment to access the Filo del Sol area from Chile or Argentina, providing that they also return to the country they entered from and do not cross out of the Treaty area into either Argentina or Chile respectively.

5.2 Climate

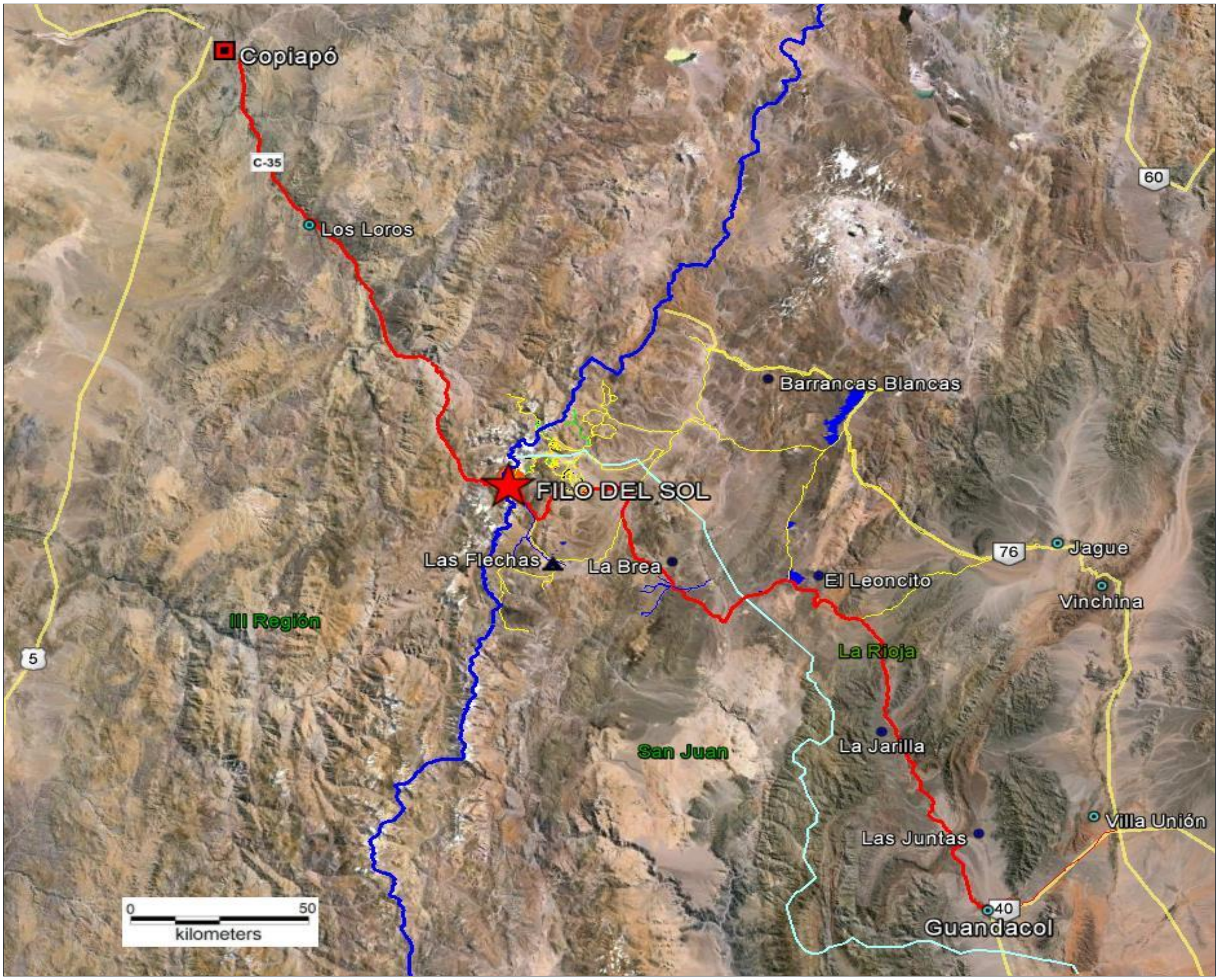
The climate is frequently cold and windy, typical of the high Andes. The exploration field season can run year round; however, during winter it is common to encounter severe operating conditions and continuous operation requires the presence of snow removal equipment to manage sudden snowfalls. Year-round access to the project is maintained with snow clearing and removal equipment utilized for road clearing and maintenance. High wind and low temperatures reduce productivity during the winter months.

Conditions for the mining operations for the Project would be comparable to those at the El Indio, Veladero, and Refugio Mines.

5.3 Local Resources and Infrastructure

Filo del Sol is a relatively new exploration development with minimal infrastructure present at site. Field staff are based out of the Batidero camp, owned and operated by Lundin Mining, with whom Filo Mining Corporation have an ongoing camp use agreement. Batidero Camp is located approximately at an altitude of 4,000 m approximately 25 km by road from the Project in Argentina. The Batidero camp can accommodate over 1,000 people with approximately 250 to 300 Filo Mining Corporation staff based there during the active field season. Facilities at the Filo del Sol worksite are remote, and no infrastructure is available other than road access.

Figure 5-1: Access to the Project from Chile and Argentina



Source: Devine et al., 2017

5.4 Physiography

Filo del Sol is in the high Andes straddling the Chile-Argentina border with the deposit centred at latitude 28°28'52.28"S and Longitude 69°39'24.90"W. Elevations on the property range from 4,500 masl at the Valley bottom to 5,500 masl at the Chile-Argentina Border. The topography is mountainous with moderately steep slopes leading to rounded ridges and peaks with varying steepness. Vehicle access with a suitable 4-wheel drive vehicle is possible to most of the property. Vegetation is generally absent in the area.

The site is situated in a glacial and periglacial belt that is characterized by permafrost and various cryoforms such as glaciers and rock glaciers.

5.5 Seismicity

The project lies in an active seismic zone, and although no specific seismic studies have been conducted, historically, seismic activity is relatively common in the area. Earthquakes are associated with the Nazca plate being subducted under the South American continental plate.

6 HISTORY

6.1 Regional History

Cyprus-Amax was the first company to conduct extensive exploration work in the area beginning in 1997, based on recognition of auriferous silica and a Cu-Au porphyry occurrence on the Chilean side of the border (now the Tamberías part of the deposit). Cyprus–Amax's work during the 1998/1999 season consisted of 1:10,000 geologic mapping, talus fine sampling, rock chip sampling, road construction from near the El Potro bridge to their camp, and from the camp to Cerro Vicuña, and a drill program of 2,519 m in 16 reverse circulation (RC) drill holes. The drilling discovered high-grade copper oxide and moderate-grade gold values, including 40 m at 1.19% Cu and 0.33 g/t Au in RCVI-02 and 20 m at 0.66% Cu and 0.63 g/t Au in RCVI-07. All holes ended in mineralization. Talus fine sampling detected a strong gold anomaly in the eastern portion of the alteration halo, associated with a large, silicified cap (Cerro Vicuña), which they did not drill. Upon discovering this feature, and losing interest in the copper potential, Cyprus-Amax decided to take on a partner to explore the gold potential. Cyprus-Amax spent approximately $800,000 USD on the property.

Filo Mining Corp. became involved in the Project through its predecessor company, Tenke Mining Corp., which negotiated a purchase arrangement with Cyprus-Amax in August 1999. Tenke operated from 1999 to 2007 and subsequent field seasons were carried out by Filo Mining's predecessor companies, Suramina and NGEx Resources. The first season of field work for Filo Mining was 2016/2017.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

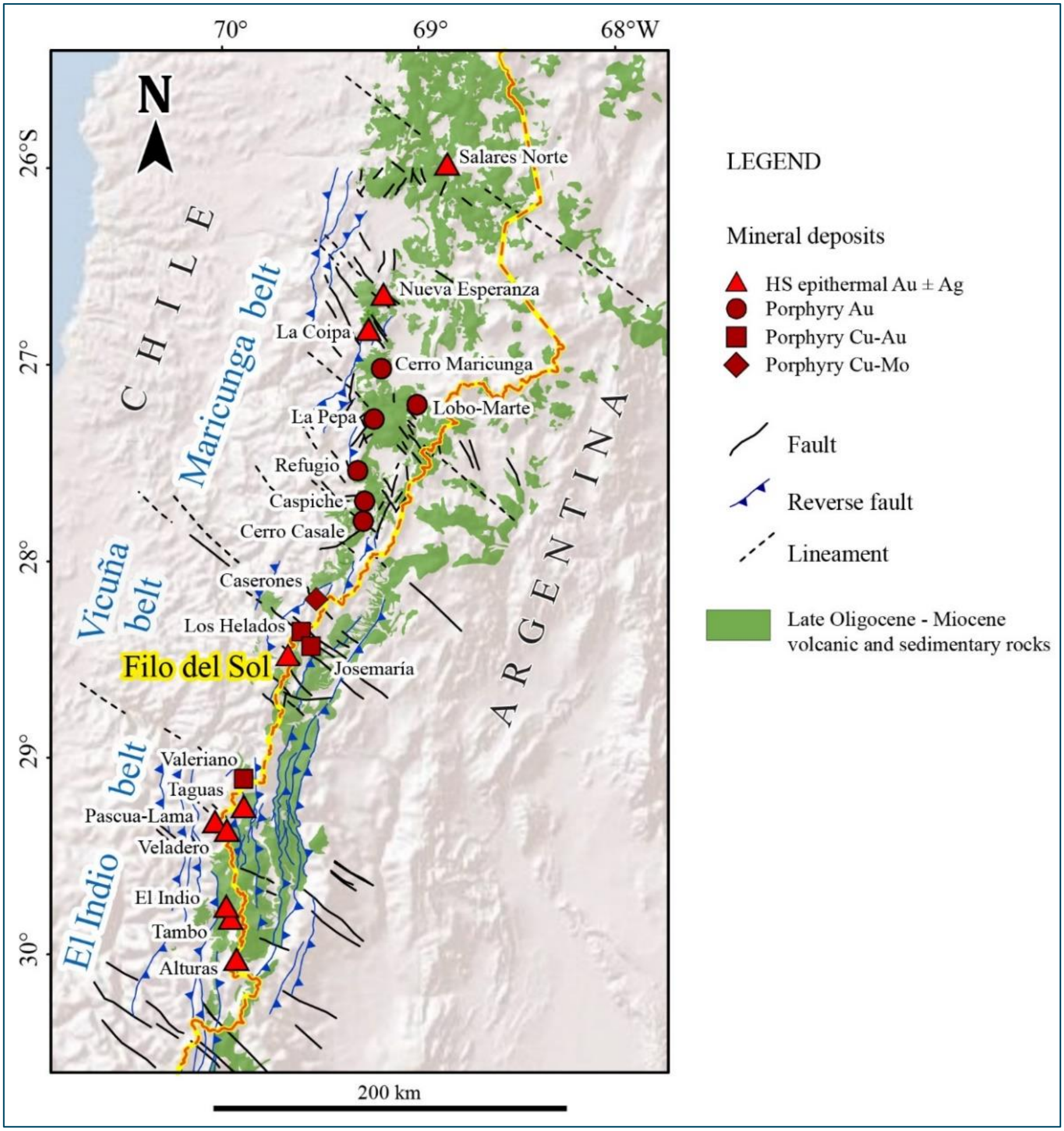
The Filo del Sol deposit is part of the Vicuña belt, a metallogenic belt of porphyry and high-sulphidation epithermal deposits that lies within the Frontal Cordillera at the northern limit of the Chilean-Pampean flat-slab segment of the southern Central Andes (~27°30´–33°S) (Jordan et al., 1983; Cahill and Isacks, 1992). It is located roughly midway between the porphyry and high-sulphidation epithermal deposits of the Maricunga and El Indio metallogenic belts (Vila and Sillitoe, 1991; Sillitoe et al., 1991; Bissig et al., 2001), thereby making the effective metallogenic connection between them (Figure 7-1). At this latitude, the Frontal Cordillera is dominated by large, north-to-northeast-trending, basement-cored, fault-bounded blocks of predominantly Permian to Triassic plutonic and volcanic rocks, with localized, structurally controlled and partially preserved Mesozoic and Cenozoic sedimentary and volcanic cover sequences (Martínez et al., 2015; Perelló et al., in prep.). The Permo-Triassic volcano-plutonic assemblages are part of the Choiyoi Magmatic Province (Kay et al., 1989). The principal, basement block-bounding reverse faults responsible for regional, wholesale uplift of the Frontal Cordillera were active during the early to the middle Miocene (~21–13 Ma) (Perelló et al., in prep.). Magmatism and related hydrothermal alteration and mineralization in the Vicuña belt took place discontinuously from the late Oligocene to the middle Miocene (25–14 Ma) (Perelló et al., 2003; Yoshie et al., 2015; Devine et al., 2019; Sillitoe et al., 2019), while in the Filo del Sol area magmatism, alteration, mineralization, tectonic uplift and denudation were active during the middle Miocene, from 16 to 14 Ma (Devine et al., 2019; Perelló et al., in prep.).

7.2 Project Geology

The Filo del Sol alignment is an ~8-km-long, north- to northeast-trending series of prospects of porphyry Cu-Au, Cu-Mo, and related epithermal Cu-Ag-Au mineralization. The area is highlighted in satellite images by the remnants of a once much larger advanced argillic alteration lithocap, the main part of which corresponds to the present-day surface expression of the Filo del Sol porphyry and high-sulphidation system (Figure 7-2). The lithocap occurs predominantly in Cretaceous siliciclastic, Permian felsic volcanic, Triassic monzogranite, and middle Miocene porphyry dikes and related hydrothermal breccias. Zones of steam-heated alteration are common.

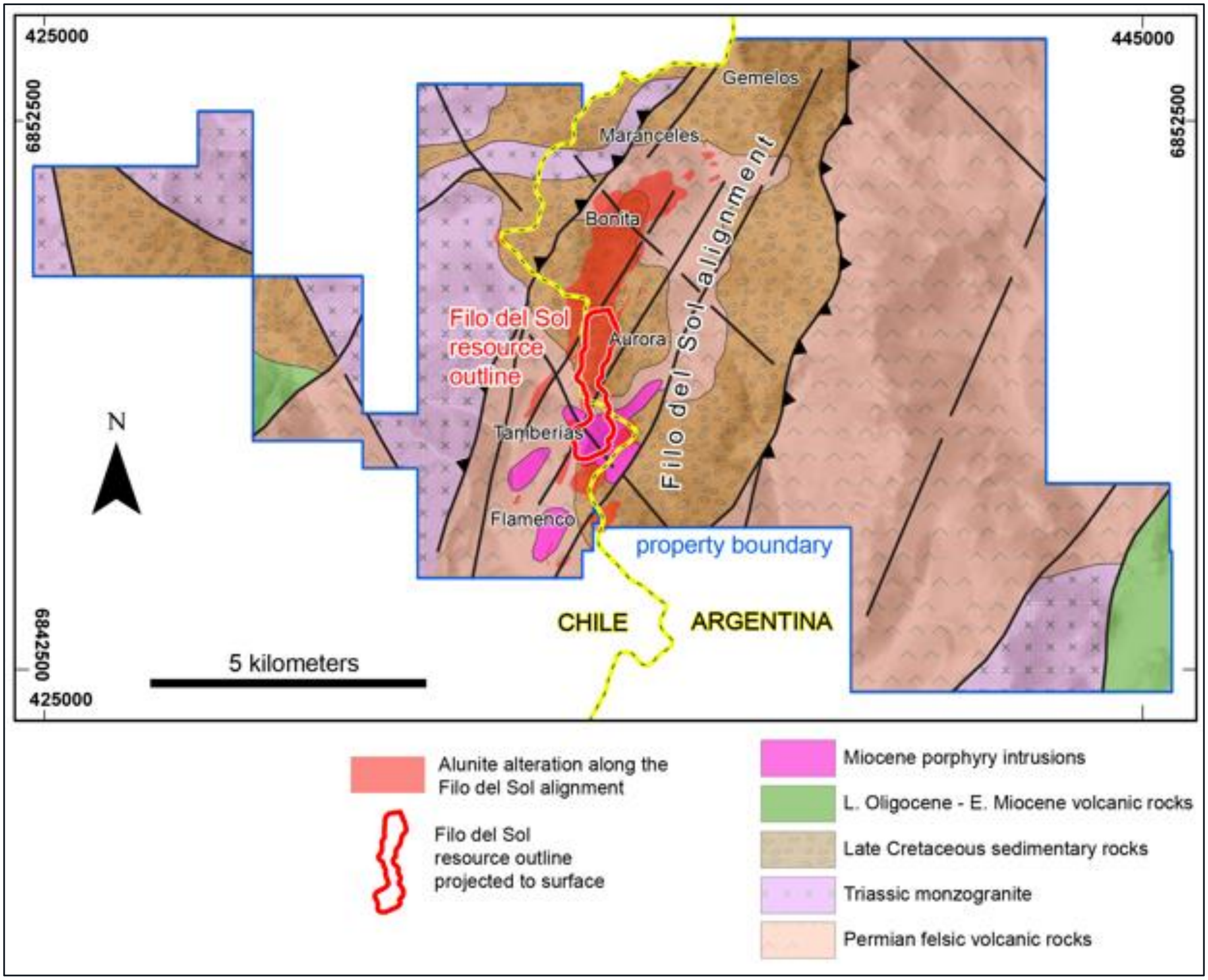
Three broad lithologic assemblages occur in the area of Filo del Sol: premineral country rocks of various assemblages and geologic events; several discrete, intermineral porphyry phases and related magmatic-hydrothermal breccias; and late-mineral hydrothermal breccia bodies of predominantly phreatic affinities. Extreme telescoping is an important feature of the deposit area with potassic-altered and Cu-Au mineralized porphyry intrusions overprinted by high-sulphidation alteration and related Cu-Au-Ag mineralization, all of which is capped by a leached zone with underlying oxidized and locally supergene-enriched mineralization. The Filo del Sol deposit, per se, is a coherent volume of Cu ± Au ± Ag mineralization which encompasses all these types of mineralization resulting in a variety of mineral assemblages and grade distributions throughout the deposit. The currently-defined mineral resource includes only the oxidized portion of the deposit.

Figure 7-1: Part of the Late Oligocene to Miocene Porphyry-Epithermal Belt in Chile and Argentina



Source: Devine, 2023

Figure 7-2: Property Geology



Source: Filo Mining Corp., 2023

In the south, the deposit is hosted predominantly by the Tamberías porphyry stock and its immediate country rocks. There, mineralization formed at the expense of magnetite- and chalcopyrite-rich potassic assemblages that were overprinted by hydrolytic alteration associations, with one or more of quartz, white mica, clays, and alunite. In the northern part of the deposit, supergene Cu mineralization largely formed from a series of north- to northeast-trending, high-sulphidation epithermal lodes emplaced during transgressive advanced argillic alteration.

7.3 Lithology

7.3.1 Country Rocks

Country rocks to the mineralization comprise Permo-Triassic felsic volcanic and monzogranitic basement units of the Choiyoi magmatic province, which are unconformably overlain by a sequence of terrigenous sedimentary and volcanic rocks (Figure 7-3). These units are intruded by a series of mafic dikes and sills, microdioritic in texture and composition that define a north-to-northeast-trending swarm all along the deposit and beyond.

7.3.2 Porphyry Intrusions

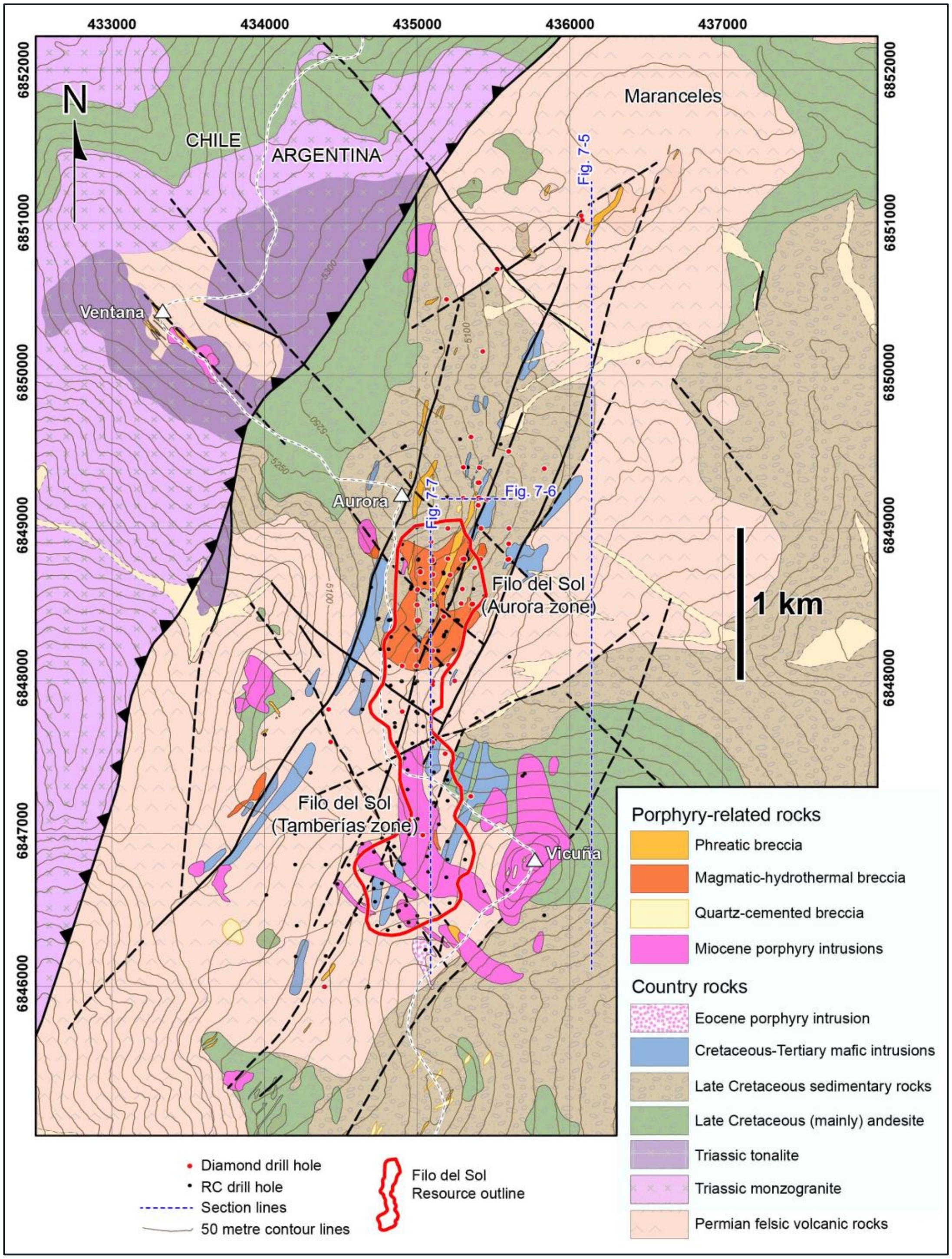
Several intermineral porphyry phases are distinguished in the project area and form a large swarm with greater than 1 km of vertical extension and at least 3 km of strike length, coincident with the more broadly defined north-to-northeast-trending Filo del Sol alignment. However, the Tamberías stock and associated hypogene Cu-Au mineralization are discordant to this trend and display a clear northwest attitude. In general, porphyry dikes have medium-to-coarse-grained (up to 6 mm) porphyritic textures and an overall dioritic to quartz-dioritic composition, with phenocrysts of plagioclase, biotite, and amphibole, in addition to minor quartz. All were originally potassic altered.

7.3.3 Hydrothermal Breccias

Two principal types of breccias are recognized in the area: early-stage magmatic-hydrothermal and late-stage phreatic. A large body of magmatic-hydrothermal affiliation occurs all along the porphyry dike swarm at Filo del Sol-Aurora. The breccia is predominantly clast-supported, polymictic, and includes clasts of all country rock units and early-stage porphyry intrusions. Subrounded to angular, pebble-sized fragments dominate, although, large blocks of several tens of metres in diameter also occur. The breccia typically contains fragments of previously formed quartz veinlets of A type (Gustafson and Hunt, 1975) and is cut by new generations of similar A-type veinlets. Pegmatoidal facies are common at depth and are comprised of aggregates of K-feldspar, biotite, and anhydrite. The breccia was originally emplaced under potassic-stable conditions and drilling shows that it is the main host to both early porphyry-related Cu-Au mineralization and transgressive high-sulphidation Cu-Au-Ag mineralization (Perelló et al., in prep.).

Late hydrothermal phreatic breccia bodies occur as irregular dikes at the surface in both zones of the deposit, at Tamberías and Filo del Sol-Aurora, where they follow the main northwest and north-to-northeast trends of the of the porphyry intrusions, respectively. Although several varieties occur, the most common is of poorly selected and sorted, matrix-supported, subangular lithic clasts and, characteristically, an abundant proportion of clasts of refractory A-type quartz veinlets. The clastic matrix is fine-grained (rock flour in places), mostly massive and disaggregated, with abundant lithic particles, crystaloclasts of quartz, biotite, and feldspars, and impregnated with alunite, clay minerals, pyrite, and Cu-As sulphosalts. All these phreatic breccias were emplaced during advanced argillic alteration and nowhere are they cut by new generations of quartz veinlets.

Figure 7-3: Surface Lithology Map – Filo del Sol Deposit Area



Source: Filo Mining Corp., 2023

7.3.4 Hydrothermal Alteration and Hypogene Mineralization

Two principal alteration types are modelled in the project area: early potassic at depth and late advanced argillic overprinting at higher levels. The boundary between the two types of alteration and their related mineralization is relatively abrupt and is clearly differentiated by geochemistry, notably As and sequential leach Cu analyses. Intervening zones of quartz and sericite (fine-grained white mica) are also present. Laterally, at the surface, propylitic alteration fringes the system on both sides, but has not been intersected by drilling in the main body of the deposit. The advanced argillic alteration assemblages are part of the large lithocap mentioned above, with its best expression coincident with the northern Aurora zone, of the deposit (Figure 7-4). The uppermost part of the lithocap underwent intense steam-heated alteration, with corresponding transformation of original components to friable, poorly indurated and highly disaggregated, pulverulent quartz and quartz-alunite associations.

7.3.5 Potassic Alteration and Associated Cu-Au Mineralization

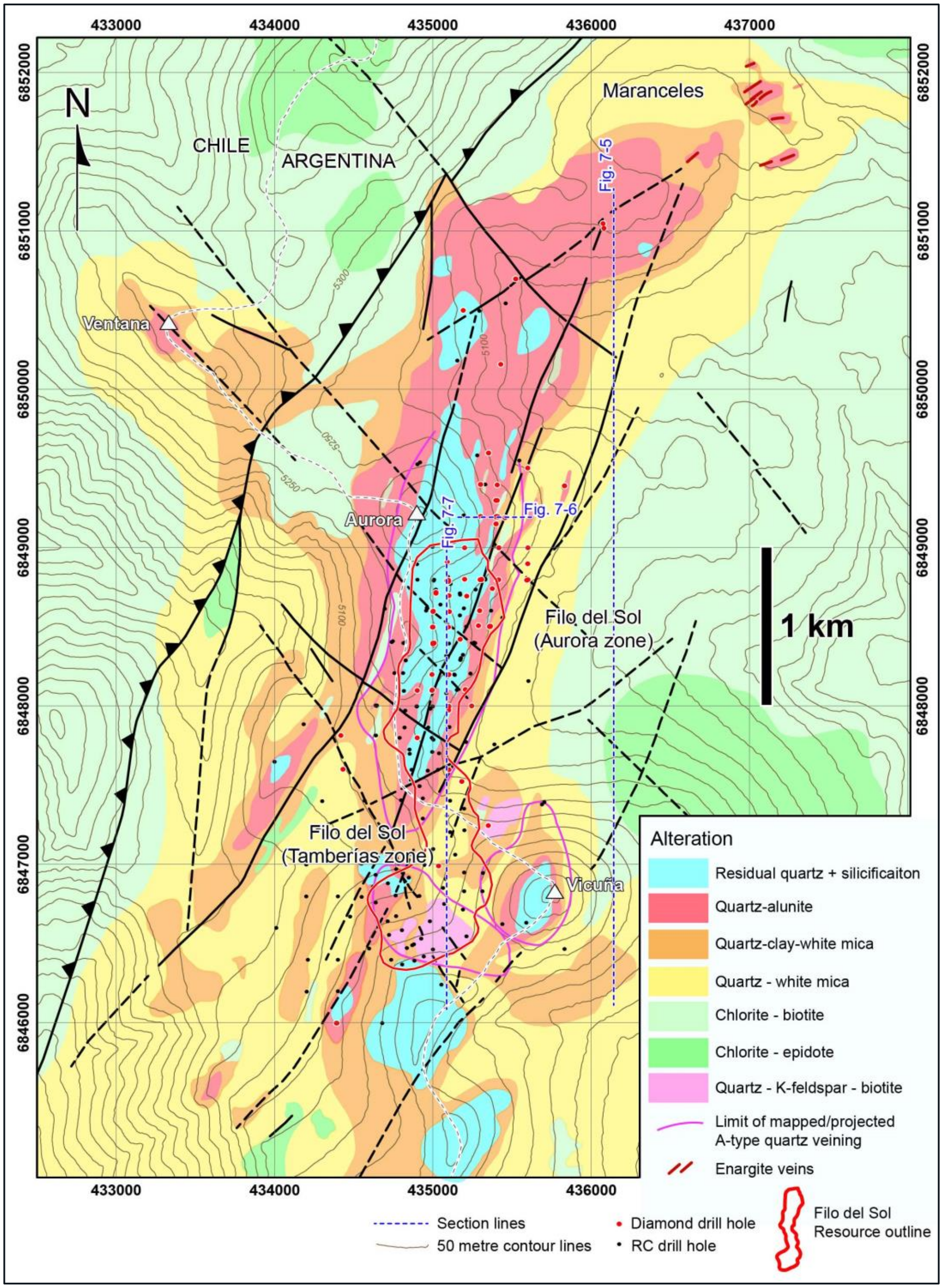
Two distinct potassic alteration associations are characteristic in the Tamberías and Filo del Sol-Aurora zones: early, biotite dominated, and late, with quartz, K-feldspar, anhydrite, and biotite. The early associations are predominantly present in hornfelsed andesitic country rocks and mafic intrusions, whereas the later assemblage is most spectacularly developed in magmatic-hydrothermal breccia at depth in the Filo del Sol-Aurora zone. Hydrothermal magnetite and/or mushketovite are integral parts of the potassic associations. A-type quartz veinlet stockworks are common in potassic altered rock and are cut by planar, molybdenite-bearing, B-type quartz veinlets (Gustafson and Hunt, 1975). The accompanying sulphides are chiefly chalcopyrite and pyrite, with localized bornite, although the Cu mineralization in potassic centres at Tamberías and Filo del Sol-Aurora is largely dominated by chalcopyrite.

7.3.6 Advanced Argillic Alteration and Associated High-sulphidation Cu-Au-Ag Mineralization

Advanced argillic alteration forms the predominant alteration type in the Filo del Sol-Aurora zone of the deposit, and its associated high-sulphidation-state sulphide lodes and disseminations contain the bulk of the Cu-Au-Ag mineralization discovered to date. The type comprises three principal, concentrically-zoned, associations, including a central zone of vuggy residual quartz, an intervening zone of quartz-alunite, and an external zone of quartz-white mica-clay minerals. The central zone consists of a large, single or more typically, composite, swarm of steeply-dipping to vertical residual vuggy quartz and silicified ledges, with intervening and flanking zones of quartz-alunite alteration. The external zone contains quartz, fine-grained white mica (sericite), and clay (illite, kaolinite) as principal components, with quartz and white mica also being predominant in depth and defining the roots of the lithocap-related alteration. The advanced argillic and intervening quartz-white mica associations are completely transgressive to the products of the early potassic alteration-mineralization event, which they destroyed partially to totally. Exceptions are the flanking microdioritic dikes which, as at Tamberías, even within the advanced argillic zone, wholly or partially preserve the original biotite-dominated potassic assemblage. In the Filo del Sol-Aurora zone, the advanced argillic alteration associations define a conventional, deeply rooted (~1000 m) and steeply dipping, funnel-shaped body but, at Tamberías, they are far more restricted and irregular.

In the Filo del Sol-Aurora zone, the mineralization is comprised of multiple high-sulphidation-state sulphide associations with one or more of pyrite, melnikovite, marcasite, Cu sulphides (bornite, covellite, chalcocite, digenite) and Cu-As-Sb sulphosalts (enargite, luzonite, famatinite, tennantite), plus numerous Cu-bearing Ag-As sulphides and sulfosalts. Native Au, calaverite, electrum, and auricupride Au are also present. The sulphides occur in a variety of forms, including metre-wide massive sulphide lodes, hydrothermal breccia cements, veins, veinlets, and disseminations. In all cases pyrite is the earliest-formed sulphide and is progressively replaced by the Cu-As sulphosalts and/or Cu-bearing sulphides.

Figure 7-4: Surface Alteration Map – Filo del Sol Deposit Area



Source: Filo Mining Corp., 2023

7.3.7 Steam-heated Alteration

Steam heated alteration is prominently developed for ~1 km along the continental divide and international frontier between Argentina and Chile (Figure 7-4). The blanket-like zone, developed above the paleo-groundwater table, typically occupies the shallowest parts of the system exposed at ~5300–5400 m, and attains a thickness of ~150–200 m beneath the ridge crest. Downward-penetrating roots, up to ~400 m, possibly caused by a descending water table, are also present. The deep roots can be guided by vertical structures and zones of damage, lithologic units (e.g., phreatic breccia), and previously formed vuggy residual quartz. The steam-heated zone is composed of a white, powdery rock comprised of cristobalite, chalcedony, kaolinite, and alunite, with additional native S and cinnabar occurring in places. Most of the steam-heated zone is barren of metals, except for localized pockets of earlier formed, high-sulphidation Cu, Au, and Ag. The steam-heated zone was likely associated with a subhorizontal landscape, of which the Filo surface is a remnant. Similar paleosurfaces are widespread in the El Indio belt, approximately 100 km to the south of Filo del Sol.

7.3.8 Supergene Mineralization

The zone of near-surface supergene sulphide oxidation forms the bulk of the currently defined resource at Filo del Sol but comprises a relatively small part of the overall mineral deposit. It reaches its maximum thickness (~300 m) below the ridge crest in the northern Filo del Sol zone, but it is more irregularly distributed at Tamberías. In general, it is comprised of three subzones, including upper leached, intermediate oxidized, and lower mixed oxide-sulphide, followed at depth by localized sulphide enrichment. The upper leached zone is mainly developed at the expense of advanced argillic vuggy residual quartz and steam-heated alteration, within which Cu has been completely removed but some Au remains. Fe-bearing sulphates are common. The intermediate, Cu-rich oxide zone is characterized by the presence of Fe, Fe-Cu, Cu, Mo, and Co oxides and hydroxides and hosts the bulk of the soluble Cu mineralization contained in the Filo del Sol resource. Chalcanthite and cuprocopiapite are dominant, and brochantite occurs locally. Enargite is present upon approaching the lower mixed oxide-sulphide zone, and sooty chalcocite occurs in localized pockets of supergene sulphide enrichment.

7.3.9 Structure

The structure of the Filo del Sol region is characterized by a series of north-to-northeast-trending, steeply-to-moderately east and west dipping reverse faults that control the distribution of equally oriented blocks of basement rocks, including both granite and felsic volcanic sequences, in an overall thick-skinned contractional tectonic style (e.g., Martínez et al., 2015). Two principal faults, El Potro in Chile and Mogotes in Argentina appear to constitute the bounding structures of a large, pop-up-like, basement-cored block affected by internal, subsidiary reverse faults. These basement highs are inferred to be expressions of large and deep seated, easterly-propagated, thick-skinned ramps that accommodated the regional shortening (Martínez et al., 2015). Numerous northwest-trending lineaments are apparent on satellite images, but only a few can be inferred or mapped as faults in the field. Some of them likely formed as conjugate structures to the predominant east-oriented reverse fault motion.

At the project scale, two principal families of faults, pre- and post-mineral in timing, are present, with many of the pre-mineral structures being inferred to have been utilized by the magmatic-hydrothermal system. The dominant north-to-northeast trend of the system is congruent with the regional structural grain described above and constitutes a first-order control in the Filo del Sol alignment. These and other east-and-west-dipping reverse faults mapped in the region were inverted from original normal faults that controlled the site of marine and continental basin construction and corresponding sedimentation during successive events of Mesozoic and Cenozoic extension (Martínez et al., 2015; Sillitoe et al., 2019).

A series of steep, north-to-northeast-striking, post-mineral faults transect the Filo del Sol system and display recent activity, enhanced by solifluction processes. They are evidenced by topographic breaks and drainage offsets at the surface, as well as metric zones of intensely damaged rock and gouge where intersected by drilling. Offsets are in the range of a few metres, and most are considered to be normal faults with both east-and-west-side down displacements.

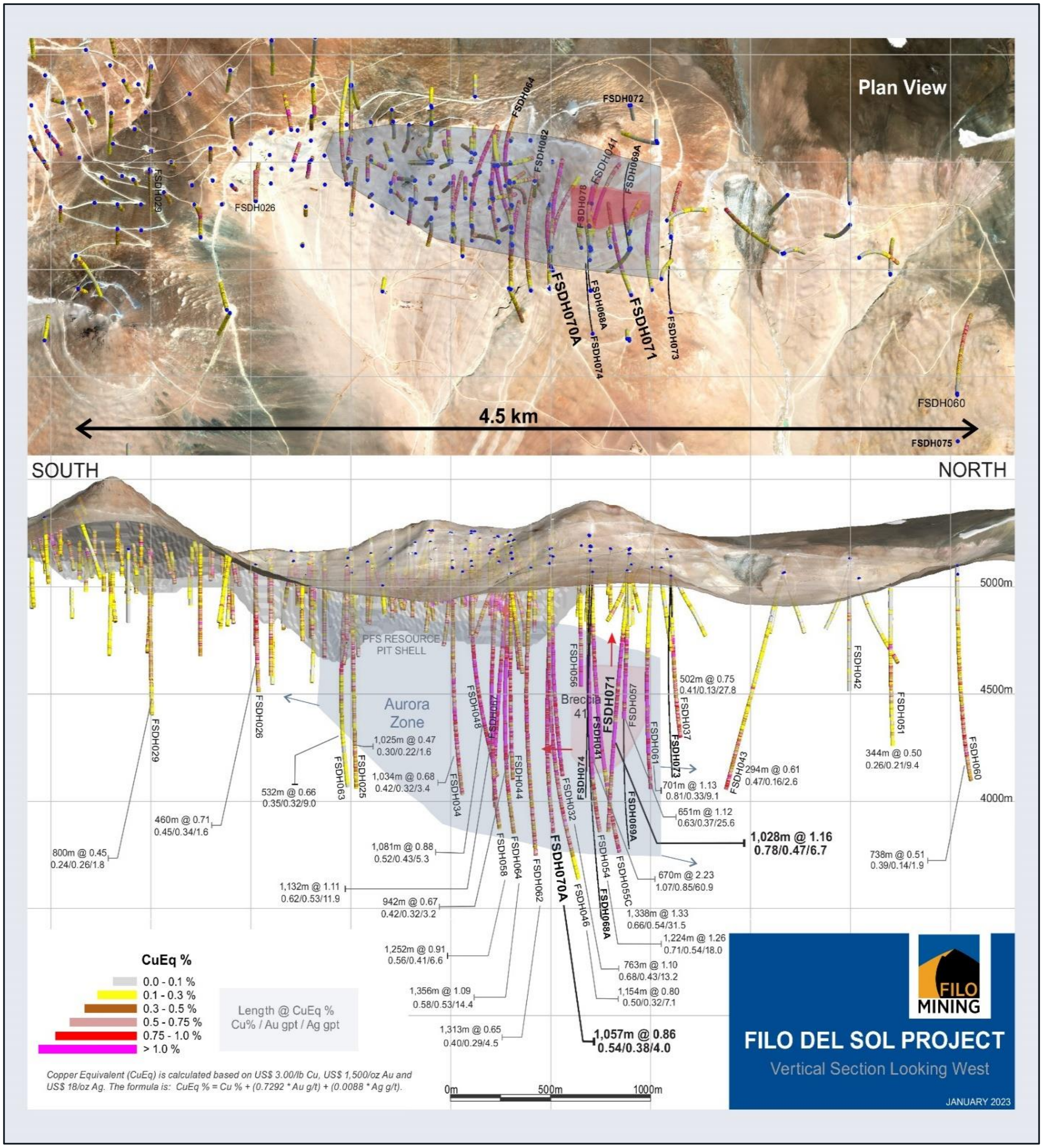
7.4 Deposit Description

The Filo del Sol deposit comprises a large porphyry Cu-Au and high-sulphidation epithermal Cu-Au-Ag system that formed during rapid uplift and erosion in the Middle Miocene. A north-northeast trending alignment of porphyry intrusions at least 3 km long in the deposit area includes an older, more deeply eroded porphyry Cu-Au mineralized domain in the Tamberías area, with slightly younger, partly blind to the surface porphyry Cu-Au mineralized intrusions in the Filo del Sol-Aurora zone to the north. Extreme telescoping in the Filo del Sol-Aurora zone has led to the overprinting of the Cu-Au mineralized porphyry domain by high-sulphidation Cu-Au-Ag epithermal mineralization within a large area of advanced argillic alteration.

In the Filo del Sol-Aurora zone, a suite of north-northeast trending, pre-mineral mafic dykes were guided by pre-existing faults through the host rhyolite and overlying clastic rocks. At higher levels in the host rock sequence, mafic sills were guided by layering within the clastic rocks. Porphyry intrusions (~15.4 Ma) and related intermineral magmatic-hydrothermal breccia were subsequently emplaced along the same structural trend, with associated Cu-Au mineralization in potassic alteration. A high rate of uplift and syn-mineral erosional unroofing of the system is inferred as the potassic porphyry Cu-Au mineralization is largely overprinted by advanced argillic alteration with associated high-sulphidation Cu-Au-Ag mineralization as pyrite, enargite, bornite, chalcocite, covellite and Ag-bearing sulphosalts in a core zone with vuggy residual silica, silicification and surrounding quartz-alunite alteration. Metal distribution within the hypogene part of the deposit is controlled by these two types of alteration and mineralization, with a relatively sharp boundary between the two at depth. A later style of pyrite mineralization with high silver grades is related to late, higher-level phreatic breccias along the north-to-northeast mineralized corridor. Steam heated alteration is preserved as the uppermost part of the lithocap domain, forming the ridgetop at Filo del Sol. Drilling is currently ongoing in the Aurora zone at depth beneath, and north of, the currently defined resource, and has so far demonstrated that the Filo del Sol deposit is at least 3 km N-S, 400 m E-W and extends at least 1.5 km below surface (Figure 7-5 and Figure 7-6). The deposit remains open to the north, south, east, west and at depth, although mineralization appears to be weakening in the deepest holes drilled to date.

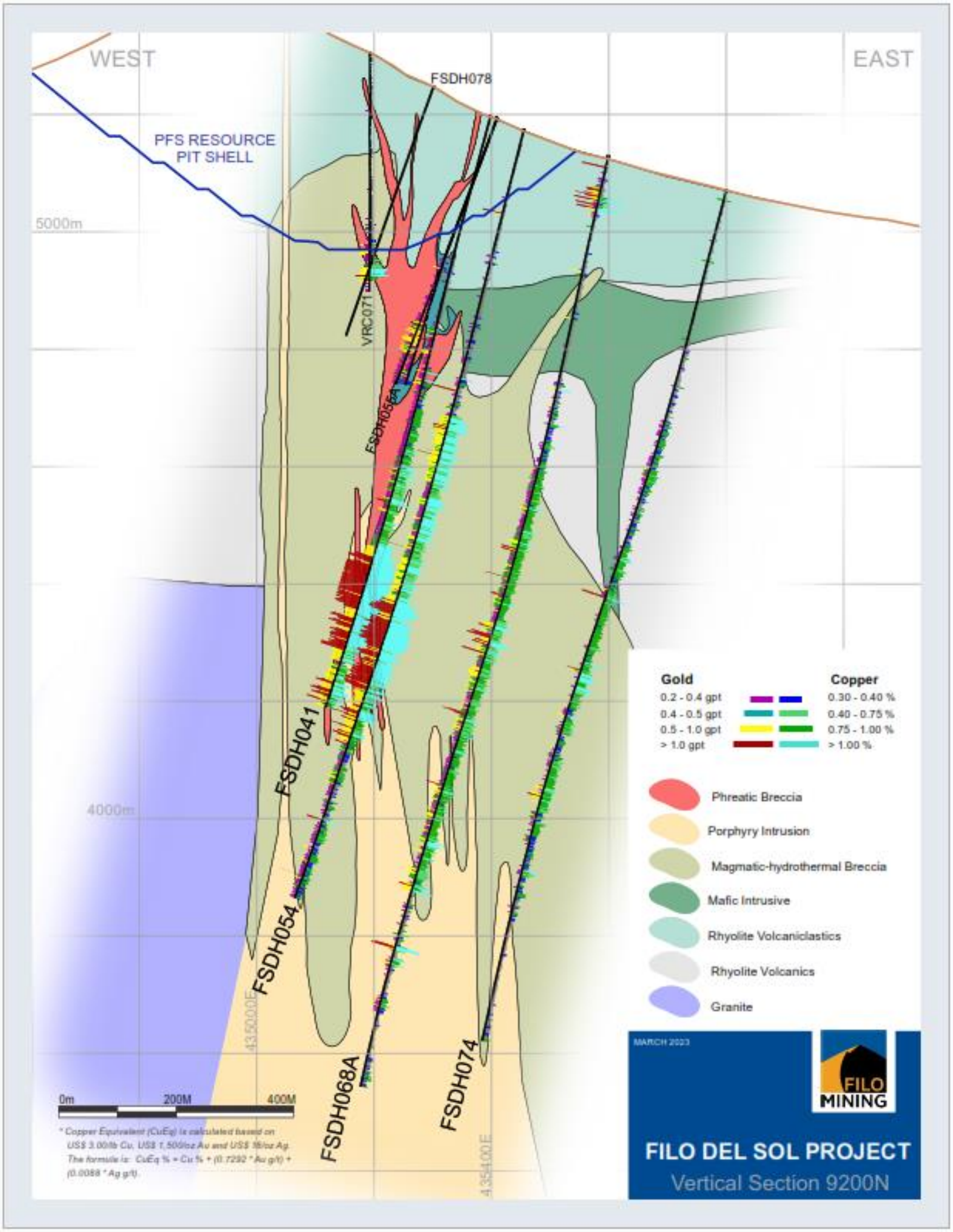
A high-grade Ag zone is a key part of the Filo del Sol deposit, occurring as a shallowly north (20°) and west-dipping (10° to 15°) zone 10 m to 130 m thick and extending at least 1,200 m N-S and 400 m to 600 m E-W within the resource area, overprinted by the oxide zone (Figure 7-7). However, to the north of the currently defined resource, the Ag geochemical zone extends another 1800 m along the same trend. Ag grades in this zone are above 60 g/t, averaging 200 g/t with a maximum value of 6980 g/t Ag over a 2-m sample length. It includes Au mineralization, generally increasing from south to north (Figure 7-7. In the southern part of the deposit, it commonly appears in drill cuttings as unconsolidated grayish to black sandy mud, commonly with associated soluble Cu mineralization as Cu-sulphates. Ag mineralization in this zone is mostly composed of chlorargyrite (AgCl) and Ag and Cu sulphosalts of proustite - pyrargyrite [Ag, (As, Sb), S] (Di Prisco, 2014). It has a distinct geochemical anomaly pattern characterized by anomalous values of metals such as Cu, Ag, Mo, Sb, (±Au), As, Hg, W, (±Bi, Sn) and low values of Al, Ca, Sr, V, (±Th). The mineral resource for this zone is shown in Table 14-19 and Figure 14-5; it remains open to the north and east.

Figure 7-5: Section Looking East to Filo del Sol



Source: Filo Mining Corp., 2023
Note: Figure 7-5 shows drilling beneath and to the north of the currently defined resource pit outline. It includes available drill results to February 2, 2023.

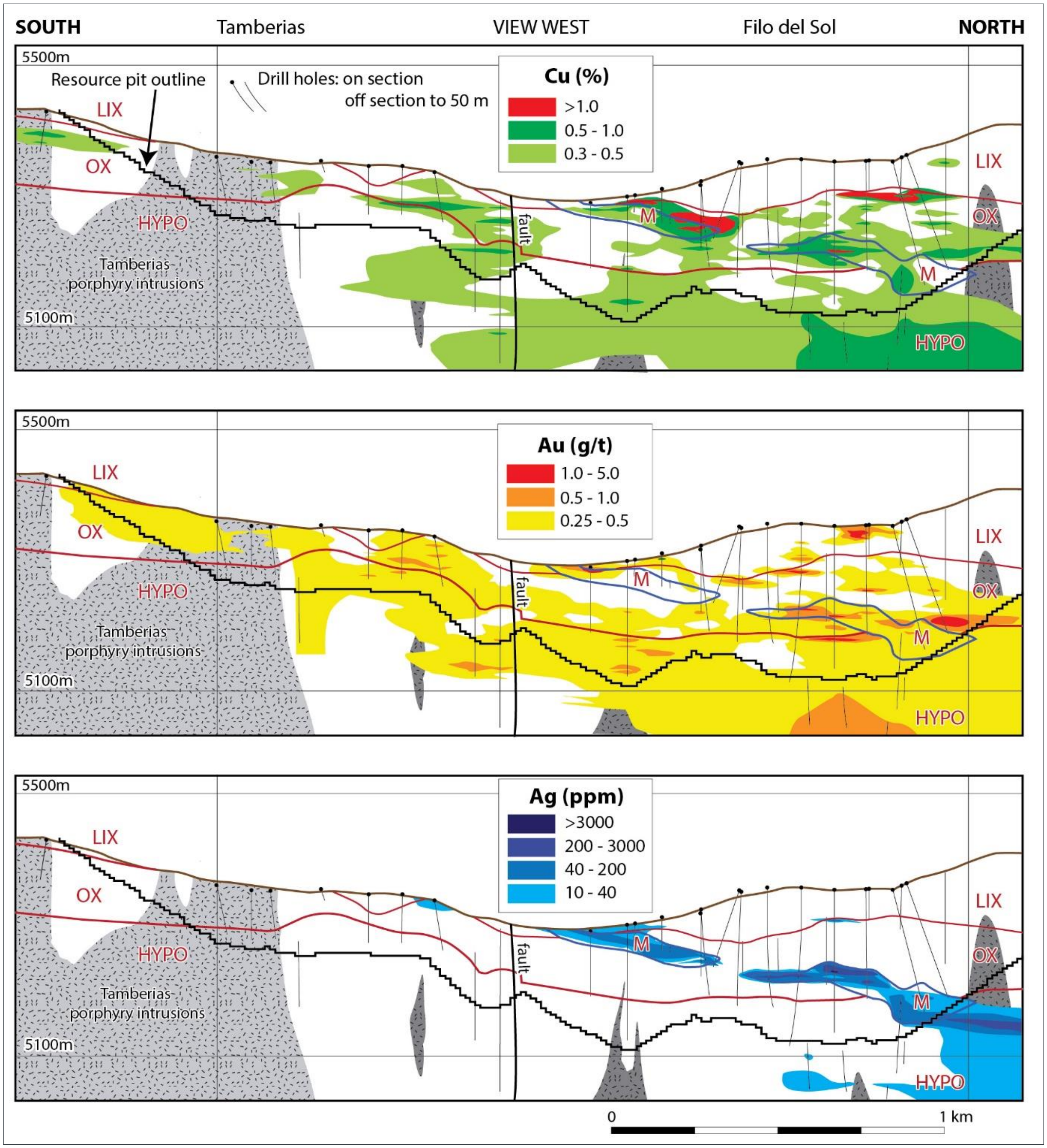
Figure 7-6: E-W Section 6849200N (View North)



Source: Filo Mining Corp., 2023

Note: Figure 7-6 shows a vertical section through the recent and ongoing drilling on the Aurora zone of the deposit that lies outside of the currently defined resource pit outline.

Figure 7-7: Section 435100 (View West)

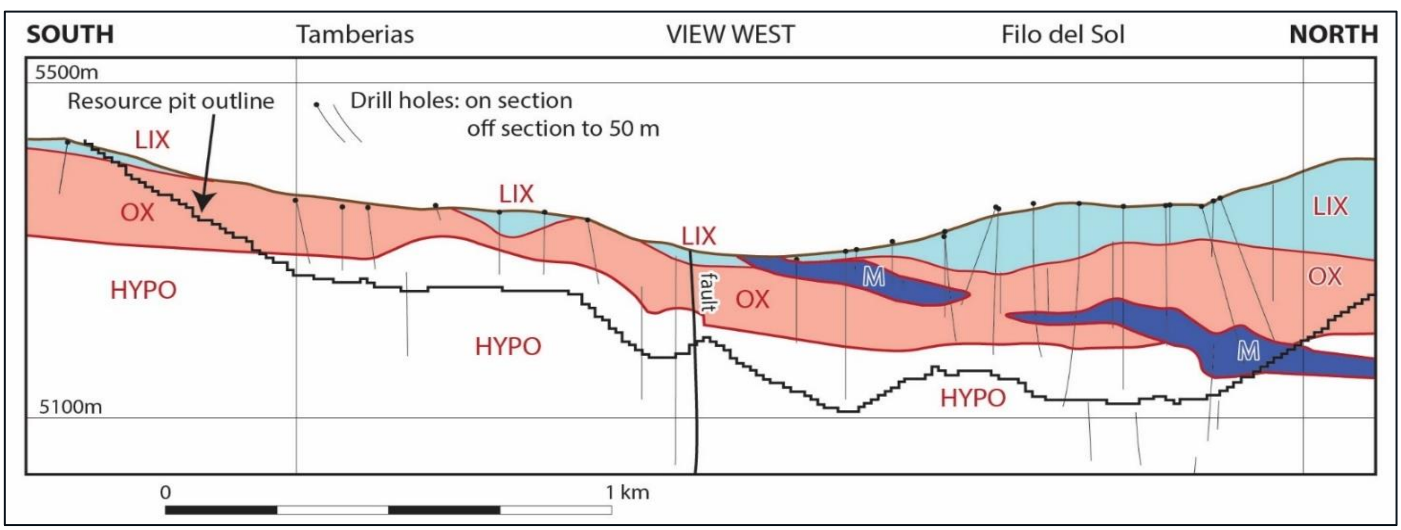


Source: Devine, 2023
Note: View to the west, a long section through the currently defined Filo del Sol resource. The resource pit outline is shown for reference, this includes only the uppermost part of the deposit at Filo del Sol. This figure is based on a 100-m wide slice though a Leapfrog 3D numeric model that includes influence from drill holes that are off section. The same section location as in Figure 7-8.

The epithermal system at Filo del Sol-Aurora extends southward into the Tamberías area, where zones of quartz-alunite and quartz-white mica-clay alteration with Au and Ag mineralization overprint the older (~16 Ma) host dacitic Tamberías porphyry intrusions. Telescoping of the younger Filo del Sol-Aurora system over the older Tamberías porphyry is inferred to largely be responsible for the juxtaposition, although normal faulting along north- and northeast-trending faults through the system also break geologically different domains. Alteration in the epithermal zone is composed of residual silica, which corresponds to a leached domain with no Cu. No high-grade Ag domain occurs in Tamberías; the leached zone is underlain by an oxide zone with Cu-sulphates that progresses down to a Cu-Au hypogene sulphide domain that has only been drill tested in a limited area. In Tamberías, dacitic plagioclase-hornblende-biotite porphyry intrusions intrude the rhyolite basement and have associated biotite-magnetite (potassic) alteration. These porphyries are intruded by younger feldspar-phyric porphyry phases that are only partly exposed, are largely blind to the surface, and are associated with Cu sulphide mineralization and elevated Au values.

The currently defined resource at Filo del Sol predominantly includes the uppermost oxidized part of the deposit at Filo del Sol and Tamberías (Figure 7-8). Leaching of the uppermost parts of the system (LIX) has resulted in the development of an Au-only oxide zone (AuOx). This zone includes drill intersections in holes VRC097 (84 m @ 1.36 g/t Au) and VRC099 (78 m @ 1.02 g/t Au). The leached zone is underlain by supergene Cu enrichment at the top of an oxide zone (OX) that ranges from 40 m to 300 m in thickness, deeper in the north. The oxide zone is characterized by the presence of Fe, Fe-Cu and Cu oxides and hydroxides. It hosts the important soluble Cu mineralization comprising hydrated sulphate minerals (chalcanthite, copiapite, cuprocopiapite) that form a bright blue blanket across the surface and extend to depth (CuAuOx Zone). This zone formed as a result of the combination of the highly acidic environment generated by oxidation of abundant marcasite and pyrite and the arid climatic conditions. The M zone (M) corresponds to the high-grade silver zone that occurs as a moderately north-northwest dipping tabular zone ranging from 10 to 130 m thick (Figure 7-8). It is overprinted by the oxide zone in the resource area. The hypogene zone (HYPO) is characterized by the presence of sulphides and the absence of oxide minerals. It includes both porphyry Cu-Au and high-sulphidation Cu-Au-Ag epithermal mineralization. The resource is bound to the west by a sharp cutoff in grade, possibly a fault. Mineralization extends to the eastern limit of the pit as the current resource does not include the most recent drilling into hypogene Cu and Au mineralization to depth.

Figure 7-8: N-S Section 435100E (View West) Showing the Mineral Zonation Profile within the Resource Pit Outline



Source: Devine, 2023
Note: The same section location as in Figure 7-7.

8 DEPOSIT TYPES

Mineralization in the Filo del Sol area includes both porphyry copper-gold-molybdenum and high-sulphidation gold-silver epithermal systems. The mineralized system in its entirety is a telescoped porphyry – epithermal system, with multiple intrusive and breccia centres, and so combines aspects of both deposit types. The currently defined mineral resource presented in this report is best classified as the upper oxidized part of the high-sulphidation epithermal Cu-Au-Ag part of the deposit.

8.1 Porphyry Cu-Au

Porphyry systems are found in intrusive belts associated with subduction generated magmatism. They are formed in the ascending magmas below volcanic systems. Broad alteration of the surrounding rocks and intrusions takes place as hot fluids are pumped through by the convective heat engine in the core of the system. Concentric shells of alteration and mineralization can develop around porphyry systems and this systematic zonation is an important characteristic of porphyries that enables geochemistry and alteration mapping to provide vectors to mineralization. Mineralization contains both disseminated sulphides and various veinlet and stockwork systems, which also host sulphides. An important characteristic of porphyry districts is that they do not form deposits in isolation but tend to occur in "clusters." This is known to be the case in Maricunga Belt systems; for example, such as Refugio where mineralized porphyry bodies are spaced in the order of 1 km apart. This spacing is important to consider when evaluating step out exploration at Filo del Sol.

8.2 High-Sulphidation Epithermal Au-Ag

Many features of the Filo del Sol deposit are typical of high-sulphidation epithermal systems produced by volcanism-related hydrothermal activity at shallow depths and low temperatures. In these systems, deposition normally takes place within 1 km of the surface in the temperature range of 50°C to 200°C, although temperatures up to 400°C are not uncommon. Most deposits occur as siliceous vein fillings, irregular branching fissures, stockworks, breccia pipes, vesicle fillings, and disseminations. The fissures have a direct connection with the surface, which allowed the mineralizing fluids to flow with comparative ease. In many cases, the deposits are related directly to deeper intrusive bodies; it is typical for most mineralization to be in or near areas of Tertiary volcanism. The country rocks located near epithermal veins are commonly extensively altered. Relatively high porosity and open-channel permeability allow fluids to circulate in the wall rocks for great distances. Favourable temperature gradients promote reactions between cool host rocks and warm to hot invading solutions. As a result, wall-rock alteration is both widespread and conspicuous. Among the principal alteration products are alunite, pyrophyllite, illite, dickite, kaolinite, silica and pyrite, as well as other metal bearing sulphides and oxides.

Remobilization of copper, and possibly silver, particularly through weathering processes (oxidation, leaching, and replacement) appears to have significantly altered the original metal zonation patterns in the upper part of Filo del Sol.

9 EXPLORATION

Filo Mining, or its predecessor companies, have been exploring at Filo del Sol since the 1999/2000 field season. A total of 20 work programs have been completed over these years, and there have been four seasons (2001/2002, 2002/2003, 2008/2009, 2009/2010) where no work was done. Apart from the 2021/2022 season, exploration has been limited to the summer season, typically between November and April, and so exploration seasons are described by the years which they bridge.

Table 9-1 summarizes the surface work done during each field season. Drilling is described in the following chapter.

Table 9-1: Exploration Summary by Year

Season	Surface	Geophysics	Drilling (m)
1998/1999	1:10000 geological mapping Talus fine and rock sampling		2,519
1999/2000	1470 talus fine samples 3720 trench samples 1150 rock channel samples	153 km MAG 37.8 km IP-CSAMT	
2000/2001	462 rock chip samples	100 km MAG	2,662
2003/2004	216 talus fine samples		1,171
2004/2005	149 talus fine samples	30.4 km IP-Res 29.4 km MAG	1,762
2005/2006	83 talus fine samples 11 rock chip samples		1,708
2006/2007			578
2007/2008	310 talus fine samples	30.0 km IP-Res. 77.6 km MAG	2,890
2010/2011	Geological mapping 1:5000		156
2011/2012		36.2 km P-DP IP	1,853
2012/2013			821
2013/2014			8,406
2014/2015	Geological mapping 1:5000 and 1:7500; PIMA sampling	23 km P-DP IP	7,320
2015/2016	Geological mapping 1:5000, Geochem and PIMA sampling	27.7 km P-DP IP	
2016/2017	Metallurgical sampling, trenching		8,616
2017/2018	RC and core drilling, metallurgical sampling		9,411
2018/2019	Core drilling	Drone mag – 146km	4,631
2019/2020	RC and core drilling	3D IP – 20.5km^2; Drone mag – 146km	8,361
2020/2021	Core drilling		11,280
2021/2022	RC and Core drilling	MT	25,187

Surface work completed on the project to date has included talus fine sampling, rock chip sampling, geological mapping, and induced polarization (IP) and magnetic geophysical surveys.

9.1 Talus Samples

Extensive talus fine sampling has been effective at outlining the main mineralized zones on the property. Over 2,000 samples have been collected, focused on areas of alteration identified through satellite image analysis.

Results indicate three broad anomalies over the Filo del Sol, Tamberías and Maranceles areas, with several other, less-distinct areas of interest. Anomalies are typically defined by Cu, Au, Ag, As, Bi, Mo and Sb. Of particular interest is that both anomalies are larger than the drilled extent of the known mineralization indicating potential for expansion.

9.2 Rock Samples

In addition to the talus fine samples, limited rock chip and channel sampling has been conducted in the main mineralized areas. Sampling was much more restricted in area than the talus fine sampling, covering mainly the Filo del Sol and Tamberías areas with a few samples at Maranceles. Several strongly anomalous (Au, Cu, Ag, As) areas were outlined, both as clusters of float samples and contiguous chip/channel samples along road cuts.

Encouraging historic road cut intervals included: 10 m at 1.96 g/t Au; 24 m at 1.28 g/t Au; 74 m at 0.74 g/t Au; 108 m at 0.72 g/t Au; 34 m at 1.75% Cu, and 0.52 g/t Au. These samples are all in the Cerro Vicuña area.

During the 2015/2016 season, systematic follow-up sampling was completed which confirmed and expanded upon these results with the collection of 378 additional samples. Four road cuts were systematically mapped and sampled identifying a northwesterly-trending zone along the western margin of the Tamberías intrusion. Results from this sampling are shown in Table 9-2. The highest-grade portions of these trenches are characterized by stockwork and brecciated stockwork of smoky quartz veinlets. These surface trenches were extended and sampled during the 2016/2017 season, with a total of 316 additional samples collected.

Table 9-2: Tamberías Trench Sample Results

Trench	Length (m)	Grade (g/t Au)	Grade (%Cu)	Grade (g/t Ag)
TR2	230	0.36	0.02	0.9
TR3	470	0.30	0.18	0.7
incl	198	0.45	0.21	0.7
TR4	227	0.45	0.46	2.2
incl	153	0.54	0.25	2.5
and incl	114	0.35	0.84	0.8
TR5	90	0.35	0.01	1.3

9.3 Geophysical Surveys

Several generations of Induced Polarization (IP) surveys have been completed at Filo del Sol, notably in the 1999/2000, 2004/2005, 2005/2006, 2006/2007, 2007/2008, 2011/2012, 2014/2015 and 2015/2016 seasons. Surveys were completed by Zonge Ingeniería y Geofísica (Chile) S.A. for the 1999/2000 and 2011/2012 surveys and by Quantec Geoscience Argentina S.A. for the others.

Following the collection and processing of data from the 2014/2015 season, the entire historical data package was sent to Grant Nimeck for compilation and 3D Inversion. This inversion resulted in a 3D data set with modelled chargeability and resistivity values and was reported in Nimeck (2015).

This history of surveying resulted in a data set that was somewhat disjointed and inconsistent; however, there were clearly geophysical responses that could be linked to observed geological features and it was thought that a more coherent data set may be of value. In conjunction with this observation, it was recognized that geophysical surveying and data processing had evolved since the last survey in 2016, and DIAS Geophysical Limited was contracted to carry out a 3D DC resistivity and induced polarization (DCIP) survey over the entire target area, including the area covered by the historical surveys.

This survey was completed between November 2019 and February 2020 and the dataset was compiled, processed, and inverted by Condor North Consulting with the final report delivered in November 2020.

The geophysical program carried out by Dias Geophysical Limited was designed to detect the electrical resistivity and chargeability signatures associated with potential targets of interest. This was achieved using the DIAS32 acquisition system, entirely managed by the Dias field crew, in conjunction with one to two GDD transmitters, -depending on ground resistivity- connected to a GDD TxCB controller, to produce up to 10.0 kW of total power. The survey was completed using a rolling distributed partial 3D DCIP array with a pole-dipole transmitter configuration. The survey covered approximately 20.5 km^2.

The survey layout was as follows:

- The survey grid was comprised of a total of four receiver lines, spaced at 300 m.
- Along the receiver lines, the electrode stations were spaced 150 m apart.
- One current injection line was set up between the two northern-most receiver lines, starting between the two southern-most receiver lines. The northern-most transmitter line was 300 m south of the northern-most receiver line.
- Injection stations were spaced 150 m apart.

One of the clearest results from this survey is a high-conductivity anomaly associated with the silver zone between 7500N and 8400N on the 5,000 m depth slice. This is interpreted to be related to the high pyrite content of the zone in this area. A second high-conductivity anomaly overlies the Aurora Zone, although several other similar anomalies have so far not been associated with deeper high-grade mineralization.

In addition to IP, surface magnetic surveys were completed in 2000/2001, 2004/2005, 2005/2006, and 2007/2008. During the 2018/2019 and 2019/2020 seasons, a drone magnetic survey was completed by Pioneer Aerial Surveys Ltd. The survey was completed using a Gem Systems Canada GSMP-35UA potassium vapor sensor mounted on a Matrice 600 drone. Line spacing was 100 m over most of the surveyed area, with 50-m lines flown in a small area in the southeast corner. The total line-length flown was 292 km.

The magnetic data confirms the NNE trend of structures and porphyries at Filo del Sol and highlights the porphyry intrusions in the vicinity of Cerro Vicuña.

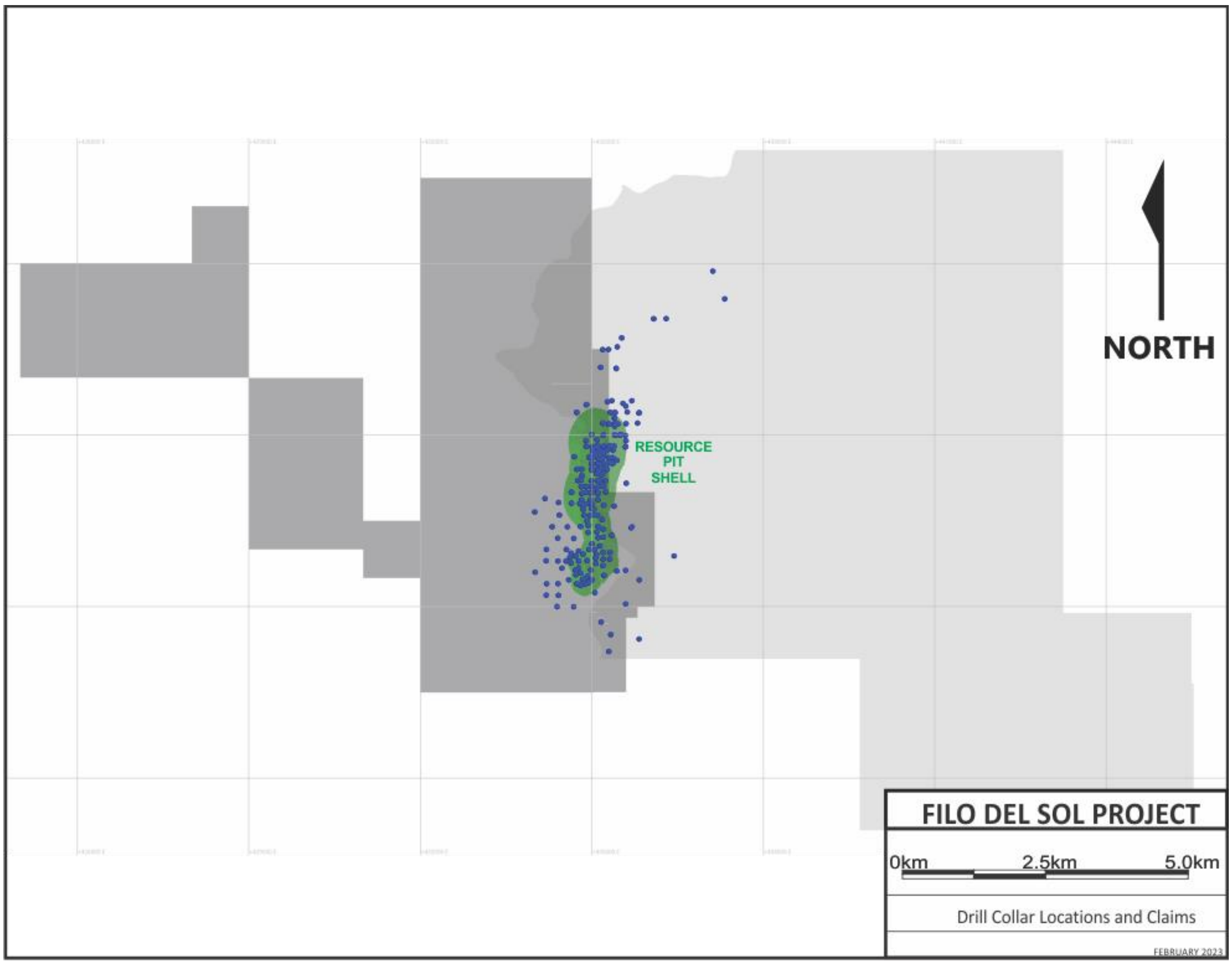
10 DRILLING

10.1 Exploration Drilling

Drilling at Filo del Sol was initiated by Cyprus in 1998/99 and to the end of 2022 a total of 44,950 m of reverse circulation (RC) drilling in 185 holes and 52,064 m of diamond drilling (DD) in 106 holes has been completed on the property.

Drill collar locations are shown relative to the property boundary in Figure 10-1.

Figure 10-1: Drill Hole Collar Locations



Source: Carmichael, 2023

10.2 Drill Methods

Drilling conditions at Filo del Sol are challenging due to the deep weathering profile and thick zone of leached and steam-heated alteration. Diamond drilling (DD), in particular, has experienced difficulties with completing holes and high costs related to lost equipment.

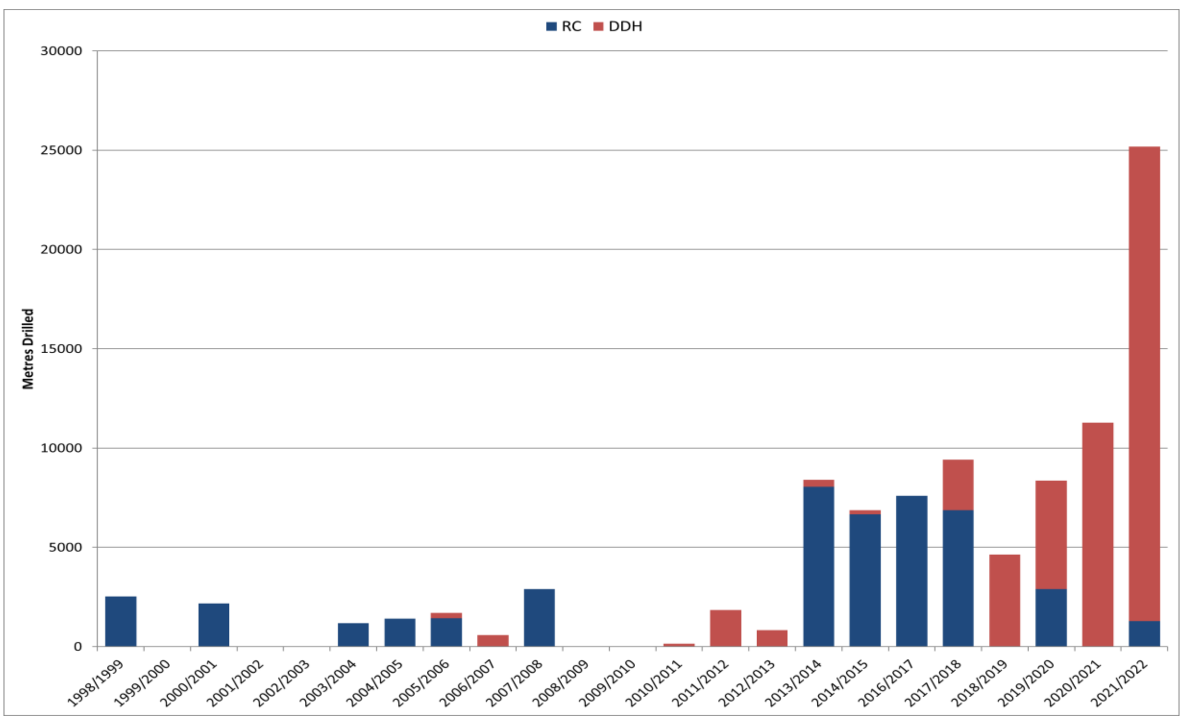
Early drilling was primarily done using RC drills in order to effectively penetrate the leached and steam-heated zones, and to avoid having to haul water to the drills particularly on the Chilean side.

An increased emphasis was put on diamond core drilling starting with the 2017/2018 season in order to better understand the geology and collect coarse sample material for column leach metallurgical testwork. Drilling utilized a triple tube system, which allowed for very good core recovery and a good final sample; however, drilling continued to be challenging, particularly in the steam-heated and oxidized zones, due to poor ground conditions and expansion of the sulphate-rich rocks.

Starting in the 2018/2019 season, diamond drill holes were targeted on exploring the hypogene sulphide mineralization at depth beneath and adjacent to the oxide portion of the deposit. Rock quality and drill penetration rates improve dramatically once the oxidized cap is penetrated and unweathered rock is encountered.

The emphasis on DD resulted in a step-change in the understanding of the deposit geology due to the ability to review geological features in drill core rather than RC chips.

Figure 10-2: Metres Drilled and Drilling Methods Used by Year



Source: Filo Mining, 2023

10.3 Recovery

Recovery for RC drilling was estimated by comparing the ideal weight of the sample (calculated as drilled volume multiplied by expected density) and the recovered material weight. This method is not exact as it relies on an estimation of the bulk rock density in order to determine the ideal weight of the sample. Poor recoveries (below 50%) are often related to fault zones or highly porous intervals in the steam-heated and residual silica zones. Recoveries over 100% are to be monitored as these may indicate sample contamination from material that has been introduced to the drilled interval (e.g., wall crumbling or hole cleaning).

Detailed recovery records from holes drilled before 2008 are missing; however, the Company´s internal reports indicate that the overall average was 72% recovery (intervals with greater than 100% recovery ignored), with a minimum of 0% recovery. There were 81 samples with greater than 100% recovery, or 1.7% of the total samples (Bassan and Rossi, 2009). Recoveries for RC holes drilled during the 2013/2014 and 2014/2015 campaigns were similar. Recovery from RC drilling during the 2016/2017 campaign averaged 69%, with 175 out of 8,616 samples (2%) greater than 100%. Recovery from RC drilling during the 2017/2018 campaign averaged 74%.

The overall average core recovery for the diamond drill holes is 91%. Data analysis shows no correlation between recovery and grade.

10.4 Collar Surveys

Collars of holes in the Filo del Sol area have been surveyed by company personnel using differential GPS. Holes drilled in Maranceles and Gemelos were surveyed by hand-held GPS. The drill platforms are easily visible on the orthorectified World View 3 satellite images and provide good confirmation of the accuracy of the collar surveys.

10.5 Downhole Surveys

Downhole surveys were not completed on holes prior to the 2013/2014 season. During that season, hole surveying using an SRG-gyroscope by Comprobe Limitada was initiated and continued into 2017/2018, starting with hole VRC056. Gyroscope surveys at 10-m intervals have been standard practice since 2018. On average, measurements were collected at 25-m intervals down the hole, decreasing to 5 m in 2016/2017 starting with hole VRC097, and increasing to 10 m in 2017/2018 starting with hole VRC135. Holes started at -90° tend to flatten between 1° and 5° per 100 m, while holes started shallower than -90° (between -85° and -70°) tend to steepen 1° to 2° per 100 m to about 300 m, and then shallow at about 0.8° per 100 m. Azimuths tend to increase 3º per 100 m; however, there is a lot of variability in the averages.

10.6 Sample Length/True Thickness

The Filo del Sol deposit is comprised of several different zones, typically with different origins and different geometries. Copper tends to occur either disseminated throughout or in flat-lying higher-grade zones likely due to supergene enrichment. Silver occurs primarily as a shallow-dipping zone of high-grade mineralization. Drilled widths for both of these metals are essentially true widths, as the steep to vertical drill holes pierce the zones at close to perpendicular. The distribution of gold is more complex, and includes disseminated, sub-horizontal zones and suspected steep structurally controlled zones. The drilled width of the disseminated and sub-horizontal zones are essentially true widths, as with copper and silver. The drilled width of the structurally controlled zones is likely to be greater than the true width. More work is required before the geometry of these structures is understood and the relationship between their drilled and true widths can be established.

10.7 Significant Results and Interpretation

Results and interpretation derived from this drilling are presented in Sections 7 and 14 of this report.

10.8 Logging Procedures

Drill core is moved from the rigs to the Batidero Camp for photography and recovery/RQD logging. The core is then trucked to the Company's core logging facility near the town of Rodeo, San Juan Province. Core is logged by Company geologists for lithology, alteration and mineral zonation, and samples are marked out for cutting. Following sampling, the core is stored at the logging facility on palettes inside a large warehouse.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Surface Sampling

Soil and Talus Samples were collected from small holes deep enough to sample the interval below the iron-cemented horizon. Talus samples were composited from ten stations located within 5 m along a 100 m-long line. Talus lines were oriented either north-south or east-west. Sampled material was finer than #10 mesh. All samples were labelled and identified before being shipped for geochemical analyses.

Rock samples involved collecting approximately 1 kg to 3 kg of representative chips from outcrops or trenches. The sample length as well as a geologic description was recorded and entered into the database. Sample location was annotated on the sample booklet and the geologist's GPS.

Rock, talus, and soil samples collected at the Filo del Sol Project were analyzed by ALS Chemex and ACME laboratories in Chile. In detail, sample preparation and analytical methodology is poorly documented in the existing reports. Control samples such as duplicates, blank or standards were not inserted in the sequence. Rock samples were not used in the resource estimate.

ALS procedures included 27-element four-acid ICP-AES, Au fire assay Atomic Absorption finish and trace Hg by cold vapour/Atomic Absorption.

ACME procedures included 35-element four-acid or aqua regia digestion ICP-AES, Au fire assay Atomic Absorption finish, and trace Hg by cold vapour/Atomic Absorption.

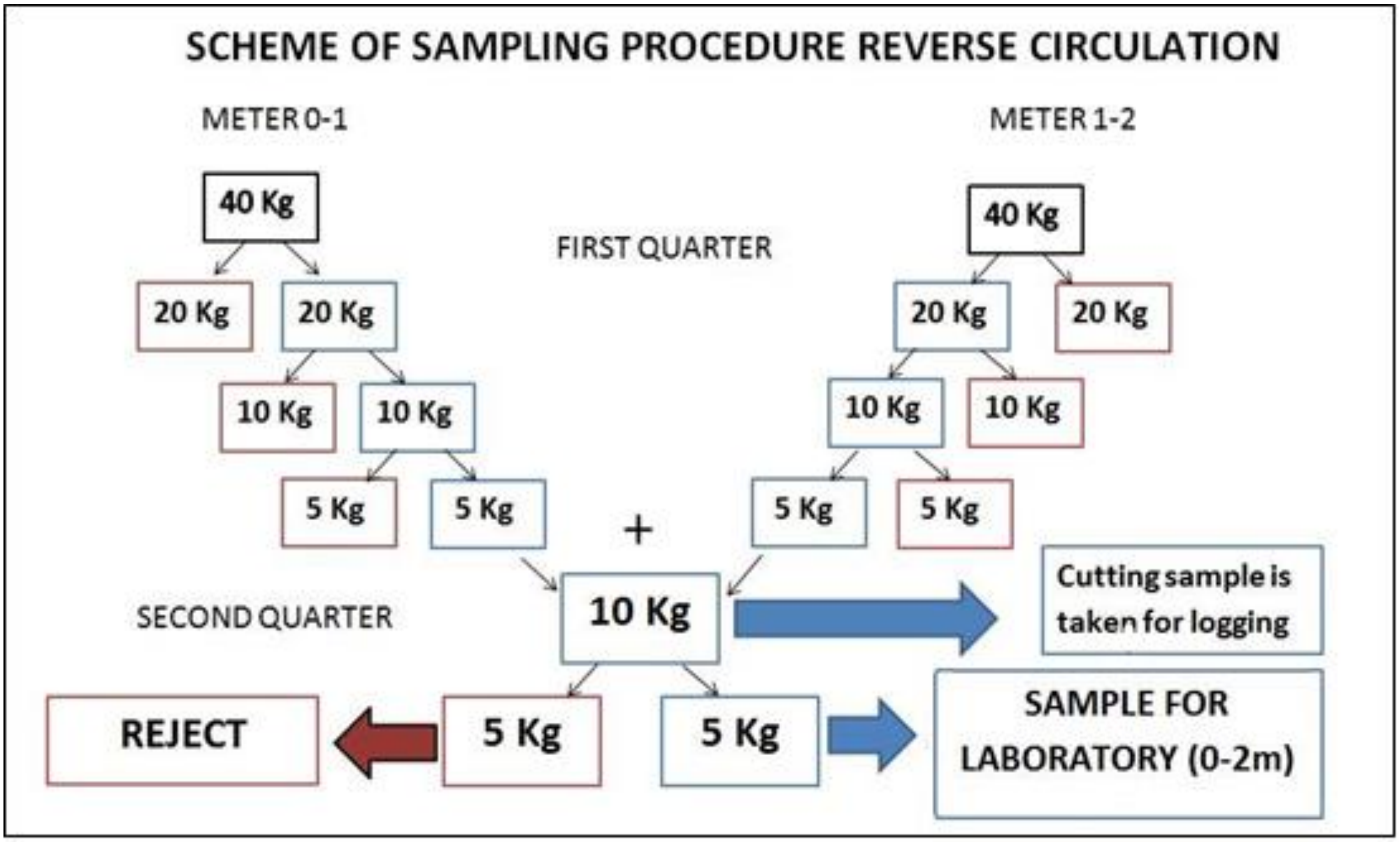
11.2 Drillhole Sampling

11.2.1 Reverse Circulation

For most of the drill programs to date, the sampling procedure for RC holes at the Filo del Sol project follows industry standards. Details regarding Cyprus's 1998/1999 procedures are not documented. The RC sampling method for holes drilled after 2000 is described below and represented in Figure 11-1. The procedure includes dividing the material homogeneously using a riffle splitter and combining two consecutive metres into one sample to be submitted for geochemical analysis.

The procedure began at the drill; the drill rig cyclone provided one sample per metre, of around 30 to 40 kg on average. After receipt of each one-metre sample, a primary quartering was manually made by technicians using a riffle splitter, thereby reducing the volume to 50%. At 50% of each drilled metre, a secondary quartering was conducted to reduce the volume to 25% of the initial 50%; this means recovering 5 kg from the initial 40 kg. The secondary quartering, in turn, enabled the preparation of a final representative sample of two drilled metres, and these two sample metres (5 kg each) are homogenized and result in a final weight of 5 kg for each two-metre sample for analysis and a second 5 kg sample for storage as coarse reject.

Figure 11-1: Flowchart of Sampling Process for RC Drilling



Source: Charchaflié and Gray, 2014

Samples were transported by truck from the splitting site near the drilling locations to the laboratory's preparation facilities. Samples dispatches are documented by the company's transportation bills in order to ensure sample tracking.

11.2.2 Core Management

Diamond drilling carried out in Filo del Sol in the 2005/2006, 2006/2007, 2010/2011, 2011/2012, 2012/2013, 2013/2014, 2014/2015, and 2017/2018 campaigns utilized the Copiapó core processing facility as a base. The more recent 2019/2020, 2020/2021, and 2021/2022 campaigns utilized a new core processing facility located near Rodeo, Argentina. For work programs prior to 2019, drill core was transported by Filo Mining personnel to the Company's core facility in Copiapó. Core was sampled continuously from the beginning of recovery to the end of the hole. Samples are generally two metres long (except for DDHV-01 that was sampled in one metre intervals). Drill core was initially cut in half using a circular, water-cooled rock saw. Starting in 2013/2014, DDH core was split using a manual core splitter under dry conditions as to minimize the soluble sulphate dissolution. Beginning in the 2017/2018 season, only core from the CuAuOx and M zones was split this way, other zones with no soluble copper were cut with a rock saw to better preserve the core. This is current practice with respect to splitting / cutting samples.

One half of the core was used as a geochemical sample and the other was stored in boxes or trays for reference and future revisions. Beginning in the 2019/2020 season where PQ and HQ diameter core was drilled, only ¼ of the core was sampled for geochemical analysis with the remaining ¾ stored in the core trays to facilitate eventual sampling for metallurgical testwork. The sampled material was put in a resistant plastic bag, labelled with sample number paper tickets identical to the ticket to be stapled on the core box or tray. Samples were then weighed and organized by number before being placed in rice sacks. These sacks were assigned an identification number that corresponds to the batch sent to the laboratory. Rice sacks were then delivered to the lab using a private courier with dispatch tracking. Beginning in 2011 and up to the 2018/2019 season, samples were delivered directly to ACME's preparation facilities in Copiapó by company personnel.

Starting in the 2019/2020 season, samples were delivered to the ALS prep lab in Mendoza for sample prep and shipment to ALS laboratory facilities either in Lima, Peru or Santiago, Chile for analysis.

No original records or indication from DDHV-01 and DVI-701-B samples are available.

11.3 Sample Analyses

Almost all holes were sampled in 2 m-intervals, and all were analyzed by either ALS Chemex Chile (prior to 2009/10 and from 2016 to 2018/2019), ACME Laboratories Chile (since 2010/2011 up to 2015) or ALS Global in either Lima, Peru or Santiago, Chile (2018/2019 to current).

ACME is an internationally certified laboratory. In 1994, ACME began adapting its Quality Management System to an ISO 9000 model. ACME implemented a quality system compliant with the International Standards Organization (ISO) 9001 Model for Quality Assurance and ISO/IEC 17025 General Requirements for the Competence of Testing and Calibration Laboratories. In 2005, the Santiago laboratory received ISO 9001:2000 registration and in July 2010 the Copiapó facility was added to the Santiago registration. The Santiago hub laboratory is also ISO 17025:2005 compliant since 2012 (http://acmelab.com/services/quality-control/). ISO/IEC 17025 includes ISO 9001 and ISO 9002 specifications, CAN-P-1579 (Mineral Analysis) for specific registered tests by the Standard Council of Canada (SCC). CAN-P-1579 is the SCC's requirements for the accreditation of mineral analysis testing laboratories.

ALS facilities operate to the higher of ISO 9001-2008 or ISO 17025 standards as appropriate to the services offered at each.

Both laboratories are completely independent of Filo Mining.

The analytical package used was multi-element, four-acid digestion ICP-AES, Au fire-assay Atomic Absorption finish and trace Hg by could vapour/Atomic Absorption. Beginning with the 2011/2012 season, the analytical package was changed to include Cu and Ag by AAS with a multi-acid digestion and Cu was also analyzed by sequential leach. Hg analyses were discontinued from drill samples.

RC holes drilled during the 2014/2015 season used the same sample preparation method and as described above; however, sample rejects from this campaign were stored in vacuum-sealed bags to preserve the samples from oxidation and enable them to be used for metallurgical testwork.

Laboratory sample preparation (either in Copiapó, Chile or Mendoza, Argentina) began with organizing the received batch and assigning a job order. Samples were sorted and weighed. If the number of samples differed from that indicated on the Requisition, the company was contacted. Sample preparation continued with:

- Drying in a large electric oven with temperature control
- Crushing to better than 85% passing 10 mesh
- Splitting to a 0.5 kg subsample
- Pulverizing the subsample to 95% passing 200 mesh
- Screen to pass 200 mesh

Bags with 150 g of pulp were submitted internally to the laboratory assaying facilities in Santiago, Chile or Lima, Peru. Gold was determined by fire assay with an AAS finish based on a 30 g sample. A suite of 37 (ACME) or 36 (ALS) elements, including copper, was determined by ICP-ES analyses. Starting in 2011/2012, Cu and Ag determinations in all samples were done by both ICP and AAS with a multi-acid digestion and Cu was also analyzed by sequential leach for ICP Cu assays greater than 500ppm.

11.4 Quality Control/Quality Assurance

11.4.1 Surface Sampling

No quality control program was implemented in relation to surface samples.

11.4.2 Drillhole Samples

Details of QA/QC programs for drilling campaigns prior to the 2016/2017 season are contained in Devine et al., 2016 and are only summarized here.

11.4.2.1 1998/99: Cyprus Drilling, RCV-02 to RCV-17

The quality control program applied to the Cyprus RC drill program consisted of one field duplicate inserted every 20 samples. No blank or standard material was used in the sampling program. Au, Ag and Mo duplicates show good correlation ($R2> 0.81$) whereas Cu duplicates display moderate correlation ($R2 = 0.61$), a result most likely associated with a single sample pair that assays 1,452 and 9,668 ppm. These results seem acceptable for all elements.

11.4.2.2 2000/01 Program: VRC01 to VRC21

The quality control program applied during the 2000/2001 drill campaign included same-laboratory (ALS) reject assaying and second laboratory (ACME) rig duplicates.

11.4.2.3 2003 to 2008 Programs: VRC25 to VRC55

The quality control program applied during the 2003 to 2008 drill campaigns included field duplicates only. A total of 185 (4.8%) field duplicates of 3,804 samples were randomly selected and analyzed as normal samples. Au, Ag, Cu and Mo duplicate samples show good correlation factors. Second laboratory analyses on a sub-set of samples collected between 2000 and 2008 was completed and is described in Section 12 of this report.

11.4.2.4 2013/14 Program: VRC56 to VRC79

A more rigorous quality control protocol was implemented in 2013, beginning with VRC56. The program included blanks, duplicates and standards inserted in the sampling sequence as well as second-laboratory analyses of a sub-set of samples. A total of 16 control samples were inserted every 174 submitted (9.1%). The control samples of every 174 sample-batch were:

- 2 Standard 1 (medium - about deposit average)
- 2 Standard 2 (low - about expected Cutoff)
- 2 Standard 3 (high – over expected Cutoff)
- 2 Field duplicate (second half core)
- 4 Blank (coarse material)
- 2 Preparation duplicate (make second pulp)
- 2 Assay duplicate (second assay)

This program has continued with minor modifications to the current drilling campaigns.

In total, 114 blank samples were analyzed and only one Cu failure occurred. The sample was re-assayed with similar results.

No failures were recognized in preparation and assay duplicates in 114 pairs of samples. Field duplicates have good correlation factors (Cu R2: 0.999 and Au R2: 0.876) and absolute differences expected in natural systems.

A total of 165 standards were included in the sampling sequence and only two failures were detected. All re-assayed samples were accepted as the grades fell within compliance limits.

A set of 160 pulps from the 2013/2014 drill campaign were selected for re-assaying at ALS laboratory Chile. In total, six standards were included in the sample stream. Grades ranged from 0.009% Cu to over 10% Cu, 0.062 ppm Au to 11.3 ppm Au, and 0.5 ppm Ag to 3,391 ppm Ag. High, medium and low-grade intervals were selected. Results indicate a very good correlation in copper, gold and silver (R2> 0.934) between ALS and ACME analyses. No bias between laboratories is observed, and results provided by both companies appear to be similar.

11.4.2.5 2014/15 Program; VRC80 to VRC96, RCVI18 to RCVI22

In total, 90 blank samples were analyzed. No Cu or Au failures occurred.

No failures were recognized in preparation and assay duplicates in 136 pairs of samples. Assay duplicates have Cu, Au and Ag R2 >0.991 whereas preparation duplicate's R2 >0.994. One duplicate pair returned an absolute difference higher than 0.5% Cu (original sample = 6.67%). Most likely the semi-failure is caused by inhomogeneous mineral dissemination in the sample and should be considered a natural event. Field duplicates have good correlation factors (Cu R2: 0.993, Au R2: 0.838 and Ag R2: 0.980) and absolute differences expected in natural systems.

In detail, 46 STD1, 47 STD2 and 46 STD3 were included in the sampling sequence. No copper or gold failures were detected. Copper and gold failures detected during the campaign would have generated a non-compliance report. The batch of samples comprised between failed and non-failed standards were to be re-assayed either by Cu AAS, Au FA or both.

Standards prepared for the 2014/2015 campaign were selected to produce Ag grades of > 2 (above detection limits), 50 ppm Ag and 150 ppm Ag. In detail, STD02 resulted to have a very narrow acceptance range (25 samples from the round robin comprised between 139 ppm Ag and 156 ppm Ag). Given the grade discrepancies, all semi-failures detected with STD02 were interpreted to represent assay uncertainty rather than true deviation from expected values.

11.4.2.6 2016/17 Program; VRC097 to VRC134

In total, 110 blank samples were analyzed. No Cu or Au failures occurred.

No failures were recognized in preparation and assay duplicates in 109 pairs of samples. Assay duplicates have Cu, Au and Ag R2 >0.981, whereas preparation duplicate's R2 > 0.985.

Field duplicates have good correlation factors (Cu R2: 0.993, Au R2: 0.838 and Ag R2: 0.980) and absolute differences expected in natural systems.

In detail, 56 STD1, 55 STD2 and 53 STD3 were included in the sampling sequence. No copper failures were detected. Gold failures were detected on report ME16226012 on STD2 and three generated a non-compliance report. The batch of samples comprised between failed and non-failed standards were re-assayed by Au FA on report ME17028750.

Standards prepared for the 2014/2015 campaign were selected to produce Ag grades of > 2 (above detection limits), 50 ppm Ag and 150 ppm Ag. In detail, STD02 resulted to have a very narrow acceptance range (25 samples from the round robin comprised between 139 ppm Ag and 156 ppm Ag). Given the grade discrepancies, all semi-failures detected with STD02 were interpreted to represent assay uncertainty rather than true deviation from expected values.

11.4.2.7 2017/18 through to 2022/23 Seasons

As described previously ,except for the introduction of certified reference material (CRM) being sourced from OREAS during the 2017/18 season, work continued to follow the established and proven procedures. All specifications for CRMs used can be found at their website, https://www.oreas.com.

Blanks are summarized below in Table 11-1. Accepted failures were scrutinized against the location in sequence of assays to determine if contamination was present. Contamination in Ag, Au and Cu was less than a percent of the assays preceding and following the blank material. Further, the inserted blanks preceding and following the Accepted Failure Blanks were also validated to ensure that any contamination was isolated. Performance of the 2021/22 and 2022/23 seasons is illustrated below in Figure 11-2.

Table 11-1: Summary of Blank Performance 2017/2018 to 2022/2023

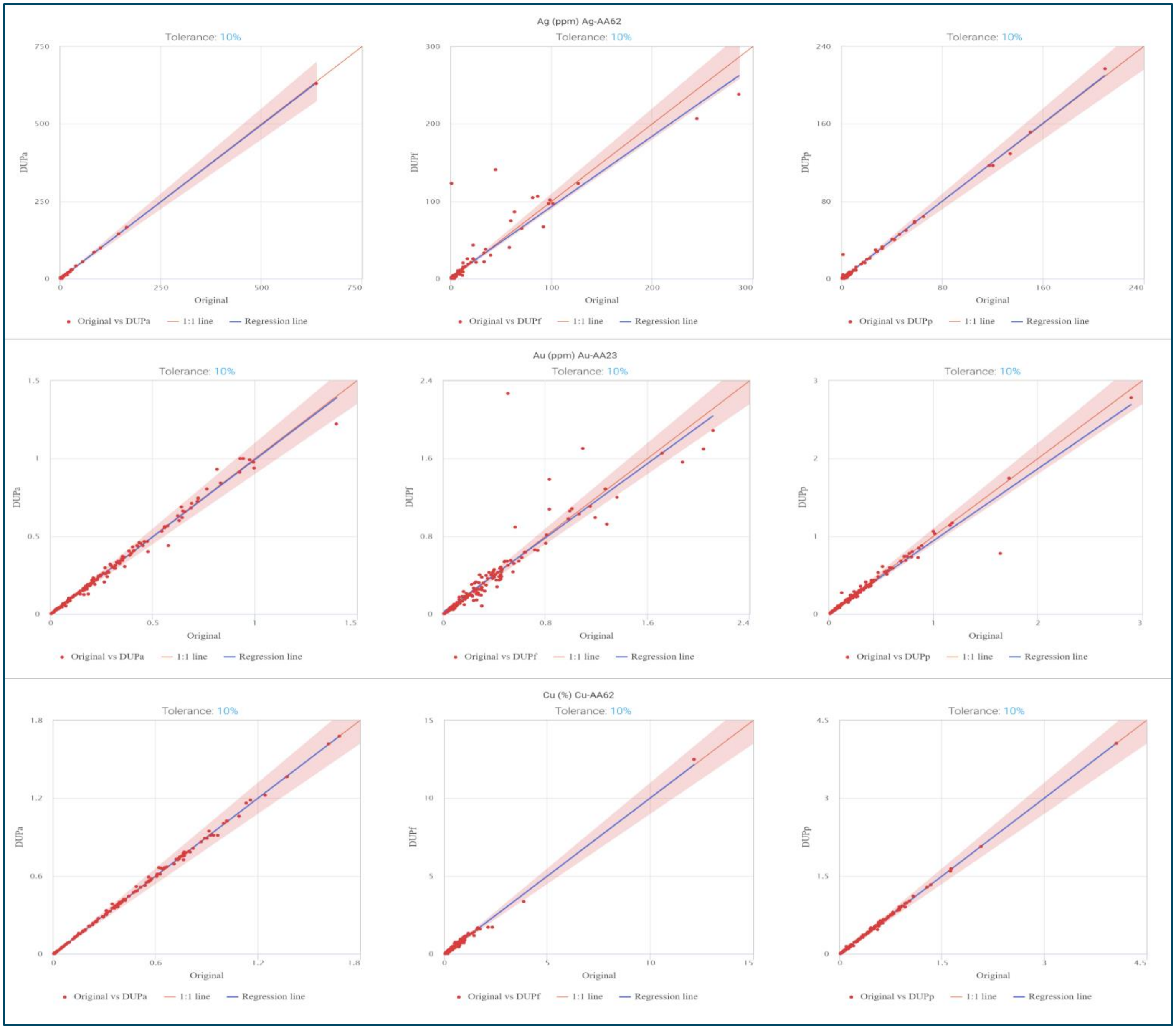
	Ag	Au	Cu
2017/2018	**113**	**113**	**113**
Accepted with Failure	1	6	3
Passed	112	107	110
2018/2019	**61**	**61**	**61**
Accepted with Failure		3	1
Passed	61	58	60
2019/2020	**50**	**50**	**50**
Accepted with Failure	2		
Passed	48	50	50
2020/2021	**133**	**133**	**133**
Accepted with Failure		1	2
Passed	133	132	131
2021/2022	**202**	**202**	**202**
Accepted with Failure	4	4	2
Passed	198	198	200
2022/2023	**97**	**97**	**97**
Passed	97	97	97
Grand Total	**656**	**656**	**656**

Figure 11-2: Illustration of Blank Performance for the 2021/22 and 2022/23 Seasons

Source: Filo Mining, 2023

Duplicates have been inserted in the sampling stream as described above. -Performance of the duplicates is illustrated with scatterplots below in Figure 11-3. The assay and preparation duplicates show a higher correlation than the field duplicates, specifically in the precious metals Au and Ag, which is typical.

Figure 11-3: Illustration of Duplicate Performance 2017/2018 to 2022/2023



Source: Filo Mining, 2023

Standards were inserted into the sample stream using CRM purchased from OREAS. Performance is summarized below in Table 11-2. Failures that are deemed inaccurate are followed up at the ALS laboratory for re-assay. Accepted CRM failures have been scrutinized against the preceding and following assays to determine the significance of the failure as well as how far outside the three standard deviation tolerance limit the value lies. Most of the accepted failures are very close to the tolerance limits as illustrated below in Figure 11-4. One exception is the performance of Ag in a low-grade CRM that has a value close to the detection limit of 1ppm Ag using assay method Ag-AA62. That CRM is failing at 12%

of the time. The Ag value is below 6 ppm with tolerance of +/- 0.6 ppm (3 standard deviations). The failures are not of concern because the atomic absorption method is accurate at higher grades of Ag, which are of interest.

Table 11-2: Summary of Standard Performance 2017/2018 to 2022/2023

	Ag	Au	Cu
2017/2018	**165**	**166**	**165**
Accepted with Failure	6	3	2
Passed	147	156	157
Undefined*	1		1
Warning	11	7	5
2018/2019	**88**	**88**	**88**
Accepted with Failure	2	1	
Passed	76	83	88
Warning	10	4	
2019/2020	**74**	**74**	**74**
Accepted with Failure	1		
Passed	67	71	71
Warning	6	3	3
2020/2021	**200**	**198**	**200**
Accepted with Failure	3	1	6
Passed	190	187	184
Warning	7	10	10
2021/2022	**301**	**301**	**301**
Accepted with Failure	27	5	2
Failed**	1	1	
Passed	179	214	187
Warning	94	81	112
2022/2023	**139**	**147**	**140**
Accepted with Failure	26	1	3
Passed	92	104	113
Warning	21	42	24
Grand Total	**967**	**974**	**968**

* Undefined: Analysis only Performed on Cu and Ag; Au not analyzed.
** Failed: Mislabelled Sample Number – Not a Failed Assay, Since Corrected.

11.4.3 Quality Control/Quality Assurance Summary

More than 83% of current RC and DDH dataset had a rigorous follow up with blanks, standards and laboratory duplicates. Another 5% has been checked with a second lab but does not have blank and standard controls. The remaining 12% of the dataset has only been verified (satisfactorily) with duplicates. No sample appears to be misplaced or intentionally deleted from the database. Sample preparation and security procedures are in place and adequate. In our opinion, the current drillhole dataset for the Filo del Sol Project is consistent and is of adequate quality to be used for Indicated resource estimation.

Figure 11-4: Illustration of Standard Performance 2017/2018 to 2022/2023

Source: Filo Mining, 2023

12 DATA VERIFICATION

J. Gray visited the core logging facility in 2014 and collected a suite of six coarse reject samples for independent analysis and comparison with the original values.

F. Devine was directly involved in the update of the geological model for the project area in 2015 - 2019, including completing extensive surface geological mapping and core logging, data and interpretation review and discussion with Company personnel. She visited the project again from October 9-11, 2022, to review the most recent drilling from 2019 – 2022 and geological model updates. Ten samples of quartered core were taken from diamond drill holes drilled over the past three years, from a range of Cu, Au, and Ag grade domains.

In 2014, the results of RC drilling from between 2000 – 2008 were checked with a large pulp resampling study.

Independent assaying of individual samples used to create metallurgical test composites was carried out by SGS Lakefield in 2016 and 2017. These results compare well with the original sample analyses.

In the opinion of the QPs, the data contained in this report is adequate to estimate an Indicated and Inferred Mineral Resource.

12.1 Verification of Geochemical Analyses for Diamond Drilling from 2019-2022

During a site visit to the core logging facility in Rodeo, Argentina from October 9 – 11, 2022, F. Devine collected ten samples of quartered core from diamond drill holes as a verification of geochemical drill data. The samples were packaged, labelled, and sealed by F. Devine and prepared for shipment by the crew. They were taken to Mendoza, Argentina by F. Devine and a geology team member, and delivered to the ALS prep lab by the geology team member the next day. They were run using the same protocol as was used for the original samples, as a duplicate.

The results for select elements are shown in Table 12-1 below and show that the duplicate samples agree well with the original values, taking into consideration the potential anomalies in sampling quartered core.

Table 12-1: Results of Duplicate Samples of Drill Core

Drill Hole	From (m)	To (m)	Cu (%)		Au (ppm)		Ag (ppm)		Mo (ppm)	
			orig.	dup.	orig.	dup.	orig.	dup.	orig.	dup.
FSDH064	534	536	0.587	0.610	0.432	0.458	1	1.51	26.6	23.5
FSDH047	336	338	0.345	0.333	0.076	0.12	1	2.11	51	48.1
FSDH063	506	508	0.392	0.421	0.386	0.392	22	27	16.15	14.25
FSDH070A	362	364	0.213	0.195	0.107	0.103	0.43	0.39	34	36.1
FSDH061	618	620	0.964	0.877	0.256	0.262	141	>100	130.5	114.5
FSDH069A	654	656	0.669	0.676	0.322	0.362	22.8	27.2	130	136
FSDH067	438	440	0.899	0.805	0.714	0.729	126	>100	107.5	116.5
FSDH067	596	598	0.992	0.970	1.05	1.095	1	1.72	34.3	35.5
FSDH062	684	686	0.658	0.689	0.285	0.345	3	1.92	64	47.6
FSDH068A	762	764	0.471	0.550	0.337	0.393	4.74	8.49	31.2	27.3

Drill Hole	From (m)	To (m)	As (ppm)		Acid Soluble Cu (%)		CN Soluble Cu (%)		Residual Cu (%)	
			orig.	dup.	orig.	dup.	orig.	dup.	orig.	dup.
FSDH064	534	536	581	635	0.054	0.052	0.468	0.483	0.052	0.05
FSDH047	336	338	947	979	0.03	0.033	0.16	0.175	0.12	0.109
FSDH063	506	508	1340	1565	0.01	0.008	0.34	0.356	0.01	0.008
FSDH070A	362	364	2.8	4.9	0.019	0.014	0.054	0.063	0.135	0.121
FSDH061	618	620	2800	2410	0.041	0.031	0.638	0.536	0.191	0.205
FSDH069A	654	656	2320	2400	0.034	0.026	0.53	0.527	0.026	0.031
FSDH067	438	440	3030	2670	0.056	0.052	0.712	0.663	0.027	0.027
FSDH067	596	598	328	330	0.094	0.083	0.722	0.754	0.139	0.116
FSDH062	684	686	914	881	0.079	0.079	0.502	0.532	0.06	0.07
FSDH068A	762	764	1275	1640	0.038	0.023	0.359	0.459	0.022	0.033

Source: Devine, 2023

12.2 Verification of RC Drilling Geochemical Analyses in 2014

A visit to the Copiapó office and support facilities was carried out by J. Gray and P. Geo. between 16th June 2014 and 21st June 2014; the project site was not visited by J. Gray. Martin Sanguinetti was the main contact; however, discussions were held with several geologists and sampling personnel. The focus of the visit was to gain an understanding of the processes and procedures related to geological interpretation of the project.

Site personnel provided a detailed overview of property geology and of the development of various components of the geological interpretation. Communication among staff and the leadership provided was very good.

The storage and sampling facilities in Copiapó were also visited; the site was well organized and tidy. Sampling staff explained the RC sample splitting process in a logical and concise manner. Six samples were taken from a variety of geological settings. Samples were coarse rejects and approximately 5 kg in size. Results of these independent samples are shown in Table 12-2 results agreed closely with the original values.

Table 12-2: Results of Six Independent Samples

Hole-ID	From (m)	To (m)	Gold Assay (g/t Au)		Silver Assay (g/t Ag)		Moly Assay (ppm Mo)		Arsenic Assay (ppm As)	
			Orig.	Indep.	Orig.	Indep.	Orig.	Indep.	Orig.	Indep.
VRC60	438	440	0.171	0.182	5.0	4.4	43	43	1014	615
VRC62	174	176	0.138	0.134	1.0	1.4	49	55	475	476
VRC65	44	46	2.458	2.316	45.0	45.1	6	6	1548	1398
VRC65	296	298	0.374	0.356	0.5	2.5	46	55	1226	902
VRC69	224	226	0.091	0.084	16.0	10.6	80	59	486	506
VRC77	374	376	0.174	0.246	4.0	4.0	26	26	844	535
Hole-ID	**From (m)**	**To (m)**	**Copper Assay (% Cu)**		**Acid Sol Cu Assay (% Cu)**		**CN Sol. Cu Assay (% Cu)**		**Resid. Cu Assay (% Cu)**	
			Orig.	**Indep.**	**Orig.**	**Indep.**	**Orig.**	**Indep.**	**Orig.**	**Indep.**
VRC60	438	440	0.463	0.429	0.046	0.046	0.3	0.277	0.109	0.124
VRC62	174	176	0.284	0.300	0.164	0.164	0.109	0.106	0.021	0.029
VRC65	44	46	0.271	0.272	0.263	0.260	0.008	0.006	0.002	0.004
VRC65	296	298	0.424	0.443	0.017	0.021	0.096	0.088	0.335	0.342
VRC69	224	226	0.222	0.217	0.209	0.201	0.006	0.006	0.004	0.007
VRC77	374	376	0.245	0.248	0.011	0.010	0.132	0.133	0.106	0.098

Source: Gray, June 19, 2014

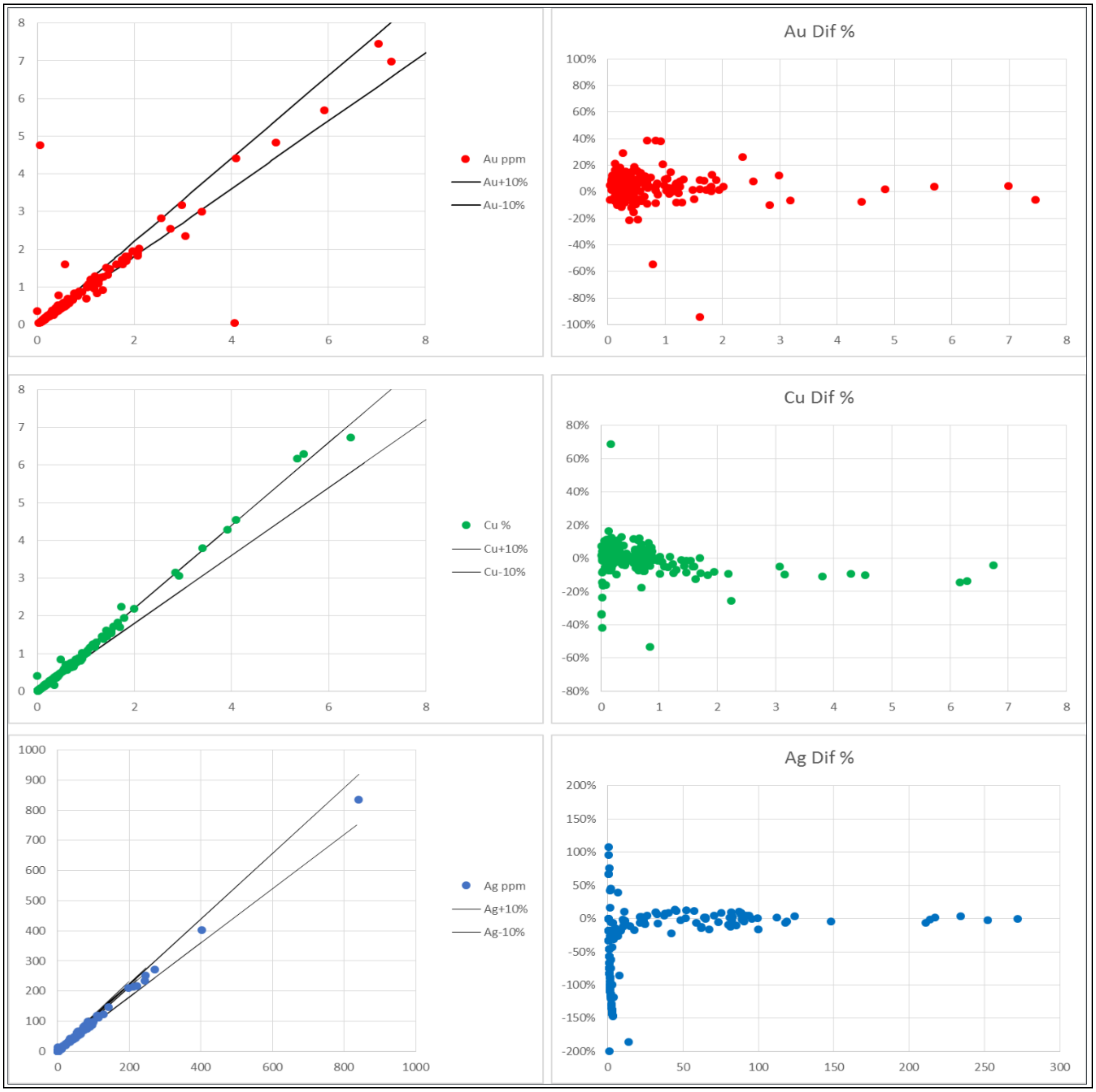
12.3 Verification of Au, Ag, and Cu Analyses for RC holes from 2000 to 2008

A total of 206 pulps from RC holes drilled between 2000 and 2008 were selected to perform an independent geochemical study in 2014 aimed to verify Au, Ag and Cu grades provided by the Company. Pulps were stored in Filo Mining facilities in San Juan, Argentina. The Company provided an inventory of available material to the 2014 Qualified Person and a list including sample numbers only was developed and pulps on the list were delivered to ACME Laboratories facilities in Mendoza, Argentina. Laboratory results were sent directly from the lab to D. Charchaflié via email in spreadsheet format under the Certificate number MEN14000462. Analytical methods used were FA430 (30 g Lead Collection Fire Assay Fusion - AAS Finish) and FA530 (Lead collection fire assay 30G fusion – Gravimetry finish) for Au, MA402 (4 Acid Digest AAS Finish) for Cu and Ag. The Mendoza and Santiago labs have ISO 9001:2008 accreditation issued by IRAM (Instituto Argentino de Normalizacion y Certificado).

Results from the original sampling and the re-assaying are compared in Figure 12-, which shows the results cluster mostly within the lines ± 10% uncertainty. Relative differences average 6%, -1% and -41% for Au, Cu and Ag respectively (negative when original < re-assay). In detail, Au and Cu grades show strong correlation factors. It must be noted that most of the original Cu and Ag grades were determined by ICP analyses whereas the re-assaying involved AAS. Silver grades have a moderate correlation factor, strongly influenced by the samples with grades below 10 ppm. If grades > 10 ppm are considered, then Ag reflects a very strong correlation.

Considering the uncertainties involved in pulp re-assaying and ICP methodologies, these results are considered a satisfactory confirmation of the results reported by Filo Mining.

Figure 12-1: Re-assay Results from Pulps Drilled Prior to 2008



Source: Charchaflié, 2014

12.4 Verification of Collar Locations

Most of the drill platforms for the RC drilling are visible on the GEOEYE satellite images (0.5 m resolution) acquired by Filo Mining in 2010. From 2015 to 2019, F. Devine visited numerous drill platforms during the course of field mapping and confirmed their location within the project area. In addition, in 2014, seven drill hole sites were visited by Company personnel and their location measured by a hand-held Garmin GPS. Filo Mining measurements, by hand-held GPS and later differential GPS, are also shown in Table 12-3. In general, the agreement in eastings and northings between the verification measurements and differential GPS data is excellent (< 4 m). Altitude agreement, as expected, is acceptable but less accurate given the handheld GPS vertical uncertainty.

Table 12-3: Confirmation of Drill Hole Collar Locations

Hole ID	Database Coordinates			GPS Check			Difference		
	East	North	Altitude	East	North	Altitude	E Diff (m)	N Diff (m)	Alt Diff (m)
VRC61	435294.73	6848604.80	5120.50	435293	6848606	5102	2	-1	19
VRC64	435100.00	6848500.20	5216.54	435097	6848496	5225	3	4	-8
VRC66	434901.02	6848799.01	5275.92	434902	6848800	5283	-1	-1	-7
VRC67	434995.88	6848408.29	5265.14	434995	6848410	5266	1	-2	-1
VRC69	435099.56	6848695.59	5201.74	435100	6848696	5206	0	0	-4
VRC74	435269.44	6848737.91	5146.70	435268	6848741	5150	1	-3	-3
VRC75	435084.38	6848321.16	5200.95	435085	6848319	5213	-1	2	-12

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

To date, the metallurgical test programs on the Filo del Sol deposit have been carried out in four phases. The first phase was conducted in 2001 by Novatech S.A. of Santiago, Chile, on various samples of the oxide and mixed zones. The second phase was conducted by SGS Minerals (Lakefield) in 2016 on one sample of each of the oxide gold, oxide copper, and mixed silver mineralization. The third phase was conducted at SGS Minerals (Lakefield) in 2017 on samples from several different zones of mineralization within the deposit. The fourth, more comprehensive, phase was conducted at SGS Minerals (Lakefield) in 2018 on various samples from the four main zones (Tamberías gold oxide (TMB AuOx), Filo del Sol gold oxide (FDS AuOx), Tamberías copper-gold oxide (TMB CuAuOx) and Filo del Sol copper-gold oxide (FDS CuAuOx) + M-Zone (M-Ag)).

13.1.1 Phase I: Novatech 2001

A preliminary test program was completed in 2001, consisting of bottle rolls and diagnostic leaches, on 20 samples of RC chips. Chips were collected from four holes drilled during the 2000/01 season, from depths between 100 and 300 metres below surface. Four of the holes, VRC002, 004 and 006, were drilled on the same section (8600N) and span an east-west distance of 500 metres. The fourth hole, VRC005 was drilled 380 metres to the south of this section. All holes are in the Filo del Sol portion of the deposit.

Results of the bottle roll tests are presented in Table 13-1 below. The metallurgical zones reported are based on the current interpretation of mineral zonation.

Excellent results were obtained for the recovery of copper with dilute sulphuric acid solution, including several samples which leached with only water and generated acid. Average copper extraction was 76%.

This work is superseded by the subsequent programs in 2016/2017/2018.

Table 13-1: Bottle Roll Test Results

Sample	Head Grade (%CuT)	2018 Mineral Zone	Sulphuric Acid Cons (kg/t)	Total Cons. (kg/t)	Net Cons. (kg/t)	Copper Recovery (%)
4368 VRC-02	0.51	CuAuOx	30.13	18.47	13.40	64.58
4363 VRC-02	0.91	CuAuOx	11.07	-8.82	-21.05	87.22
4597 VRC-04	0.73	M-Ag	0.00	-158.07	-166.44	98.30
4521 VRC-04	1.17	CuAuOx	0.83	-39.50	-55.34	89.23
4611 VRC-04	0.90	CuAuOx	12.52	-11.84	-16.40	47.24
4601 VRC-04	1.71	M-Ag	0.00	-57.76	-72.57	56.22
4578 VRC-04	0.60	M-Ag	0.00	-51.30	-59.05	91.21
4598 VRC-04	1.26	M-Ag	0.00	-146.64	-163.52	87.02
4588 VRC-04	0.83	M-Ag	0.00	-107.01	-118.42	93.15
4559 VRC-04	0.33	CuAuOx	2.72	-1.80	-4.36	50.42
4690 VRC-05	1.08	CuAuOx	0.00	-64.46	-80.20	94.64
4694 VRC-05	1.06	M-Ag	0.00	-148.02	-162.61	91.97
4667 VRC-05	1.91	M-Ag	0.00	-40.25	-66.88	90.53
4711 VRC-05	0.93	CuAuOx	3.33	-6.49	-15.33	78.38
4700 VRC-05	0.56	M-Ag	5.88	-16.55	-22.87	91.77
4661 VRC-05	0.76	M	7.64	-6.80	-17.04	92.27
4675 VRC-05	4.11	CuAuOx	0.00	-32.32	-92.32	94.79
4718 VRC-05	0.45	CuAuOx	4.55	-5.10	-9.47	63.08
5309 VRC-06	1.23	CuAuOx	0.00	-4.89	-9.51	36.99
5312 VRC-06	1.58	CuAuOx	3.08	0.12	-5.64	27.32

Source: Novatech, 2001

13.1.2 Phase II: SGS Minerals (Lakefield), 2016

Bottle roll tests were completed on three composite samples created from RC chips (crushed to 100% minus 10 mesh) of three different types of mineralization from seven drill holes within the deposit. These holes span a distance of 1,300 metres from south to north. Table 13-2 shows the holes and intervals that were used to create the composites.

Table 13-2: Phase II Sample Selection

Zone	Drillhole	From	To
AuOx	VRC082	158	188
	VRC085	144	168
M	VRC080	206	250
	VRC081	278	292
	VRC086	308	330
CuAuOx	VRC080	182	200
	VRC088	106	156
	VRC089	214	234

All bottle rolls tests were conducted at 20% solids for 96 hours, with pH ~1.8 for the copper oxide sample, and 1 g/L NaCN for the gold oxide and the mixed silver sample. Results are presented in Table 13.3 below.

Table 13-3: Phase II Bottle Roll Test Results

Zone	Head Grade	Recovery	Reagent Cons. (kg/t)	
			H_2SO_4	NaCN
Oxide Copper-Gold (CuAuOx)	0.33 g/t Au; 0.44% Cu	95.1% Cu, 87% Au	0	-
Oxide Gold (AuOx)	0.49 g/t Au; 0.02% Cu	93.2% Au	-	0.67
Silver (M)	0.34 g/t Au; 0.29% Cu; 103 g/t Ag	88.6% Au; 92.4% Cu; 92.7% Ag	-	10.0

In the CuAuOx zone, copper was readily soluble using just water, with very fast leaching kinetics. After copper leaching, a test was conducted to cyanide leach the gold in the copper leach residue (after thorough washing and neutralization). This sequential leach process recovered 87% of the gold from the CuAuOx sample. Metal extractions from the mixed silver sample by cyanide leach were good but at the cost of high cyanide consumption.

A SART test (sulphidization-acidification-recycling-thickening) indicated that >98% of the cyanide consumed could be regenerated and recycled, while the bulk of the copper could be removed from solution as a copper sulphide precipitate, assaying approximately 65% Cu.

13.1.3 Phase III: SGS Minerals (Lakefield), 2017

Following information learned during the 2016/2017 field season, updated drill results, and metallurgical testwork completed in 2016, the deposit was reclassified into four zones based on the metallurgical characteristics. These zones are described in more detail in Section 7.3, and include: a gold oxide zone (AuOx) (two areas: Filo del Sol (FDS AuOx) and Tamberías (TMB AuOx); a copper-gold oxide zone (CuAuOx) (two areas: Filo del Sol (FDS CuAuOx) and Tamberías (TMB CuAuOx), a copper-rich “M” zone (FDS M-Cu) and a silver-rich “M” zone (FDS M-Ag).

For process planning purposes, a fifth type of mineralization, CuOx, was differentiated. This material is a low-gold subset of the CuAuOx in which the gold content was expected to be uneconomical to recover. This material was not tested separately, as the relevant recovery parameter is the acid-leach recovery of copper, which was adequately tested with the CuAuOx samples.

Samples selected for this phase of testwork were a combination of bulk samples collected from surface exposures and RC chips from several drill holes. Coarse bulk sample material was used for column leach tests, while both surface and RC samples were used for bottle roll tests to evaluate variability within the deposit. RC samples ranged from 4 to 330 metres below surface.

For the AuOx zone, two surface samples from Filo del Sol, two surface samples from Tamberías, five RC samples from Filo del Sol, and five RC samples from Tamberías were collected. For the CuAuOx zone, two surface samples from Filo del Sol, two surface samples from Tamberías and four RC samples from Filo del Sol were collected. Sample locations are given in Table 13-4 and Table 135.

Table 13-4: Phase III RC Sample Locations

Zone	Hole ID	From	To
FDS AuOx	VRC67	132	160
	VRC69	2	32
	VRC70	116	140
	VRC82	110	130
	VRC85	102	114
FDS CuAuOx	VRC64	206	214
		220	232
	VRC65	6	18
	VRC75	138	166
	VRC76	100	124
FDS M-Ag	VRC62	266	292
	VRC63	226	248
	VRC64	260	292
	VRC72	166	188
	VRC76	224	254
	VRC86	300	328
FDS M-Cu	VRC70	146	150
		156	158
	VRC73	148	176
TMB AuOx	VRC133A	4	6
	VCR133B	50	52
	VCR134A	12	14
	VCR134B	108	110
	VCR109A	14	16

Table 13-5: Phase III Bulk Surface Sample Locations

		From		To	
VRC065	FDS AuOx	434,865	6,847,598		
VRC068	FDS AuOx	435,098	6,848,702		
TR4	TMB AuOx	434,945	6,846,496	434,987	6,846,493
TR2	TMB AuOx	434,686	6,846,772	434,725	6,846,773
VRC059 "tanque"	FDS CuAuOx	434,991	6,847,800		
VRC020	FDS CuAuOx	434,995	6,847,714		
TR3	TMB CuAuOx	434,893	6,846,618	434,927	6,846,628
TR4	TMB CuAuOx	434,791	6,846,446	434,829	6,846,469

Details of the results of the Phase III program are described in Devine et.al. 2017 (Independent Technical Report for a Preliminary Economic Assessment on the Filo del Sol Project, Region III, Chile and San Juan Province, Argentina).

13.2 Phase IV: SGS Minerals (Lakefield), 2018

13.2.1 Geometallurgical Domains and Mineralogy

The 2018 test program was designed to test the mineralization domains based on their preferred processing strategy, as suggested by their mineralogical characterization and host geology. Samples were differentiated based both on mineralization type and location (Filo vs. Tamberías), with the location also reflecting differences in overall geological setting (see Section 7.3). Samples were collected from the following zones:

- Gold Oxide (AuOx) – Filo and Tamberías;
- Copper-Gold Oxide (CuAuOx) – Filo and Tamberías;
- Silver-rich Mineralization (M-Ag) – Filo only (this mineralization does not exist in Tamberías sector).

Samples were a mix of bulk surface samples from trenches, diamond drill core (PQ, HQ and NQ size), and RC chips. In addition to the individual variability samples for each mineralization type, master composites were created by combining several of the individual samples. Details of the makeup and naming of these composites are provided in the sections below.

A sub-sample of the FDS CuAuOx was created to try to characterize material with a high cyanide-soluble copper component. This sample was called FDS CuCN and comprises four drill core samples and one composite. The final composite sample contained 40.7% cyanide-soluble copper, which is within the overall range of the FDS CuAuOx mineralization type, and therefore this sample represents a part of the FDS CuAuOx material and is not a separate mineralization type.

Surface samples were collected with an excavator, with approximately 300 kg of material collected from each location. This material was transported to the Filo Mining facility in Copiapó where it was screened to minus 2.5 inch, homogenized and divided into 20 kg vacuum-sealed sample bags for shipment. A total of three surface samples were collected for TMB AuOx, 3 for FDS AuOx, 6 for TMB CuAuOx, 2 for FDS CuAuOx, and none were collected for M-Ag.

Drill core samples were collected from diamond drill holes completed during the 2017/2018 season. A total of 12 holes representing 2,533 metres of core was drilled and 439 metres of this was used for metallurgical testwork (164 m of PQ, 167 m of HQ, 108 m of NQ). Sample intervals were selected based on geological characteristics supported by NITON portable XRF analysis for Cu grades and, for the AuOx samples, Au assays. For the sample intervals selected, all of the core from each 2-metre sample was homogenized and split into four equal sub-samples. One sub-sample was submitted for assay, one was kept as a reference sample, and two were combined to form the metallurgical sample. All individual samples from each overall interval were then combined to form the final sample. A total of five drill core samples were collected for FDS AuOx, 11 for FDS CuAuOx, and 4 for M-Ag. No drill core was available for Tamberías samples.

RC drill samples were collected from splits of sample rejects after homogenization. These were used exclusively for bottle roll testing due to the small particle size distribution. A total of three RC samples were collected for TMB AuOx, 5 for FDS AuOx, 3 for TMB CuAuOx, 12 for FDS CuAuOx, and 12 for M-Ag.

Table 13-6 shows the number of samples for each mineralization type.

Table 13-7 shows the head sample characterization for the 2018 program.

Table 13-6: Number of Samples for Mineral Types

Mineralization Type	Surface Samples	Drill Core Samples	RC Samples	Total
FDS AuOx	3	5	5	13
TMB AuOx	3	0	3	6
FDS CuAuOx	2	11	12	25
TMB CuAuOx	6	0	3	9
M-Ag	0	4	12	16

Table 13-7: Sample Characterization Program

Sample Name	Mineralization Type	Element Cu (%)	Sequential Copper (%)			Au (g/t)	Ag (g/t)	Fe (%)	As (%)	Al (%)	Hg (g/t)
			H2SO4	CN	Residue						
F18G-T01	FDS AuOx	< 0.01				0.04	< 0.5	1.43	0.008	0.30	< 0.3
F18G-T02	FDS AuOx	< 0.01				0.02	< 0.5	0.93	0.007	1.84	< 0.3
F18G-T03	FDS AuOx	< 0.01				< 0.02	< 0.5	0.43	0.006	3.03	< 0.3
F18G-Comp	FDS AuOx	0.02				0.35	1.0	0.66	0.011	0.86	4.5
FSDH017A (114-188)	FDS AuOx	0.02				1.18	1.2	2.11	0.002	0.69	12.8
FSDH018A (96-164)	FDS AuOx	0.02				0.26	< 0.5	0.15	0.014	0.25	12.0
FSDH019 (140-202)	FDS AuOx	0.02				0.28	5.3	0.22	0.053	0.43	3.4
FSDH020 (128-182)	FDS AuOx	0.02				2.44	1.6	0.11	0.001	0.21	3.6
FSDH024 (96-122)	FDS AuOx	0.02				0.28	1.2	0.28	0.007	0.80	2.3
VRC073 (56-68)	FDS AuOx	0.01				0.38	< 0.5	0.21	0.007	1.12	1.1
VRC097 (152-164)	FDS AuOx	0.02				5.18	1.2	0.30	0.004	0.19	3.1
VRC097 (8-18)	FDS AuOx	0.02				0.43	0.7	3.51	0.008	3.99	1.4
VRC121 (98-108)	FDS AuOx	< 0.01				1.06	0.7	1.19	0.015	4.82	18.6
VRC122B (214-224)	FDS AuOx	0.03				0.68	4.4	0.52	0.004	0.48	71.7
T18G-Comp	TMB AuOx	< 0.01				0.55	10.0	0.35	0.032	0.17	0.7
T18G-T01	TMB AuOx	< 0.01				0.30	17.9	0.29	0.068	0.19	2.7
T18G-T02	TMB AuOx	0.01				0.60	8.8	0.22	0.003	0.22	0.6
T18G-T03	TMB AuOx	0.03				0.89	5.6	0.75	0.051	0.19	<0.3
VRC111 (126-134)	TMB AuOx	0.07				0.38	1.0	3.01	0.013	8.74	0.3
VRC113 (134-144)	TMB AuOx	0.04				0.48	2.8	0.50	0.051	0.32	5.0
VRC119 (16-26)	TMB AuOx	< 0.01				0.63	1.2	2.18	0.058	5.11	1.6
F18Cu-T01	FDS CuAuOx	0.68	0.66	0.00	0.01	0.49	69.3	0.42	0.032	0.20	0.4
F18Cu-T02	FDS CuAuOx	1.05	0.96	0.01	0.01	0.56	3.3	0.15	0.028	0.21	0.6
F18 Cu-Comp	FDS CuAuOx	0.65	0.59	0.03	0.01	0.31	11.8	2.78	0.080	1.94	9.4
FSDH016 (50-68)	FDS CuAuOx	0.25	0.24	0.00	0.00	0.16	11.7	4.02	0.081	0.25	4.9
FSDH017A (256-272)	FDS CuAuOx	0.62	0.63	0.00	< 0.001	0.23	1.7	2.87	0.029	0.63	22.8
FSDH018A (264-328)	FDS CuAuOx	0.28	0.20	0.06	0.01	0.20	1.6	4.36	0.041	5.63	1.7
FSDH020 (226-291)	FDS CuAuOx	0.31	0.28	0.04	0.01	0.42	5.3	3.19	0.10	2.57	21.0
FSDH021 (110-134)	FDS CuAuOx	1.66	1.57	0.06	0.00	0.34	4.4	5.42	0.26	0.28	38.9
FSDH023 (96-130)	FDS CuAuOx	1.22	1.23	0.02	0.01	0.20	3.8	4.77	0.16	0.22	9.1
FSDH024 (150-194)	FDS CuAuOx	0.29	0.24	0.05	0.01	0.20	0.9	4.11	0.060	7.13	0.9
VRC066 (296-306)	FDS CuAuOx	0.72				0.25	0.8	3.68	0.092	3.96	3.6
VRC077 (90-100)	FDS CuAuOx	2.58				0.20	1.5	3.08	0.028	0.15	10.2
VRC079 (168-178)	FDS CuAuOx	1.00				2.21	22.7	2.24	0.290	0.18	6.4
VRC088 (118-128)	FDS CuAuOx	0.41				0.37	33.9	3.53	0.180	0.22	9.5
VRC101 (242-252)	FDS CuAuOx	0.37				0.19	1.4	2.85	0.035	4.90	4.7
VRC123 (230-240)	FDS CuAuOx	0.14				0.21	3.3	2.63	0.032	3.56	5.4
F18 CuCN-Comp	FDS CuCN	3.37	1.87	1.29	0.01	0.31	1.0	2.87	0.140	0.26	8.8
FSDH022 (106-116)	FDS CuCN	10.60	3.80	6.31	0.01	0.29	1.2	3.01	0.240	0.46	22.3
FSDH022 (116-130)	FDS CuCN	0.93	0.89	0.01	0.01	0.28	49	3.00	0.050	0.50	13.1
FSDH022 (130-140)	FDS CuCN	0.26	0.23	0.00	0.01	0.36	8.2	4.21	0.040	0.22	16.7
FSDH022 (96-106)	FDS CuCN	0.88	0.81	0.01	0.01	0.26	3.1	1.20	0.260	0.22	5.1
VRC066 (238-250)	FDS CuCN	2.65				0.10	1.8	2.69	0.032	6.61	24.7
VRC085 (226-234)	FDS CuCN	3.92				0.24	3.2	2.36	0.120	4.44	48.2
T18Cu-Comp	TMB CuAuOx	0.41	0.31	0.03	0.04	0.25	0.8	3.72	0.019	8.59	<0.3
T18Cu-T01	TMB CuAuOx	0.55	0.43	0.02	0.05	0.38	1.2	4.92	0.003	8.49	0.5
T18Cu-T02	TMB CuAuOx	0.37	0.32	0.02	0.02	0.15	< 0.5	2.38	0.064	10.1	< 0.3
T18Cu-T03	TMB CuAuOx	0.69	0.45	0.05	0.13	0.24	< 0.5	7.81	0.005	8.56	< 0.3
T18Cu-T04	TMB CuAuOx	0.29	0.23	0.02	0.04	0.37	0.8	3.62	0.002	6.96	< 0.3
T18Cu-T05	TMB CuAuOx	0.57	0.48	0.03	0.01	0.24	0.8	1.50	0.018	7.74	< 0.3
T18Cu-T06	TMB CuAuOx	0.48	0.42	0.02	0.01	0.30	0.5	2.90	0.015	5.42	< 0.3
VRC111 (58-68)	TMB CuAuOx	0.96				0.47	1.6	3.31	0.017	7.94	1.8
VRC112 (20-30)	TMB CuAuOx	0.44				0.40	0.8	5.41	0.004	9.03	< 0.3
VRC119 (40-50)	TMB CuAuOx	0.59				0.44	0.6	2.69	0.020	7.88	16.1
F18 M-Ag-Comp	M-Ag	0.95	0.84	0.07	0.01	0.30	474	4.53	0.11	3.76	284
FSDH016 (78-90)	M-Ag	0.24	0.16	0.07	0.01	0.18	478	3.93	0.089	3.39	84.7
FSDH017A (272-310)	M-Ag	1.25	1.24	0.02	0.01	0.22	88.7	3.83	0.072	0.84	111
FSDH021 (148-158)	M-Ag	1.38	1.27	0.04	0.01	0.43	824	5.50	0.11	3.02	631
FSDH023 (162-186)	M-Ag	0.58	0.45	0.10	0.01	0.34	417	4.48	0.12	6.47	149
VRC060 (82-110)	M-Ag	0.26				0.24	108	4.11	0.130	0.23	53.3
VRC062 (270-286)	M-Ag	0.48				0.37	378	5.96	0.160	0.17	68.8
VRC062 (286-296)	M-Ag	0.64				0.67	43.9	8.78	0.210	0.53	52.5
VRC063 (262-288)	M-Ag	0.22				0.34	93.4	7.68	0.093	7.04	47.7
VRC065 (86-110)	M-Ag	0.36				0.25	49.2	4.73	0.270	0.49	37.5
VRC074 (230-254)	M-Ag	0.28				0.65	200	11.8	0.065	1.27	310
VRC080 (210-250)	M-Ag	0.24				0.40	89.9	3.28	0.110	0.23	14.8
VRC086 (296-306)	M-Ag	0.63				0.35	301	3.33	0.110	0.09	59.7
VRC100 (306-330)	M-Ag	0.45				0.22	170	6.22	0.150	0.11	214
Copper Blend #1	Blend - All	0.91	0.66	0.15	0.02	0.29	9.4	3.47	0.060	4.15	7.1
Copper Blend #2	Blend - All	0.68	0.54	0.04	0.02	0.32	103	3.34	0.072	4.25	43.4

13.2.2 Physical Characterization

Various samples from the Filo del Sol and Tamberías zones were submitted to a series of industry standard physical characterization tests including Bond low energy impact test (SGS Vancouver laboratory), Bond rod mill index, Bond Ball mill index, abrasion index, and JK Drop-weight test (SMC tests).

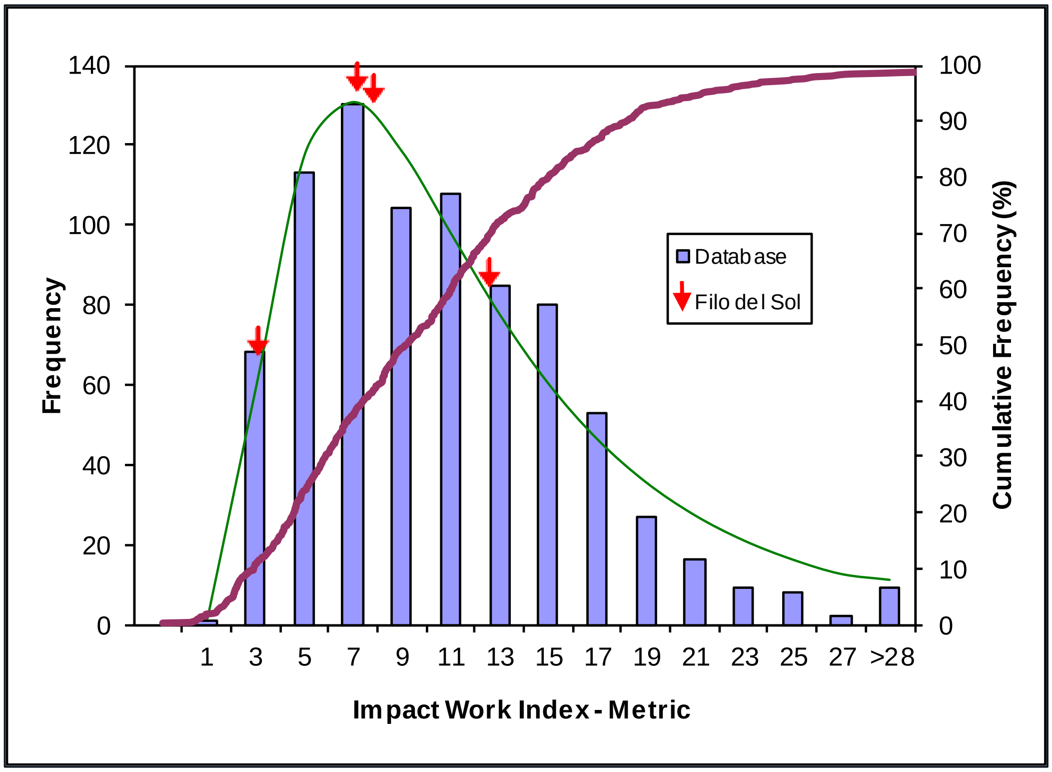
A total of 192 samples were tested and results are summarized in Table 13-8 (Bond low-energy impact), Table 13-9 (SMC), Table 13-10 (Bond rod mill grindability),

Table 13-11 (Bond ball mill grindability), and Table 13-12 (Bond Abrasion). Figure 13-1 to Figure 13-4compare the results obtained on these samples against distribution curves from SGS database.

Table 13-8: Bond Low-Energy Impact Testing (Summary)

Year	Sample Name	Zone	Number of Specimens	Work Index (kWh/t)	Min. (kWh/t)	Max. (kWh/t)	S.D. (kWh/t)	Relative Density	Hardness Percentile
2017	FDS VRC 065	FDS AuOx	18	9.6	4.9	20.1	3.3	2.20	49
	FDS VRC 068	FDS AuOx	18	6.6	2.6	15.7	3.7	2.36	28
	TMB TR2	TMB AuOx	20	9.2	2.3	18.1	4.3	2.32	46
	TMB TR4	TMB AuOx	19	8.5	4.0	17.3	3.7	2.43	41
	FDS VRC 059	FDS CuAuOx	20	9.6	2.5	24.5	4.9	2.33	49
	TMB TR2	TMB CuAuOx	19	6.9	2.1	11.1	2.7	2.53	30
2018	F18 G Comp	FDS AuOx	18	3.1	0.6	9.0	2.3	2.18	3
	F18 Cu Comp	FDS CuAuOx	20	7.2	2.1	14.6	4.1	2.33	32
	T18 Cu Comp	TMB CuAuOx	20	7.9	4.9	16.0	2.7	2.57	37
	T18 G Comp	TMB AuOx	20	12.6	2.9	22.0	4.6	2.27	69

Figure 13-1: Bond Low Energy Impact Test Comparison



Source: Ausenco, 2019

Table 13-9: SMC Testing (Summary)

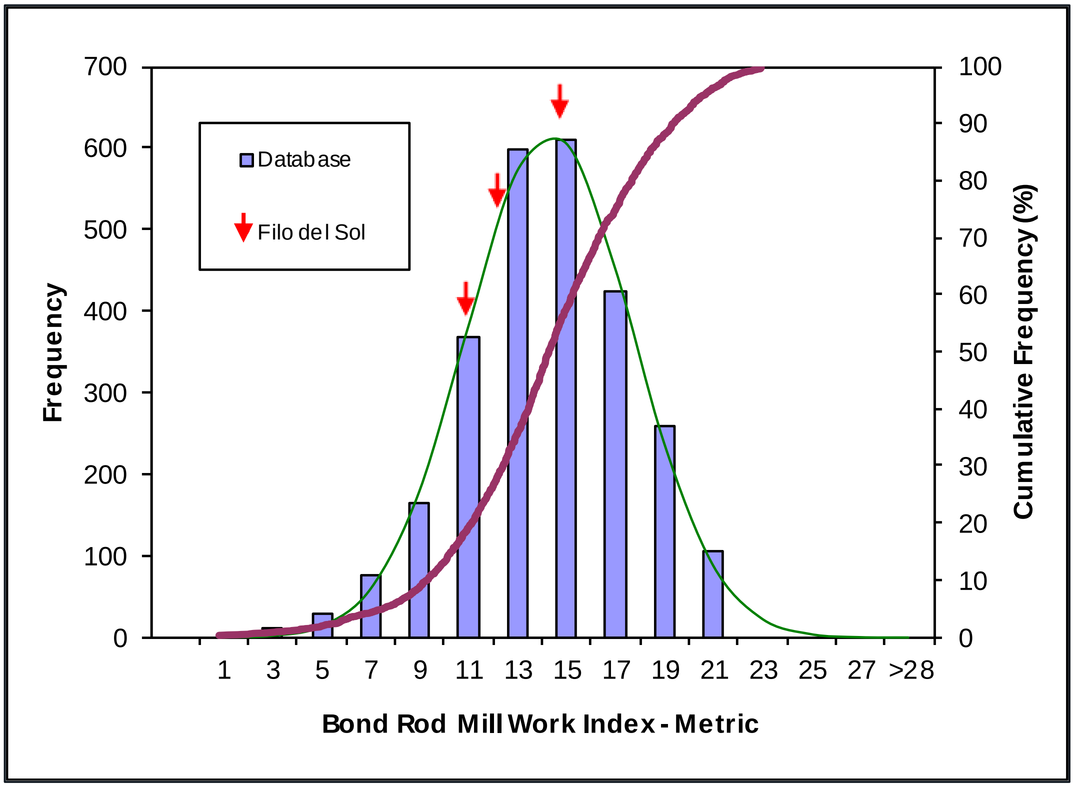
Sample Name	A	b	A x b	Hardness Percentile	T_a[1]	DWI (kWh/m³)	M_{ia} (kWh/t)	M_{ih} (kWh/t)	M_{ic} (kWh/t)	SCSE (kWh/t)	Relative Density
F18 Cu Comp	66.3	1.79	119	8	1.41	1.8	8.6	4.09	2.5	6.9	2.18
F18 M-Ag Comp	59.6	1.46	87.0	15	0.98	2.6	10.8	6.6	3.4	7.4	2.30
T18 Cu Comp	73.2	0.54	39.5	62	0.41	6.4	20.2	14.8	7.6	9.7	2.50

[1]The Ta value reported as part of the SMC procedure is an estimate.

Table 13-10: Bond Rod Mill Grindability (Summary)

Sample Name	Mesh of Grind	F_{80} (µm)	P_{80} (µm)	Gram per Revolution	Work Index (kWh/t)	Hardness Percentile
F18 Cu Comp	14	8,828	897	12.48	12.2	26
F18 M-Ag Comp	14	9,942	887	14.05	10.9	16
T18 Cu Comp	14	10,885	901	8.65	14.7	56

Figure 13-2: Bond Rod Mill Work Index Comparison

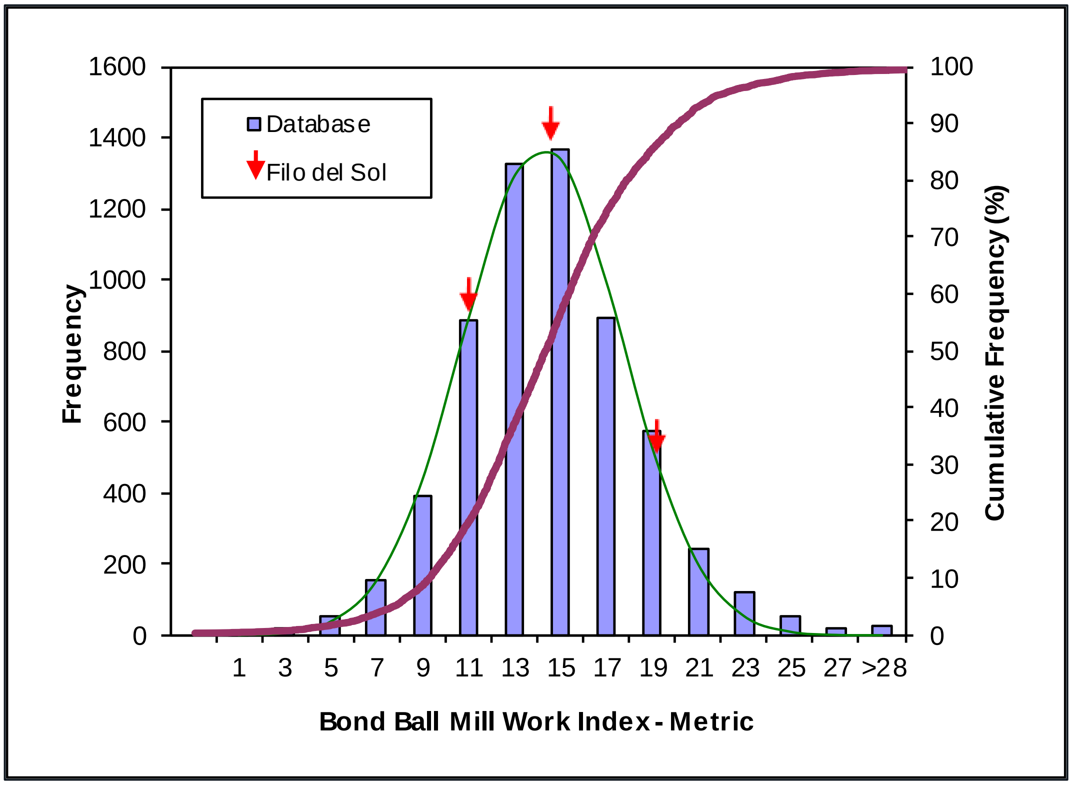


Source: Ausenco, 2019

Table 13-11: Bond Ball Mill Grindability (Summary)

Sample Name	Mesh of Grind	F_{80} (µm)	P_{80} (µm)	Gram per Revolution	Work Index (kWh/t)	Hardness Percentile
F18 Cu Comp	100	2,362	102	1.45	14.6	53
F18 M-Ag Comp	100	2,195	105	2.11	11.0	17
T18 Cu Comp	100	2,195	111	1.12	19.2	89

Figure 13-3: Bond Ball Mill Work Index Comparison

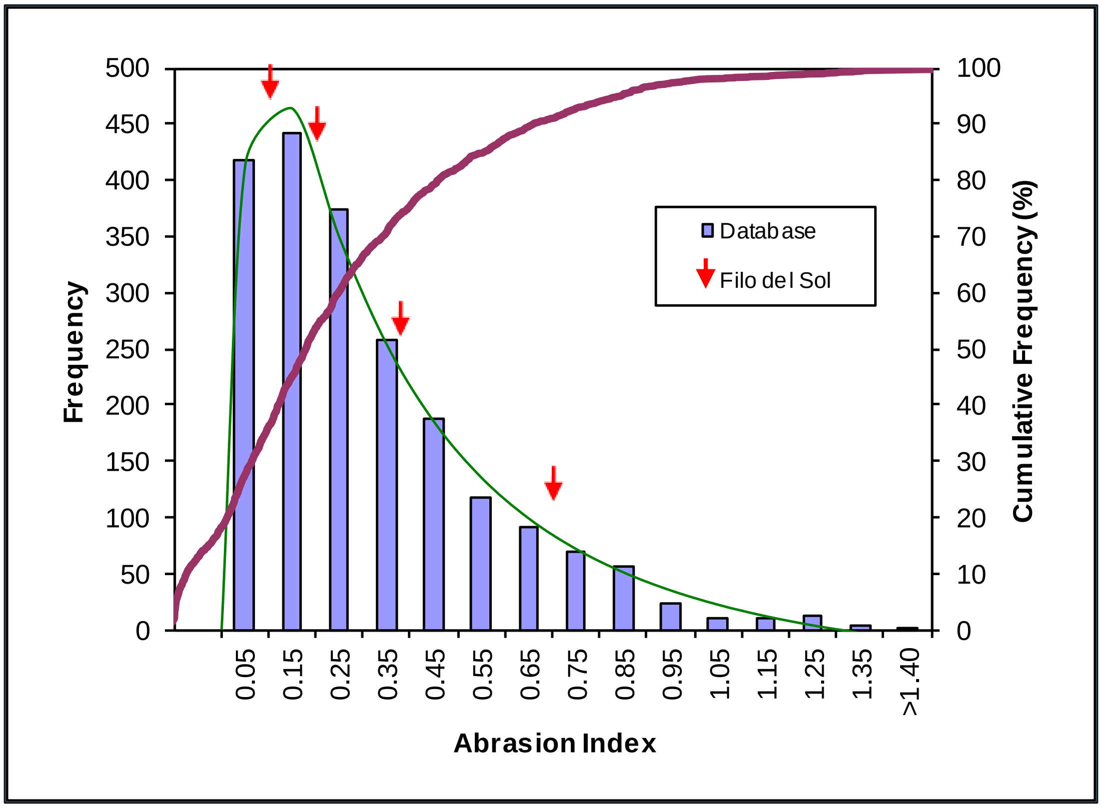


Source: Ausenco, 2019

Table 13-12: Bond Abrasion Index (Summary)

Sample Name	AI (g)	Percentile of Abrasivity
F18 G Comp	0.102	21
F18 Cu Comp	0.380	69
T18 Cu Comp	0.202	42
T18 G Comp	0.702	91

Figure 13-4: Bond Abrasion Index Comparison



Source: Ausenco, 2019

SGS results indicated that the sample F18 Cu Comp was very soft with respect to its resistance to impact breakage in SAG milling (A x b), moderately soft in terms of the CWI and Rod mill Work Index (RWI), medium soft at ball mill size (BWI) and fell in the moderately abrasive range of our database. This sample depicted an increasing trend of hardness at finer sizes, which is common.

Sample F 18 M-Ag Comp was soft with respect to its resistance to impact breakage in SAG milling (A x b), RWI and BWI. Sample T18 Cu Comp was moderately hard with respect to its resistance to impact breakage in SAG milling (A x b), moderately soft in terms of the CWI, medium in terms of its RWI and hard with respect to BWI, almost falling in the very hard range (90th percentile). This sample fell in the medium range of abrasiveness from our database.

13.3 2018 Metallurgical Program Results

13.3.1 Tamberías Gold Oxide Zone (TMB AuOx)

Three large samples were collected at the surface and were labelled T18G-T01, T18G-T02, and T18G-T03. An overall composite T18G Comp was prepared using the following proportions (Table 13-13) to approximate the average chemical composition of the Tamberías deposit over the life of mine.

Table 13-13: Preparation of T18G-Composite Sample

Component	Weight	
	(kg)	%
T18G-T01	234.8	33.5
T18G-T02	232.0	33.1
T18G-T03	234.0	33.4

13.3.1.1 Bottle Roll Tests

A series of six bottle roll cyanide amenability tests were carried out in 2018 on samples of the TMB AuOx material. For all tests, constant conditions were kept for the grind size (minus 10 mesh), retention time (96 hours) and % solids (20). Cyanide concentration ranged between 0.5 and 1.0 g/L NaCN. Results are summarized in Figure 13-4.

Table 13-14: Tamberías Gold Oxide Samples-Bottle Roll Results

Year	Test No.	Sample	Head Assay		NaCN (g/L)	Extraction		Reagent Consumption	
			Au (g/t)	Ag (g/t)		Au (%)	Ag (%)	NaCN (kg/t)	CaO (kg/t)
2017	CN-5	TMB AuOx-TR2	0.42	1.0	1	41.8	33.3	1.28	5.36
	CN-6	TMB AuOx-TR4	0.70	1.9	1	48.5	13.3	1.77	7.61
	CN-18	TMB AuOx-VRC133A	0.40	6.9	1	41.4	48.7	1.95	4.05
	CN-19	TMB AuOx-VRC133B	0.46	2.2	1	61.7	38.8	2.16	5.39
	CN-20	TMB AuOx-VRC134A	0.43	3.6	1	60.0	33.3	0.90	2.90
	CN-21	TMB AuOx-VRC134B	0.42	2.8	1	88.7	24.8	0.89	2.77
	CN-22	TMB AuOx-VRC109A	0.45	3.4	1	49.2	35.8	0.94	2.07
	Average 2017		**0.46**	**3.1**	**-**	**55.9**	**32.6**	**1.41**	**4.31**
2018	CN-32	T18G Comp	0.55	10.0	0.75	47.4	35.4	1.80	2.17
	CN-33	T18G Comp	0.55	10.0	1	43.1	34.3	2.01	2.19
	CN-34	T18G Comp	0.55	10.0	0.5	51.9	35.7	2.83	2.44
	CN-35	T18G-T01	0.30	17.9	1	46.6	33.4	1.45	3.43
	CN-36	T18G-T02	0.60	8.8	1	36.8	40.8	2.10	2.95
	CN-37	T18G-T03	0.89	5.6	1	68.5	12.1	1.89	3.86
	Average 2018		**0.59**	**10.8**	**-**	**50.6**	**28.8**	**1.81**	**3.41**

In 2018, average gold and silver extractions from the three surface samples were 50.6% and 28.8%, respectively, which was similar to the 2017 results.

The effect of cyanide concentrations (between 0.5 and 1.0 g/L) on silver extractions from the overall T18G Comp composite was not particularly significant, but there was a clear indication that increasing cyanide concentrations had an adverse effect on gold extraction.

13.3.1.2 Column Tests

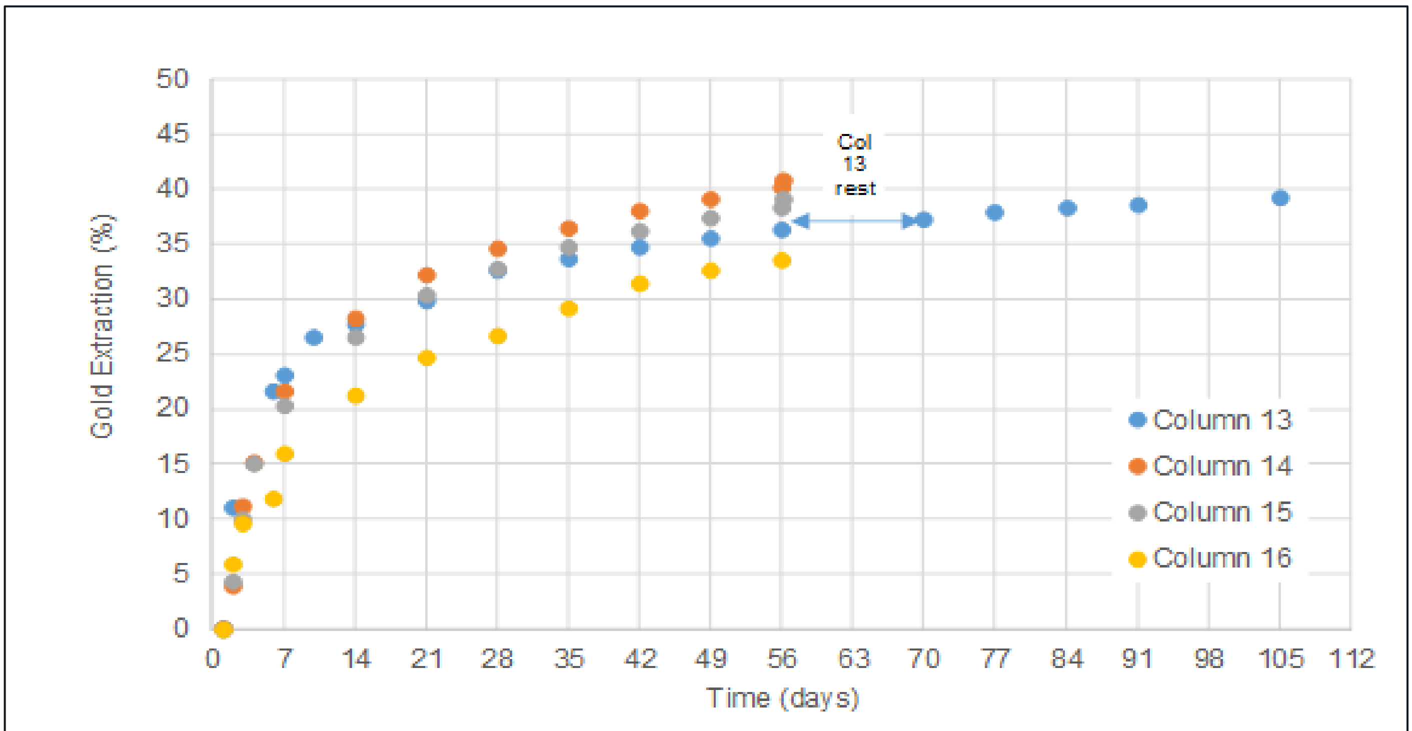
A composite of the three surface samples (T18G Comp) was cyanide leached in columns (1.8 m height) under conditions similar to the bottle roll test: 1 g/L NaCN and 10 L/h/m^2. The results are summarized in Table 13-15.

Table 13-15: Tamberías Gold T18G Comp. Column Test Results (2018)

Test #	Crush Size (100% minus)	Column Ø (mm)	Cement (kg/t)	Head Assay (g/t)		Extraction (%)		Reagent Consumed (kg/t)	
				Au	Ag	Au	Ag	NaCN	CaO
13CN	0.5 inch	150	0	0.55	10.0	39.2	21.4	1.29	2
14CN	1.5 inch	150	0	0.55	10.0	40.9	23.5	0.91	2
15CN	1.5 inch	150	5	0.55	10.0	39.1	15.5	0.25	2
16CN	2.5 inch	250	0	0.55	10.0	34.3	12.5	0.49	2
Average 1.5 inch	-	-	-			40.0	19.5	0.58	2

Column extractions were low for gold (~40%) and in particular silver (~20%) after 56 days of leaching. Kinetic curves are presented in Figure 13.5.

Figure 13-5: Tamberías Gold T18G Comp. – Kinetics of Column Leaching



Source: Ausenco, 2019

13.3.2 Filo del Sol Gold Oxide Zone (FDS AuOx)

Three surface samples (F18G-T01, F18G-T02, F18G-T03) and five drill core intervals (FSDH017A (144-188), FSDH018A (96-164), FSDH019 (140-202), FSDH020 (128-182), and FSDH024 (96-122)) were sent to the laboratory and tested. An overall composite was prepared using the following proportions (Table 13-16) to represent the life of mine average composition of the Filo del Sol gold oxide zone.

Table 13-16: Preparation of F18G Composite Sample

Component	Weight	
	(kg)	(%)
F18G-T01	96.8	12.1
F18G-T02	97.2	12.2
F18G-T03	96.8	12.1
FSDH017A (144-188)	95.2	11.9
FSDH018A (96-164)	93.6	11.7
FSDH019 (140-202)	87.6	11.0
FSDH020 (128-182)	112.0	14.1
FSDH024 (96-122)	117.6	14.8
FDS18G Composite	796.8	100.0

13.3.2.1 Bottle Roll Tests

In 2018, a series of 12 cyanide amenability bottle roll tests were conducted on the individual components of the FDS18G Comp composite, on the composite itself and on three RC drill intervals. For all tests, conditions were kept constant at 10% solids, 100% passing 10 mesh, 96 hours, 1 g/L NaCN concentration. Results are summarized in Table 13-17.

Table 13-17: Filo del Sol Gold Oxide Samples – Summary Bottle Roll results

Year	Test No.	Sample	Head Assay		Extraction		Reagent Consumption	
			Gold (g/t)	Silver (g/t)	Gold (%)	Silver (%)	NaCN (kg/t)	CaO (kg/t)
2017	CN-1	FDS AuOx-VRC065	0.55	11.9	89.9	89.9	0.92	5.39
	CN-2	FDS AuOx-VRC068	1.59	0.6	92.9	<18	1.68	3.31
	CN-7	FDS AuOx-VRC067	0.72	0.7	97.8	26.9	0.97	1.79
	CN-8	FDS AuOx-VRC069	1.18	1.0	92.6	25.5	0.90	1.85
	CN-9	FDS AuOx-VRC070	4.49	2.6	97.9	53.3	2.53	57.5
	CN-10	FDS AuOx-VRC082	0.80	1.1	89.9	39.8	1.61	29.4
	CN-11	FDS AuOx-VRC085	0.62	<0.5	88.9	19.5	0.96	1.19
		Average			**92.8**	**39.0**	**1.37**	**14.35**
2018	CN-83	VRC122B (214-224)	0.68	4.4	90.0	14.8	0.56	5.46
	CN-84	VRC097 (8-18)	0.43	0.7	67.0	31.4	0.51	20.5
	CN-85	VRC097 (152-164)	5.18	1.2	97.8	13.6	0.79	2.11

Year	Test No.	Sample	Head Assay		Extraction		Reagent Consumption	
			Gold (g/t)	Silver (g/t)	Gold (%)	Silver (%)	NaCN (kg/t)	CaO (kg/t)
	CN-104	F18G-T01	0.04	<0.5	42.0	11.5	0.17	1.94
	CN-105	F18G-T02	0.02	<0.5	52.3	11.6	0.38	3.53
	CN-106	F18G-T03	<0.02	<0.5	52.7	11.9	0.38	4.67
	CN-98	FSDH017A(114-188)	1.18	1.2	96.3	14.3	0.51	0.77
	CN-99	FSDH018A(96-164)	0.26	<0.5	80.8	30.1	0.59	5.83
	CN-100	FSDH019(140-202)	0.28	5.3	89.5	26.4	0.62	3.92
	CN-101	FSDH020(128-182)	2.44	1.6	86.3	8.2	0.33	0.93
	CN-102	FSDH024(96-122)	0.28	1.2	85.9	16.8	0.28	49.2
		Average*	**0.57**	**0.15**	**73.2**	**16.4**	**0.41**	**8.85**
	CN-103	FDS18G Comp.	0.35	1.0	89.6	25.5	0.41	9.27

* Average of the 8 components of the composite.

With the exception of the three surface samples (F18G-T01, T02 and T03) that were poorly mineralized, the samples' head assays ranged from 0.26 g/t to 5.18 g/t Au. Cyanide extractions from the mineralized samples ranged from 67.0 to 97.8% for Au. Cyanide extraction from the composite sample was 89.6% Au.

13.3.2.2 Column Tests

In 2018, seven cyanide column leach tests were carried out on the composite F18G Comp. Conditions equivalent to those employed for the other ore zones were maintained, including: cyanide concentration (1 g/L NaCN), pH (10.5) irrigation rate (10 L/h/m^2), and column height (180 cm). Ore crush size varied between 0.5 to 2.5 inches, cement addition between 0 and 15 kg/t and column diameter between 15 and 25 cm. Results are summarized in Table 13-18.

Table 13-18: Filo del Sol Gold Oxide Column Results

Year	Test (#)	Crush size (inch)	Column Diam. (cm)	Cement (kg/t)	Head Assay		Extraction		Reagent Consump (kg/t)	
					Au (g/t)	Ag (g/t)	Au (%)	Ag (%)	NaCN	CaO
2017	C5	1.5	15	12	1.15	6.25	92.7	74.4	0.51	4.6
	C6	0.75	15	12	1.15	6.25	92.9	65.2	0.40	6.2
	Average 2017		-	-	**1.15**	**6.25**	**92.8**	**69.8**	**0.46**	**5.4**
2018	35CN	0.5	15	0Δ	0.35	1.0	-	-	-	-
	35RCN	0.5	15	10	0.35	1.0	89.2	17.8	0.52	0.52
	36CN	1.5	15	5	0.35	1.0	77.7	14.5	1.15	7.86
	37CN	1.5	15	10	0.35	1.0	84.6	15.4	0.98	7.82
	38CN	1.5	15	15	0.35	1.0	81.0	15.7	0.56	7.75
	39CN	2.5	25	10	0.35	1.0	77.9	16.4	0.76	0.76
	40CN	1.0	15	10	0.35	1.0	80.2	20.4	0.94	0.94
	Average* 2018		-	-	**0.35**	**1.0**	**81.1**	**15.2**	**0.90**	**7.81**

*Average of 1.5-inch crush size column tests.
Δ Column stopped due to poor solution flow.

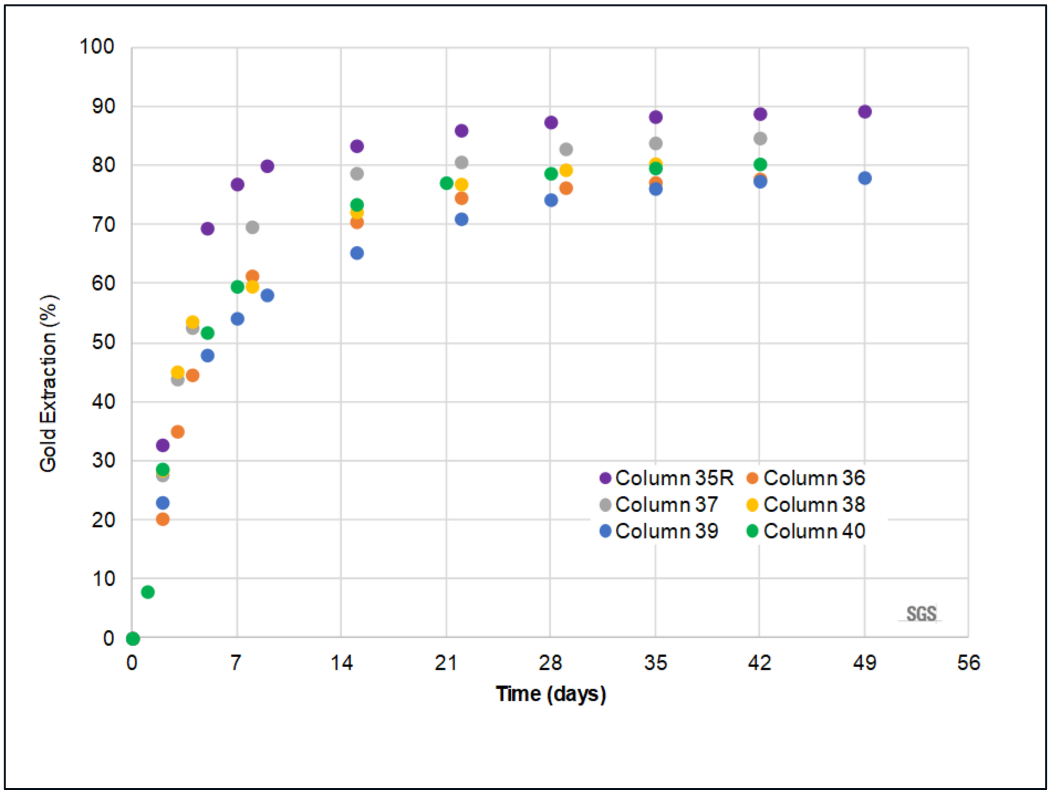
The F18G Comp. sample, crushed at 0.5 inch and without cement addition, showed poor percolation and therefore the test was terminated (Column 35CN). All other tests in the series used agglomerating cement with improved results.

The best extractions were produced at the finer crush size (0.5 inch) with cement added (10 kg/t); column 35RCN resulted in extractions of 89.2% for Au and 17.8% for Ag.

Average extractions for the 1.5-inch crush size columns were 81.1% and 15.2% for gold and silver, respectively. Kinetic curves are presented in Figure 13-16.

They indicate rapid kinetics with extraction nearing completion after only four weeks.

Figure 13-6: Filo Gold Oxide Composite – Kinetics of Column Leaching



Source: Ausenco, 2019

13.3.3 Filo del Sol Copper-Gold Oxide Zone (FDS CuAuOx)

Two surface samples (F18 Cu-T01 and F18 Cu-T02), plus seven drill core intervals [(FSDH016 (50-68), FSDH017A (256-272), FSDH018A (264-328), FSDH020 (226-291), FSDH021 (110-134), FSDH023 (96-130) and FSDH024 (150-194)] were sent to the laboratory for the test program. An overall composite was prepared using the following proportions (Table 13-19) to approximate the life of mine average for the FDS CuAuOx zone.

Table 13-19: Preparation of F18 Cu Composite Sample

Component	Weight		Cu Assay		Proportion of Total Cu*		
	(kg)	(%)	Total (%)	Acid Soluble (%)	Acid Soluble (%)	CN Soluble (%)	Insoluble (%)
F18 Cu-T01	132.0	15.3	0.68	0.66	97.1	0.5	1.5
F18 Cu-T02	132.0	15.3	1.05	0.96	91.4	1.0	1.0
FSDH016 (50-68)	92.0	10.7	0.25	0.24	96.0	0.8	0.4
FSDH017A (256-272)	32.8	3.8	0.62	0.63	100	0.3	0
FSDH018A (264-328)	99.2	11.5	0.28	0.20	71.4	19.7	3.2
FSDH020 (226-291)	70.4	8.2	0.31	0.28	90.3	11.3	1.9
FSDH021 (110-134)	111.2	12.9	1.66	1.57	94.6	3.7	0.2
FSDH023 (96-130)	8/8.0	10.2	1.22	1.23	100	1.4	0.9
FSDLH024 (150-194)	105.6	12.2	0.29	0.24	82.8	15.9	2.1
F18 Cu Composite	**863.2**	**100.0**	**0.65**	**0.59**	**90.8**	**6.1**	**1.3**

*Proportion of total copper as acid soluble, cyanide soluble, and insoluble based on copper sequential analyses.

The F18Cu Composite contained 90% of the copper as acid soluble. Several components of the composite contained a significant fraction, between 11 and 20%, of copper not soluble (or only partially soluble) in acid but soluble in cyanide (possibly attributable to the presence of chalcocite, covellite, or other copper minerals).

13.3.3.1 Bottle Roll Tests

In 2018, a total of 12 sequential (acid leach followed by cyanide leach) bottle roll tests were completed on various samples from the FDS CuAuOx zone including the individual components of the F18Cu Composite and the composite itself. The selected material was first acid leached to recover the copper, rinsed and neutralized, and then subjected to cyanide leaching for gold and silver recovery.

Test conditions were consistent for all bottle roll tests: 100% minus 10 mesh, 20% solids, 96 hours, pH~1.8 (acid leach), and 1 g/L NaCN (cyanide leach).

Results are summarized in Table 13-20.

Table 13-20: Filo del Sol Copper Gold Oxide Sample – Summary of Bottle Roll Tests

Year	Test No.	Sample	Head Assay			% Weight Loss	Extraction			Reagent Consumption		
			Cu (%)	Au (g/t)	Ag (g/t)		Cu (%)	Au (%)	Ag (%)	H_2SO_4 (kg/t)	NaCN (kg/t)	CaO (kg/t)
2017	LC-1	FDS CuAuOx-Tanque	0.31	0.69	2.0	3.7	98.0	75.9	63.6	17.8	2.3	5.0
	LC-3	FDS CuAuOx-VRC64	0.33	0.34	1.0	8.2	93.0	88.7	48.2	0	1.4	4.3
	LC-4	FDS CuAuOx-VRC65	0.43	0.52	21.5	14.7	97.1	96.4	85.3	2.3	0.7	2.6
	LC-5	FDS CuAuOx-VRC75	0.50	0.31	3.0	12.5	74.2	82.0	56.5	2.6	3.6	3.7
	LC-6	FDS CuAuOx-VRC76	0.88	1.51	0.80	19.3	98.8	98.1	46.1	0	0.7	1.8
		Average					**92.2**	**88.2**	**59.9**	**4.5**	**1.7**	**3.5**
2018	CN-40	VRC077 (90-100)	2.58	0.20	1.5	29.1	96.1	54.0	25.2	-33.1	1.9	2.5
	CN-41	VRC101 (242-252)	0.37	0.19	1.4	17.0	59.3	71.9	18.0	4.2	3.5	2.9
	CN-70	VRC066 (296-306)	0.72	0.25	0.8	11.4	75.3	60.5	32.9	-8.3	3.7	6.6
	CN-60	F18 Cu-T01	0.68	0.49	69.3	4.8	96.1	73.9	96.7	-8.9	0.3	1.6
	CN-61	F18 Cu-T02	1.05	0.56	3.3	8.4	97.3	92.9	79.5	-14.9	0.5	1.7
	CN-62	FSDH016 (50-68)	0.25	0.16	11.7	26.3	93.5	93.1	92.4	-39.7	0.3	2.4
	CN-63	FSDH021 (110-134)	1.66	0.34	4.4	34.9	98.6	90.1	73.4	-73.2	1.3	1.8
	CN-64	FSDH018A (264-328)	0.28	0.20	1.6	17.7	72.4	76.2	68.6	-2.0	1.6	2.1
	CN-65	FSDH023(96-130)	1.22	0.20	3.8	22.4	96.2	86.9	41.3	-43.5	0.7	1.7
	CN-66	FSDH017A (256-272)	0.62	0.23	1.7	23.0	98.4	90.5	80.5	-24.0	0.4	1.1
	CN-67	FSDH020 (226-291)	0.31	0.42	5.3	21.7	86.0	92.3	86.2	-22.8	1.2	2.0
	CN-68	FSDH024 (150-194)	0.29	0.20	0.9	19.6	76.2	72.1	66.0	-0.2	1.4	4.2
	CN-69	**F18 Cu Comp.**	**0.65**	**0.31**	**11.8**	**18.8**	**94.7**	**77.7**	**94.9**	**-26.4**	**0.8**	**1.7**
	Average of F18Cu Comp. individual components		**0.70**	**0.31**	**11.3**	**19.9**	**90.5**	**85.3**	**76.1**	**-28.4**	**0.85**	**2.1**

Weight loss during the acid leach phase of the tests was significant for all FDS CuAuOx samples, except for the two surface samples (F18 Cu-T01 and F18 Cu-T02). Based on the mineralogical analyses, this weight loss was attributable to the presence of significant quantities of water-soluble sulphate minerals in the feed material. In general, the leach solution resulting from the dissolution of these sulphate minerals was acidic and therefore acid consumption to maintain a pH of 1.8 was negative, i.e., acid was generated.

As expected, copper extractions were largely dependent on the amount of copper present as acid soluble copper. Only when the proportion of acid soluble copper was low, such as in samples FSDH018A (264-328) and FSDH024 (150-194), was the copper extraction was below 86%. For the overall F18Cu Comp sample, the copper extraction was 94.7%.

Gold and silver extractions for the F18Cu Comp sample were 77.7% and 94.9%, respectively.

13.3.3.2 Column Tests

Seven sequential (acid leach followed by cyanide leach) column tests were completed on the FDS CuAuOx composite (F18 Cu Comp) using consistent leach conditions (180 cm column height, 10 L/h/m^2 irrigation rate, pH ~1.8 (acid leach) and 1 g/L NaCN (cyanide leach). Some columns contained material crushed to 100% minus 1.5 inch while others included material crushed to 100% minus 0.50 inch or minus 2.5 inch. Results are summarized in Table 13-21 and Table 13-22 below.

Table 13-21: F18 Cu Composite – Sequential Column Test Conditions

Year	Test #	Sample	Acid Leach			Cyanide Leach	Head Assays		
			Crush Size 100% minus	Column Ø (mm)	Curing (kg/t H_2SO_4)	Cement (kg/t)	Cu (%)	Au (g/t)	Ag (g/t)
2017	C-1/C-7	FDS $CuAuO_x$	1.5 inch	150	0	0	0.31	0.69	2.0
	C-2/C-8	FDS $CuAuO_x$	0.75 inch	150	0	0	0.31	0.69	2.0
2018	C-24	F18 Cu Comp	0.5 inch	150	18	0	0.65	0.31	11.8
	C-25	F18 Cu Comp	1.5 inch	150	0	0	0.65	0.31	11.8
	C-26	F18 Cu Comp	1.5 inch	150	10	0	0.65	0.31	11.8
	C-27	F18 Cu Comp	1.5 inch	150	18	0	0.65	0.31	11.8
	C-28	F18 Cu Comp	1.5 inch	150	25	0	0.65	0.31	11.8
	C-29	F18 Cu Comp	2.5 inch	250	18	0	0.65	0.31	11.8
	C-30	F18 Cu Comp	1.0 inch	150	18	0	0.65	0.31	11.8
	Average F18Cu Comp (1.5 inch)		-	-	-	-	**0.65**	**0.31**	**11.8**

Table 13-22: F18 Cu Composite – Sequential Column Test Results

Year	Test #	Sample	% Recovery			Reagent Consumption		
			Cu	Au	Ag	H_2SO_4 (kg/t)	NaCN (kg/t)	CaO (kg/t)
2017	C-1/C-7	FDS $CuAuO_x$	80.5	86.4	74.4	~0	0.73	4.6
	C-2/C-8	FDS $CuAuO_x$	83.3	87.0	67.2	~0	0.76	4.6
2018	C-24	F18 Cu Comp	95.1	78.9	88.5	-20.0	1.35	1.93
	C-25	F18 Cu Comp	95.9	74.0	90.9	-12.8	2.25	1.71
	C-26	F18 Cu Comp	95.3	75.5	89.5	-22.4	1.74	1.62
	C-27	F18 Cu Comp	94.9	75.5	87.4	-21.8	0.99	1.68
	C-28	F18 Cu Comp	95.0	78.0	90.5	-16.1	0.69	2.01
	C-29	F18 Cu Comp	96.2	76.2	82.6	-32.0	1.88	1.79
	C-30	F18 Cu Comp	95.8	78.4	92.9	-18.8	2.00	2.20
	Average F18Cu Comp (1.5 inch)		**95.3**	**75.8**	**89.6**	**-18.3**	**1.42**	**1.76**

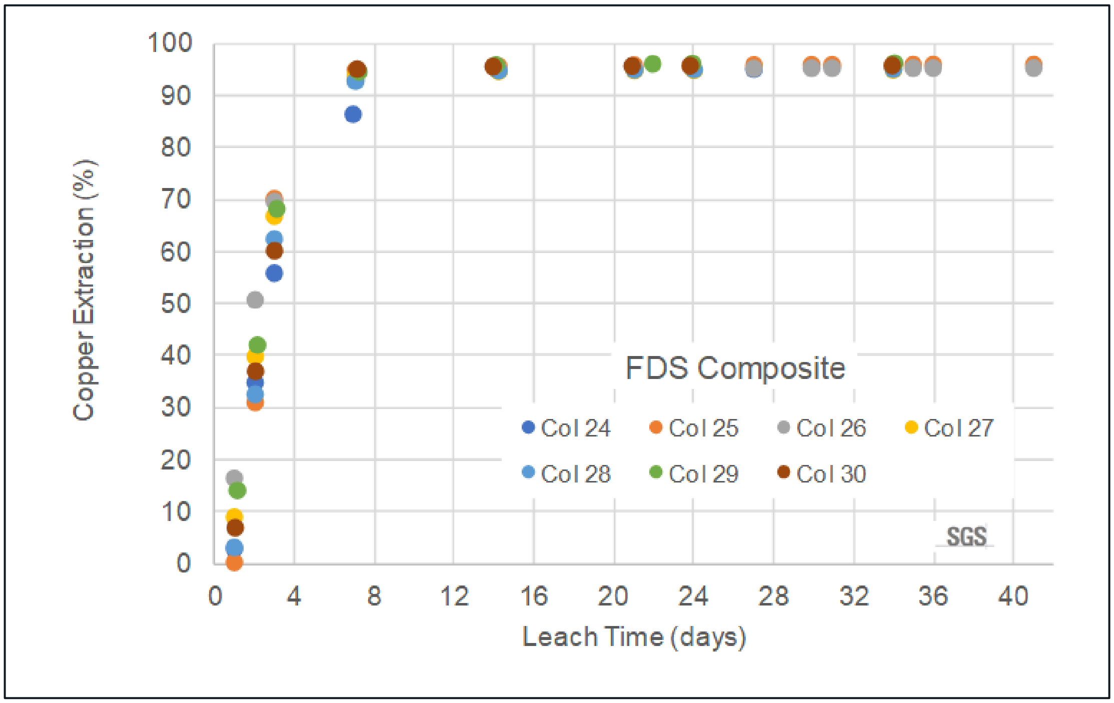
In 2018, the main parameters tested on FDS CuAuOx composite sample (F18Cu Comp) were retention time, crush size (0.5 inch, 1 inch, 1.5 inch, 2.5 inch) and curing acid addition (0, 10, 18, and 25 kg H2SO4 per tonne of sample).

Copper extraction was not particularly sensitive to acid addition, with extractions ranging from 94.9% to 95.9% for acid additions ranging from 0 to 25 kg/t.

The effect of crush size on copper extractions at constant acid addition of 18 kg/t was also fairly limited, with copper extractions ranging from 94.9% to 96.2%.

The copper leach kinetics are presented in Figure 13-17. The kinetics were very rapid, with leach completion (for 1.5-in crush size) achieved after only four weeks.

Figure 13-7: Copper Extraction for FDS CuAuOx

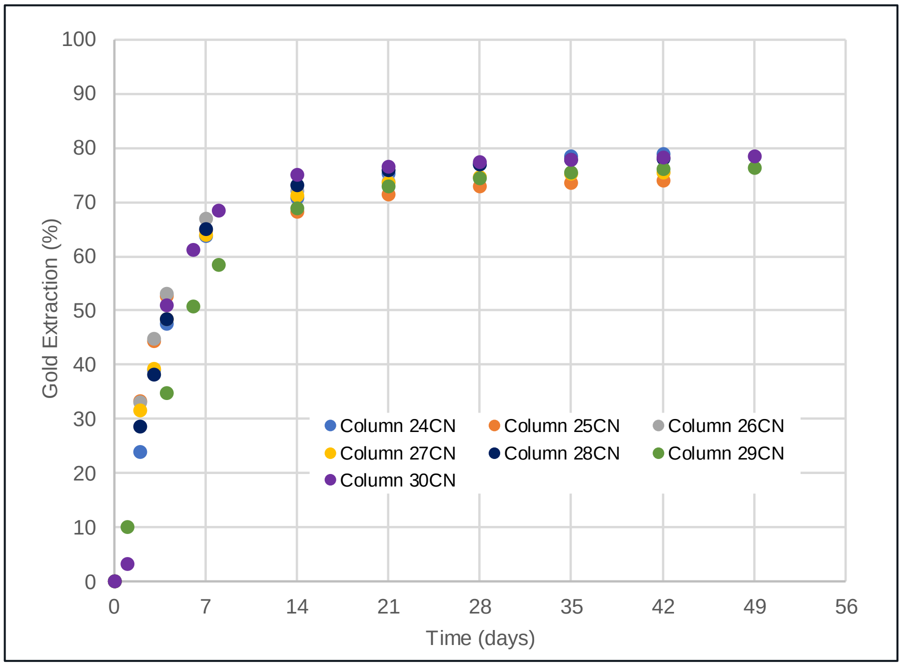


Source: Ausenco, 2019

The average gold and silver extractions at 1.5-in crush size were 75.8 and 89.6%, respectively.

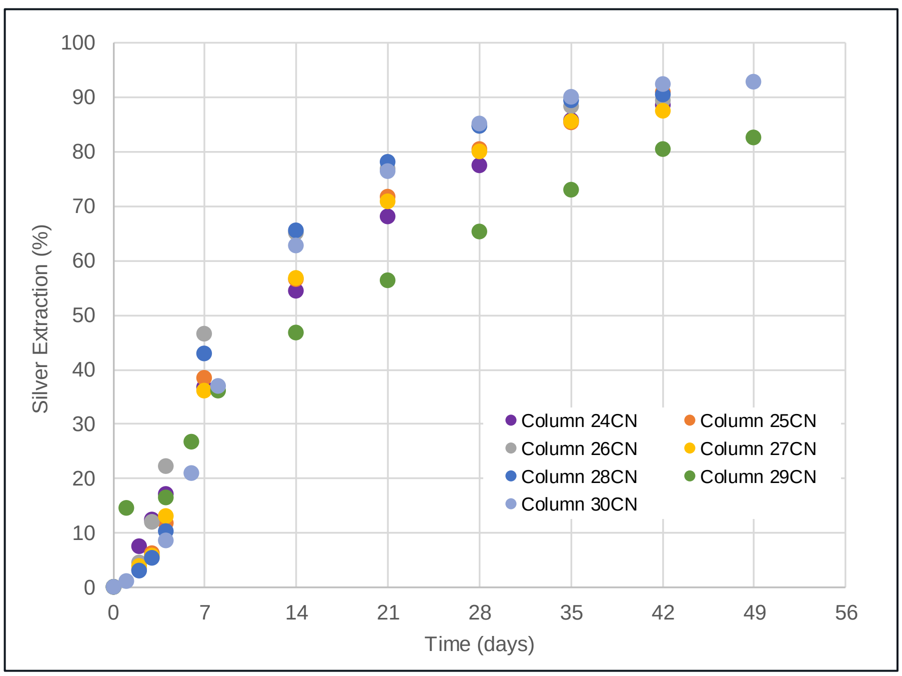
Leach kinetics for gold and silver are shown in Figure 13-8 and Figure 13-9.

Figure 13-8: Gold Extraction for FDS CuAuOx



Source: Ausenco, 2019

Figure 13-9: Silver Extraction for FDS CuAuOx



Source: Ausenco, 2019

13.3.4 Filo del Sol Copper Gold Oxide Variability

To further examine variability within the FDS CuAuOx zone, specific intervals were selected based on copper speciation assays to constitute a composite high in cyanide soluble copper (F18 CuCN Comp) and a composite high in silver (F18 M-Ag Comp).

The exact composition of each of these two composites is presented in Table 13-23 and Table 13-24.

Table 13-23: FDS 18 CuCN Composite Make Up

Component	Weight		Cu Assay		Proportion of Total Cu		
	(kg)	(%)	Total (%)	Acid Soluble (%).	Acid Soluble (%)	CN Soluble (%)	Insoluble (%)
FSDH022 (96-106)	18.0	20.0	0.88	0.81	92.0	1.3	1.4
FSDH022 (106-116)	25.6	28.5	10.6	3.80	35.8	59.5	0.1
FSDH022 (116-130)	24.9	27.7	0.93	0.89	95.7	1.2	0.5
FSDH022 (130-140)	21.4	23.8	0.26	0.23	88.5	1.2	1.9
F18 CuCN Comp	**89.9**	**100.0**	**3.37**	**1.87**	**55.5**	**38.3**	**0.3**

*Proportion of total copper as acid soluble, cyanide soluble and insoluble based on copper sequential analyses.

Table 13-24: FDS 18 M-Ag Composite Make Up

Component	Weight		Cu Assay		Proportion of Total Cu			Ag Head Grade (g/t)
	(kg)	(%)	Total (%)	Acid Soluble (%)	Acid Soluble (%)	CN Soluble (%)	Insoluble (%)	
FSDH016 (78-90)	66.4	26.2	0.24	0.16	66.7	27.5	5.8	478
FSDH017A (272-310)	62.6	24.7	1.25	1.24	99.2	1.4	0.5	89
FSDH021 (148-158)	58.6	23.2	1.38	1.27	92.0	2.5	0.7	824
FSDH023 (162-186)	65.4	25.8	0.58	0.45	77.6	17.2	1.9	417
F18 M-Ag Comp	**253.0**	**100.0**	**0.95**	**0.84**	**88.4**	**6.8**	**1.3**	**474**

*Proportion of total copper as acid soluble, cyanide soluble and insoluble based on copper sequential analyses.

13.3.4.1 Bottle Roll Tests

In 2018, a total of seven sequential (acid leaching followed by cyanide leaching) bottle roll tests were completed on various samples from the high copper cyanide soluble samples (CuCN), and 14 sequential bottle roll tests on various samples from the high silver area (M-Ag). Consistent leach conditions were used for all these tests, including: 100% minus 10 mesh, 20% solids, 96 hours, pH ~1.8 (acid leach) and 1 g/L NaCN, pH ~10.5 (cyanide leach). Results are summarized in Table 13-25 below.

Table 13-25: FDS CuAuOx (F18CuCN Composite and F18M-Ag Comp Samples) Bottle Roll Results

Year	Test No.	Sample	Head Assay			Weight Loss	Extraction			Reagent Consumption		
			Cu (%)	Au (g/t)	Ag (g/t)		Cu (%)	Au (%)	Ag (%)	H_2SO_4 (kg/t)	NaCN (kg/t)	CaO (kg/t)
2018		F18 CuCN										
	LC-20	VRC066 (240-250)	2.65	0.10	1.8	6.2	85.8	73.9	38.2	-19.2	9.51	7.89
	LC-21	VRC085 (226-234)	3.92	0.24	3.2	14.1	69.5	70.4	5.1	-20.0	20.4	1.31
	LC-54	FSDH022(96-106)	0.88	0.26	3.1	5.3	96.6	87.7	83.8	-13.4	0.44	1.60
	LC-55	FSDH022(106-116)	10.6	0.29	1.2	22.4	46.6	55.9	6.7	-53.6	18.5	0.5
	LC-56	FSDH022(116-130)	0.93	0.28	49	18.8	95.9	89.9	93.0	-12.6	0.62	1.22
	LC-57	FSDH022(130-140)	0.26	0.36	8.2	20.8	95.2	93.4	93.0	-7.2	1.38	1.83
	LC-58	**F18 CuCN Comp.**	3.37	0.31	1.0	17.5	70.9	79.4	17.3	-21.0	4.84	3.00
	Average of Individual F18CuCn Components					16.8	83.6	81.7	69.1	-23.6	5.24	1.29
2018		F18 M-Ag										
	LC-22	VRC100 (306-330)	0.45	0.22	170	17.6	97.2	78.5	92.1	-58.6	0.24	1.02
	LC-23	VRC060 (82-110)	0.26	0.24	108	27.8	96.7	85.5	94.0	-53.0	0.44	2.19
	LC-24	VRC065 (86-110)	0.36	0.25	49.2	25.4	96.4	76.6	90.8	-8.3	0.36	4.13
	LC-25	VRC074 (230-254)	0.28	0.65	200	12.4	67.1	38.4	92.9	-6.8	1.56	3.25
	LC-26	VRC080(210-250)	0.24	0.40	89.9	23.6	96.4	83.2	94.0	-33.8	0.26	2.36
	LC-27	VRC063(262-288)	0.22	0.34	93.4	13.0	52.7	57.3	87.8	-3.0	2.57	6.06
	LC-28	M-Ag Flot. Comp.	0.31	0.33	114	37.4	90.0	67.5	90.9	-29.5	0.62	1.62
	LC-29	VRC062(270-286)	0.64	0.37	37.8	28.8	60.5	62.6	84.4	-42.4	2.37	1.06
	LC-30	VRC062(286-296)	0.63	0.67	43.9	39.7	48.7	61.3	85.3	-57.9	3.74	1.06
	LC-49	FSDH016(78-90)	0.24	0.18	478	22.0	63.1	57.9	49.0	-7.8	4.28	1.60
	LC-50	FSDH017A(272-310)	1.25	0.22	89	24.4	97.9	83.4	90.0	-46.6	0.84	1.07
	LC-51	FSDH021(148-158)	1.38	0.43	824	38.8	94.5	76.1	97.8	-86.1	3.51	0.81
	LC-52	FSDH023(162-186)	0.58	0.34	417	20.9	79.4	71.8	43.0	-20.7	2.43	3.05
	LC-53	**F18 M-Ag Comp.**	0.95	0.30	474	27.2	91.8	75.8	86.1	--38.5	2.12	1.64
	Average of Individual F18M-Ag Components					26.5	83.7	72.3	70.0	-40.3	2.77	1.63

For both types of mineralization, as expected, copper extractions were directly related to the proportion of acid soluble copper in the feed material.

For the F18 CuCN, samples with a high proportion of cyanide soluble copper (e.g., sample FSDH022 (106-116)) resulted in low copper extraction during the acid leach (46.6%) and high cyanide consumption in the subsequent cyanide leach. Gold and silver extractions during the cyanide leach were highly variable and ranged from 56% to 93% for Au and 5% to 93% for Ag. One of the factors contributing to this variability was the residual concentration of cyanide during the leach, itself affected by the proportion of cyanide soluble copper. The F18CuCN Composite gave extractions of 71%, 79% and 17% for copper, gold and silver, respectively.

For the high silver samples, copper extraction during the acid leach was also a function of the proportion of acid soluble copper in the feed and varied from 49% to 98%. Gold and silver extractions were also variable ranging from 38% to 85%

(gold) and 43% to 98% (silver). The F18M-Ag Composite yielded extractions of 92%, 76% and 86% for copper, gold and silver, respectively.

13.3.4.2 Column Tests

A single column test was conducted on each of the F18 CuCN and F18 M-Ag composite samples, due to the limited amount of sample available. Conditions for both tests were identical: crush size of 100% minus 1.5 inch; 180 cm height; 15 cm diameter column; 10 L/h/m^2 irrigation rate; pH ~1.8 (acid leach), and 10 kg/t acid curing.

Both tests were abandoned very shortly after the start of the acid leach due to poor solution flow.

13.3.5 Tamberías Copper Gold Oxide Zone (TMB CuAuOx)

Six surface samples from the TMB CuAuOx zone were selected and combined to prepare an overall composite sample, representative of the TMB CuAuOx composition over the life of mine. Table 13-26 summarizes preparation of the T18 Cu Composite sample.

Table 13-26: Preparation of T18 Cu Composite Sample

Component	Weight		Cu Assay		Proportion of Total Cu		
	(kg)	(%)	Total (%)	Acid Soluble (%)	Acid Soluble (%)	CN Soluble (%)	Insoluble (%)
T18 Cu-T01	25.0	16.7	0.55	0.43	78.0	3.6	9.1
T18 Cu-T02	25.0	16.7	0.37	0.32	86.2	5.9	4.9
T18 Cu-T03	25.0	16.7	0.69	0.45	65.7	6.8	18.7
T18 Cu-T04	25.0	16.7	0.29	0.23	78.3	5.5	12.1
T18 Cu-T05	25.0	16.7	0.57	0.48	84.6	5.6	2.5
T18 Cu-T06	25.0	16.7	0.48	0.42	86.5	4.8	1.7
T18 Cu Composite	**150.0**	**100.0**	**0.41**	**0.31**	**75.1**	**6.1**	**9.8**

*Proportion of total copper as acid soluble, cyanide soluble, and insoluble based on copper sequential analyses.
The T18 Cu Composite contained 75% acid soluble copper.

13.3.5.1 Bottle Roll Tests

In 2018, a total of nine sequential (acid leaching followed by cyanide leaching) bottle roll tests were completed on samples from the TMB CuAuOx zone, including the individual components of the T18 Cu Composite and the composite itself.

Test conditions were kept constant: 100% minus 10 mesh, 20% solids, 96 hours, pH ~1.8 (acid leach), and 1 g/L NaCN, and pH ~10.5 (cyanide leach).

Results are summarized in Table 13-27.

Table 13-27: T18 Cu Composite – Summary of BR Tests

Year	Test No.	Sample	Head Assay			% Weight Loss	Extraction			Reagent Consumption		
			Cu (%)	Au (g/t)	Ag (g/t)		Cu (%)	Au (%)	Ag (%)	H_2SO_4 (kg/t)	NaCN (kg/t)	CaO (kg/t)
2017	LC-2/CN4	TMB CuAuOxTR-4	0.49	0.28	<0.5	8.2	86.7	46.6	47	25.9	1.36	6.09
2018	LC-16	VRC112 (20-30)	0.44	0.40	0.8	4.4	85.8	89.3	40.1	10.1	1.2	6.1
	LC-17	VRC111 (58-68)	0.96	0.47	1.6	1.6	92.8	89.4	87.0	-9.0	2.0	4.6
	LC-31	T18 Cu-T01	0.55	0.38	1.2	21.3	78.1	83.0	61.0	-18.5	0.8	4.0
	LC-32	T18 Cu-T02	0.37	0.15	<0.5	19.1	78.5	46.5	57.1	10.2	0.7	3.1
	LC-33	T18 Cu-T03	0.69	0.24	<0.5	30.9	67.3	83.9	47.9	2.8	1.3	6.3
	LC-34	T18 Cu-T04	0.29	0.37	0.8	28.0	79.4	85.6	64.0	3.9	0.9	5.7
	LC-35	T18 Cu-T05	0.57	0.24	0.8	15.6	92.3	55.0	40.8	-6.5	0.5	3.8
	LC-36	T18 Cu-T06	0.48	0.30	0.5	12.9	93.9	65.8	67.6	2.6	0.7	3.3
	LC-37	**T18 Cu Comp.**	**0.41**	**0.25**	**0.8**	**4.3**	**81.0**	**69.6**	**53.9**	**6.4**	**0.7**	**4.4**
	Average of Individual T18 Cu Components		**0.49**	**0.28**	**0.55**	**21.3**	**81.6**	**70.0**	**56.4**	**-0.9**	**0.8**	**4.4**

Bottle roll extractions on the composite were 81.0%, 69.6% and 53.9% for copper, gold and silver, respectively. Copper extraction ranged from 67% to 93%.

13.3.5.2 Column Tests

In 2018, seven sequential leach (acid leaching followed by cyanide leaching) column tests were conducted on the TMB CuAuOx composite (T18Cu Composite) sample. Similar to the FDS CuAuOx composite sample, parameters tested were crush size (between 0.5 and 2.5 inches), acid curing (0 to 25 kg/t), and column diameter (15 to 25 cm). For all tests, irrigation rates (at 10 $L/h/m^2$), pH (~1.8 during the acid leach and ~10.5 during the cyanide leach), and cyanide concentration (1 g/L) were kept constant. Test conditions are summarized in Table 13-28, while results are summarized in Table 13-29.

Table 13-28: T18 Cu Composite – Sequential Column Tests Conditions

Year	Test No.	Sample	Acid Leach			Cyanide Leach	Head assays		
			Crush Size (100% minus)	Column Diam. (mm)	Curing (kg/t) H_2SO_4	Cement (kg/t)	Cu (%)	Au (g/t)	Ag (g/t)
2017	C-9	TMB-CuAuOx TR-4	0.75 inch	150	24	12.2	0.48	0.28	<0.5
	C-10*	TMB-CuAuOx TR-4	0.75 inch	150	-	0	0.44	0.28	<0.5
2018	C-17	T18 Cu Comp	0.5	150	18	0	0.41	0.25	0.8
	C-18	T18 Cu Comp	1.5	150	0	0	0.41	0.25	0.8
	C-19	T18 Cu Comp	1.5	150	10	0	0.41	0.25	0.8
	C-20	T18 Cu Comp	1.5	150	18	0	0.41	0.25	0.8
	C-21	T18 Cu Comp	1.5	150	25	0	0.41	0.25	0.8
	C-22	T18 Cu Comp	2.5	250	18	0	0.41	0.25	0.8
	C-23	T18 Cu Comp	1.0	150	18	0	0.41	0.25	0.8
	Average T18 Cu Comp. @ 1.5 inch		-	-	-		**0.41**	**0.25**	**0.8**

*For this test, fines (-150 mesh) were screened off the column feed to improve percolation.

Table 13-29: T18 Cu Composite – Sequential Column Test Results

Year	Test No.	Sample	% Extraction			Reagent Consumption		
			Cu	Au	Ag	H_2SO_4 (kg/t)	NaCN (kg/t)	CaO (kg/t)
2017	C-9	TMB-CuAuOx TR-4	90.8	29.0	-	27.8	0.83	5.2
	C-10	TMB-CuAuOx TR-4	87.1*	34	-	12.7	0.79	5.1
2018	C-17	T18 Cu Comp	88.0	65.0	32.4	37.5	1.0	4.1
	C-18	T18 Cu Comp	76.9	57.7	35.4	19.1	0.6	3.6
	C-19	T18 Cu Comp	85.9	58.8	39.1	32.5	1.0	4.0
	C-20	T18 Cu Comp	88.5	55.8	38.6	36.2	0.8	3.9
	C-21	T18 Cu Comp	85.8	51.0	33.9	42.0	1.2	4.1
	C-22	T18 Cu Comp	83.5	43.7	25.1	30.2	0.9	3.6
	C-23	T18 Cu Comp	82.0	45.8	26.8	21.4	1.2	4.0
	Average T18 Cu Comp (1.5 inch)		**84.3**	**55.8**	**36.8**	**32.5**	**0.9**	**3.9**

*Including copper recovery from the fines (-150 mesh fraction).

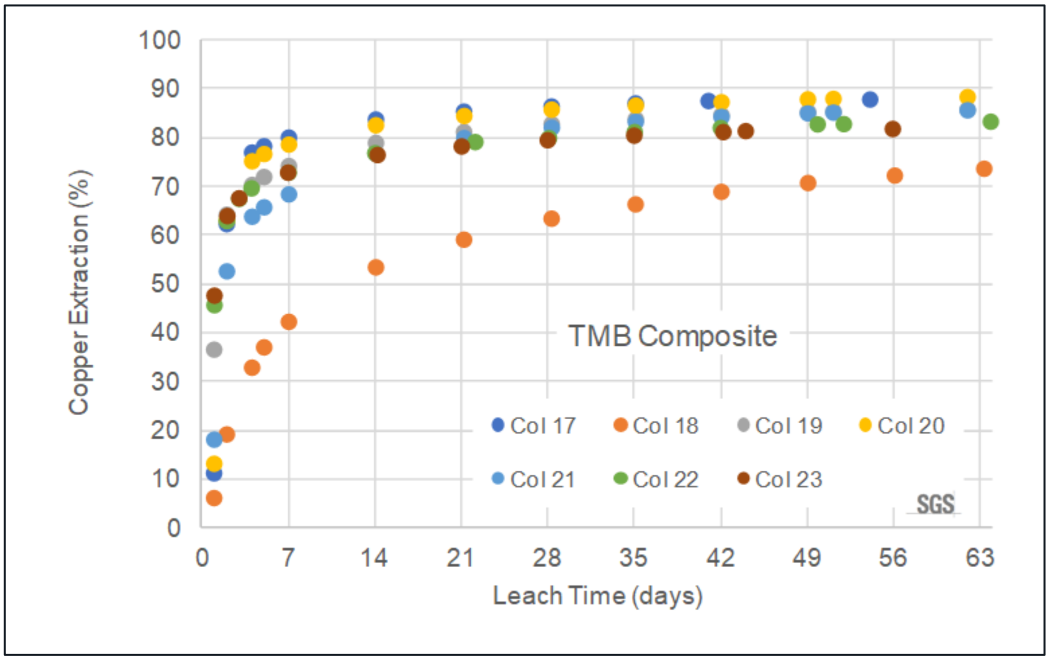
In 2018, as for the FDS CuAuOx composite sample, the main parameters tested during the column tests with the TMB CuAuOx composite were retention time, crush size (from 0.5 to 2.5 inches), and acid cure addition.

Contrary to the FDS CuAuOx composite sample, acid curing proved necessary for the TMB CuAuOx composite. The poorest copper extractions were obtained when no curing was completed (Column 18). The best copper extractions were achieved with 10 kg/t acid curing and 1.5-inch crush size (Column 19). Average extractions at a crush size of 1.5 inch were 84.3%, 55.8% and 36.8% for copper, gold and silver, respectively.

Kinetic curves for copper, gold and silver are presented in Table 13-10,

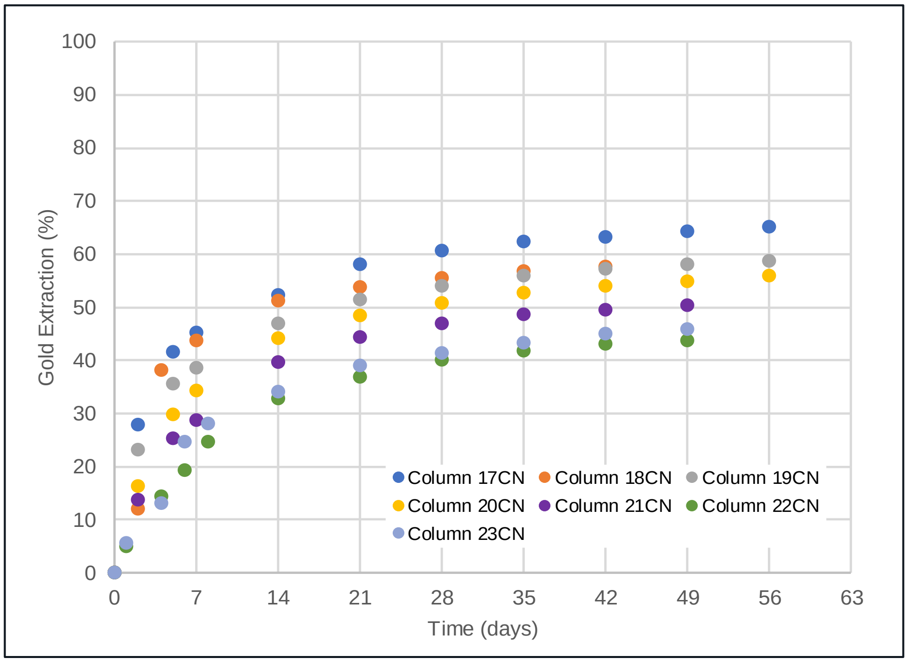
Table 13-11 and Table 13-12, respectively.

Figure 13-10: Copper Extraction for TMB CuAuOx Composite



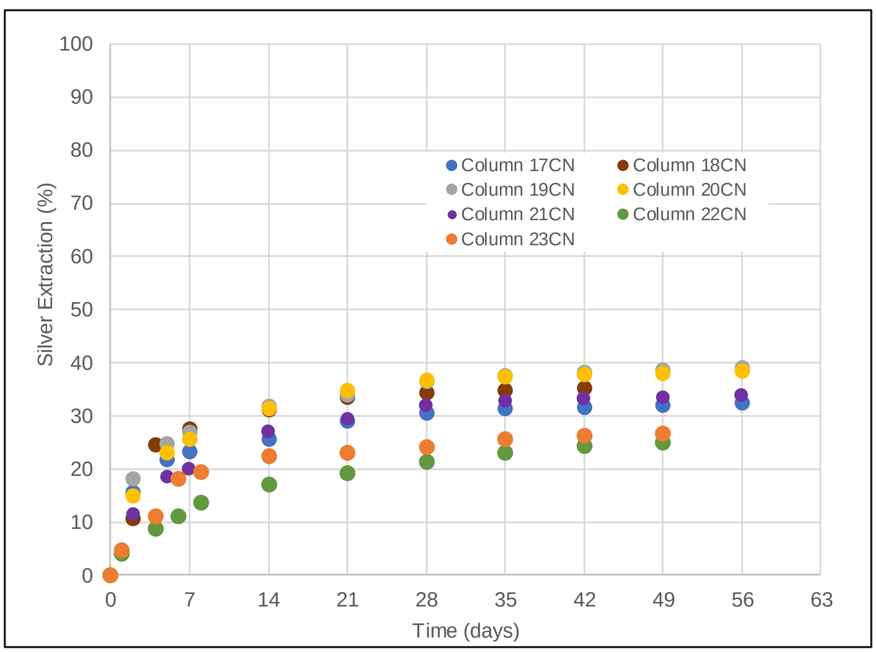
Source: Ausenco, 2019

Figure 13-11: Gold Extraction for TMB CuAuOx Composite



Source: Ausenco, 2019

Figure 13-12: Silver Extraction for TMB CuAuOx Composite



Source: Ausenco, 2019

13.3.6 Overall Copper Blends

A limited number of column leach tests were also carried out in 2018 to evaluate the response of an overall copper composite to heap leaching. The four zone composite samples were blended in varying proportions, as shown in Table 13-30 and Table 13-31.

Table 13-30: Preparation of Copper Blend #1 Composite Sample

Component	Weight		Assay			Proportion of Total Cu		
	(kg)	(%)	Cu (%)	Au (g/t)	Ag (g/t)	Acid Soluble (%)	CN Soluble (%)	Insoluble (%)
T18 Cu Comp	55.0	49.0	0.41	0.25	0.8	75.1	6.1	9.8
F18 Cu Comp	37.0	33.0	0.65	0.31	11.8	90.9	4.0	1.7
F18 CuCN Comp	18.0	16.0	3.37	0.31	1.0	55.5	38.3	0.2
F18 M-Ag Comp	2.25	2.0	0.95	0.30	474	88.4	6.8	1.3
Cu Blend #1 Composite	**112.3**	**100.0**	**0.91**	**0.29**	**9.4**	**72.5**	**16.2**	**2.1**

*Proportion of total copper as acid soluble, cyanide soluble and insoluble based on copper sequential analyses.

Table 13-31: Preparation Copper Blend #2 Composite Sample

Component	Weight		Assay			Proportion of Total Cu		
	(kg)	(%)	Cu (%)	Au (g/t)	Ag (g/t)	Acid Soluble (%)	CN Soluble (%)	Insoluble (%)
T18 Cu Comp	30.7	33.0	0.41	0.25	0.8	75.1	6.1	9.8
F18 Cu Comp	45.6	49.0	0.65	0.31	11.8	90.9	4.0	1.7
F18 CuCN Comp	1.9	2.0	3.37	0.31	1.0	55.5	38.3	0.2
F18 M-Ag Comp	14.9	16.0	0.95	0.30	474	88.4	6.8	1.3
Cu Blend #2 Composite	**93.1**	**100.0**	**0.68**	**0.32**	**103**	**79.4**	**5.7**	**2.2**

*Proportion of total copper as acid soluble, cyanide soluble and insoluble based on copper sequential analyses.

Copper Blend #2 Composite is a fair representation of the proportions of each mineralization type in the Filo del Sol project, as it was understood in 2018.

13.3.6.1 Bottle Roll Tests

A series of sequential (acid leaching followed by cyanide leaching) bottle roll tests were completed with the two copper blends, Copper Blend #1 and Copper Blend #2. Test conditions were similar to those used throughout the program, excepted where indicated.

Test conditions and results are presented in Table 13-32.

Table 13-32: Copper Blends – Bottle Roll Test Results – 2018

Sample	Test No.	Head Assay			Weight Loss (%)	Extraction			Reagent Consumption		
		Cu (%)	Au (g/t)	Ag (g/t)		Cu (%)	Au (%)	Ag (%)	H_2SO_4 (kg/t)	NaCN (kg/t)	CaO (kg/t)
Copper Blend #1	LC-59	0.91	0.29	9.4	15.0	81.6	-	-	-9.2	-	-
	LC-64	0.91	0.29	9.4			-	-	-3.6	-	-
Copper Blend #2	LC-69	0.68	0.32	10.3	16.7	90.9	76.0	85.5	-13.4	1.32	3.01
	LC-70*	0.68	0.32	10.3	17.3	92.3	75.8	81.5	-11.6	2.07	2.57

*10 kg/t acid used to cure the sample prior to copper leach.

Copper extraction from Copper Blend #2 was significantly higher (~91%) than that of Copper Blend #1 (~82%), due to the higher proportion of very soluble ore types, such as F18 Cu Comp and F18 M-Ag making up 65% of Copper Blend #2 as compared to 35% in Copper Blend #1.

13.3.6.2 Column Tests

One sequential leach (acid leaching followed by cyanide leaching) column test was completed on each of Copper Blend #1 and Copper Blend #2 samples.

Conditions for both column tests were identical, i.e., crush size 1.5 inch, column diameter 15 cm, 10 kg/t acid curing, irrigation rate 10 $L/h/m^2$, pH~1.8 (acid leach) or 10.5 (cyanide leach), sodium cyanide concentration 1 g/L and no cement addition.

Results are summarized in Table 13-33.

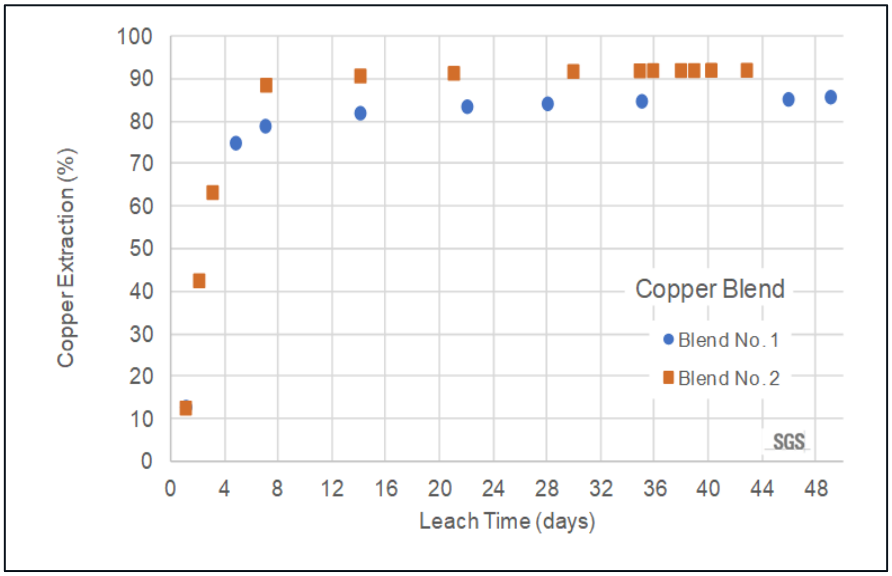
Table 13-33: Copper Blend Column Test Results

Sample	Test #	Weight Loss (%)	Extraction			Reagent Consumption		
			Cu (%)	Au (%)	Ag (%)	H_2SO_4 (kg/t)	NaCN (kg/t)	CaO (kg/t)
Copper Blend #1	C-33	14.8	86.3	64.4	59.8	3.3	2.06	2.6
Copper Blend #2	C-34	20.8	92.0	67.5	55.7	-9.4	1.52	2.6

As with the bottle roll tests, copper extractions from Copper Blend #2 were significantly higher than that from Copper Blend #1 (92% vs. 86%).

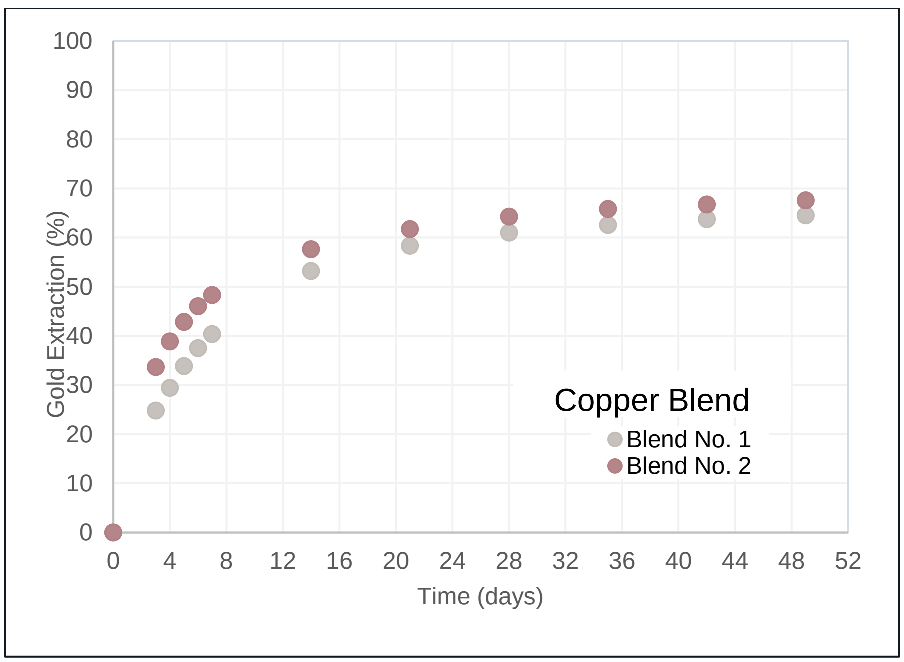
Kinetics for copper are presented in Figure 13-13. Again, the rates of copper dissolution were very rapid with copper extraction nearing completion after only two to three weeks.

Figure 13-13: Copper Extraction for Copper Blends



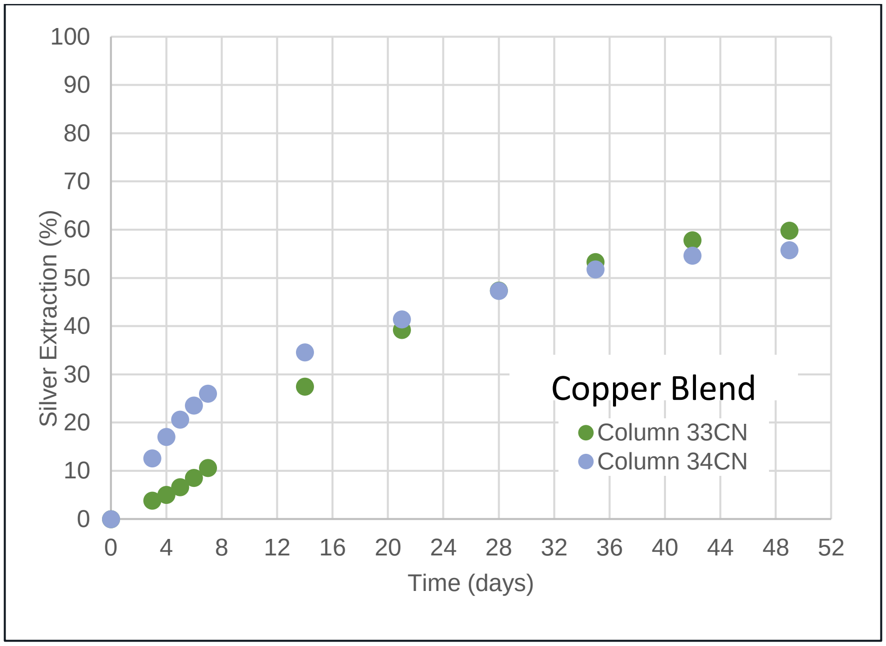
Source: Ausenco, 2019

Figure 13-14: Gold Extraction for Copper Blends



Source: Ausenco, 2019

Figure 13-15: Silver Extraction for Copper Blends



Source: Ausenco, 2019

13.3.7 Alternative Leaching Processes

The primary focus of the test program was to assess the amenability and response of the Filo del Sol mineralization types to heap leaching for the recovery of copper and precious metals. Given the apparent copper leaching kinetics of most of the ore types delineated and the proportion of acid (or water) soluble copper in the ores, two alternative leaching processes were also tested, including washing/scrubbing and grinding/tank leaching.

13.3.7.1 Washing/Scrubbing Tests

Results of the 2017 test program had indicated that the kinetics of copper dissolution were extremely rapid, particularly for the FDS CuAuOx material, due to the presence of abundant copper sulphate minerals.

In 2018, all copper composites were submitted to washing/scrubbing tests in a tumbling cement mixer simulating a trommel. Test conditions and results are summarized in Table 13-34 and Table 13-35, respectively.

Results indicated that the washing/scrubbing process was successful in leaching copper from the FDS CuAuOx composite (91-92% in 6 hours, regardless of the crush size between 0.75 and 1.5 inch). However, the kinetics were not sufficiently rapid for the process to translate to an industrial scale, and in particular the size of the required trommels/tumblers would exceed the mechanical limitations of currently available technology.

Lower copper extraction results for the TMB CuAuOx composite supported the conclusion that the washing/scrubbing process may not be preferable for Filo del Sol ores.

Table 13-34: Washing/Scrubbing Tests Conditions

Test #	Sample	Crush Size (100% minus, inch)	Feed Assay		Acid Curing (kg/t H_2SO_4)	Fe^{3+} added (g/L)	Temp (°C)	pH	ORP (mV SCE)	Ret. Time (hr)
			Total Cu (%)	Acid Sol. Cu (%)						
WSH15	F18 Cu Comp	1.5	0.65	0.59	0	0	24	1.3	473	6
WSH20	F18 Cu Comp	1.5	0.65	0.59	10	0	25	1.1	456	6
WSH22	F18 Cu Comp	1	0.65	0.59	10	0	23	1.1	456	6
WSH24	F18 Cu Comp	0.75	0.65	0.59	10	0	21	1.5	447	6
WSH16	T18 Cu Comp	1.5	0.41	0.31	0	0	30	1.5	465	6
WSH21	T18 Cu Comp	1.5	0.41	0.31	10	0	22	1.7	478	6
WSH26	T18 Cu Comp	1.5	0.41	0.31	10	10	22	1.7	520	6
WSH23	T18 Cu Comp	1	0.41	0.31	10	0	21	1.4	470	6
WSH28	T18 Cu Comp	0.75	0.41	0.31	10	0	20	1.7	470	6
WSH17	Copper Blend #1	1.5	0.91	0.66	10	0	21	1.4	473	6
WSH17b	Copper Blend #2	1.5	0.68	0.54	10	0	25	1.6	462	6
WSH18	F18 M-Ag Comp	1.5	0.95	0.84	10	0	25	1.1	446	6
WSH19	F18 CuCN Comp	1.5	3.37	1.87	10	0	24	1.3	439	6

Table 13-35: Washing/Scrubbing Tests Results

Test #	Sample	Cu Extracted at 6 hrs (%)	Acid consumption (kg/t)	Weight Loss (%)
WSH15	F18 Cu Comp	91.5	--18.7	21
WSH20	F18 Cu Comp	91.9	-75.9	20
WSH22	F18 Cu Comp	90.9	-87.5	21
WSH24	F18 Cu Comp	91.9	-81.0	20
WSH16	T18 Cu Comp	53.1	1.7	5
WSH21	T18 Cu Comp	57.7	5.3	5
WSH26	T18 Cu Comp	75.5	--46.8	7
WSH23	T18 Cu Comp	78.8	-25.5	7
WSH25	T18 Cu Comp	73.6	-35.7	6
WSH17	Copper Blend #1	77.5	-6.0	13
WSH17b	Copper Blend #2	81.0	-61.4	18
WSH18	F18 M-Ag Comp	87.0	-34.2	19
WSH19	F18 CuCN Comp	43.7	-22.8	14

13.3.7.2 Grinding/Tank Leaching

A series of five sequential (acid leach followed by cyanide leach) bottle roll tests were carried out on ground copper material (Copper Blend #2, FDS CuAuOx composite, TMB CuAuOx composite and FDS M-Ag), while three sequential leach bottle roll tests were carried out on ground gold materials from FDS AuOx and TMB AuOx.

Test conditions are detailed in Table 13-36 and results are summarized in Table 13-37.

Table 13-36: Grind/Tank Leaching Test Conditions

Test #	Sample	Acid Leach				Cyanide Leach				
		P_{80} (µm)	Solids (%)	Ret. Time (hr)	pH	Solids (%)	Ret. Time (hr)	pH	NaCN (g/L)	D.O. (mg/L)
LC-72	Copper Blend #2	47	20	24	1.8	20	48	10.5	1.0	19
LC-79	Copper Blend #2	68	20	24	1.8	20	48	10.5	1.0	17
LC-74	F18 Cu Comp	50	20	24	1.8	20	48	10.5	1.0	12
LC-76	T18 Cu Comp	53	20	24	1.8	20	48	10.5	1.0	16
LC-73	F18 M-Ag Comp	~36	20	24	1.8	20	48	10.5	1.0	20
LC-75	T18 G Comp	85	20	24	1.8	20	48	10.5	1.0	17
LC-78	T18 G Comp	143	20	24	1.8	20	48	10.5	1.0	13
LC-77	F18 G Comp	49	20	24	1.8	20	48	10.5	1.0	16

Table 13-37: Grind/Tank Leaching Results

Test #	Sample	Extraction			Reagent Consumption			Wgt Loss (%)
		Cu (%)	Au (%)	Ag (%)	H_2SO_4 (kg/t)	NaCN (kg/t)	CaO (kg/t)	
LC-72	Copper Blend #2	91.1	78.0	84.0	-16.0	1.1	2.9	18.1
LC-79	Copper Blend #2	92.6	84.0	80.6	-14.4	1.4	2.4	17.5
LC-74	F18 Cu Comp	95.1	92.3	57.2	-28.0	1.2	1.0	20.4
LC-76	T18 Cu Comp	79.9	62.6	5.6	-1.3	0.5	5.3	11.5
LC-73	F18 M-Ag Comp	90.3	78.8	35.9	-49.0	1.8	1.3	22.0
LC-75	T18 G Comp	38.4	56.5	18.0	-3.6	0.4	1.0	2.4
LC-78	T18 G Comp	34.6	52.4	15.2	-7.0	0.5	0.3	-1.3
LC-77	F18 G Comp	24.9	60.2	14.1	-4.2	0.3	0.5	5.8

Under the conditions tested grinding/tank leaching resulted in copper extractions that were equal to, or in some cases slightly inferior to those produced in heap leaching tests under comparable conditions.

Precious metal extractions, however, were generally better due to the improved liberation. A preliminary economic trade-off study indicated limited economic advantage for grinding when compared to heap leaching, and so testwork was discontinued. However, tank or vat leaching may have some operational advantages at Filo del Sol, and additional work during future phases of project development to explore these advantages may be of benefit.

13.3.8 Preliminary Evaluation of Potential Process Improvements

A few tests were also completed in an attempt to improve extractions in two particular areas:

- increase copper extractions for samples with lower proportions of acid soluble copper; and,
- increase silver extractions for silver samples containing very high values of silver.

13.3.8.1 Improvement to Copper Extractions

Optimization of the leach conditions - Bottle roll tests

Previous bottle roll tests had indicated copper extraction issues when the proportion of acid soluble copper was low. A few optimization bottle roll tests were therefore completed on selected samples to develop a better understanding of the leach mechanism(s) and to target improved copper extractions during the acid leach. In an effort to improve copper extractions, ferric sulphate additions were made to the leach, to promote and maintain an increased oxidation-reduction potential (ORP) throughout the test, as could be expected in a bacterial leach process. Apart from the ferric iron additions, all other conditions were identical to those used in the remainder of the test program. Samples used are described in Table 13-38, while results are presented in Table 13-39.

Table 13-38: Samples Used for the Ferric Sulphate Additions Tests

Mineralization	Sample	Total Cu Assay (%)	Proportion of Total Cu		
			Acid Soluble (%)	CN Soluble (%)	InSoluble (%)
FDS CuAuOx	FSDH022 (106-116)	10.8	41.2	58.7	0.1
	F18 CuCN Comp.	3.17	59.0	40.7	0.2
	VRC101 (242-252)	0.33	72.4	24.5	3.1
FDS M-Ag	F18 M-Ag Comp.	0.92	91.6	7.1	1.3
	VRC062 (270-286)	0.50	69.8	18.8	11.4
	VRC062 (286-296)	0.72	50.5	20.8	28.7
TMB CuAuOx	T18 Cu Comp.	0.37	82.6	6.7	10.7
Copper Blend #2	Copper Blend #2	0.59	90.9	6.6	2.5

*Proportion of total copper as acid soluble, cyanide soluble, and insoluble based on copper sequential analyses.

Table 13-39: Results of the Ferric Sulphate Additions Tests

Mineralization	Sample	Acid Leach #	Cu Extraction (%)		Δ Cu Extraction
			0 g/L Fe^{3+}	10 g/L Fe^{3+}	%
FDS CuAuOx	FSDH022 (106-116)	55	46.6	-	+20.7
		86	-	67.3	
	F18 CuCN Comp.	58	70.9	-	+1.0
		67	-	71.9	
	VRC101 (242-252)	19	59.3	-	+13.9
		83	-	73.2	
FDS M-Ag	F18 M-Ag Comp.	53	91.8	-	+0.2
		66	-	92.0	
	VRC062 (270-286)	29	60.5	-	-0.1
		84	-	60.4	
	VRC062 (286-296)	30	48.7	-	-2.1
		85	-	46.6	
TMB CuAuOx	T18 Cu Comp.	37	81.0	-	-0.5

Mineralization	Sample	Acid Leach #	Cu Extraction (%)		Δ Cu Extraction
			0 g/L Fe^{3+}	10 g/L Fe^{3+}	%
		65	-	80.5	
Copper Blend #2	Copper Blend #2	69	90.9	-	+0.9
		71	-	91.8	

Significantly improved copper extractions (Δ Cu Extraction of 14 to 21%) were obtained for the two samples containing high proportions of cyanide-soluble copper (LC-86, LC-83) but not for others (LC-67, LC-85).

Additional work would be required in this area, in particular mineralogical identification of the various copper species involved, if an augmented tank leach or bacterial leach process was to be further considered.

13.3.8.2 Improvement to Silver Extractions

Initial bottle roll results had indicated that several high silver samples (FDS M-Ag) gave poor silver extractions. It is believed that this was because some silver minerals (Ag, AgCl, Ag2S) dissolve more easily than other silver minerals, that the kinetics of cyanide leaching silver are typically slower than for gold, and that silver cyanidation typically requires a higher driving force (higher cyanide concentration) than was used in the testwork. Low cyanide concentration results when the addition of the reagent is low or when there are other minerals present that consume cyanide faster than silver. Accordingly, selected tests were run to try and address these issues and compare the results to the standard conditions.

A limited number of bottle roll cyanidation tests were completed on selected samples that produced poor silver extractions under the project standard conditions (1 g/L NaCN, 96 hours).

A total of five bottle roll tests were completed on selected samples during which retention time was doubled compared to typical conditions (192 hours vs. 96 hours), cyanide was maintained at 3 g/L (vs 1 g/L in the typical tests), with all other conditions being kept constant (100% minus 10 mesh, 20% solids, room temperature, and pH ~10.5).

Test conditions are presented in Table 13-40.

Improvements to silver extractions during cyanidation were spectacular for the samples [FSDH016 (78-90) and FSDH023 (162-186)] with Ag extractions increasing from 49% to 92% and from 43% to 97%, respectively. Improvements to silver extractions from the Copper Blend #2, where the proportion of F18 M-Ag was lower, was still significant from both the blend itself and the column 34 residue.

Increased silver extraction also resulted in increased cyanide consumption due to higher dissolution of copper minerals resulting from the higher cyanide level in solution and the extended leach time. Further investigation to optimize silver extraction may be warranted in future phases of the project.

Table 13-40: Results of Extended Cyanidation Tests

Test #	Sample	Retention Time (hr)	NaCN (g/L)	Feed Assay		
				Cu (%)	CN Soluble Cu (%)	Ag (g/t)
CN-71	FSDH016 (78-90)	96	1.0	0.24	0.16	478
CN-116	FSDH016 (78-90)	192	3.0	0.24	0.16	478
CN-74	FSDH023 (162-186)	96	1.0	0.58	0.10	417
CN-117	FSDH023 (162-186)	192	3.0	0.58	0.10	417
CN-95	Copper Blend #2	96	1.0	0.68	0.04	103
CN-118	Copper Blend #2	192	3.0	0.68	0.04	103
CN-119	C-34 CN residue**	96	1.0	0.68	-	49
CN-120	C-34 CN residue**	192	3.0	0.68	-	49

Table 13-41: Results of Extended Cyanidation Tests

Test #	Sample	Ag Extraction (%)		Reagent Consumption		Cu Extraction* (%)
		After 96 hr	After 192 hr	NaCN (kg/t)	CaO (kg/t)	
CN-71	FSDH016 (78-90)	49.0	-	4.3	1.6	32.9
CN-116	FSDH016 (78-90)	72.0	92.4	6.1	2.0	74.7
CN-74	FSDH023 (162-186)	43.0	-	2.4	3.1	46.0
CN-117	FSDH023 (162-186)	89.5	96.6	6.4	3.7	80.9
CN-95	Copper Blend #2	85.5	-	1.3	3.0	39.3
CN-118	Copper Blend #2	86.9	96.1	2.1	2.6	31.1
CN-119	C-34 Residue**	86.0	89.1	1.8	2.9	34.6
CN-120	C-34 Residue**	91.0	92.1	5.6	2.6	34.1

Notes: *Proportion of copper dissolved during cyanidation. **Column 34 (Copper Blend #2) residue was screened; the minus 6 mesh fraction containing more than 90% of the silver in the residue was leached in a bottle roll after crushing to minus 10 mesh.

13.3.8.3 Cyanide Regeneration Using the SART Process

Results presented earlier indicated that cyanide consumption was variable and varied principally on the amounts of cyanide soluble copper left after the acid leach step.

The SART process has been developed and applied commercially in more than ten plants around the world as a method to reduce the copper concentration of cyanide solutions and thereby reduce the net cyanide consumption from copper-gold mineralization. This is achieved by precipitating the copper in solution as copper sulphide ($Cu2S$) while regenerating the cyanide previously consumed by the formation of copper-cyanide complexes.

Standard SART tests were conducted in 2018 on cyanide leach solutions generated during the test program. Test conditions are summarized in Table 13-42 and Table 13-43.

SART results were excellent with almost complete regeneration of the CNWAD in the starting solution, near quantitative precipitation of the copper, and the production of a copper precipitate assaying between 51% and 66% Cu, and 0% to 19% Ag.

Table 13-42: Test Conditions of the SART Tests

Year	Test #	Temp (°C)	Ret. Time (Min)	pH	NaSH Addition (% Stoichiometric)*
2016	SART 2	20	20	4.0	110
2018	SART 1	20	20	4.0	120
2018	SART 2	20	20	3.5	125

Note: *Stoichiometric requirement based on Cu, Zn, Ag present in starting solutions.

Table 13-43: Results of SART tests

Year	Test #	Feed Analysis			Metal Precipitated		CN_{WAD} Regenerated (%)	Precipitate Assay		Reagent Consumption (kg/m³)		
		Cu (mg/L)	Ag (mg/L)	Au (mg/L)	Cu (%)	Ag (%)		Cu (%)	Ag (%)	NaSH (67.8%)	$Ca(OH)_2$	H_2SO_4
2016	SART 2	920	0.5	0.06	99.9	>94	~100	66.6	0.04	(0.52)	0.28	2.85
2018	SART 1	264	18.8	0.08	96.0	~100	95	65.7*	4.9*	0.15	0.91	1.52
2018	SART 2	80	28.9	0.08	99.0	~100	>99	51.4*	18.8*	0.05	2.22	4.21

Note: *Values calculated from solution assays (In/Out) and weight of Cu2S precipitate (too little weight for direct assays).

13.3.9 Miscellaneous Test Programs

13.3.9.1 Copper Solvent Extraction

Copper contained in leach solutions from heap leach operations is almost exclusively treated by solvent extraction to upgrade and purify the copper from solution and recycle the acid bound to the copper. This process is well understood and is an industry standard process for recovery of copper from acidic leach solutions.

A limited number of tests were completed to confirm that solutions originating from column leaching of FDS CuAuOx and TMB CuAuOx ores would be amenable to solvent extraction.

A readily available commercial copper extractant (LIX-984N), prepared as 10% by volume in a diluent (APCO D80) was used for all tests. Test conditions for the three tests are summarized in Table 13-44.

Table 13-44: Test Conditions for the Loading Isotherms

Test #	Feed Sample					Temp	Contact Time	Fe:Cu Ratio (g/L Fe : g/L Cu)
	Col #	Ore type	Cu (g/L)	Fe (g/L)	pH	(°C)	(min)	
CuX-Iso1	C-26	F18 CuComp	1.14	3.52	2.0	20	2	3.09:1
CuX-Iso2	C-23	T18 CuComp	0.87	0.56	2.0	20	2	0.64:1
CuX-Iso3	C-34	Copper Blend #2	0.94	2.42	2.0	20	2	2.57:1

Leach solutions originating from column leaching the FDS CuAuOx and TMB CuAuOx Composites and that from leaching the Copper Blend #2 Composite were used for this program.

Loading isotherms for all three tests indicated the copper extraction proceeded as expected. Loaded organics from all three tests were then stripped for ten minutes using a 160 g/L H_2SO_4 strip solution at a 1:1 phase ratio.

Stripping results are summarized in Table 13-45. The stripping solutions in all three cases were good or excellent quality and no transfer of major impurities to the subsequent electrowinning (EW) circuit would be expected.

Table 13-45: Results of Stripping Isotherms

Test #	Sample		
	C-26 (F18 Cu Comp)	C-23 (T18 Cu Comp)	C-34 (Copper Blend #2)
	CuX-Isol	CuX-Iso2	CuX-Iso3
Cu (g/L)	4.55	4.50	4.88
Fe (g/L)	0.017	0.012	0.024
Te (mg/L)	<1	<1	<1
Hg (mg/L)	<0.0001	<0.0001	<0.0001
As (mg/L)	<3	<3	<3
Bi (mg/L)	<6	<6	<6
Pb (mg/L)	<2	<2	<2
Sb (mg/L)	<3	<3	<3
Se (mg/L)	<3	<3	<3
Ag (mg/L)	<0.08	<0.08	<0.08
Be (mg/L)	<0.002	<0.002	<0.002
Fe: Cu Ratio (g/L Fe : g/L Cu)	3.74×10^{-3} : 1	2.67×10^{-3} : 1	4.92×10^{-3} : 1

13.3.9.2 Deportment of Mercury

The presence of mercury within the deposit is known and a program was completed to follow the deportment of mercury during the leaching process.

Table 13-46 summarizes the mercury assays of the various composites tested.

Table 13-46: Mercury Assays of Composites

Assays	Composites					
	F18 Cu Comp	T18 Cu Comp	F18 CuCN	F18 M-Ag	Copper Blend #1	Copper Blend #2
Hg (g/t)	9.4	<0.3	8.8	284	7.1	43.4
Ag (g/t)	11.8	0.8	(1.0)	474	9.4	103

Mercury content is variable but mostly present in the high silver samples (F18 M-Ag and Copper Blend #2).

Selected leach solutions from the acidic copper leach and the cyanide leach programs were analyzed for mercury to assess the deportment of mercury throughout the leach process.

For the copper acid leach, leach solutions from the highest mercury content samples (Copper Blend #2 and M-Ag) were analyzed for mercury. Results are presented in Table 13-47.

Table 13-47: Mercury Assays for Selected Acidic Copper Leaches

Test #	Feed	Hg assays in leach solution (mg/L)						Hg Extracted* (%)
		0 hr	1 hr	2 hr	4 hr	8 hr	24 hr	
LC-72	Copper Blend #2	<0.001	<0.001	0.001	0.006	0.002	<0.001	0.01
LC-73	F18 M-Ag	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.002

Note: *Based on head assay.

Very little mercury was dissolved during the acidic copper leach from the two highest Hg content samples even after grinding the feed (P80 of 36-47 µm). Selected cyanide leach solutions were analyzed for mercury. Results are presented in Table 13-48.

Table 13-48: Mercury Assays for Selected Cyanide Leach Solutions

Test	Feed	Feed Particle Size	Hg Solution Analysis (mg/L)				
		F80 µm	8 h	24 h	48 h	72 h	96 h
CN-63	FSDH021 (110-134)	-10 mesh	0.050	0.015	0.009	0.006	0.005
CN-71	FSDH016 (78-90)	-10 mesh	0.008	0.01	0.02	0.005	0.09
CN-88	FDS M-Ag Comp	-10 mesh	0.16	4.65	4.12	5.25	6.52
CN-96	Copper Blend #2	-10 mesh	0.011	0.018	NSS	0.71	0.024
			Hg Solution Analysis (mg/L)				
			3 h	6 h	24 h	30 h	48 h
CN-109	F18 M-Ag Comp	~36	0.81	0.77	1.07	1.18	1.35
CN-108	Copper Blend #2	47	0.01	0.005	0.38	0.61	0.4

Results indicated a limited dissolution of mercury during cyanide leaching, in particular for the samples high in silver/mercury (F18 M-Ag Comp and Copper Blend #2). Most of the mercury in the heap leach feeds is expected to remain in the heap.

The small amount of mercury entering the cyanide solution when processing high mercury ores is expected to report to the SART copper precipitate and/or the gold room. At the levels reported, mercury deportment in the proposed process is not expected to be an issue.

13.3.9.3 Samples for Environmental Assessment

Various samples were sent to SGS Canada (Burnaby) for specific chemical analyses relevant to the environmental program. Summary results of this work is described in more detail in Section 20.3.5.

These included cyanide leach residues for selected tests (bottle roll or columns), leach solutions (acid and cyanide, bottle roll and columns), and column wash solutions (acid and cyanide).

Moreover, samples of the column cyanide leach residues for various zone samples (T18 G Comp., T18 Cu Comp., F18 G Comp., F18 Cu Comp., and Copper Blend #2) were also provided for analysis.

13.3.10 Conclusions

Based on prior results, a comprehensive test program was conducted in 2018 to confirm and optimize those results on new samples freshly collected in early 2018 (surface/trench samples, RC chips, and diamond drill core samples).

In total, 14 surface trench samples, 32 RC chips samples and 20 diamond drill hole intervals were collected and sent to SGS (Lakefield) for various test programs. A total of more than 3,500 kg of samples was shipped to Canada, and various composites were prepared to represent the various mineralization types. Composites of the gold oxide zones (both TMB AuOx and FDS AuOx), as well as the copper-gold oxide zones (TMB CuAuOx, FDS CuAuOx and FDS M-Ag), were prepared and tested. Finally, copper blends of all the copper-gold mineralization types were also prepared and tested.

13.3.10.1 Characterization of the various ore mineralization types and composites

Several samples including the various composites and the copper blends were submitted to physical, chemical and detailed mineralogical characterization. Physical characterization included hardness, abrasivity and bulk density. Based on the results, the materials are not expected to present any undue issues for the design of a conventional crushing circuit.

Most of the metallurgical program was devoted to the leaching stage of the process, particularly heap leaching. Heap leaching was simulated by conducting column leaching of the material at coarse sizes ranging from 0.5 to 2.5-inch crush size and using ~50 to 250 kg of sample per column test. Cyanide column leaching was tested for the gold oxide mineralization (a total of 11 column tests), while sequential column leaching (acid leaching followed by washing/neutralization and cyanide leaching) was used for the copper-gold oxide mineralization (a total of 18 sequential column tests).

Variability testing, as well as some optimization programs, was carried out using bottle roll tests on minus 10 mesh material. Both cyanide leaching (a total of 21 bottle roll tests) and sequential leaching (a total of 72 sequential leach bottle roll tests) were conducted during the 2018 program.

In addition to heap leaching, other leaching methods were also tested during the program, such as: grinding-agitation leach (cyanide leaching for the AuOx mineralization or sequential acid-cyanide leaching for the CuAuOx mineralization) and washing-scrubbing for the acid leaching of the CuAuOx mineralization.

Finally, downstream processes were also briefly tested on products generated during the test program. Copper loading isotherms were prepared to confirm the suitability of solvent extraction to recover the copper selectively from chosen column-leach solutions.

SART tests were successfully conducted on selected cyanide solutions to confirm the SART process could re-generate the cyanide consumed by copper minerals, and simultaneously recover the copper dissolved during cyanidation.

13.3.10.2 AuOx Mineralization

The main parameters tested during this 2018 program were cement agglomeration (0 to 15 kg/t cement), crush size (from 0.5 to 2.5 inch) and retention time.

Selected results (average of all 1.5-inch crush size tests) are presented in Table 13-49.

Table 13-49: Tamberías and Filo del Sol Gold Oxide Column Tests Summary

Zone	Cement (kg/t)	Crush size (inch)	Head assay (g/t)		% Ave extraction		Reagent consumed (kg/t)	
			Au	Ag	Au	Ag	NaCN	CaO
TMB AuOx	0-5	1.5	0.55	10.0	40.0	19.5	0.58	2
FDS AuOx	5-15	1.5	0.35	1.0	81.1	15.2	0.90	7.8

Results indicate a much better extraction of gold from the FDS AuOx composite than for the TMB AuOx composite. This result corresponds with mineralogical examination which indicates that gold particles at Tamberías are encapsulated by silica relative to Filo del Sol mineralization. Further confirmation of this is provided by the grinding/tank leach tests, which showed increased gold recoveries from TMB AuOx mineralization with increased grinding and associated increased gold particle liberation.

13.3.10.3 CuAuOx Mineralization

Column tests were conducted on composites of the two main copper-gold zones (TMB CuAuOx and FDS CuAuOx). In addition, column tests were also carried out on blends of all the copper zones, designed to mimic a proportionally representative sample of the whole deposit.

For all these composites, sequential leaching was carried out. During the acid leach, the main parameters tested were acid curing, crush size and retention time. During the cyanide leach, no cement was added and the main parameter tested was crush size and retention time.

A total of 16 sequential column tests were carried out on the two main copper-gold composites and two more on the copper blends. Conditions and results for the 1.5-inch crush size columns are summarized in Table 13-50 and Table 13-51.

Table 13-50: Copper Gold Oxide Zones – Summary Conditions

Zone	Head assays			Acid Leach Curing Acid (kg/t)	Cyanide leach Cement (kg/t)
	% Cu	g/t Au	g/t Ag		
TMB CuAuOx	0.41	0.25	0.8	5-25	0
FDS CuAuOx	0.65	0.31	11.8	0-25	0
Copper Blend #1	0.91	0.29	9.4	10	0
Copper Blend #2	0.68	0.32	103	10	0

Table 13-51: Copper Gold Oxide Zones – Summary Results (Ave. 1.5-inch columns)

zone	% Extraction			Reagent Consumed (kg/t)		
	Cu	Au	Ag	H_2SO_4	NaCN	CaO
TMB CuAuOx	86.7	55.8	36.8	36.9	1.0	4.2
FDS CuAuOx	95.3	75.8	89.6	-18.3	1.4	1.8
Copper Blend #1	86.3	64.4	59.8	3.3	2.4	3.0
Copper Blend #2	92.0	67.5	55.7	-9.4	1.9	3.0

Copper extractions from the two copper zones ranged from 86.7% (Tamberías) to 95.3% (Filo del Sol), with rapid leach kinetics. This was particularly so for the Filo del Sol zone. Due to the mineralogy of the copper in the Filo del Sol zone, where copper is mostly present as water soluble sulphates of copper, the composite actually generates acid during leaching.

Gold extraction from the main zones ranged from 55.8% (TMB CuAuOx) to 75.8% (FDS CuAuOx), while silver extraction ranged from 36.8% (TMB CuAuOx) to 89.6% (FDS CuAuOx). The two copper blends were prepared using varying proportions of the copper zones. Copper Blend #2 represents the overall deposit (based on reserves) as it is presently known.

Extractions from Copper Blend #2 were 92.0%, 67.5% and 55.7% for copper, gold and silver, respectively. Because of the presence of large amounts of water-soluble sulphates (Cu, Fe, Al) in the copper zones, a significant weight loss was observed in the columns after the copper acid leach, ranging from 8% (TMB CuAuOx) to 19% (FDS CuAuOx) and 21% (Copper Blend #2).

13.3.10.4 Alternative Leaching Process

Alternative leaching processes were tested on the composites. The first leach alternative considered was a washing/scrubbing process on coarse material. Although the process was successful in rapidly dissolving the copper with acid, it was not fast enough to justify the use of large rotating equipment such as trommels.

The second leach alternative considered was grinding followed by agitation leaching in tanks. The results of finer grinding showed little improvement in copper extractions; however, gold and silver extractions were improved.

13.3.10.5 Testing of Downstream Processes

Copper Solvent extraction: The standard recovery process for copper from leach solutions involves the use of solvent extraction. Three tests were conducted on leach solutions produced from the test program with the aim to confirm that the leach solutions from the column tests could be processed using commercial extractants (LIX 984N).

The results confirmed a selective extraction of copper from the leach solutions.

SART process: The SART process (Sulphidization-Acidification-Recycle-Thickening) has been developed to alleviate high cyanide consumption caused by copper minerals soluble in cyanide. The process re-generates the cyanide consumed by copper (and thus decreases the overall cyanide costs) and at the same time recovers the copper present in the cyanide solutions by precipitating it as a high-grade copper sulphide compound (Cu_2S).

Two SART tests were carried out on cyanide solutions produced during the test program. For both tests CNWAD regeneration was greater than 95%, and copper recovery by precipitation was greater than 96%, resulting in copper grades in the precipitate ranging from 51 to 65% Cu.

13.3.10.6 Environmental Testwork

Selected samples of leach solutions and leach residues have been collected during the metallurgical test program and sent for geochemical testwork in support of the environmental programs.

13.4 Metal Recovery Estimates

The estimates for copper recovery from the various mineralized ore types were based on the extractions results obtained in laboratory bottle roll and column leach tests completed at SGS Lakefield under the supervision of HydroProc Consultants, as presented above.

For reference purposes, the LOM distribution of mined material is presented in Table 13-52.

Table 13-52: Summary of LOM grades per ore type

Material	Mass Distribution (%)	Cu (%)	Au (g/t)	Ag (g/t)	Mass (kt)
FDS AuOx	10	0.06	0.51	3.1	24,922
FDS M-Ag	16	0.50	0.42	78.3	40,935
FDS CuAuOx	60	0.42	0.27	3.7	154,173
TMB AuOx	0	0.18	0.37	1.7	240
TMB CuAuOx	15	0.37	0.34	1.7	38,808
Total	100	0.39	0.33	15.12	259,078

Recovery predictions were heavily weighted on the 2018 testwork program, as those samples better represented the lithologies of the deposit as developed in the block model and proposed mine production schedule.

The 2018 column tests were completed on composite samples representing each main ore type and two overall blends. Variability is observed in recoveries within the database for each ore type. No variability work has been completed on column tests. In the absence of variability data from column testing, which best simulates the performance of heap leach operations, the bottle roll test results were used to determine the recoveries for the project.

Column leach test extractions were compared to the bottle roll test results obtained on the same composite samples. A bottle roll test to column leach test correction factor was applied when analysing the bottle roll test variability test results to determine the recoveries for the project.

Test results were analysed per ore types and recoveries were calculated by ore type year-by-year in the financial model.

13.4.1 Copper Leaching Time and Extraction Model

Copper leaching kinetics, as reported above, indicated completion of extractions in the order of two weeks for the FDS CuAuOx composite Figure 13-7 and in the order of eight weeks for the TMB CuAuOx composite Figure 13-10.

The copper blends kinetics Figure 13-13 confirmed rapid copper leaching rates, in the order of two to six weeks, with the lower rate observed for the sample with higher content of TMB material.

The metal recovery was based on the bottle roll variability testwork in conjunction with the column test results. When evaluating the bottle roll variability test results, the copper sequential assays on the specific feed materials were used to determine the correlations to forecast copper extractions. The copper sequential assays, including Acid Soluble Cu (CuAS), Cyanide Soluble Cu (CuCN) and residual Cu (CuRES), are available within the block model.

The correlations to calculate copper extraction based on sequential assay for FDS CuAuOx are:

- If CuCN%<15%, Extraction = CuAS%+0.45*CuCN%
- If CuCN% between 15%-25%, Extraction = CuAS%+0.3*CuCN%
- If CuCN% between 25%-45%, Extraction = CuAS%+0.2*CuCN%
- If CuCN% >45%, Extraction = CuAS%+0.1*CuCN%

The equation to calculate TMB CuAuOx copper extraction is:

- Extraction = 0.95*CuAS+0.45*CuCN

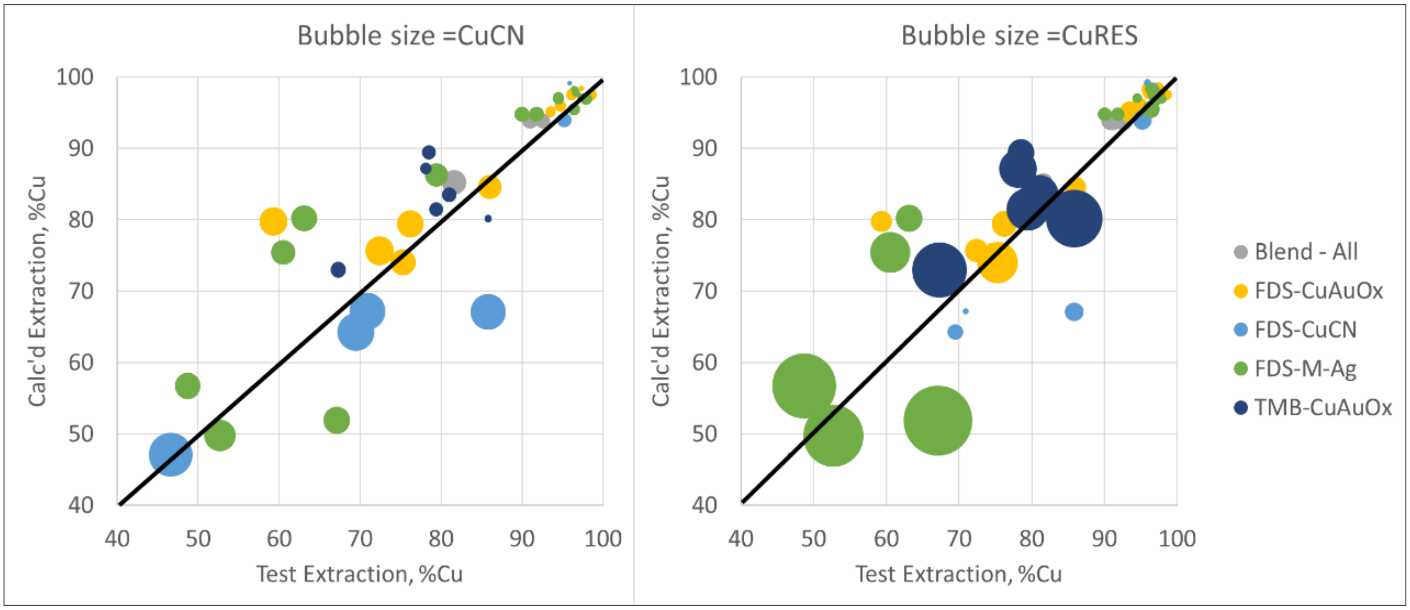
Note: CuCN%=CuCN/(CuAS+CuCN+CuRES)

And CuAS%=CuAS/(CuAS+CuCN+CuRES)

The graphs in Figure 13-16illustrate the correlation between calculated copper extraction and actual test result.

Comparative CuCN and CuRES levels can be observed as per bubble size in the graph. The comparison of bottle roll and column test results on the same composite samples and comparable conditions indicated similar results, so the bottle roll test results for copper extraction are assumed to be equivalent to column test results. The correlations when applied to current mine schedule return a LOM copper extraction of 83%.

Figure 13-16: FDS and TMB CuAuOx Cu Extraction Calculated vs. BRT Result.



Source: Ausenco, 2019

13.4.2 Gold and Silver Leaching Time and Extraction Model

Gold leaching kinetics indicated incomplete extraction of gold within the 49 to 105 days of column leaching test for either FDS or TMB gold oxide composites. Gold leach kinetic curves are presented in Figure 13-5 for the TMB AuOx composite and Figure 13-6 for the faster leaching FDS AuOx composite.

The gold and silver extraction kinetics for copper blends, shown in Figure 13-14 and Figure 13-15, confirm faster gold leaching rates with a slightly higher leaching rate observed for the sample with higher content of FDS material. Incremental gold and silver extractions could be expected with leaching times in excess of the 49 days tested.

The metal recovery forecast was based on the bottle roll variability testwork in conjunction with the column test results. Bottle roll test extractions were averaged by ore type and a factor was applied for those ore types in which the same composite sample was tested under similar conditions in both bottle roll and column leach tests.

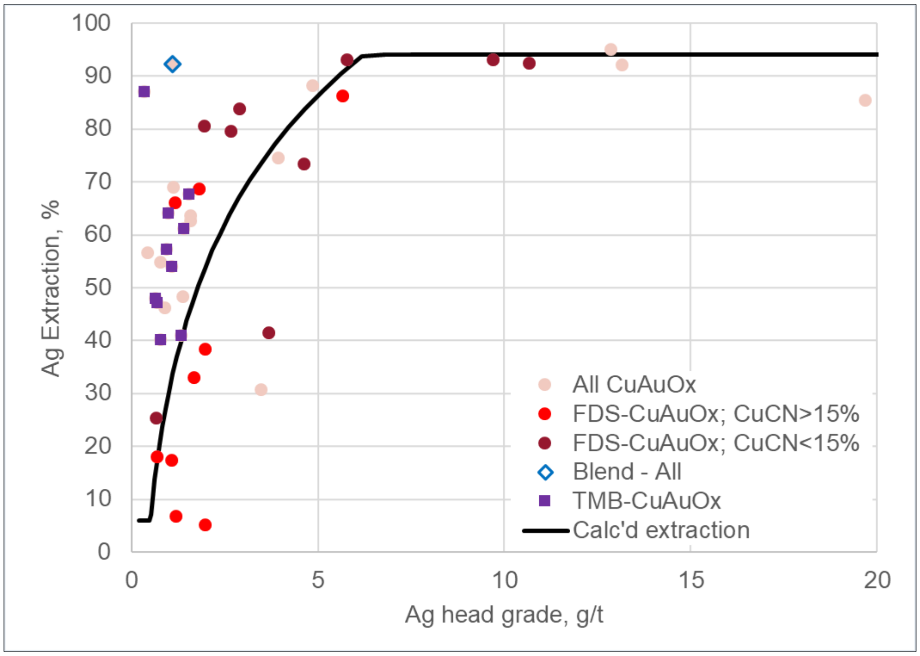
A head grade to recovery correlation was developed to calculate the silver production for the CuAuOx. The correlation was developed for FDS. The equation to calculate the Ag extraction based on Ag head grade is:

- Ag Extraction, % = 35*ln(Ag head grade, g/t) + 30
- Allow for maximum 94% and minimum 6% Ag extraction

The database shows a wide spread of extractions of silver for head grade below 5 g/t Ag. Figure 13-17 illustrates the extraction equation and the dataset by ore type.

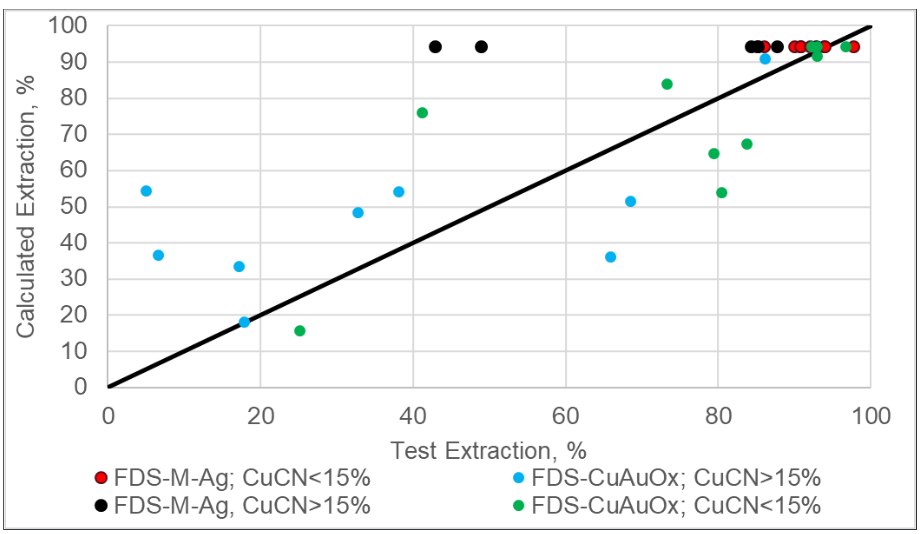
Figure 13-17Illustrates the correlation between calculated silver extraction and actual test results.

Figure 13-17: FDS CuAuOx Ag Extraction Equation and BRT Result



Source: Ausenco, 2019

Figure 13-18: FDS CuAuOx Ag Extraction Calculated vs. BRT Result



Source: Ausenco, 2019

The results for Au and Ag extraction, CLT/BRT factor and calculated CLT extractions are presented on Table 13-53.

Table 13-53: BRT to CLT Au and Ag Extraction by Ore Type

Ore type	BRT Ave. Extraction, %		CLT/BRT Factor		CLT Calc'd Extraction, (%)	
	Au	Ag	Au	Ag	Au	Ag
FDS-AuOx	89	31	87%	57%	78	17
FDS-CuAuOx (overall)	75	Function of head grade	95%	96%	78	62
TMB-AuOx	48	30	95%	69%	50	22
TMB-CuAuOx	71	58	84%	73%	60	42

13.4.3 Overall Metal Recovery

Overall metal recovery, from ore to cathode and SART precipitate for copper or doré for gold and silver, was calculated on the basis of extractions achieved by ore type in the various leach tests, with appropriate adjustments to reflect non-ideal conditions within the heaps, which include variation in:

- ore feed
- agglomeration and stacking
- solution application
- permeability within the heap, percolation of lixiviant
- edge irrigation effect
- temperature.

For Filo del Sol, it is recommended to apply a 4% reduction to copper extraction and 3% reduction to gold and silver extractions to the recovery equations for each metal for each ore type discussed above to determine the overall metal recoveries, as summarized in Table 13-54. In short, these adjustments account for physical phenomenon in the heap leaching process as well as any minor losses of metal values through the subsequent processing steps.

Table 13-54: Leach Recovery Factors for Non-Ideal Conditions

Metal	Recovery Factor	Comment
Cu	4%	Non-agglomerated, on-off pad, stacked
Au	3%	100% agglomerated, permanent pad, stacked
Ag	3%	100% agglomerated, permanent pad, stacked

13.5 SGS Minerals (Lakefield), 2020-2022

13.5.1 Filo del Sol Sulphide Mineralization (Deeper Zone) Preliminary Testwork

Preliminary metallurgical testwork was conducted on three composite samples of material from drill core originating from the 2018-2019 and 2019-2020 drilling campaigns on material that is not included in the resource model as stated in Section 14.1. The metallurgical testwork was completed at SGS Minerals Services, Lakefield, Ontario, during 2020, 2021 and 2022. The samples varied from 0.33% to 0.57% Cu, 0.38 to 0.41 g/t Au, and 1.3 to 10.3 g/t Ag. Two of the samples had a low arsenic content ("HiRes" material) of ≤10 g/t and one sample ("HiCN" material) had a high arsenic content; 1,400 g/t or 0.14%. The copper contained in the HiRes samples was mainly present as chalcopyrite with minor bornite, covellite and chalcocite. The copper contained in the HiCN sample was mainly covellite and tennantite, with minor enargite, bornite, chalcocite and chalcopyrite.

The material tested was not intended to reflect the elemental grade(s) of the oxide resource, nor the proportion of each type of material that may occur within the oxide resource. The focus of the preliminary testing was to provide insight and direction for future testing requirements for the hypogene sulphide portion of the deposit. In this context, the HiRes and HiCN material are reasonably representative of the potassic alteration associated porphyry and high-sulphidation mineralization (as described in Section 7 of this report), respectively, at least as far as their characteristics are known from the current drilling.

Preliminary scoping flotation tests were performed on all three composite samples, including rougher kinetic and batch cleaner tests. Two locked cycle flotation tests were performed, one on a combined HiRes sample and the other on the HiCN sample. The locked cycle tests utilized a flowsheet consisting of primary grinding, rougher flotation, concentrate regrinding, and 3-4 stages of cleaner flotation. The flowsheet and reagent scheme were not fully optimized for this testing program. For the HiRes sample, a concentrate containing 22% Cu, 18 g/t Au, 37 g/t Ag and 880 g/t As was produced, while the HiCN sample produced a concentrate containing 26% Cu, 14 g/t Au, 106 g/t Ag and 52,400 g/t (5.24%) As.

In addition to the preliminary flotation tests, the flotation cleaner tailings were subjected to intensive cyanide leaching tests in an effort to recover additional precious metal values. The results indicated that an additional 10-16% of the gold and 10-26% of the silver could potentially be recovered. Using this flowsheet, approximately 88% of the copper and 80% of the gold was recovered from the HiRes sample, and 90% of the copper and 75% of the gold from the HiCN sample.

Preliminary comminution testing indicated that the composite samples reflected a moderate hardness, with an indicated Bond Ball Mill Work Index of 14-15 kWh/t.

The results of the flotation testing on the HiRes samples indicated that a concentrate suitable for smelting without further treatment or penalties could be produced. Smelter penalties for arsenic content in concentrates are typically incurred for material containing >2,500 or >3,000 g/t with a limit of 5,000 g/t above which the concentrates may be rejected. As such, the flotation testing on the HiCN sample indicated that the concentrates would require further treatment for arsenic removal prior to smelting, or the HiCN material must be processed by alternative means to manage the arsenic content. Based upon the drilling completed to date, Filo expects that a significant portion of the Deeper Zone sulphide will contain elevated arsenic content. While blending of the mineralized material and/or concentrated product may be suitable to allow limited (small tonnage) processing of sulphide material with elevated arsenic content together with lower arsenic feed, it is likely that additional processing steps would be required to manage the elevated arsenic content for a significant portion of the sulphide resource.

Based upon the above, preliminary metallurgical testing of various treatment options for arsenic-bearing, copper-gold, mineralized material was initiated and completed in 2022. The treatment options included:

- alkaline sulphide leaching of concentrates to remove and stabilize arsenic and produce a saleable concentrate;
- partial roasting of concentrates to remove and stabilize arsenic and produce a saleable concentrate; and
- pressure oxidation followed by conventional SX-EW technology.

This work was performed by SGS Mineral Services, Lakefield, Ontario, Canada, and at Dundee Sustainable Technologies, Montreal, Quebec, Canada.

Small laboratory scale alkaline sulphide leaching tests were performed on samples of the high arsenic (HiCN) concentrate produced by flotation using 40 g/L sodium hydroxide and 300-400 g/L sodium sulphide at 80°C for 2 hours. The results indicated that >98.5% of the arsenic was extracted, producing an upgraded concentrate containing 31-33% Cu and <1,000 g/t (0.1%) As. These results indicate the technical feasibility of extracting arsenic from the concentrates by alkaline sulphide leaching. Further work is required to optimize leach conditions and minimize reagent consumptions, and also to develop and prove up the necessary downstream portions of the process flowsheet to effectively stabilize arsenic (in a form suitable for long term disposal) and to generate a transportable concentrate containing the copper, gold, and silver values.

Small laboratory scale partial roasting tests were completed on samples of the high arsenic (HiCN) concentrate produced by flotation by roasting at 650-703°C for between 25 and 60 minutes. The work was performed by Dundee Sustainable Technologies. The test completed at 703°C for 60 minutes indicated that approximately 96% of arsenic was volatilized and removed from the concentrate, producing an upgraded concentrate containing 38% Cu and 0.3% As. In addition, lead, zinc, antimony and mercury were also volatilized to varying extents. These results indicate the technical feasibility of extracting arsenic from the concentrates by partial roasting. Further work is required to optimize roasting conditions, and to develop and prove up the necessary downstream portions of the process flowsheet to effectively stabilize arsenic (in a form suitable for long term disposal) and to generate a transportable concentrate containing the copper, gold and silver values.

Options for stabilization of arsenic removed by alkaline sulphide leaching and partial roasting processes include precipitation as ferric arsenate (scorodite) under controlled conditions and a vitrification process whereby the arsenic is locked in glass.

Small-scale tests were performed to examine the technical feasibility of treating flotation concentrates by pressure oxidation followed by conventional SX-EW technology to recover copper and downstream treatment of the pressure oxidation solid residue by cyanide leaching for gold and silver recovery. The pressure oxidation step was performed in an autoclave at 220°C for 2 hours with a target oxygen over-pressure of 690 kPa (100 psi). The results indicated that approximately 93% copper extraction was achieved from high grade concentrate (27% Cu) and over 99% copper extraction was achieved from a lower grade concentrate (12% Cu). The better performance of the lower grade concentrate is attributable to the more favourable Fe:As ratio in the feed. No regrinding of the flotation concentrate was required. After separation and removal of solution containing copper, the pressure oxidation residue was neutralized and subjected to a hot lime boil procedure to break down silver jarosite. The resulting slurry was adjusted to pH 10.5-11 and 10% solids, and the slurry was leached using 1 g/L sodium cyanide for 48 hours. Under these conditions, 95-98% of the contained gold and 35-94% of the contained silver were extracted. The environmental stability of the cyanide leaching residues was examined by conducting standard US EPA TCLP tests (TCLP-1311 procedure). The results indicated that, pending optimization of the process flowsheet and conditions, the concentrations of arsenic and other elements in the test leach solution would meet the applicable standards.

In addition to the three process options identified above for treatment of arsenic-bearing copper concentrated product, there has been significant interest and progress within the copper industry towards the development of crushed mineralized material heap and run-of-mine (ROM) heap leaching technology to effectively extract and recover copper from primary copper sulphide mineralized material (containing dominantly chalcopyrite, enargite, tennantite, tetrahedrite, bornite and covellite copper mineralization). If successful, such efforts could result in process technology applicable to treat Filo del Sol sulphide mineralization. Filo continues to monitor the progress in this area and plans to further evaluate and potentially test one or more options on representative samples of sulphide mineralization from Filo del Sol.

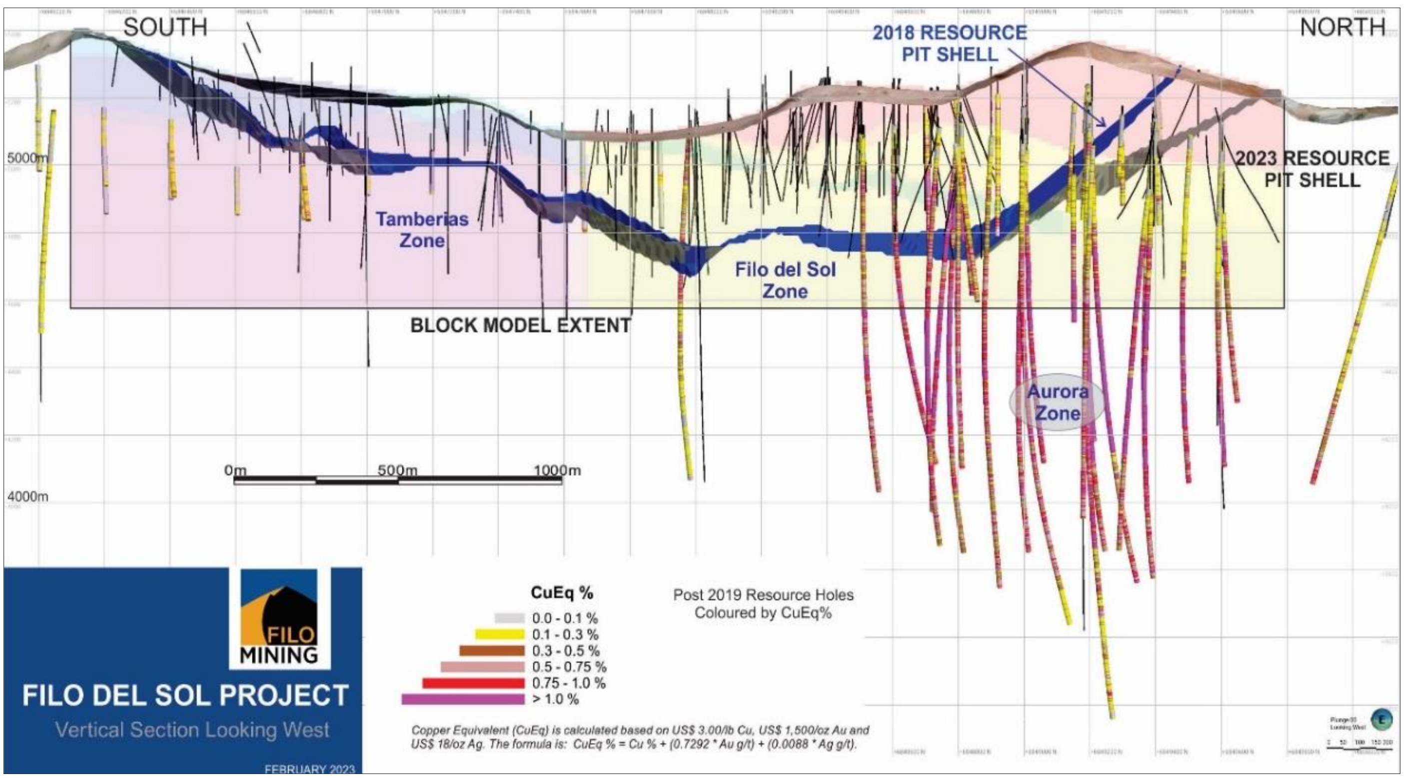
Based on the preliminary testing on a small number of samples of deeper zone sulphides from Filo del Sol, all three options described above (alkaline sulphide leaching and partial roasting to upgrade concentrates, and pressure oxidation and residue treatment to recover copper, gold and silver as saleable products) are considered to be technically viable approaches to the treatment of concentrates with elevated arsenic concentrations. Additional work is required to test these approaches on representative samples of feed from a range of locations within and across the sulphide mineralization. Also, additional work is required to fully develop the process flowsheet(s), to further optimize operating conditions and reagent consumptions, to determine the preferred method for removal, recovery and/or stabilization of arsenic and other species, and to determine process economics (metal recoveries, operating costs, capital costs, and sustaining capital costs). Further, the environmental, social and governance aspects of the preferred process(es) selected for development need to be assessed in detail. Such work is ongoing.

14 MINERAL RESOURCE ESTIMATE

14.1 Introduction

This resource update replaces the resource estimate released in February of 2019. Although this update considers the results of 60 new holes completed since the previous mineral resource estimate it should be noted that the block model limits were not changed from the 2019 model and the new resource does not include the deeper, high-grade mineralization constituting the Aurora Zone (see Figure 14-1) as it was determined that, at the cutoff date, there was insufficient data density to warrant including the mineralization outside of the model limits.

Figure 14-1: Aurora Zone Drilling Below Reported Resource



Source: Filo Mining Corp., 2023

Copper, gold and silver grades were estimated by ordinary kriging using Geovia Gems™ software. Implementation of geologic control for grade estimation is consistent with that used for the 2019 mineral resource estimate; an updated geological model was used as control for grade interpolation of the three metals. The distribution of assay and composite grades were statistically well-behaved for all elements. High-grade capping was applied, with a generally low impact on metal content. The reporting of the resource inside an optimized pit ensures reasonable prospects of eventual economic extraction. Table 14-1 summarizes the Mineral Resources at Filo del Sol.

There are no known legal, political, environmental, or other risks that could materially affect the potential development of the mineral resource described herein. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 14-1: Estimated Mineral Resource: Filo del Sol Deposit

Min. Type	Cutoff	Category	Tonnes (millions)	Cu (%)	Au (g/t)	Ag (g/t)	lbs Cu (millions)	Ounces Au (thousands)	Ounces Ag (thousands)
AuOx	0.20 g/t Au	Indicated	54.4	0.06	0.40	3.0	72	705	5,250
		Inferred	24.0	0.10	0.31	2.1	52	241	1,640
CuAuOx	0.15% CuEq	Indicated	265.0	0.37	0.30	3.5	2,179	2,558	29,750
		Inferred	97.3	0.27	0.28	2.8	588	889	8,670
Ag	20 g/t Ag	Indicated	42.8	0.46	0.42	87.1	432	576	119,670
		Inferred	11.4	0.34	0.42	87.5	85	154	32,060
Hypo	0.30% CuEq	Indicated	70.4	0.31	0.35	2.5	473	790	5,710
		Inferred	78.9	0.31	0.33	3.1	542	834	7,960
Total		Indicated	432.6	0.33	0.33	11.5	3,156	4,629	160,380
		Inferred	211.6	0.27	0.31	7.4	1,267	2,118	50,330

14.2 Available Drill Data and Model Set Up

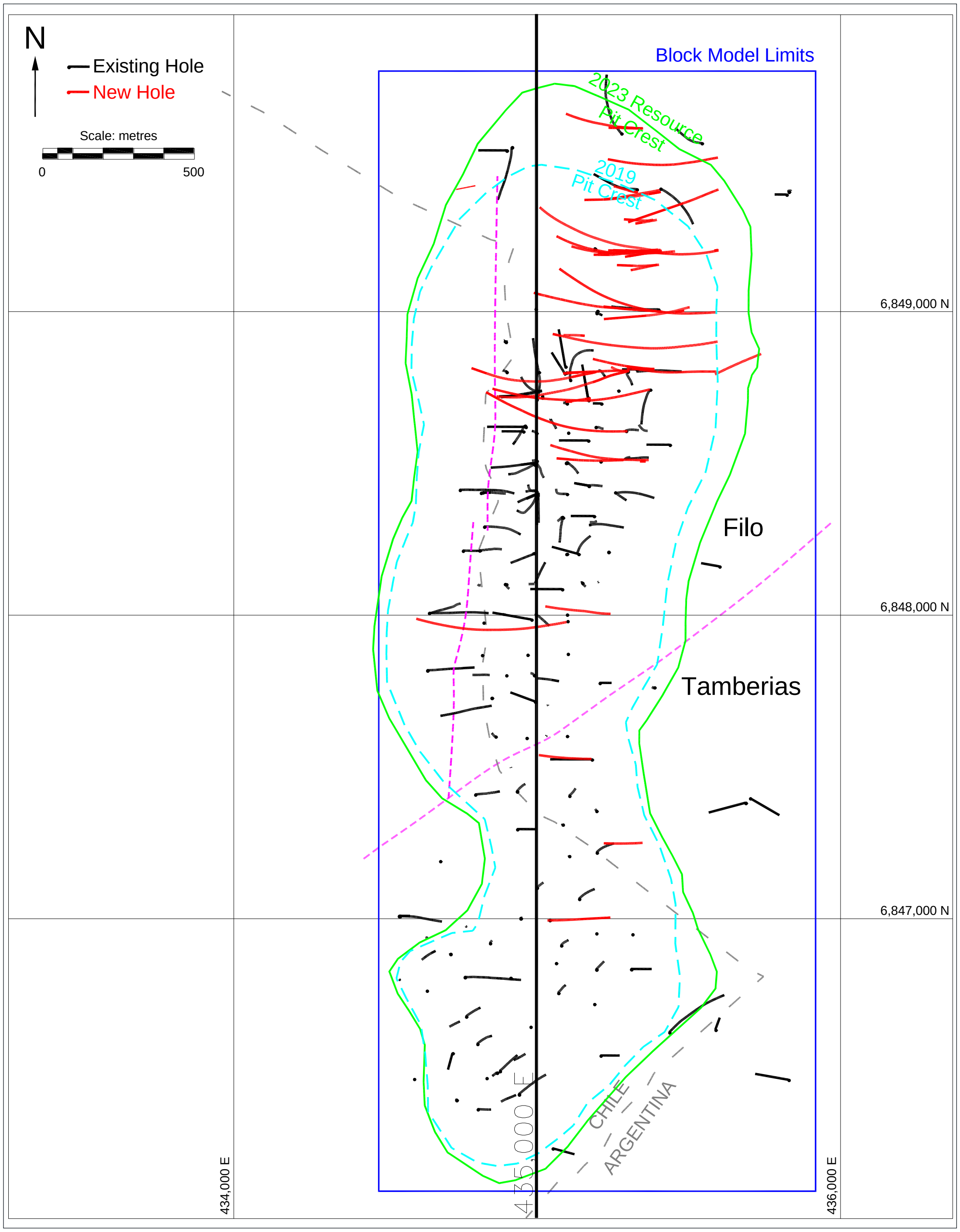
This Filo del Sol mineral resource update has an effective date of January 18, 2023. The update is based on a total of 61,800 metres of drilling in 247 holes including an additional 1,156 metres of reverse circulation drilling in 6 new holes and 18,725 metres of diamond drilling in 54 new holes from drilling completed in since the 2017/2018 field season. Figure 14-2 illustrates drill hole locations as well as: the block model outline, the crest of the resource pit and the national border; holes drilled since the 2019 resource estimate are shown in red. For comparative purposes, the limit of the 2019 resource pit is shown in light blue. The latest drilling has returned positive results at the north end of the resource area resulting in an extension of inferred blocks and the resource pit shell in that direction.

The block model setup is unchanged from that used for the 2019 resource estimate. The 15x15x12 m block size is deemed appropriate based on the anticipated production rate and drill spacing; block model configuration details are provided in Table 14-12.

Table 14-2: Block Model Setup

Block:	X	Y	Z
origin[(1)]	434,480	6,846,100	5,525
size (m)	15	15	12
no. blocks	96	246	79
no rotation; 1,865,664 blocks			
[(1)] SW model top, block edge			

Figure 14-2: Filo del Sol Exploration Drilling and Block Model Limits



Source: Advantage Geoservices, 2023

14.3 Geological Model

Resource estimation is controlled by a geologic model based on three-dimensional interpretation of drill results. The project area is divided into a north - Filo del Sol Area (FDS) and a south - Tamberías Area (TMB). FDS mineralization is bounded to the west by a sharp cut off in grade, possibly a fault. Units are consistent with those used for grade estimation in 2019.

Over the entire property, surfaces tied to drill intercepts have been generated to bound mineralized zones. These surfaces generally separate a leached cap from an oxide zone above the hypogene basement.

A zone of silver enrichment, has been outlined above the hypogene zone in the Filo Mining area. In the Tamberías area, a zone of silica alteration correlates with elevated gold grades.

14.4 Assay Grade Capping

Grade capping is used to control the impact of extreme, outlier high-grade samples on the overall resource estimate. For this estimate, assay intervals were back-tagged by the mineralization wireframes and grades examined in histograms and probability plots to determine levels at which values are deemed outliers to the general population. These cap values (Table 14-3) were applied by metal, by mineralized zone. Uncapped and capped composite statistics are presented in Table 14-4 to Table 14-6.

The impact of grade capping can be measured by comparing uncapped and capped estimated grades above a zero cutoff. Metal removed by capping is generally low reflecting the fact that relatively few composite grades included capped assay intervals (see composite statistics below). Estimated metal removed through capping amounts to 0.7% Cu, 3.2% Au and 10.5% Ag.

Table 14-3: Assay Grade Capping Levels

Min Zone			Cu	Au	Ag
			(%)	(g/t)	(g/t)
Filo	1	- Lix	1	7	40
	3	- Oxide	10	11	100
	4	- Hypogene	1.8	3	120
	11	- Ag Zone	*uncap*	3	2000
Tamberías	31	- Lix	0.6	1	10
	33	- Oxide	*uncap*	2	20
	34	- Hypogene	1.6	2	12
	23	- Silica Alt'n	0.4	2	*uncap*

14.5 Assay Compositing

Assays were composited to a target length of two metres within the bounds of the mineralization domain wireframes. The composite length was chosen because 95% of samples within the resource model volume were two metres in length.

Compositing to a constant length within geology units would result in the generation of shorter-length intervals at the down-hole edge of the solids; less than half-length (1.0 m in this case) samples would commonly be discarded prior to grade estimation. For this estimate, composite lengths across solid intersections were calculated such that they were equal, and as close to 2.0 m as possible. This technique resulted in composites averaging 2.0 m in length, ranging between 1.0 and 2.9 m, but includes all sampled material in the interpreted mineralization domains.

Approximately 31,000 Cu, Au and Ag composites were tagged with units of the geologic model and used for grade estimation. In cases where composite intervals spanned unsampled portions of holes, those missing intervals were assigned very low values of: 0.001% Cu, 0.001 g/t Au and 0.01 g/t Ag.

Table 14-4: 2m Composite Statistics – Copper

Min Zone			Count	Cu(%)			CuCap(%)			
				mean	max	CV	# Cap	mean	max	CV
FDS	1	- Lix	7,338	0.04	4.41	3.3	29	0.04	1.00	2.1
	3	- Oxide	9,890	0.33	16.41	1.8	17	0.33	10.00	1.7
	4	- Hypogene	4,854	0.38	3.25	0.7	12	0.38	1.80	0.7
	11	- Ag Zone	2,521	0.40	4.35	0.9	0	0.40	4.35	0.9
FDS Total:			*24,603*	*0.26*			*58*	*0.26*		
TMB	31	- Lix	491	0.10	1.43	1.6	15	0.09	0.60	1.3
	33	- Oxide	2,424	0.30	3.77	1.1	1	0.30	3.77	1.1
	34	- Hypogene	2,546	0.22	4.54	0.9	8	0.22	1.60	0.7
	23	- Silica Alt'n	798	0.05	0.88	1.7	13	0.05	0.40	1.5
TMB Total:			*6,259*	*0.22*			*37*	*0.22*		

Table 14-5: 2m Composite Statistics Gold

Min Zone			Count	Ag(g/t)			AgCap(g/t)			
				mean	max	CV	# Cap	mean	max	CV
FDS	1	- Lix	7,338	1.8	208.2	2.7	20	1.7	40.0	1.8
	3	- Oxide	9,890	4.4	2,410.5	8.6	44	3.2	177.2	2.6
	4	- Hypogene	4,854	5.7	443.9	2.7	24	5.5	120.0	2.2
	11	- Ag Zone	2,521	99.4	6,190.7	3.2	28	89.4	2,000.0	2.2
FDS Total:			*24,603*	*13.60*			*116*	*12.01*		
TMB	31	- Lix	491	1.9	31.0	1.4	11	1.7	10.0	1.0
	33	- Oxide	2,424	1.5	49.8	1.5	10	1.5	20.0	1.3
	34	- Hypogene	2,546	1.4	41.3	1.7	31	1.3	12.0	1.1
	23	- Silica Alt'n	798	4.6	43.2	1.2	2	4.6	43.2	1.2
TMB Total:			*6,259*	*1.88*			*54*	*1.82*		

Table 14-6: 2m Composite Statistics – Silver

Min Zone			Count	Au(%)			AuCap(%)			
				mean	max	CV	# Cap	mean	max	CV
FDS	1	- Lix	7,338	0.18	14.47	2.5	13	0.17	6.82	2.1
	3	- Oxide	9,934	0.26	16.46	1.8	9	0.26	10.90	1.7
	4	- Hypogene	4,854	0.32	8.94	1.0	18	0.32	3.00	0.8
	11	- Ag Zone	2,521	0.46	36.46	2.6	35	0.39	3.00	0.9
FDS Total:			*24,647*	*0.27*			*75*	*0.26*		
TMB	31	- Lix	491	0.23	2.21	0.8	6	0.23	1.00	0.7
	33	- Oxide	2,424	0.29	9.24	1.0	10	0.28	1.91	0.6
	34	- Hypogene	2,546	0.24	2.58	0.7	8	0.24	1.98	0.7
	23	- Silica Alt'n	798	0.34	3.98	0.8	3	0.34	2.00	0.7
TMB Total:			*6,259*	*0.27*			*27*	*0.27*		

14.6 Variography

Spatial continuity of capped composite data was analysed using Supervisor® software. Data were subdivided by modelled geologic zones, to establish suitable variogram model parameters for use in estimation by ordinary kriging. The variogram models used are listed in Table 14-7 to Table 14-9 for copper, gold and silver respectively.

Directions of continuity were determined from variogram maps. The nugget effect and sill contributions were derived from down-hole experimental variograms followed by final model fitting on directional variogram plots.

Table 14-7: Copper Variogram Models

FDS Domain	Axis	Direction (dip/azimuth)	Nugget Effect	Spherical Component 1 Sill	Spherical Component 1 Range(m)	Spherical Component 2 Sill	Spherical Component 2 Range(m)
1. Lix	X	00/355	0.11	0.44	60	0.45	375
	Y	00/26			105		250
	Z	90/000			10		130
3. Oxide (flattened)	X	43/355	0.20	0.55	120	0.25	280
	Y	43/205			20		175
	Z	15/100			15		120
4. Hypo	X	-10/092	0.09	0.14	15	0.77	155
	Y	-17/358			10		175
	Z	70/030			15		200
11. Ag Zone	X	00/100	0.13	0.38	60	0.49	240
	Y	70/190			30		150
	Z	-20/190			40		60
TMB Domain	**Axis**	**Direction (dip/azimuth)**	**Nugget Effect**	**Spherical Component 1 Sill**	**Range(m)**	**Spherical Component 2 Sill**	**Range(m)**
31. Lix	too few pts, use Filo						
33. Oxide (flattened)	X	00/165	0.28	0.43	45	0.29	100
	Y	-85/255			45		80
	Z	-05/075			45		70
34. Hypo	X	-05/330	0.23	0.29	130	0.48	195
	Y	-09/240			60		145
	Z	80/270			20		350
23. Sil. Alt'n	X	00/070	0.06	0.37	50	0.57	130
	Y	00/340			170		270
	Z	90/000			20		185

Table 14-8: Gold Variogram Models

FDS Domain	Axis	Direction (dip/azimuth)	Nugget Effect	Spherical Component 1		Spherical Component 2	
				Sill	Range(m)	Sill	Range(m)
1. Lix	X	21/008	0.10	0.52	75	0.38	190
	Y	38/261			50		160
	Z	45/120			20		100
3. Oxide	X	14/249	0.30	0.49	40	0.21	145
	Y	-05/161			95		250
	Z	75/090			25		130
4. Hypo	X	44/321	0.11	0.33	60	0.56	270
	Y	19/212			100		350
	Z	40/105			40		295
11. Ag Zone	X	00/040	0.15	0.41	50	0.44	250
	Y	00/310			50		340
	Z	90/000			15		170
TMB Domain	**Axis**	**Direction (dip/azimuth)**	**Nugget Effect**	**Spherical Component 1**		**Spherical Component 2**	
				Sill	**Range(m)**	**Sill**	**Range(m)**
31. Lix	too few pts, use Filo						
33. Oxide	X	00/185	0.24	0.34	35	0.42	165
	Y	00/275			15		230
	Z	-90/000			25		140
34. Hypo	X	00/240	0.15	0.17	10	0.68	220
	Y	00/330			10		250
	Z	-90/000			15		380
23. Sil. Alt'n	X	-14/356	0.24	0.36	20	0.40	145
	Y	-42/253			110		140
	Z	45/280			10		80

Table 14-9: Silver Variogram Models

FDS Domain	Axis	Direction (dip/azimuth)	Nugget Effect	Spherical Component 1 Sill	Spherical Component 1 Range(m)	Spherical Component 2 Sill	Spherical Component 2 Range(m)
1. Lix	X	00/350	0.31	0.36	70	0.33	285
	Y	00/260			40		140
	Z	90/000			10		100
3. Oxide	X	54/127	0.31	0.47	40	0.22	150
	Y	-08/206			25		115
	Z	-35/110			10		90
4. Hypo	X	62/167	0.26	0.43	40	0.31	150
	Y	-19/217			60		210
	Z	-20/120			25		70
11. Ag Zone	X	-02/024	0.06	0.58	30	0.36	80
	Y	30/296			30		50
	Z	60/110			15		65
TMB Domain	**Axis**	**Direction (dip/azimuth)**	**Nugget Effect**	**Spherical Component 1 Sill**	**Spherical Component 1 Range(m)**	**Spherical Component 2 Sill**	**Spherical Component 2 Range(m)**
31. Lix	too few pts, use Filo						
33. Oxide	X	-02/229	0.33	0.39	115	0.28	265
	Y	30/141			55		190
	Z	60/315			15		110
34. Hypo	X	00/020	0.39	0.35	85	0.26	185
	Y	00/110			20		130
	Z	-90/000			25		235
23. Sil. Alt'n	X	11/049	0.19	0.35	210	0.46	345
	Y	43/309			40		135
	Z	45/150			25		105

14.7 Grade Interpolation

Grades were estimated in a single pass by ordinary kriging. Blocks were estimated using a minimum of five samples, and maximum of 24 samples using a maximum of six samples per hole. Check models were estimated by inverse distance weighting and by nearest neighbour. Search orientations and distances are listed in Table 14-10. Search directions were chosen to best fit the orientation of the different mineralized zones.

To appropriately capture the slightly uneven geometry of the copper grade distribution in the oxide zone, copper in that zone was estimated using a transform coordinate system. Block and composite elevations were adjusted using the top of the oxide zone (bottom of Lix) as a datum. Search orientation for copper in the oxide zone was therefore unrotated. Block grades were relocated back to their real elevations after estimation for pit optimization and reporting.

Table 14-10: Estimation Search Parameters

Min Zone			Search Direction (dip/azimuth)			Axis Radii (m)		
			X	Y	Z	X	Y	Z
FDS	1	- Lix	0/8	9/278	-81/278	150	150	75
	3	- Oxide (Au, Ag)	0/341	2/251	-88/251	150	150	75
		- Oxide (Cu)	0/90	0/0	90/0	150	150	75
	4	- Hypogene	0/266	1/176	-89/176	150	150	75
	11	- Ag Zone	0/91	-11/1	79/1	150	150	75
TMB	31	- Lix	0/39	-20/309	70/309	150	150	75
	33	- Oxide (Au, Ag)	0/35	-20/305	70/305	150	150	75
		- Oxide (Cu)	0/90	0/0	90/0	150	150	75
	34	- Hypogene	0/30	-9/300	81/300	150	150	75
	23	- Silica Alt'n	0/172	14/82	-76/82	150	150	150

Contact plots of composites by interpolation domain were used to establish hard/soft boundary relationships for grade estimation. These boundary conditions are listed in Table 14-11.

Table 14-11: Grade Interpolation Contact Relationships

Min Zone			Match Codes on Estimation		
			Cu	Au	Ag
FDS	1	- Lix	1	1	1
	3	- Oxide	3	3,4	3,4
	4	- Hypogene	4	3,4,11	3,4
	11	- Ag Zone	11	4,11	11
TMB	31	- Lix	31	31	31
	33	- Oxide	33	33,34	33,34
	34	- Hypogene	34	33,34	33,34
	23	- Silica Alteration	23	23	23

14.8 Density Assignment

Density measurements continued with drill campaigns since the 2019 resource estimate resulting in more than a six-fold increase in the number of samples contributing to the averages presented in Table 14-12. Samples were coded with “MinZone” based on the geologic model and examined statistically. Sixty-seven samples, judged to have spurious values, were removed from the dataset used to derive the averages per geologic division. The mean values listed in Table 14-12 have been assigned to resource blocks based on their MinZone value and used for resource tonnage calculation.

Table 14-12: Average Bulk Density

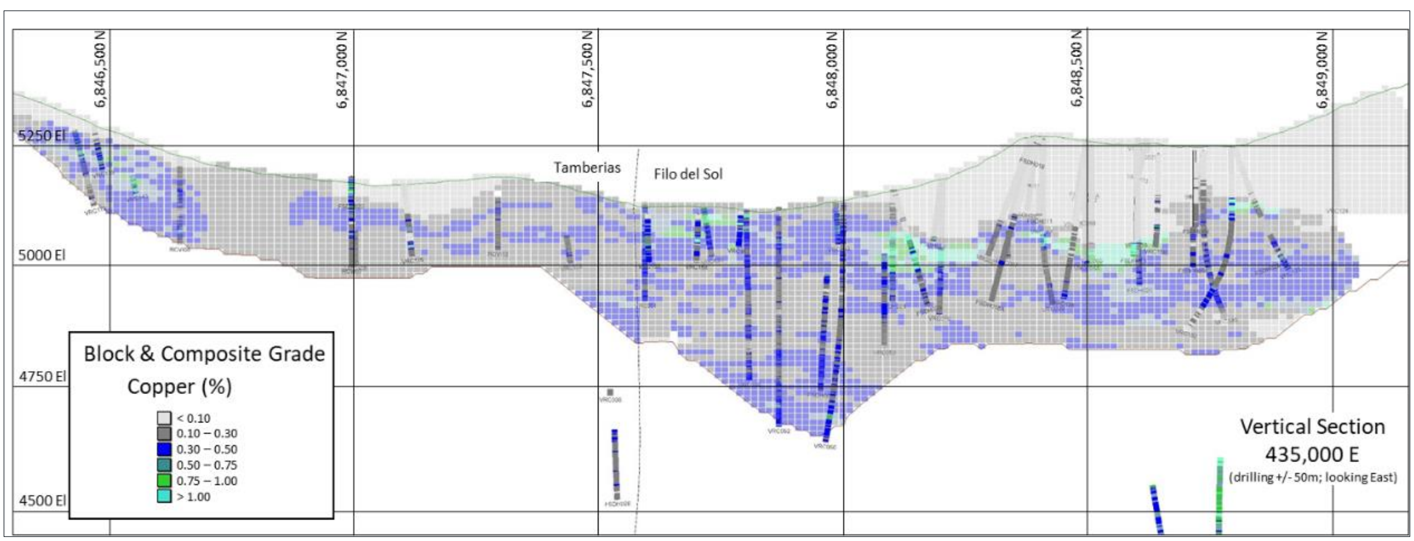
MinZone		Count	Bulk Density (t/m³)				Comment		
			Mean	Median	Min	Max			
1, 31	- Lix	937	**2.21**	2.23	1.60	2.80			
3, 33	- Oxide	1,585	**2.27**	2.25	1.61	3.04			
4, 34	- Hypogene	5,413	**2.45**	2.44	1.86	3.05			
11	- AgZone	464	**2.31**	2.31	1.81	2.81			
23	- Sil Alt'n	0	**2.40**				Av. non-Lix		
	Total:	8,399							

14.9 Model Validation

Estimated grades for all elements were validated visually by comparing composite to block values in plan view and on cross-sections. Example vertical sections comparing drill hole composites with block grades for the copper, gold and silver estimates are shown in Figure 14-3 to Source: Advantage Geoservices, 2023

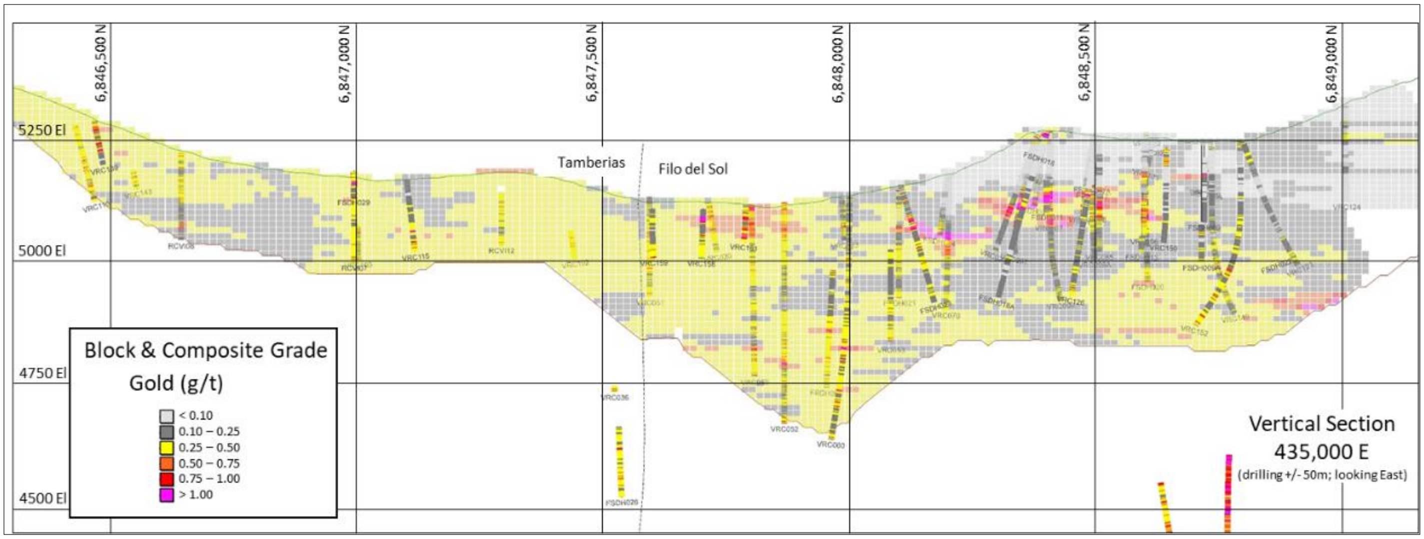
Figure 14-5 respectively. There is good visual correlation between composite and estimated block grades for all modelled elements. See Figure 14-2 for 435,000E Section location.

Figure 14-3: Copper Block and Composite Grades



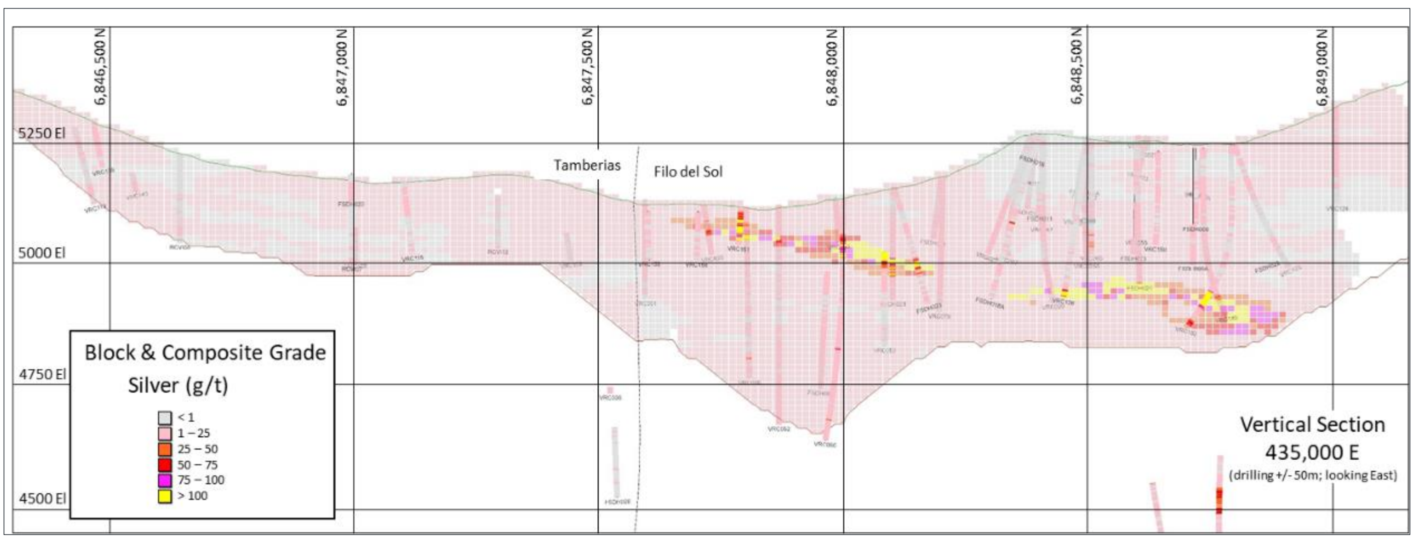
Source: Advantage Geoservices, 2023

Figure 14-4: Gold Block and Composite Grades



Source: Advantage Geoservices, 2023

Figure 14-5: Silver Block and Composite Grades



Source: Advantage Geoservices, 2023

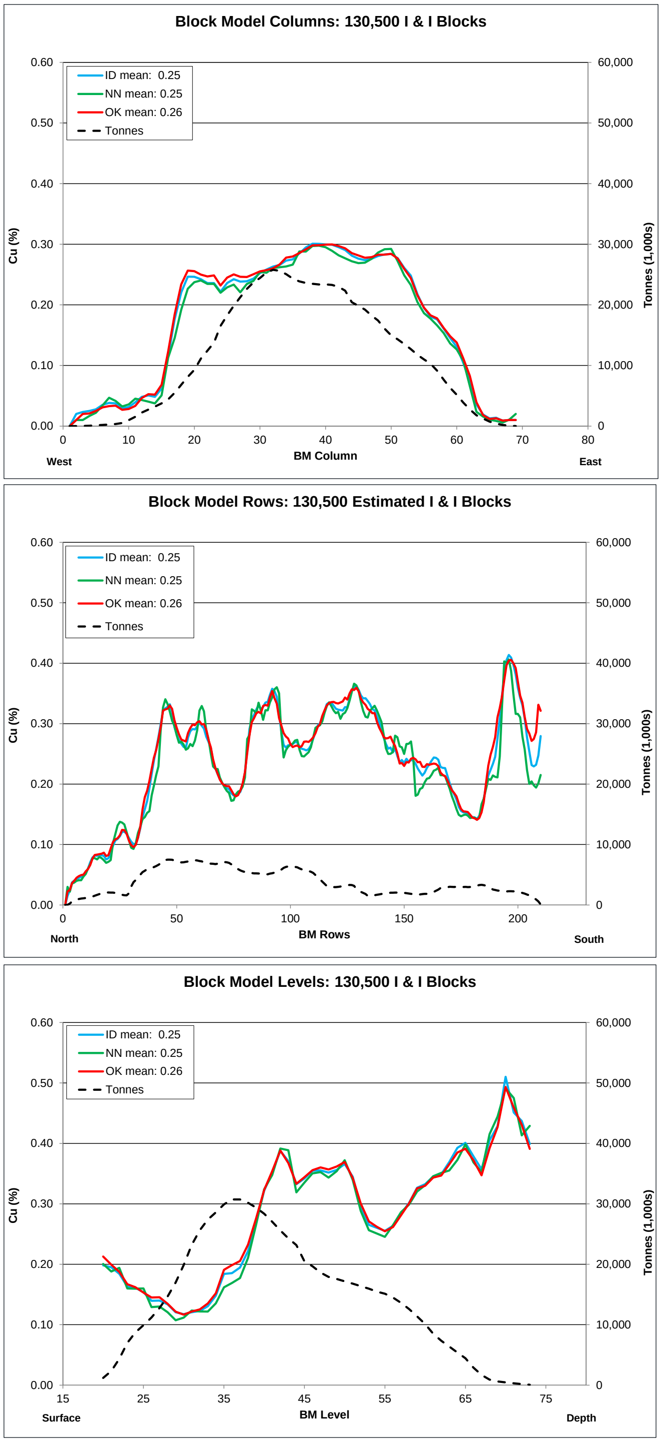
Nearest neighbour (NN) and inverse distance (ID) validation models were also estimated for all metals using parameters consistent with those used for ordinary kriging. Rather that reduce the block size to match composite length and then re-block, the NN model was estimated by a minimum of four and a maximum of six samples thereby approximately matching block size and sample length.

Copper, gold and silver estimates are compared spatially against NN and inverse distance estimates in swath plots. As an example, copper swath plots of indicated and inferred blocks are included in Figure 14-6. The OK estimates are appropriately smooth in comparison to the nearest neighbour models. Globally, model average grades above zero cutoff compare very closely indicating no bias; mean grades at zero cutoff are shown on the swath plots and listed by mineralized zone in Table 14-13.

Table 14-13: Check Models Grade Comparison

Min Zone			Block	Cu (%)			Au (g/t)			Ag (g/t)		
			Count	OK	NN	ID^2	OK	NN	ID^3	OK	NN	ID^2
FDS	1	- Lix	27,922	0.04	0.04	0.04	0.17	0.17	0.17	1.8	1.8	1.8
	3	- Oxide	40,923	0.36	0.34	0.35	0.29	0.29	0.29	3.8	3.6	3.6
	4	- Hypogene	18,558	0.30	0.31	0.31	0.33	0.33	0.33	3.5	3.3	3.3
	11	- Ag Zone	9,983	0.42	0.41	0.41	0.41	0.40	0.40	79.3	82.3	82.5
	FDS Total:		*97,386*	*0.26*	*0.26*	*0.26*	*0.28*	*0.27*	*0.27*	*10.9*	*11.1*	*11.1*
TMB	31	- Lix	3,883	0.11	0.12	0.11	0.27	0.26	0.26	1.6	1.7	1.6
	33	- Oxide	18,942	0.30	0.28	0.30	0.30	0.29	0.30	1.6	1.5	1.6
	34	- Hypogene	6,945	0.26	0.27	0.26	0.32	0.32	0.32	1.3	1.3	1.3
	23	- Silica Alt'n	6,010	0.05	0.05	0.05	0.36	0.36	0.36	3.8	3.8	3.9
	TMB Total:		*35,780*	*0.23*	*0.22*	*0.23*	*0.31*	*0.31*	*0.31*	*1.9*	*1.9*	*1.9*

Figure 14-6: Copper Grade Swath Plots Comparing OK, NN, and ID Estimates



Source: Advantage Geoservices, 2023

14.10 Resource Classification and Tabulation

The mineral resource is classified based on spatial parameters related to drill density and configuration and the generation of an optimized pit. An example section showing block classification is included in Figure 14-7.

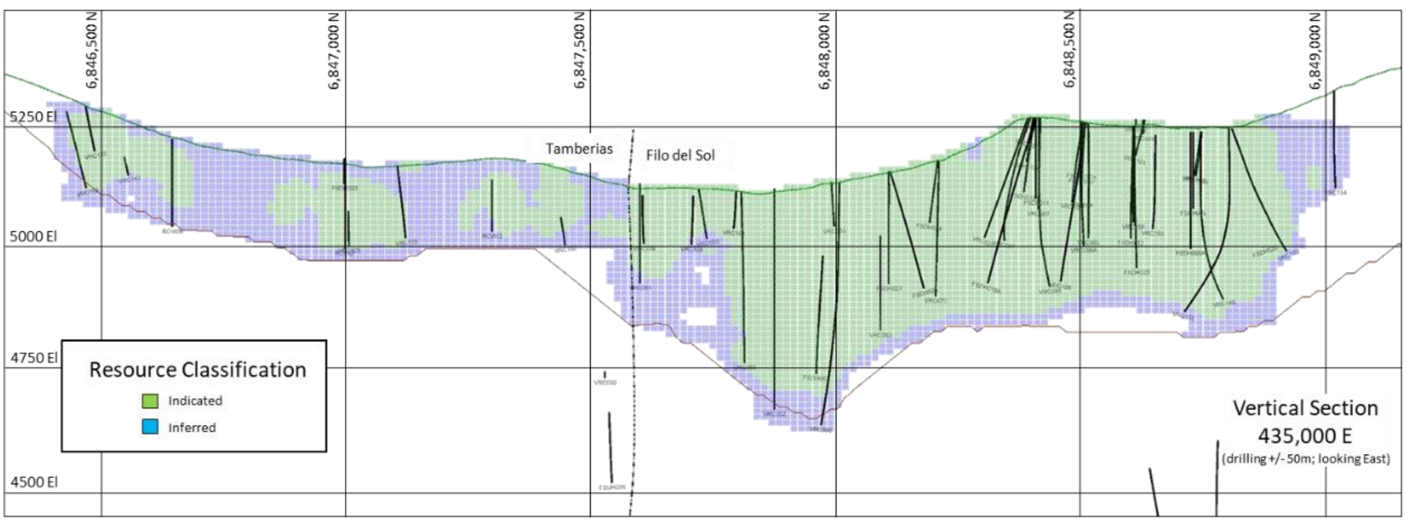
In order to ensure appropriate classification of contiguous blocks, blocks were classified inside a solid volume. As utilized for earlier resource estimates, that solid was generated such that blocks were initially classified as Inferred Mineral Resource where those blocks:

- Have sample data in at least three octants of a 150 m spherical search, and/or
- Are within 50 m of sample data

Blocks within the classification solid were assigned as Indicated Mineral Resource where those blocks were:

- Greater than 25 m inside the solid volume and estimated by ≥ 3 holes, and
- Were within 65 m of the closest hole or have samples in ≥5 octants of a 150 m spherical search

Figure 14-7: Mineral Resource Classification



Source: Advantage Geoservices, 2023

Measures were taken to ensure the resource meets the condition of "reasonable prospects of eventual economic extraction" as required under NI 43-101. An optimized pit shell was generated using Minesight® software's Lerchs–Grossmann (LG) optimizer and only blocks within the pit volume are included in the Mineral Resource. Pit optimization was carried out by AGP Mining Consultants using parameters listed in

Table 14-14. Metal recoveries and process costs were variable by mineralization type; averages are included in Table 14-16. The effective date of the mineral resource estimate is January 18, 2023.

Table 14-14: Pit Optimization Parameters

Metal	Metal Price	Av.Recovery
Cu	US$4.0/lb	78%
Au	US$1800/oz	70%
Ag	US$23/oz	84%
Mining Cost:	$2.72/t	
Av. Process Cost:	$9.86/t, including G & A	
Pit slope:	29° to 45°, by rock type/pit sector	

The Filo del Sol mineral resource estimate is tabled by mineralization type based on metallurgical testwork to date. Four mineralization types are anticipated; their correspondence to the MinZones used for estimation is listed in Table 14-15. Mineralization types were assigned per block and used in final resource tabulation.

Table 14-15: Mineralization Type Assignment

Min Zone				Min. Type
Filo	1	- Lix		AuOx
	3	- Oxide		CuAuOx
	4	- Hypogene		Hypogene
	11	- Ag Zone	≥ 20 g/t Ag	Ag
			< 20 g/t Ag	CuAuOx
Tamberias	31	- Lix		AuOx
	33	- Oxide		CuAuOx
	34	- Hypogene		Hypogene
	23	- Silica Alt'n		AuOx

Copper equivalence was calculated per block for tabulation of the copper gold oxide zone and the hypogene (sulphide) zone where there is anticipated revenue from multiple metals. Equivalence parameters, using average oxide and sulphide recoveries, are listed in Table 14-16.

Table 14-16: Copper Equivalence Parameters

Min. Type	Metal Prices (US$ per)			Recoveries (%)			Formula
	Cu (lb)	Au (oz)	Ag (oz)	Cu	Au	Ag	
CuAuOx (Oxide)	4	1800	23	77	72	71	Cu+Ag*0.0077+Au*0.6136
Hypogene (Sulphide)	4	1800	23	84	70	77	Cu+Ag*0.0077+Au*0.5469

Cutoff grades were chosen based on preliminary expected mining and processing costs per mineralization type. Cutoffs are specified with the updated mineral resource in Table 14-17. The four mineralization types are tabled at a range of cutoff grades in Table 14-18 to Table 14-21.

Table 14-17: Filo del Sol Mineral Resource by Mineralization Type

Min. Type	Cutoff	Category	Tonnes (millions)	Cu (%)	Au (g/t)	Ag (g/t)	lbs Cu (millions)	Ounces Au (thousands)	Ounces Ag (thousands)
AuOx	0.20 g/t Au	Indicated	54.4	0.06	0.40	3.0	72	705	5,250
		Inferred	24.0	0.10	0.31	2.1	52	241	1,640
CuAuOx	0.15% CuEq	Indicated	265.0	0.37	0.30	3.5	2,179	2,558	29,750
		Inferred	97.3	0.27	0.28	2.8	588	889	8,670
Ag	20 g/t Ag	Indicated	42.8	0.46	0.42	87.1	432	576	119,670
		Inferred	11.4	0.34	0.42	87.5	85	154	32,060
Hypo	0.30% CuEq	Indicated	70.4	0.31	0.35	2.5	473	790	5,710
		Inferred	78.9	0.31	0.33	3.1	542	834	7,960
Total		Indicated	432.6	0.33	0.33	11.5	3,156	4,629	160,380
		Inferred	211.6	0.27	0.31	7.4	1,267	2,118	50,330

Table 14-18: Gold Oxide Zone by Gold Cutoff

Min. Type	Cutoff	Category	Tonnes (millions)	Cu (%)	Au (g/t)	Ag (g/t)	lbs Cu (millions)	Ounces Au (thousands)	Ounces Ag (thousands)
AuOx	***0.20 g/t Au***	***Indicated***	***54.4***	***0.06***	***0.40***	***3.0***	***72***	***705***	***5,250***
		Inferred	***24.0***	***0.10***	***0.31***	***2.1***	***52***	***241***	***1,640***
	0.40 g/t Au	Indicated	22.0	0.06	0.58	3.3	26	411	2,360
		Inferred	4.1	0.09	0.49	2.9	8	65	390
	0.50 g/t Au	Indicated	12.4	0.05	0.69	3.2	14	274	1,250
		Inferred	1.4	0.08	0.58	3.5	3	27	160

Table 14-19: Copper Gold Oxide Zone by Copper Equivalent Cutoff

Min. Type	Cutoff	Category	Tonnes (millions)	Cu (%)	Au (g/t)	Ag (g/t)	lbs Cu (millions)	Ounces Au (thousands)	Ounces Ag (thousands)
CuAuOx	***0.15% CuEq***	***Indicated***	***265.0***	***0.37***	***0.30***	***3.5***	***2,179***	***2,558***	***29,750***
		Inferred	***97.3***	***0.27***	***0.28***	***2.8***	***588***	***889***	***8,670***
	0.30% CuEq	Indicated	248.9	0.39	0.31	3.6	2,127	2,474	29,020
		Inferred	79.5	0.31	0.31	2.9	540	792	7,330
	0.50% CuEq	Indicated	136.5	0.51	0.36	4.8	1,520	1,599	21,130
		Inferred	33.0	0.43	0.38	3.7	313	402	3,880
	0.70% CuEq	Indicated	57.6	0.71	0.43	6.1	901	790	11,320
		Inferred	10.4	0.61	0.46	4.6	140	156	1,540

Table 14-20: Silver Zone by Silver Cutoff

Min. Type	Cutoff	Category	Tonnes (millions)	Cu (%)	Au (g/t)	Ag (g/t)	lbs Cu (millions)	Ounces Au (thousands)	Ounces Ag (thousands)
Ag	***20 g/t Ag***	***Indicated***	***42.8***	***0.46***	***0.42***	***87.1***	***432***	***576***	***119,670***
		Inferred	***11.4***	***0.34***	***0.42***	***87.5***	***85***	***154***	***32,060***
	50 g/t Ag	Indicated	29.0	0.47	0.43	111.7	301	400	104,030
		Inferred	7.5	0.35	0.43	114.9	57	104	27,690
	60 g/t Ag	Indicated	24.8	0.48	0.43	121.2	263	346	96,690
		Inferred	6.4	0.36	0.43	125.3	50	89	25,700
	80 g/t Ag	Indicated	18.7	0.49	0.43	138.0	204	262	83,150
		Inferred	4.8	0.36	0.44	144.3	38	67	22,080

Table 14-21: Hypogene Zone by Copper Equivalent Cutoff

Min. Type	Cutoff	Category	Tonnes (millions)	Cu (%)	Au (g/t)	Ag (g/t)	lbs Cu (millions)	Ounces Au (thousands)	Ounces Ag (thousands)
Hypogene	***0.30% CuEq***	***Indicated***	***70.4***	***0.31***	***0.35***	***2.5***	***473***	***790***	***5,710***
		Inferred	***78.9***	***0.31***	***0.33***	***3.1***	***542***	***834***	***7,960***
	0.40% CuEq	Indicated	54.9	0.34	0.37	2.8	405	659	4,940
		Inferred	66.3	0.33	0.34	3.4	487	727	7,140
	0.50% CuEq	Indicated	31.7	0.38	0.42	3.5	268	430	3,520
		Inferred	35.8	0.39	0.37	4.6	306	424	5,320
	0.60% CuEq	Indicated	13.4	0.46	0.51	4.8	135	220	2,080
		Inferred	16.0	0.45	0.40	7.2	158	207	3,690

Table 14-22 compares this updated resource estimate to that completed in 2019. The impact of the additional drilling was modest in terms of change to indicated resource numbers. Drilling in the north of the deposit area resulted in the extension of inferred mineral resource. As illustrated in Figure 14-2, the revised pit shell extends approximately 250m north, beyond the 2019 pit limits.

Table 14-22: Comparison to Previous Mineral Resource

Mineral Resource	Category	Tonnes (millions)	Cu (%)	Au (g/t)	Ag (g/t)	lbs Cu (millions)	Ounces Au (thousands)	Ounces Ag (thousands)
2023	Indicated	432.6	0.33	0.33	11.5	3,156	4,629	160,380
	Inferred	211.6	0.27	0.31	7.4	1,267	2,118	50,330
2018	Indicated	425.1	0.33	0.32	10.7	3,107	4,436	146,738
	Inferred	175.1	0.27	0.33	6.2	1,054	1,834	34,811
Difference	Indicated	+1.8%	0.0%	+3.1%	+7.5%	+1.6%	+4.4%	+9.3%
	Inferred	+20.8%	0.0%	-6.1%	+19.4%	+20.2%	+15.5%	+44.6%

15 MINERAL RESERVE ESTIMATE

The Filo del Sol deposit is a large near surface, bulk mineable deposit that is well suited for extraction by conventional open pit methods. The Mineral Reserves for Filo del Sol, and with an effective date of February 28, 2023, were based on the project Mineral Resource estimate with an effective date of January 18, 2023, which is discussed in Section 14. The work was performed using metal prices of Cu $3.50/lb, Ag $20/oz, Au $1600/oz. Only Measured and Indicated Mineral Resources were considered for processing. Inferred Mineral Resources were treated as waste.

This section describes the economic and technical parameters used, including, geotechnical considerations, dilution and mining loss adjustments, Net Value per Tonne (NVPT) and cutoff application, Lerchs Grossmann nested pit shells, the Ultimate Pit Design that contains the reserves, and the Mineral Reserves statement.

15.1 Geotechnical Considerations

A preliminary pit slope geotechnical assessment was performed by BGC Engineering, Inc. in 2018. Based on geotechnical mapping, core logging, and geotechnical laboratory test results, slope design recommendations were provided for the identified structural domains. Overall pit slopes varied from 29 to 45 degrees, inclusive of geotechnical berms and ramp allowances. A detailed discussion of pit slope design parameters is provided in Section 16.1.

15.2 Dilution and Mining Loss Adjustments

The 15 x 15 x 12 block size used in the resource model is a good match to the Selective Mining Unit for the envisioned mining method. The mineralization is generally gradational across the ore/waste contacts, except for limited areas where a fault delineates a hard boundary between mineralized material on one side and barren material on the other side.

Based on this gradational nature of the mineralization near the ore/waste contacts, dilution and mining loss adjustments were applied using a mixing zone approach, where the volumes of dilution gain and ore loss would ‘wash out’, resulting in diluted grades lower than the in-situ resource grades, but tonnage remaining the same. A mixing zone extending three metres on a vertical block edge was chosen considering anticipated blast pattern dimensions, ore control methods, blast heave mixing/movement, and high precision GPS guided digging accuracy. The diluted grades were calculated on a tonnage weighted basis with inferred materials being treated as barren. The resulting average reductions in grades from the in-situ resource grades are 1.0%, 1.3% and 1.0% for Cu, Au and Ag respectively.

15.3 Net Value Per Tonne Calculations

Revenue will be generated from the sale of copper cathode resulting from the acid leaching of copper, and gold/silver doré from cyanide leaching. To assess the value of material with three payable metals, recoveries that vary with grade and rock type, and variable process costs by rock type, NVPT estimates were performed at the block level via a script and verified with spreadsheet calculations. The inputs to the NVPT estimates are as follows:

15.3.1 Rock Type Independent Parameters

The metal prices and selling costs used for mine planning are shown in Table 15-1 below.

Table 15-1: Metal Prices and Selling Costs

	Copper $/lb	Gold $/oz	Silver $/oz
Price	3.50	1600.00	20.00
Selling Cost	0.20	0.50	0.50
Net Price	3.30	1599.50	19.50

As of the effective date of the Mineral Reserves (Feb 28, 2023), the metal prices used are all lower than current spot prices ($3.58/lb Cu (LME), $1737/oz Au (COMEX) and $20.85/oz Ag (COMEX)), and lower than the three-year trailing averages ($3.63/lb Cu (LME), $1779/oz Au (COMEX) and $22.31/oz Ag (COMEX)).

The selling costs were based on values used in the 2019 PFS study for Filo and are different from those developed later and presented in Section 22.

A 3% San Juan province 'mine head' royalty was applied to the revenue, net of process and G&A operating costs, for the blocks in Argentina.

15.3.2 Metallurgical Recoveries

Metallurgical recoveries were provided by Ausenco and are discussed in Section 13. The metallurgical domains are the same as the resource estimation domains as discussed in Section 14. The recoveries are a mix of formulas and fixed values by domain, as shown in Table 15-2.

Table 15-2: Metallurgical Recoveries Used for Mine Planning

Domain	Min Zone	Recoveries (%)		
		Au	Ag	Cu
FDS-AuOx	1	78	17	Formula A
FDS-CuAuOx	3	78	Formula C*	Formula A
FDS-M-Ag	11	65	Formula C*	Formula A
TMB-AuOx	23 & 31	50	22	Formula B
TMB-CuAuOx	33	60	42	Formula B
Formula A:	If CuCN%<=15; Ext = CuAS% + 0.45 * CuCN%			
	If 15% <= CuCN% < 25%; Ext = CuAS% + 0.30 * CuCN%			
	If 25% < =CuCN% < 45%; Ext = CuAS% + 0.20 * CuCN%			
	If 45% <= CuCN%; Ext = CuAS% + 0.10 * CuCN%			
Formula B:	Ext = 0.95 * CuAS% + 0.45 * CuCN%			
Formula C*:	Ext = 0.96*(35 * ln(Head Ag g/t) + 30): set minimum = 6%, maximum = 90%.			

Note: CuCN% is the percent cyanide soluble Cu grade divided by the sum of the sequential copper grades. CuAS% is the percent acid soluble Cu grade divided by the sum of the sequential copper grades.

The above recoveries differ from those used in the financial analysis, which are LOM averages of 80% for Cu, 70% for Au and 82% for Ag. Post mine planning, operational efficiency adjustments were introduced, which reduced the recoveries applied in the financial model. AGP is of the opinion that if the mine planning recoveries were similarly reduced, it would have a non-material effect on the pit shapes and quantity of material above cutoff.

15.3.2 Operating Costs

The process costs applied for the 60,000 t/d throughput rate, were constant across domain groups, as shown in Table 15-3.

Table 15-3: Process Operating Costs

Domains	Min Zone	Process Cost ($/t)
FDS Domains	1, 3, & 11	9.65
TMB Domains	23, 31, & 33	9.65

The G&A cost used was $1.46/t processed. The average mining cost was $2.72/t mined, which is the result of cost escalators applied for material above and below a reference pit-exit bench.

The above costs differ from those presented in Sections 21 and 22, due to refinement that occurred after the mine planning started. In all cases, the final operating costs were lower than those used for mine planning.

15.3.3 Cutoffs

Pit-limits analysis was performed at a marginal breakeven cutoff of $0.01/t net block value. The mine schedule however employed an elevated cutoff as a results of leach pad constraints which meant that not all ore present in the chosen pit design could be processed. The cutoff chosen was $4.50/t net block value, which includes process and G&A costs. Mining costs are not included in the cutoff analysis. Approximately 22.5 Mt or ore between the marginal cutoff of $0.01/t net block value, and $4.5/t block value was wasted, grading 0.086% Cu, 0.218gpt Au, and 2.5gpt Ag.

15.4 Pit Shell Optimization

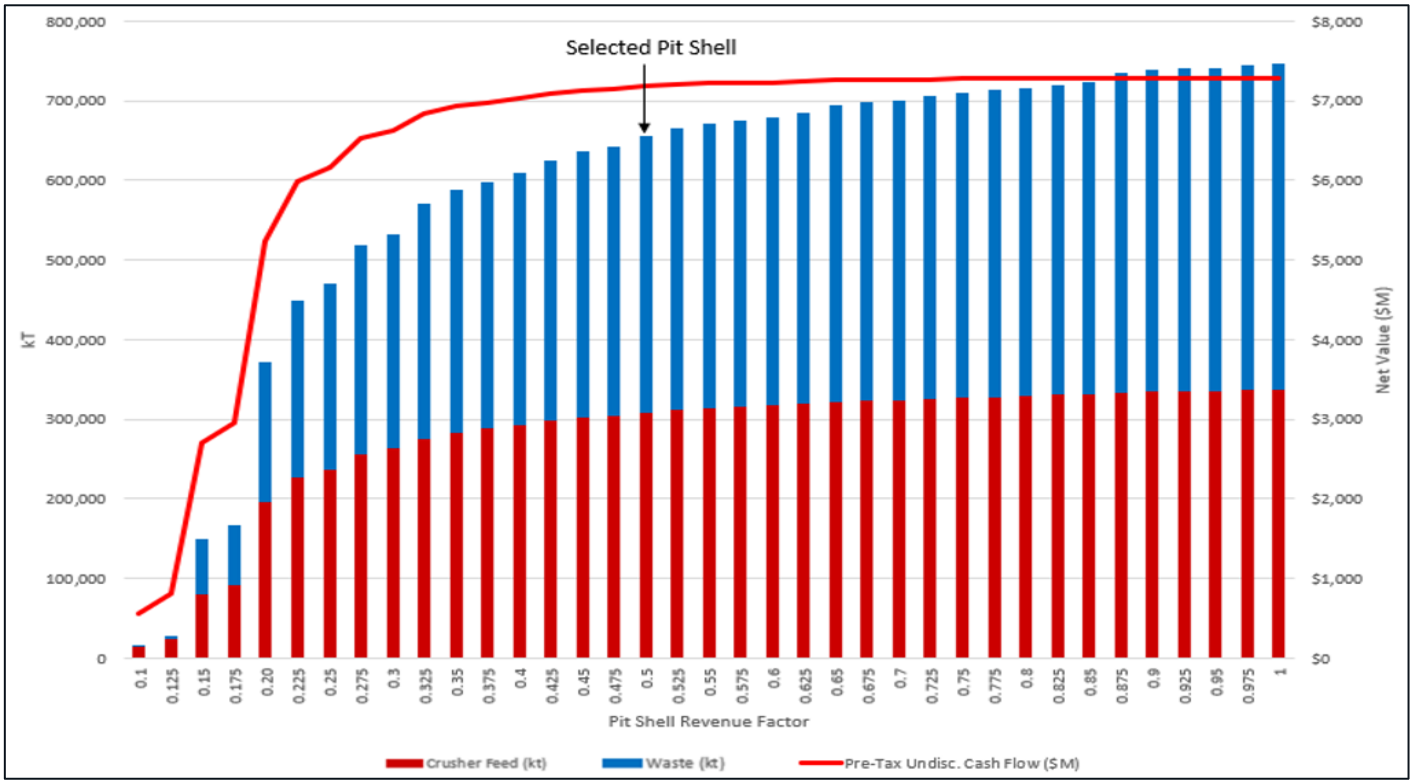
The ultimate pit design and internal pit phases were guided by Lerchs-Grossman (LG) optimized pit shells generated using the Hexagon Mining's MinePlan^{TM}, (formerly known as MineSight) mine planning software package and the technical and cost parameters described above. A series of 'Revenue Factor' nested shells were generated by multiplying the block gross revenue by the unitless revenue factor that was varied from 0.1 to 1.0 by 0.025 increments. The volumetric results of the set of nested shells are shown in Table 15-4 and below.

Table 15-4: Nested LG Pit Shell Volumetrics

Revenue	Crusher Feed	Grades				Waste	Total Material	SR	Pre-Capex Undiscounted Cash Flow
Factor	kt	NVPT ($/t)	Cu (%)	Au (g/t)	Ag (g/t)	kt	kt	(W:O)	$M
RF_0.10	15,109	40.0	0.45	0.41	14.8	1,191	16,300	0.08	561
RF_0.125	23,767	37.3	0.43	0.39	13.6	4,532	28,299	0.19	812
RF_0.15	80,814	38.4	0.43	0.35	18.6	68,844	149,658	0.85	2,711
RF_0.175	90,865	37.3	0.43	0.34	17.7	76,521	167,387	0.84	2,952
RF_0.20	196,371	31.7	0.37	0.33	16.0	176,154	372,524	0.90	5,228
RF_0.225	227,145	31.6	0.37	0.33	15.6	221,639	448,784	0.98	5,985
RF_0.25	236,176	31.4	0.37	0.33	15.5	233,379	469,555	0.99	6,164

Revenue	Crusher Feed	Grades				Waste	Total Material	SR	Pre-Capex Undiscounted Cash Flow
RF_0.275	256,354	30.8	0.37	0.33	15.1	261,955	518,309	1.02	6,528
RF_0.30	262,955	30.5	0.37	0.33	14.9	269,141	532,096	1.02	6,618
RF_0.325	275,889	30.3	0.37	0.33	14.9	295,095	570,985	1.07	6,843
RF_0.35	283,888	29.9	0.36	0.33	14.7	303,541	587,430	1.07	6,929
RF_0.375	288,806	29.6	0.36	0.33	14.5	308,378	597,184	1.07	6,974
RF_0.40	293,505	29.5	0.36	0.33	14.4	316,088	609,593	1.08	7,027
RF_0.425	298,420	29.3	0.36	0.33	14.4	327,164	625,585	1.10	7,087
RF_0.45	301,520	29.2	0.36	0.33	14.4	335,056	636,576	1.11	7,125
RF_0.475	304,484	29.1	0.36	0.33	14.3	337,478	641,962	1.11	7,142
RF_0.50	307,329	29.0	0.36	0.33	14.3	347,978	655,307	1.13	7,179
RF_0.525	311,387	28.8	0.36	0.33	14.2	354,973	666,361	1.14	7,206
RF_0.55	313,446	28.7	0.36	0.33	14.2	357,260	670,706	1.14	7,216
RF_0.575	315,768	28.6	0.36	0.33	14.1	360,143	675,911	1.14	7,227
RF_0.60	317,106	28.5	0.35	0.33	14.0	361,992	679,099	1.14	7,233
RF_0.625	319,194	28.4	0.35	0.33	14.0	366,657	685,851	1.15	7,244
RF_0.65	321,636	28.3	0.35	0.33	13.9	372,692	694,328	1.16	7,257
RF_0.675	322,930	28.2	0.35	0.33	13.9	374,727	697,656	1.16	7,261
RF_0.70	324,426	28.1	0.35	0.33	13.8	376,563	700,988	1.16	7,265
RF_0.725	325,776	28.1	0.35	0.33	13.8	379,820	705,596	1.17	7,270
RF_0.75	326,570	28.1	0.35	0.33	13.8	383,931	710,501	1.18	7,275
RF_0.775	328,322	27.9	0.35	0.33	13.8	385,791	714,112	1.18	7,278
RF_0.80	328,927	27.9	0.35	0.33	13.8	387,043	715,971	1.18	7,279
RF_0.825	330,442	27.8	0.35	0.33	13.7	389,186	719,628	1.18	7,282
RF_0.85	332,050	27.7	0.35	0.33	13.6	391,848	723,898	1.18	7,284
RF_0.875	333,589	27.7	0.35	0.33	13.7	402,148	735,736	1.21	7,289
RF_0.90	334,419	27.7	0.35	0.33	13.7	404,338	738,757	1.21	7,290
RF_0.925	335,209	27.6	0.35	0.33	13.6	405,394	740,603	1.21	7,290
RF_0.95	335,527	27.6	0.35	0.33	13.6	405,890	741,417	1.21	7,291
RF_0.975	336,885	27.5	0.35	0.33	13.6	408,139	745,024	1.21	7,291
RF_1.00	337,470	27.5	0.35	0.33	13.6	409,041	746,511	1.21	7,291

Figure 15-1: Nested LG Pit Shell 'Pit by Pit' Graph



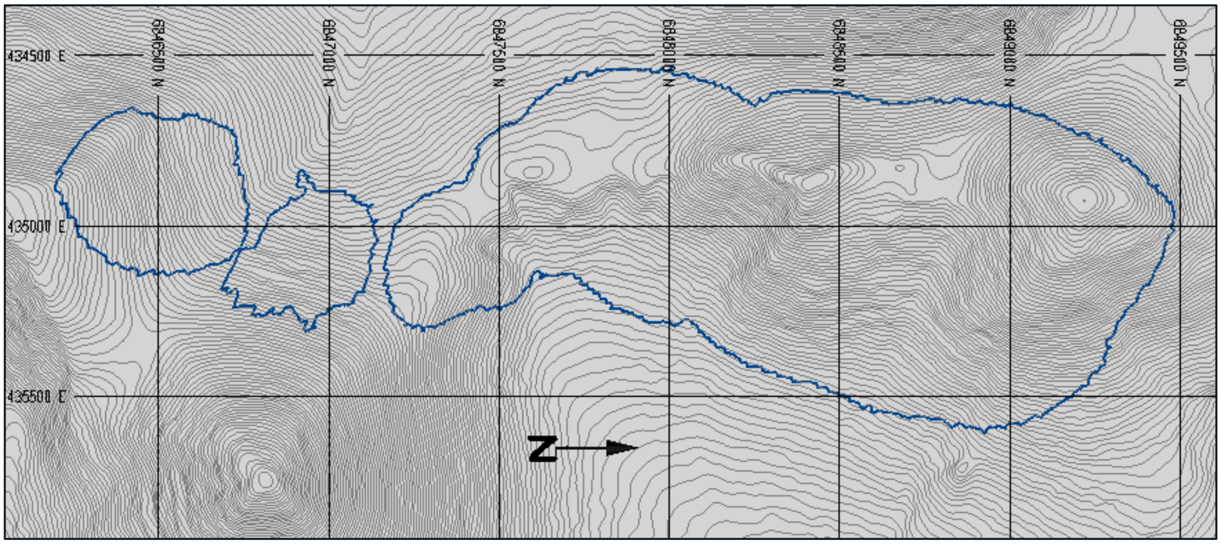
Source: AGP, 2023

The Revenue Factor (RF) 0. 50 LG shell was selected to guide the ultimate pit design. This shell was selected based on the following considerations:

- The requirement for ore tonnes to be capped at 260 Mt to match the available capacity for a single cyanide leach pad
- The desire to maintain the pit phase and ultimate pit designs employed in the 2019 PFS
- As a result of the higher metal price assumptions and additional drilling employed since the 2019 PFS was completed, a larger reserve pit could be employed if desired.

The outline of the RF 0.50 LG shell outline is shown with 5m topography contours is shown in Figure 16-2.

Figure 15-2: RF 0.775 LG Shell Outline

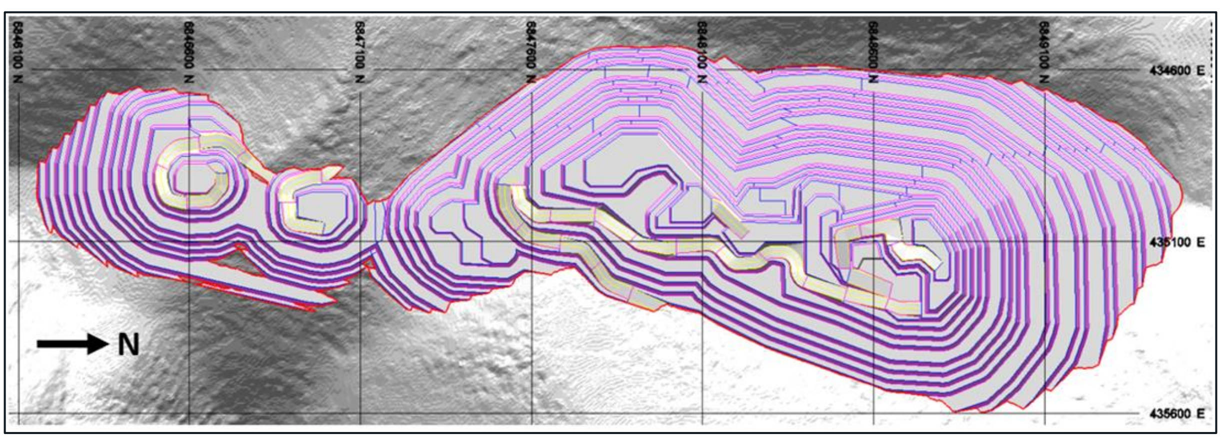


Source: AGP, 2023

15.5 Ultimate Pit Design

The open pit has been designed for large-scale truck-and-shovel operations. There are two pit areas: the larger multi-phase Filo pit to the north and the smaller single phase Tamberías pit to the south. Multiple phases are required to release ore in a timely manner and to smooth out stripping requirements on an annual basis. The overall dimensions of the ultimate pit are approximately 3,400 m in the north-south direction, 1,000 m in the east-west direction and 468m maximum depth at the north end of the Filo pit. Haulage roads are designed at 33.7 m with a maximum 10% uphill loaded grade and 8% downhill loaded grade. A minimum mining width of 60m was used. Additional design criteria are summarized in Section 16.1.2. The ultimate pit is shown in Table 15-3.

Figure 15-3: Ultimate Pit Design



Source: AGP, 2023

15.6 Mineral Reserves Statement

Mineral Reserves have been modified from Mineral Resources by taking into account mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors and are therefore classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves.

The Mineral Reserves were prepared under the supervision of Gordon Zurowski, P.Eng. of AGP Mining Consultants Inc. who is a QP as defined under NI 43-101. The Mineral Reserve has an effective date of February 28, 2023. The Mineral Resources are inclusive of Mineral Reserves.

Table 15-5: Filo del Sol Mineral Reserve Estimate @$0.01/t NVPT Cutoff (Effective February 28, 2023)

Category (All Domains)	Tonnage (Mt)	Grade				Contained Metal		
		Cu (%)	Au (g/t)	Ag (g/t)	NVPT ($/t)	Cu (Mlb)	Au (koz)	Ag (koz)
Proven	-	-	-	-	-	-	-	-
Probable	259.6	0.39	0.34	16.0	32.5	2,220	2,867	133,334
Total Proven and Probable	259.6	0.39	0.34	16.0	32.5	2,220	2,867	133,334

Notes: **1.**The qualified person for the estimate is Mr. Gordon Zurowski, P.Eng. of AGP Mining Consultants, Inc. **2.** The mineral reserves were estimated in accordance with the CIM Definition Standards for Mineral Resources and Reserves. **3.** The mineral reserves are supported by a mine plan, based on a pit design, guided by a Lerchs-Grossmann (LG) pit shell. Inputs to that process are metal prices of Cu $3.50/lb, Ag $20/oz, Au $1600/oz; mining cost average of $2.72/t; an average processing cost of $9.65/t; general and administration cost of $1.46/t processed; pit slope angles varying from 29 to 45 degrees, inclusive of geotechnical berms and ramp allowances; process recoveries were based on rock type. The average recoveries applied were 83% for Cu, 73% for Au and 80% for Ag, which exclude the adjustments for operational efficiency and copper recovered as precipitate which were included in the financial evaluation. **4.** Dilution and mining loss adjustments were applied at ore/waste contacts using a mixing zone approach. The volumes of dilution gain and ore loss were equal, resulting reductions in grades of 1.0%, 1.3% and 1.0% for Cu, Au and Ag, respectively. **5.** Ore/waste delineation was based on a net value per tonne (NVPT) cutoff of $4.5/t considering metal prices, recoveries, royalties, process and G&A costs as per LG shell parameters stated above, elevated above break-even cutoff to satisfy processing capacity constraints. **6.** The life-of-mine stripping ratio in tonnes is 1.57:1. **7.** All figures are rounded to reflect the relative accuracy of the estimate. Totals may not sum due to rounding as required by reporting guidelines.

15.7 Factors that May Affect the Mineral Reserves Estimate

Factors that may affect the Mineral Reserves estimate include dilution; metal prices; metallurgical recoveries and geotechnical characteristics of the rock mass; capital and operating cost estimates; and effectiveness of surface and groundwater management.

The QPs are of the opinion that these potential modifying factors have been adequately accounted for using the assumptions in this report, and therefore the Mineral Resources within the mine plan may be converted to Mineral Reserves.

16 MINING METHODS

The Filo del Sol deposit is a large, near surface, bulk mineable deposit that is well suited for extraction by conventional open pit methods. Ore and waste will be drilled, blasted and loaded by diesel hydraulic face shovels and front-end loaders from 12-metre benches. Haul trucks will haul the material to the ore crusher, a short-term stockpile, or the waste dump as required. Based on the results of a throughput trade-off study, the mine plan is based on a nominal 60,000 t/d processing rate. The peak mining capacity is 68 million tonnes per annum.

This section describes the pit phase design, waste dump design, mining schedule, equipment selection, and other operational considerations.

16.1 Pit Design

There are two pit areas: the larger multi-phase Filo pit to the north and the smaller single phase Tamberías pit to the south. Multiple phases are required in the Filo pit to release ore in a timely manner and to smooth out stripping requirements on an annual basis.

16.1.1 Slope Design Angles

Geotechnical open pit slope design criteria were developed by BGC Engineering Inc. (BGC) for the current study. The basis of the slope design parameters are as follows:

- The geological model for the mine developed by Filo Mining Corp. (Filo Mining)
- Geotechnical unit model developed by BGC
- A structural geology model developed by BGC
- Slope stability assessments completed by BGC
- The rock mass and structural geology models for the open pit slope designs are based on the following data:
- Geotechnical core logging from one geotechnical drill hole conducted by BGC
- Geotechnical core logging from seven exploration holes conducted by Filo Mining staff trained and supervised by BGC
- Surface geotechnical and structural geological mapping conducted by BGC in 2018
- Surface structural geological mapping conducted by Devine (2016) and provided by Filo Mining
- Laboratory tests of uniaxial compressive strength (8), indirect tensile strength (16), direct shear strength (8) of rock core samples and natural discontinuities sampled from the drill core and tested by BGC
- Point load index tests (152) and Leeb hardness tests (171) conducted by Filo Mining personnel from the geotechnical and exploration drill holes
- Specific gravity tests (527 tests) provided by Filo Mining

Groundwater was not encountered in any of the drillholes, and it is assumed for this study that the phreatic surface is below the bottom of the pit.

BGC used the project scale three-dimensional model of major faults and mineralogical zones, together with rock mass data collected from geotechnical logging, to divide the open pit into structural domains and geotechnical units.

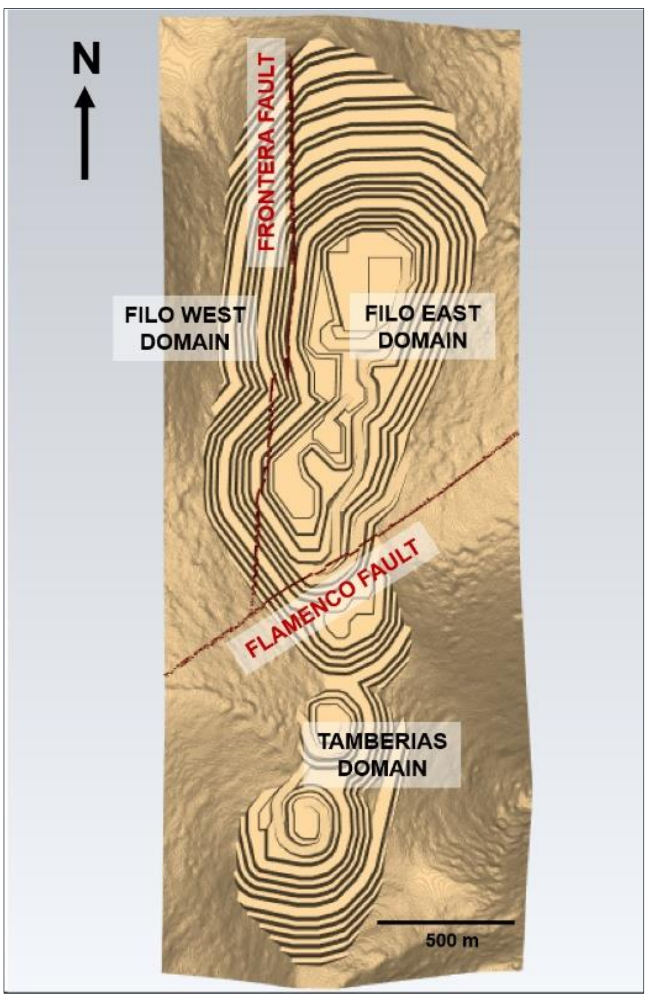
Three structural domains, as shown in Figure 16-1, were identified as follows:

1. Filo East Domain. This domain comprises the main Filo deposit and is bounded by the Frontera fault to the west and the Flamenco fault to the south. The North and East Walls of the proposed Filo pit are situated in the Filo East Domain. Only the lower portion of the West Wall falls within this domain.

2. Filo West Domain. This domain lies west of the Frontera fault and north of the Flamenco fault and comprises the unmineralized rock that will form the upper portion of the West wall of the proposed Filo pit.

3. Tamberías Domain. This domain represents the portion of the project area south of the Flamenco fault. The southern portion of the proposed Filo pit and all the proposed Tamberías pit are located within this domain.

Figure 16-1: Structural Domain Boundaries Shown with Ultimate Pit



Source: AGP, 2023

No geotechnical or exploration core drilling has been completed since 2018 in either the Filo West or Tamberías structural domains. Material in these areas was characterized from surface outcrops. The Filo East structural domain was subdivided into four (4) geotechnical units representing zones of differing rock mass properties. The resulting geotechnical domains used in the development of open pit slope design parameters are:

- Leached (LIX) Unit, present at the top of the deposit and up to 315 m thick, this zone includes heavily leached and altered material with low rock mass strength.
- Oxide (OX) Unit, located beneath the LIX, this zone is characterized by the presence of oxide mineralization and has increased rock mass strength relative to the LIX
- Silver (M-AG) Unit, a sub-zone of the OX, this zone is typically present near the base of the OX and has a higher rock mass strength than the OX Unit.
- Hypogene (HIPO), this zone is the deepest modelled unit of the Filo deposit and will only be present in the floor of the proposed open pit. Due to the lack of drilling in this unit and for the purpose of this study, the HIPO Unit was assigned the same strengths as the OX Unit.
- Filo West (FW) Unit, comprising the rock mass located west of the Frontera Fault and north of the Flamenco Fault, this unit is geologically distinct from the four mineralized and altered units described above.
- Tamberías (TAM) Unit, comprising the rock mass located south of the Flamenco fault, this unit forms the Tamberías pit walls, generally lacks the hydrothermal alteration observed in the Filo pit area and is considered the strongest Unit.

BGC completed kinematic and limit equilibrium stability analyses using the structural geology and rock mass data available to develop slope design criteria for the Filo and Tamberías deposits. The open pit slope design parameters are outlined in Table 16-1. To account for the weak rock mass present in the LIX Unit, a 12 m high single-bench configuration and 48 m maximum inter-ramp slope height is recommended for design sectors within this unit. Inter-ramp (i.e., toe-to-toe) slope angles range from 40° to 42°, excluding geotechnical berms, within the LIX unit. For all other geotechnical units encountered in the pit walls (i.e., the FW, TAM and OX units), a 24 m high double-bench configuration with a maximum inter-ramp slope height of 96 m is recommended. Inter-ramp slope angles, excluding geotechnical berms, within these units range from 40 to 47°. Recommended widths for geotechnical berms, which separate inter-ramp slope segments, range from 25 to 40 m and were designed to achieve the overall slope stability acceptance criteria.

The recommended open pit slope design parameters assume the following:

- Geotechnical slope monitoring systems and a ground control management plan are in place during operation of the proposed open pits.
- Pore pressures do not develop within the pit walls.
- Controlled blasting techniques are used for interim and final walls to minimize damage from production blasting.

Table 16-1: Filo del Sol Open Pit Slope Design Parameters

Structural Domain	Design Sector	Slope Azimuth		Bench Geometry			Inter-Ramp Geometry			Slope Design Control
				Design Height	Face Angle	Width	Maximum Height	Angle	Geotechnical Berm Width	
		Start (°)	End (°)	Bh (m)	Ba (°)	Bw (m)	Ih (m)	Ia (°)	(m)	
Filo West	FW-235	210	260	24	65	13.7	96	44	35	Inter-ramp (Wedge FW2-FW4)
	FW-310	260	000	24	65	11.3	96	47	35	Inter-ramp (Bench geometry)
Filo East	LIX-250	220	280	12	65	7.9	48	42	35	Inter-ramp (Bench geometry)
	LIX-325	280	010	12	65	7.9	48	42	40	Geotechnical berm geometry designed for overall rockmass stability; Inter-ramp control is bench geometry
	LIX-043	010	075	12	65	8.7	48	40	40	Geotechnical berm geometry designed for overall rockmass stability; Inter-ramp control is Toppling FE9
	LIX-100	075	125	12	65	8.7	48	40	35	Inter-ramp (Toppling FE6)
Filo East	OX-250	220	280	24	65	11.3	96	47	35	Inter-ramp (Bench geometry)
	OX-325	280	010	24	65	11.3	96	47	35	Inter-ramp (Bench geometry)
	OX-068	010	125	24	65	17.4	96	40	35	Inter-ramp (Toppling FE9 & FE6)

Structural Domain	Design Sector	Slope Azimuth		Bench Geometry			Inter-Ramp Geometry			Slope Design Control
				Design Height	Face Angle	Width	Maximum Height	Angle	Geotechnical Berm Width	
		Start (°)	End (°)	Bh (m)	Ba (°)	Bw (m)	Ih (m)	Ia (°)	(m)	
	OX-140	125	155	24	65	16.4	96	41	35	Inter-ramp (Toppling FE4)
	OX-188	155	220	24	65	11.3	96	47	35	Inter-ramp (Bench geometry)
Tamberías	TAM-193	145	240	24	65	16.9	96	41	25	Inter-ramp (Bench geometry)
	TAM-263	240	285	24	65	16.4	96	41	25	Inter-ramp (Toppling T1)
	TAM-318	285	350	24	65	11.3	96	47	25	Inter-ramp (Bench geometry)
	TAM-020	350	050	24	65	17.4	96	40	25	Inter-ramp (Toppling T4)
	TAM-065	050	080	24	65	11.3	96	47	25	Inter-ramp (Bench geometry)
	TAM-113	080	145	24	65	11.3	96	47	25	Inter-ramp (Bench geometry)

The pit phase designs presented in the following section utilized double benching rather than single benching in the Filo East Lix domain to simplify the pit phase design process, with resulting minor volumetric changes. This is considered an acceptable deviation from the design criteria at prefeasibility level. Designs for future feasibility level analysis will require single bench designs in this domain.

The presence of potentially deep permafrost in the walls of the proposed Filo del Sol open pit requires further study in more advanced stages of design. Melting near-surface permafrost can result in increased slope ravelling and rockfall hazard, particularly for north-facing slopes. In the zones where the permafrost will remain frozen, it may be possible to incorporate higher strengths into slope stability models to account for the additional cohesion provided by the ice bonds.

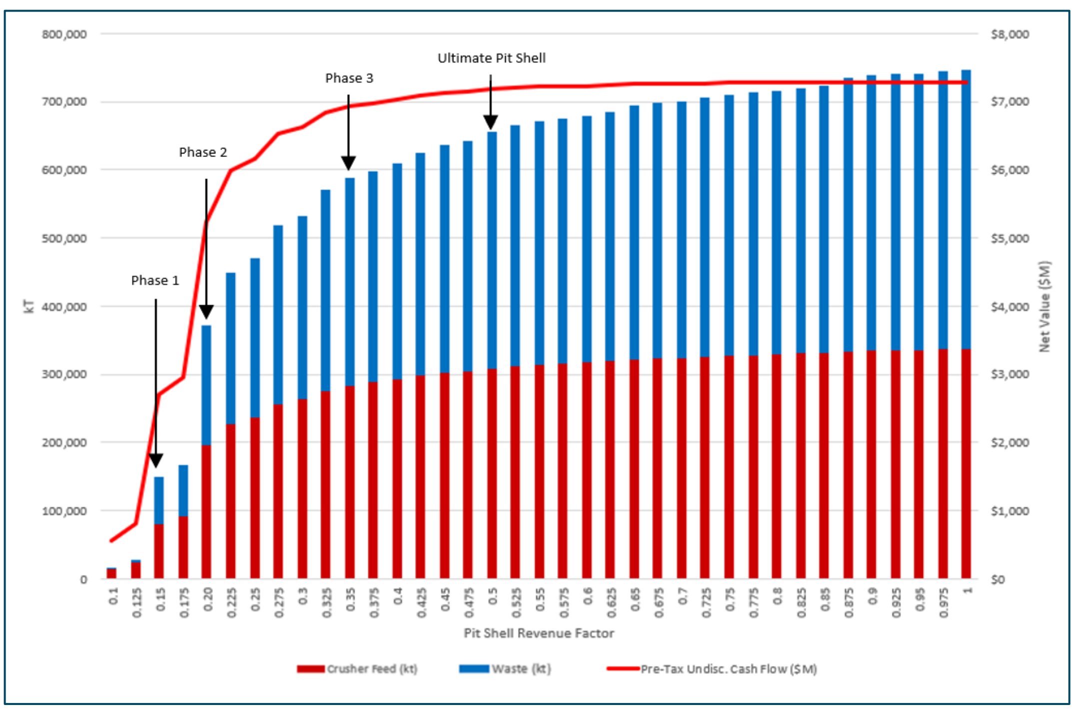
16.1.2 Pit Phase Designs

Sections 15.4 and 15.5 presented the pit optimization analysis used to develop and select the LG shell used to guide the ultimate pit design. The nested shells used to guide the internal phase designs were selected based on:

- 2 to 3 years of crusher feed in the starter pit (Filo Phase 1)
- An even distribution of crusher feed tonnes per phase
- Access and minimum mining width considerations

The LG shells used to guide the Filo pit phase are shown in Figure 16-2 below.

Figure 16-2: Pit by Pit Graph with Selected Filo Internal Phase Guidance



Source: AGP, 2023

The pit design criteria used to develop the pit phase designs are as follows:

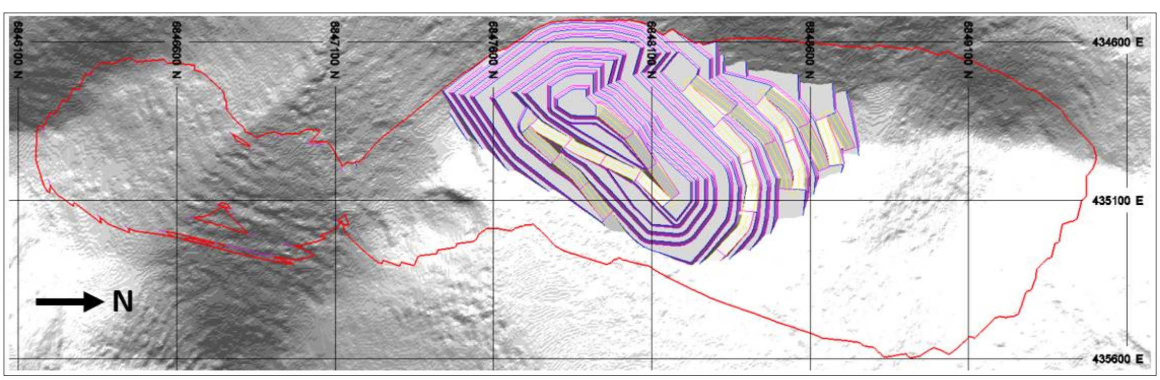
- All bench face angles are 65 degrees.
- Bench height is 12 m.
  - All design sectors will use double benching including the Filo LIX sector for which BGC recommends single benching. This variance granted for the Filo LIX sector for PFS level design purposes is discussed in Section 16.1.1 above.
- All other pit slope angles as presented in Table 16 1 above.
- Haul ramps:
  - 33.7-m wide double lane and 24 m wide single lane, based on the design 220mt capacity truck.
  - No steeper than 10% on shortest ramp segment (inside corner) for uphill loaded hauls
  - For downhill loaded hauls, no steeper than 8%
- Minimum Mining width 60m including one 7.2m outside berm and 12m wide drill access ramp
  - Can reduce to 40m for distances less than 150m
  - Minimum mining width for 26 m^3-size shovels is 37 m for double side loading
  - Minimum turning diameter for the design 220mt capacity truck is 29 m.

The resulting pit phase designs are as follows:

16.1.2.1 Filo Phase 1

The Filo phase 1 is the starter pit of the project. A portion of the west wall is final. In all other directions, this pit is expanded outwards by subsequent pit phases. Stripping starts in Year -2 at the 5,309 m elevation and mining to the 4,949 m bottom is completed in Year 3. The ramp from the bottom daylights to the east at the 5,075 m elevation. A ramp has been designed in the north wall to provide access to the upper portions of Filo phase 2. The Filo phase 1 design is shown in Figure 16-3 below.

Figure 16-3: Filo Phase 1 Design with Ultimate Pit Outline in Red

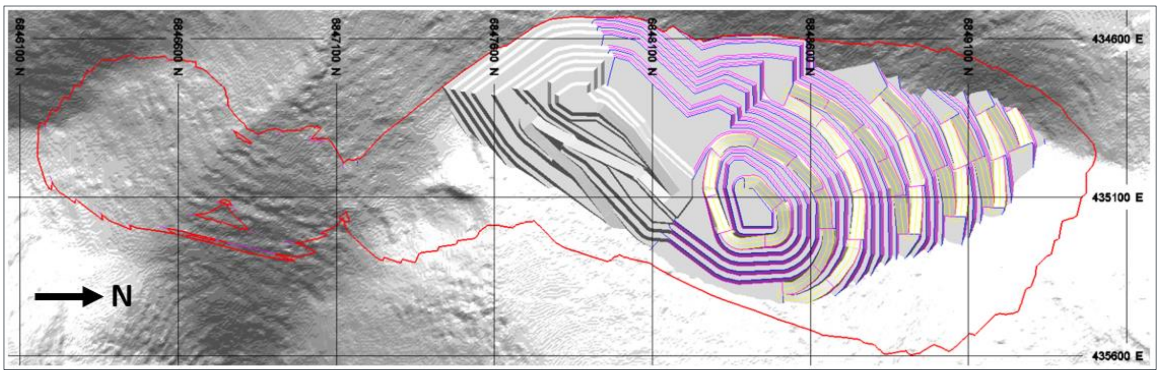


Source: AGP, 2023

16.1.2.2 Filo Phase 2

The Filo Phase 2 is a pushback to the north and west of Filo phase 1. The West portion pushes the phase 1 wall further west to the final wall limits. Stripping starts in Year 1 at the 5,405 m elevation and mining to the 4,937 m bottom is completed in Year 7. The ramp from the bottom daylights to the east at the 5,069 m elevation. A ramp has been designed in the north wall to provide access to the upper portions of Filo Phase 3. The Filo Phase 2 design is shown in Figure 16-4below.

Figure 16-4: Filo Phase 2 Design

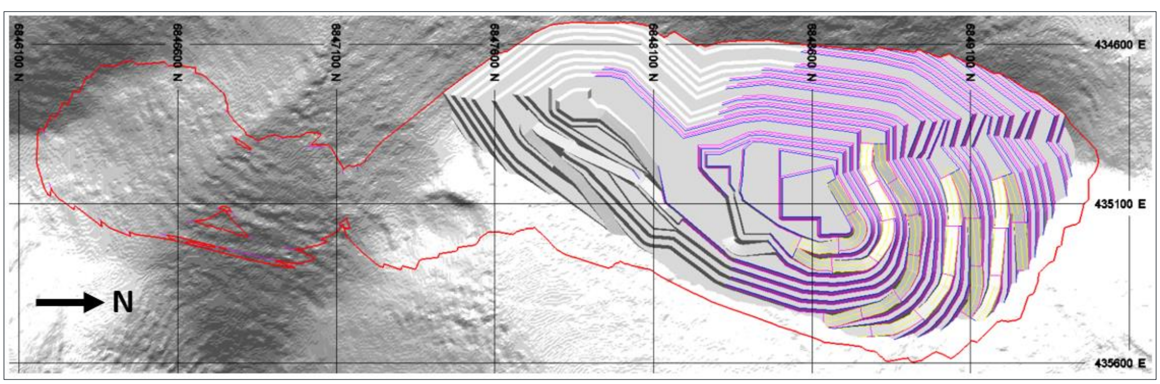


Source: AGP, 2023

16.1.2.3 Filo Phase 3

The Filo phase 3 is a pushback to the north of Filo phase 2. The West portion is final wall. Stripping starts in year 1 at the 5,381 m crest elevation and mining to the 4,901 m bottom is completed in year 9. The ramp from the bottom daylights to the east at the 5,067 m elevation. A ramp has been designed in the north wall to provide access to the top of Filo phase 4. The Filo phase 3 design is shown in Figure 16-5below.

Figure 16-5: Filo Phase 3 Design

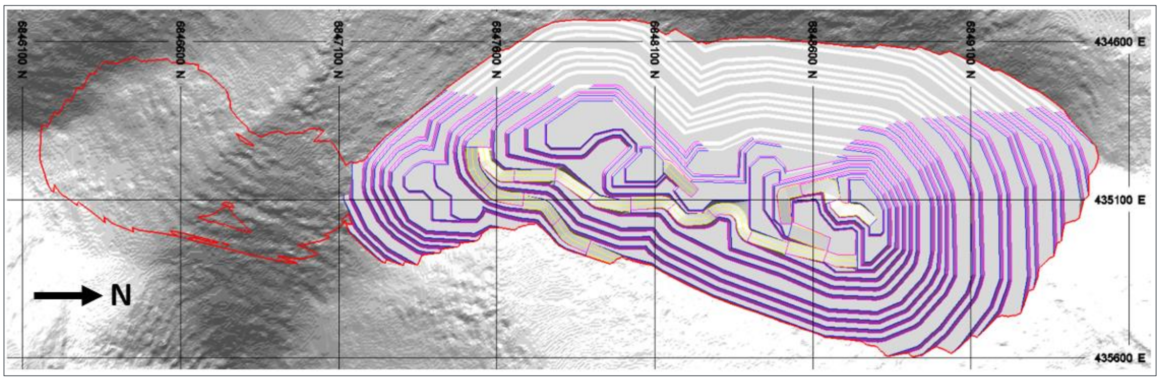


Source: AGP, 2023

16.1.2.4 Filo Phase 4

The Filo Phase 4 is the final Filo pushback to the north of Filo Phase 3, and to the west east and south of the previous Filo pushbacks. Stripping starts in Year 3 at the 5,333 m elevation and mining to the 4,841m bottom is completed in Year 12. The ramp from the bottom daylights to the east at the 5,055 m elevation. The Filo Phase 4 design is shown in Figure 16-6 below.

Figure 16-6: Filo Phase 4 Design

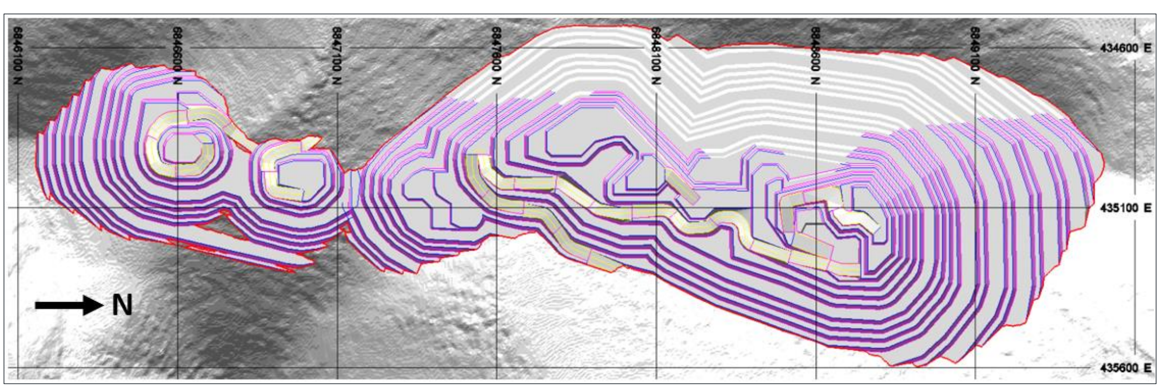


Source: AGP, 2023

16.1.2.5 Tamberías Pit

The Tamberías pit is a small single phased pit to the south of the Filo pit. Stripping starts in Year 2 at the 5,393 m elevation and mining to the twin 5,105 m bottoms is completed in Year 13. In the mine schedule, the Tamberías pit serves as an in-pit stockpile, filling ore feed gaps that arise in the sequencing of the higher-grade Filo pushbacks, resulting in a long active period throughout the mine life. The ramps from the southern lobe pit bottom daylights to the west at the 5,153 m elevation, and ramp from the bottom of the northern lobe daylights at the 5,133 m elevation. The Tamberías pit design is shown in Figure 16-7below.

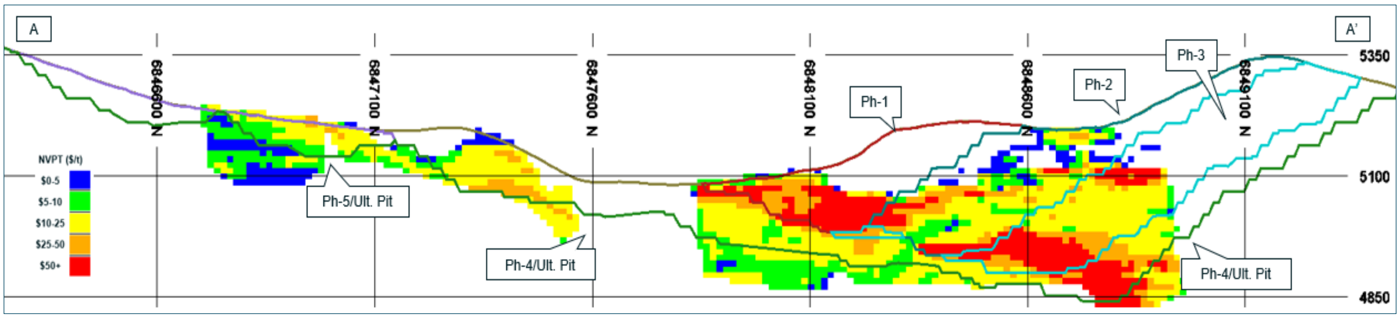
Figure 16-7: Tamberías Pit Design



Source: AGP, 2023

Figure 16-8 shows a vertical section through all the pit phases, with the ore blocks colour coded by the NVPT, along the section line shown in Figure 16-7. Note, the section does not pass through all the phase pit bottoms.

Figure 16-8: West Looking Vertical Section Showing Pit Phases and Above Cutoff Mineralization



Source: AGP, 2023

The volumetrics by pit phase are shown in Table 16-2 below.

Table 16-2: Pit Volumetrics by Phase

	Ore					Waste	Total	SR
	Kt	Cu (%)	Au (g/t)	Ag (g/t)	NVPT ($/t)	Kt	Kt	(W/O)
Filo Ph1	53,522	0.44	0.36	21.9	42	85,033	138,158	1.59
Filo Ph2	48,679	0.30	0.37	6.6	24	79,737	127,119	1.64
Filo Ph3	59,392	0.43	0.31	21.4	37	134,671	193,529	2.27
Filo Ph4	77,136	0.36	0.35	17.5	30	80,564	157,700	1.04
Tamberías	23,364	0.41	0.33	1.3	24	27,099	50,238	1.16
Total	**259,640**	**0.39**	**0.34**	**16.0**	**32**	**407,104**	**666,744**	**1.57**

16.2 Waste Dump Design

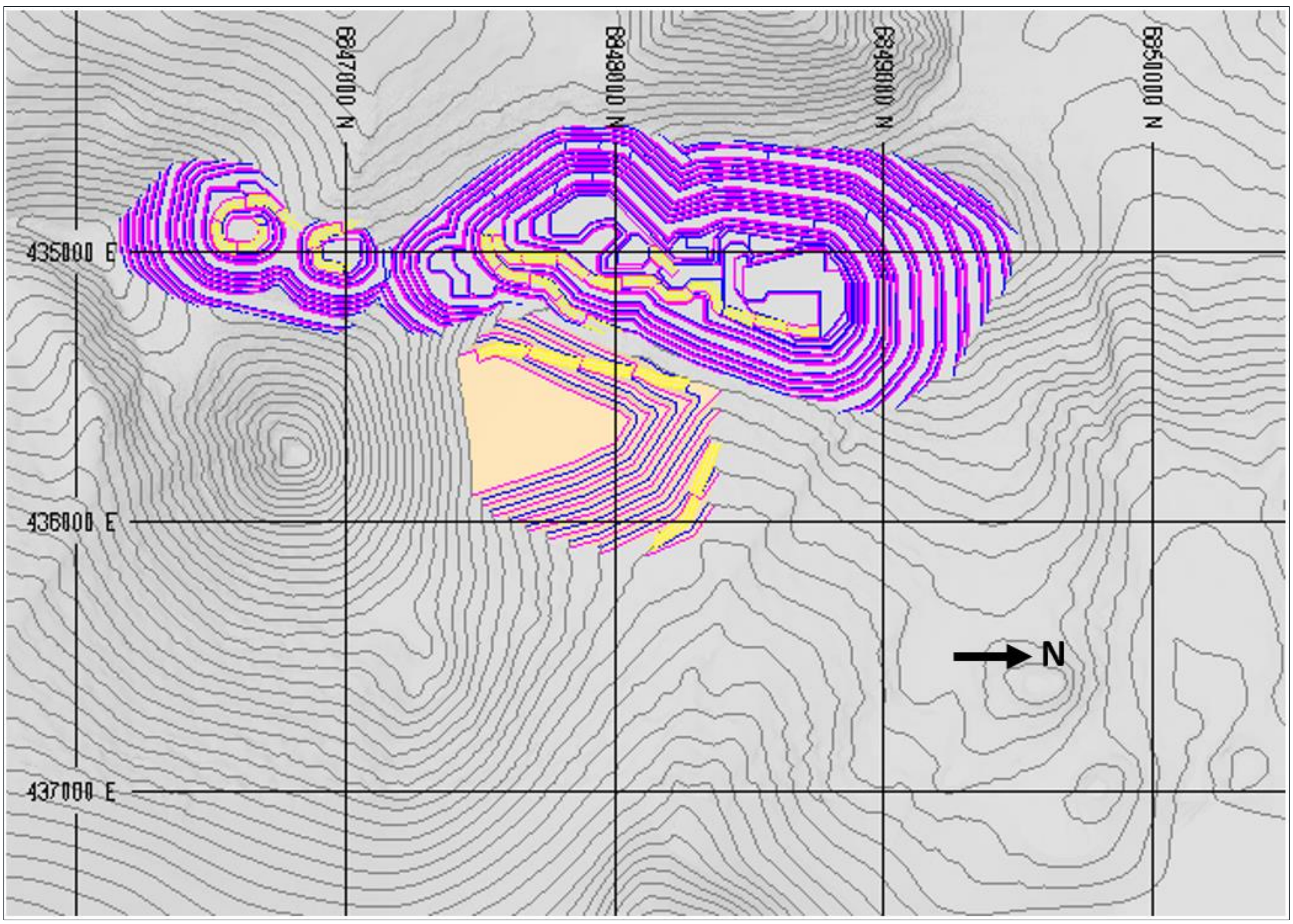
A waste dump was designed to hold the waste rock generated during the mine life, excluding the 14 million tonnes of waste used as construction fill. The facility is located immediately east of and generally downslope from the Filo pit. Due to the presence of near surface permafrost throughout the dump footprint, 'bottom up' construction and excavation in the area (key) to provide good contact and stability are required. To mitigate very long downhill waste hauls in the early mine life and provide some scheduling flexibility, the dump was designed in two phases: a first smaller phase located on shallowly sloping topography, and an ultimate dump that 'toes out' much lower in the valley and largely encompasses the phase 1 dump. The design criteria provided by Ausenco is shown in Table 16-3 below.

All waste rock is assumed to be potentially acid generating (PAG). No special waste handling has been contemplated at this time. There will be minor pit backfill opportunities that have not been utilized at this time.

Table 16-3: Waste Dump Facility Design Criteria

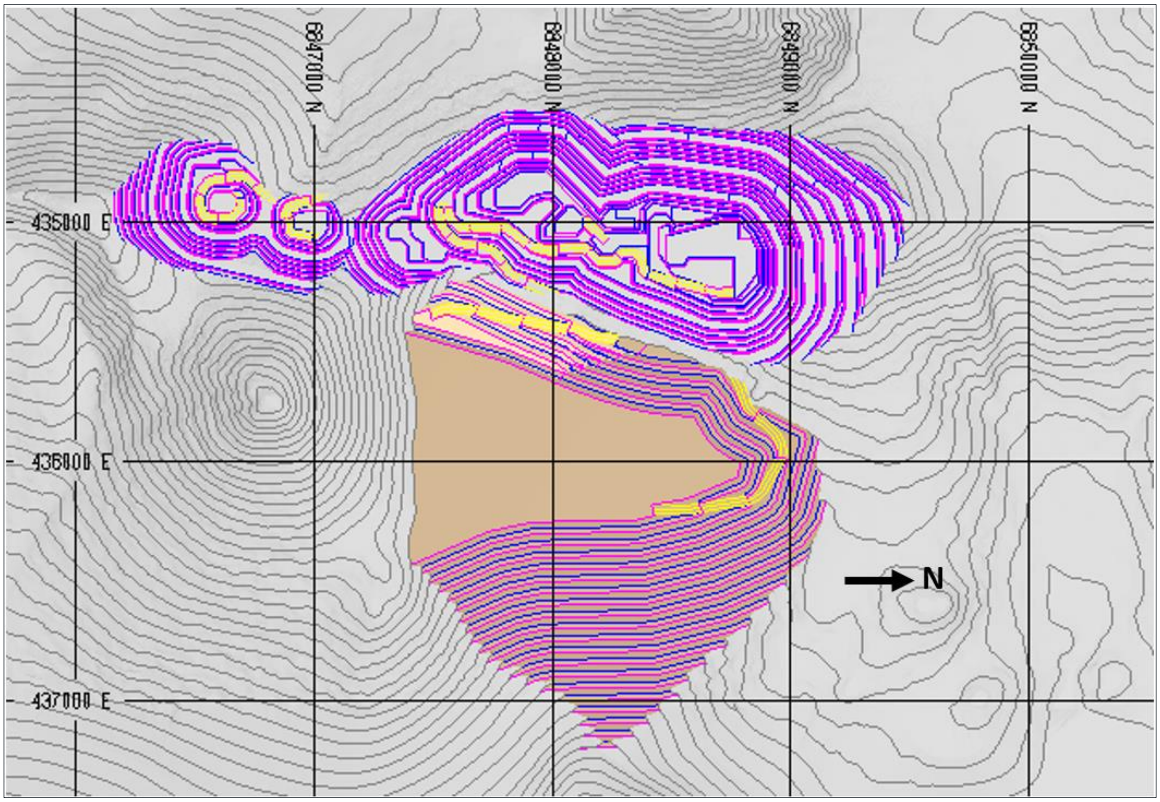
Parameter	Value
Disposal Method	Upslope Construction
Angle of Repose	A.O.P. = 36
Overall Fill Slope Angle	O.S.A. = 22 (2.5:1 H:V)
Lift Height	H = 20 m
Swell Factor	30%

Figure 16-9: Phase 1 Waste Dump



Source: AGP, 2023

Figure 16-10: Phase 2 Waste Dump



Source: AGP, 2023

A 40m stand-off distance was used between the pit and the waste dump. The stand-off distance should be confirmed by geotechnical analysis during the next stage of study.

16.3 16.3 Production Schedule

The mine plan presented in this report was developed using MinePlan's Schedule Optimizer. Descent rates were limited to 10 benches per year. The mine is scheduled to work 365 days a year (d/a), with thirteen days of delay time to weather disruptions. The plant is scheduled to operate 365 d/a.

16.3.1 Pre-Production

One and a half years of pre-production mining are required to carry out the following tasks:

- Develop approximately 8 km of cut and fill haul roads to connect the upper elevation of the Filo phase 1 pit to the bottom of the phase 1 waste dump and crusher area.
- Dump footprint preparation, consisting of 120,000 m^3 of key excavation at the toe of the phase 1 dump, and dump underdrain installation

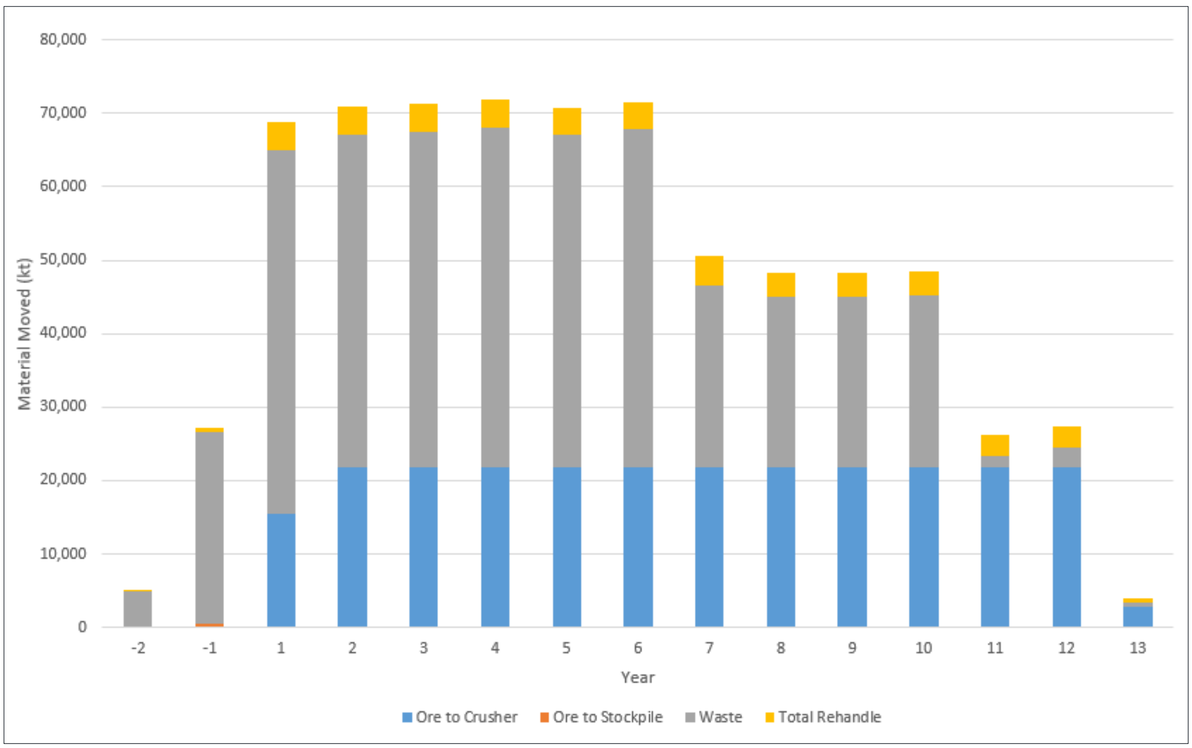
- Strip 31.3 Mt of waste rock from Filo Phase 1, exposing sufficient ore to allow continuous ore delivery during production
- Stockpile 289 kt of preproduction ore for rehandle to the crusher during Year 1
- Deliver 14 Mt of waste rock for construction fill.

16.3.2 Production

Ore delivery to the crusher in the first production year is 16,000 kt, of the which is inclusive of the pre-production stockpiled ore reclaim. In production Year 2 through 12, the full 21,900 kt (60 kt/d) are delivered to the crusher area. The last year of production, Year 13, is a partial year with 2,740 kt to be processed.

It is assumed that 90% of the ore can be direct tipped to the crusher, with the remaining 10% being placed in a nearby short-term stockpile and rehandled to the crusher by front end loader and trucks as required. The peak mining capacity ex-pit of 68 Mt/a or 186.3 kt/d is reached in Year 4, with an average ex-pit mining rate of 67 Mt/a or 183.5 kt/d in Years 1-6. Total material movement averages approximately 70 Mt/a from years 1-6. The material moved from the mine is shown in Figure 16-11below, and material delivered to final destinations is shown in Table 16-4 below.

Figure 16-11: Material Moved in Mine Schedule



Source: AGP, 2023

Table 16-4: Material Delivered to Final Destination

	Ore Delivered to Crusher					Waste	Total Material	SR
-2						4,998	4,998	
-1						25,999	25,999	
1	16,000	0.29	0.40	4.6	24.8	49,557	65,557	3.10
2	21,900	0.47	0.35	16.0	40.7	45,100	67,000	2.06
3	21,900	0.47	0.35	35.1	48.9	45,505	67,405	2.08
4	21,900	0.33	0.38	1.3	22.3	46,100	68,000	2.11
5	21,900	0.33	0.36	3.1	23.1	45,100	67,000	2.06
6	21,900	0.40	0.32	5.4	26.5	45,842	67,742	2.09
7	21,900	0.36	0.24	8.7	23.9	24,714	46,614	1.13
8	21,900	0.43	0.28	12.4	31.6	23,100	45,000	1.05
9	21,900	0.48	0.40	48.8	56.8	23,100	45,000	1.05
10	21,900	0.35	0.30	9.1	23.5	23,218	45,118	1.06
11	21,900	0.36	0.31	13.7	27.5	1,453	23,353	0.07
12	21,900	0.37	0.43	32.1	40.1	2,617	24,517	0.12
13	2,740	0.33	0.31	1.5	17.0	701	3,441	0.26
Total	**259,640**	**0.39**	**0.34**	**16.0**	**32.5**	**407,104**	**666,744**	**1.57**

16.4 Mining Operations

The Filo del Sol ore bodies are generally large and relatively continuous in grade, allowing a bulk mining scenario. The pit operations will work two 12 hour shifts per day with four crews on a one week in, one week out rotation. Engineering, geology and some operations supervisory / support positions will be on day only 12 hour shifts which will also rotate weekly.

The below sections discuss the selection of equipment and peak requirements. A summary table of the primary production equipment is shown in Table 16-5.

Large sized diesel-powered equipment has been selected for this study, however a trade-off between diesel vs. electrified shovels and drills should be investigated as part of a feasibility study.

16.4.1 Loading

Production loading duties will be performed by 26 m^3 diesel hydraulic face shovels, with 18 m^3 front-end loaders assisting with pit loading as well as ore rehandle from the short-term stockpile. The equipment is well matched to the 12m bench height. The peak loader requirements are three shovels and two front end loaders.

Table 16-5: Primary Production Equipment

Equipment Type	Equipment Class	Maximum Fleet Size
Haul Truck	220 t	25
Hydraulic Shovel	26 m^3	3
Front End Loader	18 m^3	2
Track Dozer	4.7 m blade	4
Grader	4.9 m blade	2
Rubber Tired Dozer	5.2 m blade	1
Support Backhoe	3.0 m^3	1
Water Truck	136 t	2
Blast Hole Drill	34,000 kg pulldown, 200 mm bit	4
Small Drill	22 t operating wt., 140 mm bit	2

16.4.2 Hauling

The geometric shapes of the pit phases, mountainous terrain and relative location of the ore crusher and waste dumps result a high percentage of downhill loaded hauling vs. uphill loaded hauling. Electric drive haul trucks were selected over mechanical drive as a better fit for the significant downhill hauling requirements. A 220 t truck was selected as it matched well to the loading tools.

A trade-off study was performed comparing autonomous haulage vs. conventional haulage using performance and cost parameters provided by the primary equipment vendor. The capital component for the autonomous case consisted of the installed haulage network control system, add-on components for the haul trucks, components to be installed at the crusher and add on components for all other mobile equipment in the pit. Operational cost components consisted of user fees, licensing fees and a monthly service fee for the vendor to provide system operators and system maintenance technicians. Assumed performance improvements due to autonomous haulage included:

- a 2% increase in mechanical availability,
- a 25% improvement in tire life (from 4,500 hrs to 5,625 hrs),
- an increase in operator efficiency from 83% to 90%, and
- an increase in operating time of two hours per day (with extra loading and crusher operators to allow loading and hauling during breaks and lunch)

The autonomous case showed a net present cost savings (at an 8% discount rate), of US$19.8 million to the mine operations department. Further savings are recognized to the project G&A, camp and transport costs due to a reduction in manpower. AGP are of the opinion that autonomous haulage is sufficiently proven in operations to be used to support a mineral reserves disclosure.

The peak truck requirement is 25 units.

16.4.3 Drilling and Blasting

Blasting will have a significant effect on slope performance and achievable pit slope angles. The recommended open pit slope design parameters assume that controlled blasting (e.g., trim, buffer, modified trim) techniques will be applied to interim pit walls, with pre-splitting applied to final pit walls to reduce disturbance to the rock mass comprising the pit slopes. Loose rocks that may represent a hazard to equipment or personnel working in the mine should be removed through scaling of the final bench faces and proper bench clean-up procedures should be implemented to preserve rockfall catchment.

16.4.3.1 Production Drilling

For the 12 m benches, the drill bit size selected for main production holes was 200 mm diameter. A production drill rig with 34,000 kg pulldown was selected which could drill holes in a single pass, without the need to add or remove steel, to improve productivity.

The pattern size for ore was determined by using a fragmentation prediction model with the goal of producing a fragmentation distribution curve with a P80 passing size of approximately 700 mm. Waste did not need to meet this size specification, so the pattern used for waste was expanded slightly. The drill pattern specifications are shown in Table 16 6 below.

Table 16-6: Drill Pattern Specifications

Specification	Unit	Ore	Waste
Bench Height	m	12	12
Sub-drill	m	1.3	1.3
Blasthole Diameter	mm	200	200
Pattern Burden - Staggered	m	6.4	6.9
Pattern Spacing- Staggered	m	7.4	7.6
Hole Depth	m	13.3	13.3

The recommended primary drill has the capability of drilling the 12 m bench plus subdrill in a single pass, thus improving the cycle time compared to a smaller drill. Based on a drill productivity of 29.9 m/working hour, a peak of 4 drills is required. Two secondary drills capable of drilling a 140 mm hole will be used for pioneering work, presplitting and secondary blasting as required.

An opportunity exists to investigate autonomous drilling during the next stage of project planning.

16.4.3.2 Blasting

A bulk loaded emulsion blended product will be used for blasting and is expected to give better performance and have better water resistance compared to ANFO. The product selected is composed of 70% emulsion and 30% AN by weight and will have a loaded density of 1.2 g/cc. The powder factors used were 0.27 kg/t and 0.23 kg/t for ore and waste, respectively.

Buffer blasting and pre-shear will be employed for wall control. The buffer row will be drilled on a 3.5 m burden by 7.6 m spacing pattern, with a subdrill of 1.0 m. The pre-shear row will be drilled with a smaller DTH drill using a 140 mm diameter bit. The pattern will be 2.2 m burden by 1.7 m spacing and only 600 mm (11 kg) of explosive will be placed in the hole to reduce energy that may be directed into the wall.

The blasting cost is estimated using quotations from local vendors, adjusted to reflect levels above long-term price history, but below current elevated price levels which are expected not to persist for the duration of the project. Unit costs for bulk, packaged and initiating explosives, delivered to site, were provided.

The vendors also quoted a monthly service fee to cover the cost of capital and personnel to provide a full blasting service (priming, loading, stemming, sequencing, firing and magazine management). The blasting supplier will provide three mobile manufacturing units (MMUs), magazine storage capacity for two weeks, offices, storage tanks and pumps.

The mine will be responsible for providing the following at no cost to the blasting vendor: meals, accommodation, electricity, water, diesel and stemming aggregate, and any other special accessories.

16.4.4 Support and Ancillary Equipment

Roads, pit floors and dumps will be maintained by a fleet of track dozers, wheel dozers, and graders support equipment, as shown in Table 16-5 above. The ancillary equipment specified for the mine is shown in Table 16-7 below.

Table 16-7: Ancillary Equipment

Equipment	Maximum Fleet Size
Tire Manipulator	1
Lube/Fuel Truck	1
Mechanic's Truck	1
Welding Truck	1
Blasting Loader	1
Blasters Truck	1
Integrated Tool Carrier	1
Compactor 2.1 m drum	1
Lighting Plants	8
Track Dozer 2.7 m blade	1
Man Bus	2
Pickup Trucks (3/4 ton)	15
Crane 50 t	1
Crane 35 t	1
Pump Truck	1
Dump Truck 20 ton	2
Lowboy and tractor 75- 100 ton	1

16.5 Ore Control

Ore control will be performed by a group of geologists and geologic technicians within the mine operations department. Samples will be collected from the blastholes during the drilling process and delivered to the process facility for sample preparation and assay determinations. Assay results for total Cu, Au and Ag, plus sequential Cu determinations will be used to estimate recovered metals and NVPT in a similar manner to the long-range planning process. An estimate of annual sample quantities was developed assuming all ore plus 70% of waste blastholes would require assay determinations. No waste characterization determinations were considered. An ore control block model will be developed and used to create 'diggable' homogeneous ore control 'packets' which will be uploaded to the shovels and loaders for 'stakeless' ore and waste digging.

The ore control group will also be responsible for performing regular reconciliations between the resource model, the ore control model and process production reporting.

16.6 Hydrogeological Considerations

Slope design criteria assume fully depressurized conditions in the proposed open pit slopes, as they are primarily above the regional groundwater table. Observations during drilling and in open holes from previous programs indicate that water is greater than 150 m below ground surface. No hydrogeological testing data were collected for this study. If groundwater is encountered in future studies, the recommended slope design criteria may need to be revised.

Significant surface water management structures in the open pit at Filo del Sol are not anticipated to be required based on climate and the current groundwater regime. However, storm water will need to be managed intermittently. Surface water runoff should be diverted away from the pit slopes, especially those developed in the extremely weak LIX unit. Ditches and ponds at or near the pit slope crests should be avoided. If water is to be conveyed near the pit crests, pipelines should be used. Secondary containment via ditches could be considered. If groundwater seepage is noted in the open pit, a series of in-pit ditches and sumps are recommended to collect water to be pumped out of the open pit into the mine surface water management system.

Annual Precipitation

The Filo del Sol Mine Study Area is comprised of the Los Mogotes River watershed and the Upper Montoso River watershed, totalling an area of approximately 205 km^2. Estimated long-term monthly and annual precipitation data was provided by Knight Piésold (KP) in Table 2.9 of the Hydrometeorology Assessment dated May 14, 2018. From this data, a mean value of 131 mm of precipitation per year was used for dewatering calculations.

Groundwater Inflow

KP estimated that an extreme, one in ten-year, rainfall event could produce short duration inflows of 1,000 L/s. As storm water would be collected in sumps, it was recommended that the dewatering system be designed to have a peak capacity of 100 L/s, to enable the excess water to be removed over a number of days.

A high-level estimate for ground water inflow of 8 L/s was used for dewatering calculations and the system was designed to handle a peak capacity of 100 L/s.

The peak the annual dewatering requirement has been estimated to be 762,000 m^3. A 265 hp electrical pump was selected for pit dewatering. In cases where the total head is too great for a single pump, two pumps will be connected in series.

The peak number of electric pumps required to meet the dewatering requirements over the life of the mine is nine, including one spare and replacement units.

16.7 Pit Slope Monitoring

Deformation monitoring of the pit slopes during mining will be undertaken to:

- Maintain safe operational practices for personnel, equipment, and near-pit facilities
- Provide warning of slope instability
- Confirm design assumptions
- Provide geotechnical information for slope designs to assist in making subsequent modifications, should they be required, to achieve the desired slope performance

A ground control management plan will be developed and implemented for the pit slopes of the proposed Filo del Sol mine during operations including: daily visual inspections of pit crest and slopes by mine staff with results recorded in a slope hazard log book to be reviewed on a regular basis by the site geotechnical engineer; monitoring of slope movements using total stations to survey a network of reflector prisms; a trigger action response plan (TARP) associated with the slope monitoring; and, a monitoring database to store the prism survey records with the ability to plot the time-series graphs. The need for more complex monitoring systems, such as slope stability radar, LiDAR monitoring, or subsurface instrumentation, should be assessed throughout the mine's operation. If slope instabilities develop, the monitoring system should be upgraded to allow for continued safe operation of the mine.

16.8 Workforce

The peak mine operations workforce will consist of 250 hourly operators and maintenance workers and 56 staff. Additionally, there will be 13 blasting contractors and 6 dispatch/autonomous system operators on site at all times. The peak total mine operations workforce in camp is 177 people.

17 RECOVERY METHODS

17.1 Summary

The process plant for the Filo del Sol project is designed to treat 60,000 t/d of ore through a sequential heap leach process, to produce copper cathodes and gold/silver doré.

Key operating criteria for the process plant are listed below:

- Nominal throughout of 60,000 t/d or 21.9 Mt/a
- Crushing plant availability of 72%
- Heap Leach plant availability of 98%
- Plant availability of 95% for solvent extraction, electrowinning, and neutralization

Figure 17-1 shows a simplified schematic of the process flowsheet.

17.2 Process Design Criteria

Key process design criteria listed in Table 17-1 and flowsheet selection was based upon results of laboratory test work as summarized in Section 13.

The on/off pad has been designed considering cost and footprint constraints. Ausenco evaluated the column test extractions as a function of solution to ore ratio, as well the extractions as a function of leach time to determine the appropriate on/off leach cycle time. Based on this analysis a leach time of 52 days was selected as the design basis.

The permanent leach pad has been designed considering cost and footprint constraints. Ausenco evaluated the column test extractions as a function of the solution to ore ratio, as well the extractions as a function of leach time. Based on this analysis, a leach time of 60 days was selected as the design basis for the recoverable metals, followed by additional irrigation time as each successive overlying lift is leached. The additional time needed to reach the ultimate precious metal recovery for a given lift was not considered at this stage and should be further resolved in the future stages of the Project.

Table 17-1: Process Design Criteria

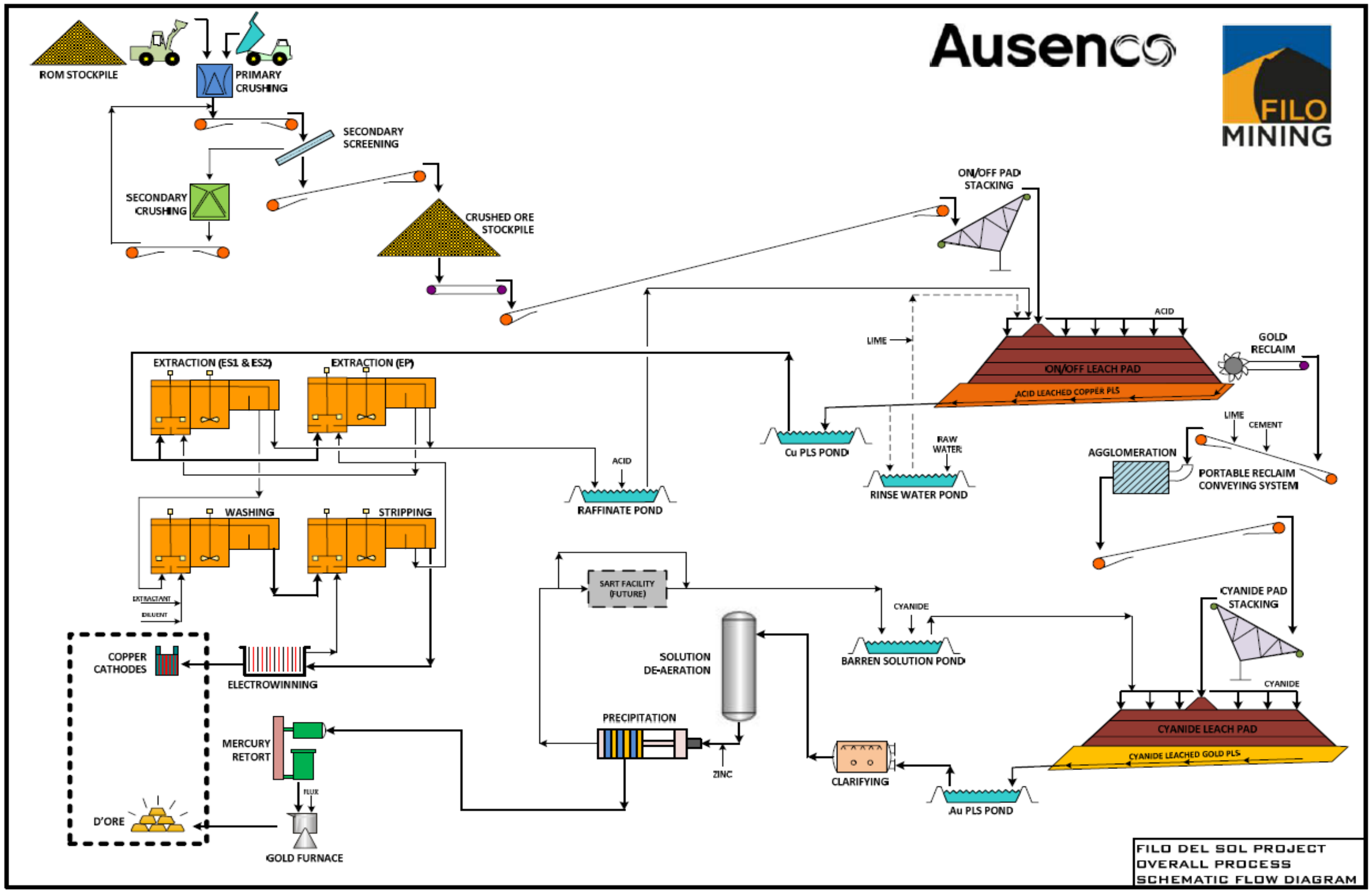
Description	Units	Value
Throughput	Mt/a	21.9
Throughput	t/d	60,000
Copper Grade – LOM	%	0.38
Gold Grade – LOM	g/t	0.33
Silver Grade – LOM	g/t	14.7
Material Specific Gravity	t/m^3	2.7
Moisture Content	%	3
Crushing Area Availability	%	72
Heap Leach Area Availability	%	98
Plant Availability	%	95
Crushing Work Index – design	kWh/t	9.1
Abrasion Index (Ai)	g	0.46
Leaching – Copper On/Off Pad		
Type of Pad		Re-usable flat pad with 12 cells in various stages of operation
Residence Time	Days	52
Rinse Time	Days	12
Copper Extraction	%	78
Leaching – Cyanide Permanent Pad		
Type of Pad		Valley Fill
Residence Time	Days	60
Gold Extraction	%	70
Solvent Extraction Circuit Configuration		Series-Parallel

17.3 Process Flowsheet

The process plant includes the flowing:

- two-stage crushing of run-of-mine (ROM) material
- copper on/off leach pad
- copper solvent extraction with two stages of extraction, stripping and washing followed by electrowinning
- cyanide leach pad followed by Merrill-Crowe circuit and gold refinery.

Figure 17-1: Overall Process Schematic Flow Diagram



Source: Ausenco, 2019

17.4 Process Description

Ore will be trucked from the mine and either stockpiled or direct tipped into the primary crusher. The ore will be further crushed through a closed-circuit secondary crushing system to a stockpile.

Crushed ore will be processed at an on/off heap leach pad where the copper will be leached in acid and then recovered from the leach solution by solvent extraction and electrowinning to produce LME grade copper cathodes.

Once the copper has been leached, the ore will be rinsed, neutralized and removed from the on/off leach pad by a bucketwheel reclaimer. The material will be agglomerated and stacked on a permanent heap leach pad where gold and silver will be leached in a cyanide solution. Gold and silver will be recovered from the pregnant gold leach solution by a Merrill-Crowe zinc precipitation process and smelted to produce doré.

17.4.1 Crushing

The crushing plant is designed to operate for 6,307 hours or 72% availability at a capacity of 60,000 t/d. No distinction is made between ore types; all ore types will be processed through the same crushing plant.

Run-of-mine (ROM) ore will be delivered by mine trucks to a stockpile or directly into the crusher feed hopper. Front end loaders will transfer the ore from the stockpile to the primary crusher at 3,472 t/h. the primary crusher discharge is conveyed to a vibrating double-deck secondary screen. The secondary screen oversize from both decks is fed to the secondary cone crusher. The cone crusher discharge is conveyed to the secondary screen. The secondary screen undersize is conveyed to the fine ore stockpile that provide 24 hours of live storage. The stockpile disconnects crushing from the on/off pad stacking mill to allow for crusher maintenance.

The fine ore stockpile is equipped with two reclaim feeders to regulate feed onto the on/off leach pad.

The material handling and crushing circuit includes the following key equipment:

- primary jaw crusher
- secondary cone crusher
- secondary screen
- reclaim feeders
- material handling equipment.

17.4.2 Copper On/Off Leaching

Crushed ore will be reclaimed from the stockpile, transferred to the on/off leach pad, stacked with 12 cells to a height of 7.5 m. The crushed ore will be leached for 52 days. At the end of the leach period, the heap will be allowed to drain and be rinsed with water and neutralized with lime slurry over 12 days.

For the copper on/off pad, irrigation will be provided by a series of drippers which distribute the acidic leach solution at a rate of 10 L/h/m^2. Pregnant leach solution (PLS) from the heap will flow to the copper on/off PLS pond with a residence time of 12 h. The PLS will be pumped to the solvent extraction plant for recovery of copper.

Once the copper leach and rinse cycle are completed, the ore will be reclaimed by a bucketwheel conveyor and conveyed to the gold heap leach facility.

Further discussion on the on/off leach pad and copper on/off PLS pond are included in Section 18.10.

17.4.3 Solvent Extraction and Electrowinning

Solvent extraction and electrowinning will be used to extract copper-rich PLS from the on/off leach pad. The solvent extraction (SX) process consists of three stages of extraction, wash stage and single strip stage.

Copper will be extracted from the PLS into the organic phase at a rate of 4,500 m^3/h. The organic will flow in a continuous closed loop around the extraction and strip mixer-settlers, and the loaded organic tanks. Stripped organic flows counter-current to the PLS in the extraction stages and extracts soluble copper from the PLS. Organic leaving the extraction mixer-settlers is called loaded organic which will report to the loaded organic tank. The loaded organic is pumped to the strip stages, counter current to the electrolyte flow. Organic leaving the strip stages, will report to the extraction stages.

Copper will then be removed from the organic using acidic spent electrolyte. This will generate a concentrated copper solution which will be filtered in polishing filters to remove trace amounts of solids and sent to the electrowinning cell house.

Copper will be deposited on stainless steel cathodes in the electrowinning cells. The cathodes will be lifted by a crane from the cells and fed to an automatic cathode stripping machine to separate the product copper sheets which will then be packed for export.

The solvent extraction and electrowinning circuit includes the following key equipment:

- extraction mixer-settlers, washing mixer-settler and stripping mixer-settlers;
- electrolyte heat exchanger;
- electrolyte filters;
- electrowinning cells with rectifiers; and
- anodes and cathodes.

17.4.4 Permanent Cyanide Leaching

Leached ore from the copper on/off pad will be transferred to a stockpile prior to being mixed with cement in the agglomerator and stacked on the permanent cyanide leach pad. The permanent leach pad is based on a valley fill design.

For the cyanide permanent pad, irrigation will be provided by a set of drippers which distribute a solution containing cyanide to leach the gold and silver. Pregnant leach solution (Au PLS) from the heap will flow to the gold PLS pond and then be pumped to the gold recovery plant.

Further discussion on the gold permanent pad is included in Section 18.10.

17.4.5 Gold Recovery

PLS from the gold PLS pond will be treated by the Merrill Crowe process to recover the contained precious metals. PLS will be discharged to the pregnant solution tank which will provide approximately 1 hour of surge capacity to cater for the semi-continuous nature of the clarification and precipitation stages in the Merrill Crowe circuit.

The Merrill Crowe circuit will be provided as a vendor package, and will include:

- clarifier filters;
- de-aeration tower;
- air/water separator;
- de-aeration tower vacuum pump;
- zinc mixing cone, including a hopper and a feeder;
- precipitation filter press units;
- pre-coat preparation tank;
- body feed preparation tank; and
- associated material handling and storage systems (pumps, sump pumps, pump boxes, feed conveyors).

The clarifying filter feed pumps will pump the PLS from the tank to the clarifying disk filters to remove any residual solids. Pre-coat will be required to enhance capture of the fine solids at the start of each cycle. Filtrate from the clarifying filters will feed the de-aeration tower. De-aeration of the solution will prevent the excessive zinc consumption by minimizing side reactions that oxidize zinc.

De-aerated pregnant solution will be contacted with the zinc dust slurry and pumped to the precipitate filters using precipitate filter feed pumps. The precipitate filters will be recessed plate filter presses furnished with filter cloths. Pre-coat will be used at the beginning of the filter cycle to prevent cloth blinding and body feed will be required to provide acceptable filtration rates. Filtrate will report to the barren solution pond.

At the end of the filtration cycle, feed pumps will be shut down, filters drained, and compressed air may be used to further dewater the cake. The filter cake, containing precious metals, will be dropped onto precipitate carts for transfer to the doré room for smelting.

Zinc precipitates from the Merrill Crowe circuit will be loaded into a mercury retort for removal of mercury and further treated by smelting into gold–silver doré. The smelting process will be performed in batch mode.

The smelting circuit will be a vendor package, and the main equipment will include:

- electric retort and adsorption skid;
- induction furnace;
- flux dosing and flux mixer system;
- gold–silver doré safe;
- mechanized slag handling; and

- associated material handling and other systems (molds, dryers, dust collection system).

17.4.6 SART

A sulphidization, acidification, recycle and thickening process (SART) will be installed in the second year of operation. The SART unit operation will treat a portion of the barren gold leach solution before it is recycled to the permanent cyanide leach pad. The SART process will reduce the copper load in the leach solution and regenerate cyanide which is bound to the dissolved copper thus reducing overall cyanide consumption and providing revenue from the corresponding copper sulphide precipitate.

17.4.7 Reagents

Package plants will be provided to supply the following reagents required for the process:

17.4.7.1 Lime

The quicklime slaking system is a proprietary slaking system comprising storage silo, feeder slaking mill and hydro cyclone, agitate storage tank and distribution pumps.

Quicklime is delivered to site in isotainers and unloaded into a storage silo at the lime slaking plant. Quicklime is transferred from the silo at a controlled rate via a rotary vale and screw feeder and fed to a lime slaker. Quicklime is slaked with process water to produce a milk of lime slurry. Mill discharge is pumped to a cyclone to remove grit. Cyclone underflow returns to the mill and overflow at 25% w/w solids gravitates to an agitated storage tank.

The slurry is stored in a tank with a 24-hour residence time and is circulated by dosage pumps.

17.4.7.2 Sulphuric acid

Sulphuric acid is used as a copper leach agent as well as an acidification agent in the SART circuit. The reagent is delivered to the site in a 40-tonne container in liquid form. It is dosed without dilution to the Cu_2S precipitation reactor.

17.4.7.3 Salt

Salt is delivered to the site in a 25 kg bag as a crystalline powder. It is used as an electrolyte modifier.

17.4.7.4 Sodium Cyanide

Sodium Cyanide is delivered to site in secured boxes containing reagent bags. Bags are lifted into the sodium cyanide bag breaker on top of the mixing tank. The solid reagent discharges into the tank and is dissolved in water to achieve the required dosing concentration. After the mixing period is complete, cyanide solution is transferred to the cyanide storage tank using a transfer pump. Sodium cyanide is delivered to the gold leach pad with dedicated dosing pumps. An extraction fan is provided over the sodium cyanide bag breaker/mixing tank to remove reagent dust that may be generated during reagent addition/mixing.

17.4.7.5 Cobalt sulphate

Cobalt sulphate is received on site in solid form. The activator is mixed with water to prepare a solution with 5% by weight in an agitated tank. Cobalt sulphate is an anode stabilizer during the electrowinning process.

17.4.7.6 Organic Solvents

Diluent is delivered to site as a liquid in 1,000 kg tote. Dosing pumps deliver this reagent to the copper solvent extraction stage. Diluent is used as a vehicle to further disperse the extractant organics and optimize viscosity of the organic phase.

Extractant is delivered to site as a liquid in 1,000 kg tote. Dosing pumps deliver this reagent to the copper solvent extraction stage.

17.4.7.7 Smoothing agent

Smoothing Agent is used in the copper electrowinning stage. It is delivered to site as a solid in a 25kg bag. The agent is mixed with water to prepare a solution with 20% by weight in an agitated tank.

17.4.7.8 Zinc Dust

Zinc dust is used to precipitate precious metals in the Merrill-Crowe circuit. It is supplied in 100 kg drums at 99.5% purity. The zinc dust is added via a hoist to the zinc dust hopper and is mixed in the zinc preparation tank with process water. The zinc solution is then dosed into the precipitate filter feed line via a dosing pump.

17.4.7.9 Lead Nitrate

Lead nitrate is used to activate the zinc dust in the Merrill-Crowe circuit. It is supplied in 1000 L intermediate bulk containers ('IBC's) as a 40% w/v solution. It is dosed to the precipitate filter feed tank by a positive displacement dosing pump.

17.4.7.10 Fluxes

Fluxes are used in the smelting process to remove impurities from the filtered and dried precipitate. Borax is the major flux used and it is delivered as a powder, in 25 kg bags. The other fluxes: nitre, silica, and soda ash are also delivered in powder form in 25 kg bags on pallets. Bags of fluxes are transferred to the refinery as required and mixed with the dried precipitate in measured quantities, typically as a ratio of precipitate to be smelted.

17.4.7.11 Sodium Hydrosulphide

Sodium Hydrosulphide is used as a sulphidation agent in the SART circuit. It is mixed to 43% solution strength and dosed to the Cu_2S precipitation reactor.

Detailed breakdown of annual reagent use is provided in Table 21-15 and Table 21-16.

17.4.8 Services

17.4.8.1 Plant and Instrument Air

High pressure air at 650 kPa is produced by compressors to meet plant requirements. The high-pressure air supply is dried and used to satisfy the instrument air demand. Dried air is distributed via the air receivers located throughout the plant.

17.4.8.2 Raw Water Supply

Raw water will be supplied from a ground well and will be stored in a raw water tank and pumped to a distribution piping system. Raw water is used for all purposes requiring clean water with low dissolved solids and low salt content, primarily as follows:

- process plant
- potable water treatment
- fire water for use in sprinkler and hydrant system
- a total of 270 m^3/h of fresh water is required as makeup water for the process plant.

A fire water system will be included with its own tank, electric pump, diesel pump, and jockey pump.

A treatment plant will collect raffinate bleed, storm water runoff, pit water and on/off rinse water. The plant will treat the water and provide treated water to the process water tank. Process water will be supplied to the crushing plant, rinse water plant, SX raffinate pond and barren solution pond.

Further discussion on the water supply is included in Section 18.7

17.4.8.3 Power

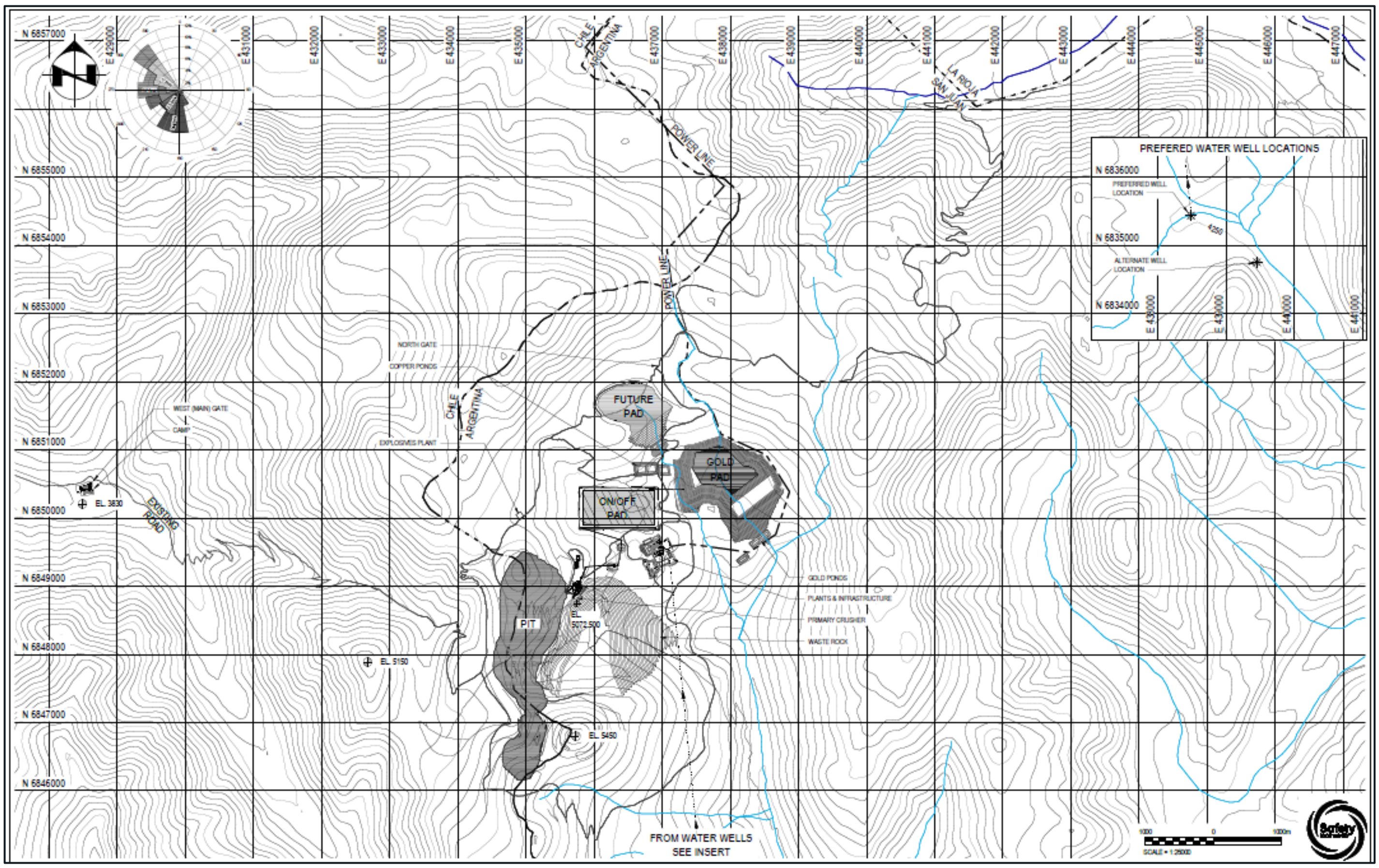
The operating load required for the process plant is 56 MW. The installed power capacity is 75 MW. Further discussion on the power requirements is included in Section 18.5.3.

18 PROJECT INFRASTRUCTURE

18.1 General Site Layout

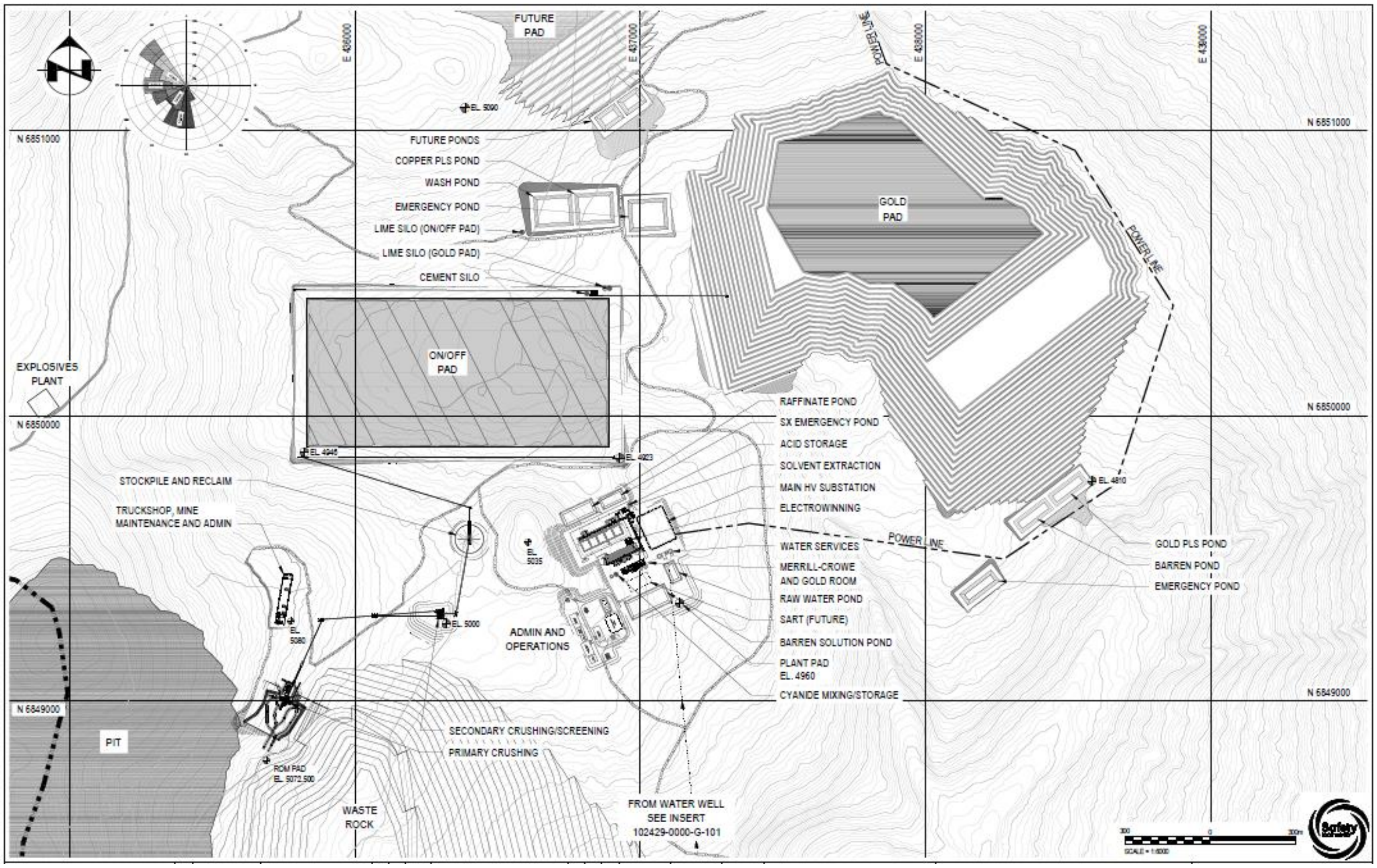
The overall site plan, included as Figure 18-1, shows the general arrangement of the plant, the mine and major infrastructure.

Figure 18-1: Infrastructure Layout Plan



Source: Ausenco, 2019

Figure 18-2: General Arrangement



Source: Ausenco, 2019

18.2 Site-wide Geotechnical Investigation

As part of the design of the heap leach facilities, primary crusher, waste dump facility, and stockpiles, a geotechnical program was carried out. The field program included surface mapping and a test pit program to take samples of soil and rock from plant, leach pads, ponds. primary crusher and waste dump facility sites along with a corresponding laboratory testing program to understand the foundation conditions for these site facilities and material properties of borrow sources. A surface mapping program was also conducted at the aforementioned sites.

The Filo project infrastructure is situated on alluvium and colluvium that is underlain by weathered bedrock. Most of the mine site has permafrost located 0.5 to 1.0 metres below the surface. The design of mine infrastructure took this into account.

18.3 Road and Logistics

18.3.1 Road

The travel distance from the mine site to Puerto Caldera, the nearest port, is approximately 245 km. Approximately 48 km of light vehicle road will require upgrading to a 9-m-wide, two-lane, dirt road to connect the Filo del Sol mine site to the national highway system at Iglesia Colorada. The route continues on C-35, through Nantoco, until it connects with Ruta 5 (Panamericana Norte) in Copiapo. Ruta 5 passes the city of Caldera, which is located approximately 77 km from Copiapo, and accesses Puerto Caldera.

Copper cathodes will be transported by flatbed trucks to Puerto Caldera, and doré will be transported approximately 175 km to the airport (Aeropuerto Desierto de Atacama) for ongoing airfreight.

Operating consumables required by the mine that have foreign supply will be imported to Puerto Caldera. The route to access the mine will be the same used by the cathode shipments.

Roads will connect various mine facilities, including the camp, open pit, truckshop, crushers, process plants, heap leaches, electrical substations, and administrative buildings.

18.3.2 Port

Puerto Caldera was chosen as the preferred option for inbound and outbound requirements, primarily as a result of having the shortest trucking distance from the project site. In addition, the selected port has several suitable existing terminals for the export of product and import of consumables.

18.4 Camps and Accommodation

Due to its remote location, the construction and operations workforce at Filo del Sol will be housed in an accommodation camp. The camp is planned to be located in Chile approximately 6.5 km west of the pit location, at an elevation of approximately 3,800 m amsl, and adjacent to the main site access road. The camp will be built from modular structures with infrastructure for water distribution, sewage treatment, catering, first-aid, and other facilities required for the personnel. The camp will be powered through an overhead power line connection from the main substation and will have a backup diesel generator at its location.

The construction accommodations have been sized based on a preliminary manning schedule showing approximate peak requirements for a 1,000-person camp. As the construction workforce decreases, parts of the camp will be reassigned to operations personnel and for use as operations offices. The construction camp will become the operations camp upon project completion. During operations, it is expected that the camp will accommodate approximately 250-300 persons.

18.5 Power and Electrical

The site will be supplied with electricity through a 127-km-long, 110-kV, single circuit power transmission line connected to the Los Loros substation in Chile. Average electrical demand is estimated to be 56 MW.

18.5.1 Transmission Line

The overhead transmission line will need to be connected to an existing substation in Chile. Argentina was not considered for a connection point as the distance to existing substations was much farther than substations in Chile.

Two substations (Los Loros and Alto del Carmen) were identified within relative proximity to the Filo del Sol site. Indicative transmission line routes were plotted from both substations to the project site and are shown in Figure 18-3 below.

Figure 18-3: Transmission Line Route Options

Source: Ausenco, 2019

The transmission line route from Los Loros to Filo del Sol, referenced as Transmission Line 2, was selected due to its lower estimated capital cost and its relative ease to permit. Its elevation profile is shown in Figure 18-4 below. The length of the route is 127 km.

Figure 18-4: Transmission Profile

Source: Ausenco, 2019

The lower estimated capital cost for the Los Loros option is a result of a 40-km section of the route over relatively low and flat terrain. There is also a 50-km section that will run alongside the existing Caserones transmission line. It has been assumed that existing right-of-way clearing and access for the Caserones line will contribute to lower the cost for this section as well.

Routing the transmission line alongside the already impacted Caserones transmission line right-of-way may is anticipated to contribute positively to the permitting process.

18.5.2 Main Substation

The incoming electrical power from the 110-kV transmission line will be stepped down at the main Substation switchyard to 13.8 kV for in-plant distribution through two 110/13.8-kV step-down transformers.

The prefabricated main substation will house the 13.8-kV distribution switchgear and the controls and protection systems for the high-voltage equipment. The switchgear arrangement provides dual sources of supply to the process plant in the event of loss of one of the incoming transformers.

All required auxiliary services, including emergency generator, electrical room and control room for substation operation, will be housed within the substation perimeter fence. The main substation control and automation system is designed for centralized operation of the substation, with a communication link to the plant-wide Process Control System (PCS). The main control room for the plant will be in the administration and operations building.

18.5.3 Power Distribution

From the 13.8 kV distribution switchgear at the main substation, power will be supplied to all electrical rooms within the plant site through cable trays mounted on structures such as building and conveyor galleries, or via underground duct banks as needed. Overhead power lines will feed distant facilities such as pond pumps, water well pumps, and camp.

Prefabricated electrical rooms have been considered for the various crushing and processing areas.

Variable frequency drives have been allowed where required and will be fed from the main 13.8 kV switchgear location. All medium-voltage motors or drives will be fed from 4.16 kV switchgears, and starters for low-voltage motors will be grouped in motor control centres (MCC), with incoming breakers. The MCC's will be in the electrical room and will include intelligent combination starters, with circuit breakers for instantaneous fault protection.

Rectiformers for the electrowinning plant, which represent the largest single power draw, are located near the electrowinning building.

All critical loads at the process plant will be powered by a three 2 MW emergency diesel generators, and uninterruptible power supply systems will also be located at each electrical room, control room and operator cabin.

18.6 Fuel

Diesel fuel will be delivered to the mine site using tanker trucks. The fuel storage tanks will be single-walled within a lined containment berm. Tank design will comply with appropriate regulatory requirements.

Provisions will be made for fuel storage and dispensing prior to permanent facilities being completed. Fuel for construction will be the responsibility of each individual contractor.

18.7 Water Supply

Water will be supplied from local aquifers in Argentina, located near the proposed plant site. The water makeup requirement is estimated to be 75 L/s based on a 60,000 t/d nominal feed rate.

Knight Piésold has identified locations of three potential water supply sources that are under consideration for the Filo del Sol Project. The locations have been identified based on regional geology and topography, and they range between 14 km to 25 km away (direct) from the plant site. The selected aquifer, directly south of the project site, is approximately 16 km away, with the pipeline following an existing road access and running cross country where possible.

Water will be pumped from the wells to an intermediate fresh water holding tank for distribution to process water, fire water, camp water treatment, and other facilities. The assumption at this phase of the Project is that two wells will be located relatively proximate to each other, at the same aquifer, and will produce sufficient water supply to meet the water demands of the Project. The water supply capacity of the selected aquifer (and alternate aquifers) will need to be tested and confirmed during the next phase of the Project.

One vertical turbine pump and one booster pump station is required to transport the water from the source to the process plant. Due to the relatively high operating pressure, ANSI 600 class flange rating and schedule 60 carbon steel pipe is required. The pipe will be buried for most of its length.

Due to the arid region, water recovery processes will be reviewed and further optimized during the feasibility study.

18.8 Mine Water Management

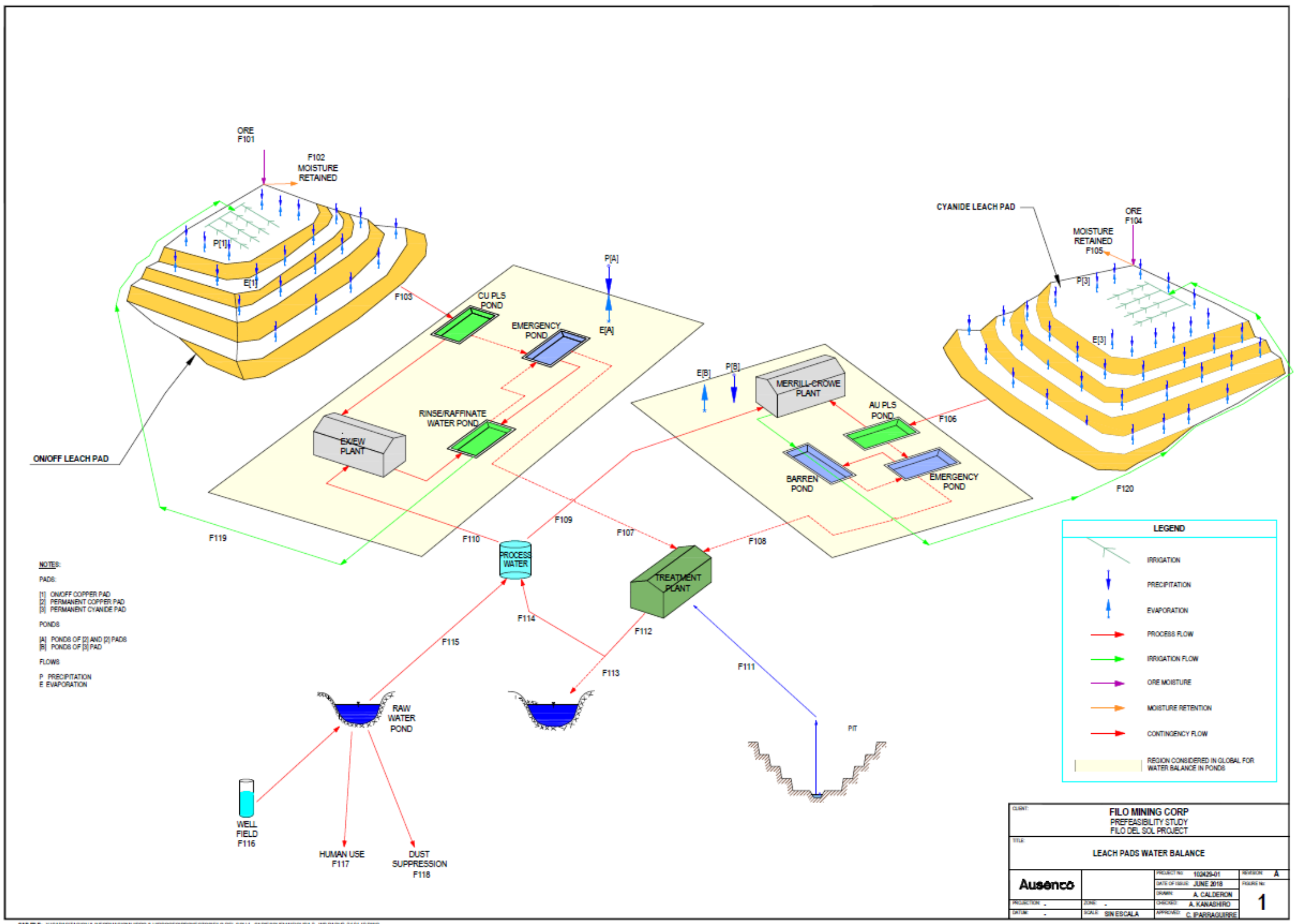
The site water balance was evaluated through a deterministic model developed in GoldSim® on a monthly timestep basis for the life of mine operations. Three precipitation scenarios were considered in the model; dry months, average months, and wet months.

The site water balance study provides a conceptual water management strategy mainly focused on estimating water makeup requirements for the entire operating life of the Project. The model distinguishes between contact and non-contact water flows along with integrating the flows between different mine components such as the open pit, process plant, leach pads, ponds, underdrain sumps, and seepage collection wells.

Available information such as hydrogeology, hydrology, and the climate data were used in the model (PFS level). Production ramp-up, irrigation rates, evaporation losses due to irrigation and ponds, heap stacking plan, and open dewatering were included in the model.

The main components for the site water balance are shown in Figure 18-5 below.

Figure 18-5: Water Balance



Source: Ausenco, 2019

18.9 Surface Water Management

The climate at the project site is classified as sub-tropical semi-arid, typical of the high-altitude Central Andes Mountain Range. This climate is characterized by low precipitation, low relative humidity and high solar radiation, and is notably influenced by cyclical patterns such as the El Nino-Southern Oscillation (ENSO) and the Pacific Decadal Oscillation. The interannual variability of precipitation and air temperature in the Andean Mountain range is highly influenced by the ENSO climate cycles. ENSO cycles are characterized by frequent and intense rainfall events during the warmer El Nino phase, and more dry periods and droughts during the cooler La Nina phase.

During the austral winter, snow accumulates in the Andean Cordillera when cold precipitation fronts arrive from the Pacific Ocean, accounting for 85%-95% of annual precipitation (Masiokas et al., 2016). The frequency and magnitude of the storms determine the amount of precipitation that falls over the Andes, which are then spatially distributed by processes typical of mountainous catchments, such as orographic effects, the preferential deposition and wind redistribution of snow. During the austral summer, the ice and seasonal snowpack accumulated during the winter melt due to drier and warmer conditions, higher incoming solar radiation, and generally lower relative humidity.

18.9.1 Objectives

The main objectives of surface water management at the mine site are summarized below:

- Minimize mine-contact water to prevent this water from entering the receiving environment by surface discharge. This is achieved by routing clean surface water runoff around disturbed areas and minimizing sediment discharge from the site to the environment by entrapping and retaining eroded sediment as close as possible to disturbed areas.
- Provide adequate protection to internal infrastructure and personnel from the uncontrolled effects of surface water runoff during storm events into mining facilities.
- Maximize the internal recycle of contact and process waters in ore processing and thereby minimize the use of external water sources.
- Minimize the generation of sediment due to mining activities and develop structures to capture sediment and prevent it from being released into the environment.
- Achieve environmental compliance.

18.9.2 Proposed Controls

A number of water control structures have been proposed for the surface water management within the project. These structures correspond to standard Best Management Practices which have been adopted for the project. To assure continued performance and functionality all control structures should be inspected regularly.

Control techniques adopted to prevent stormwater damage to facilities, the releases of mine- contact water into the environment and to supply water for process are:

- Recycling water used for processing ore in order to reduce the volume of makeup water demand for process.
- Intercepting and diverting surface water from entering the mine site by building diversion channels structures to reduce the potential for water coming in contacting with exposed ore, and waste and mine facilities.

- Impounding as much ephemeral runoff volume as possible in water retaining ponds to diversion structures for use in operations.
- Collecting contact water from waste rock facility in a sediment collection pond as part of zero release program and utilize in operations.

It will be necessary to alter the current flow path of surface water flows to reduce the potential for harm to infrastructure and/or to minimize the potential for mixing clean water with runoff from disturbed sites.

Surface runoff which can be intercepted and directed by the diversion works will be considered non-contact water. Any water stream runoff that cannot be captured within the area of influence of the project facilities and has the potential for its quality to be adversely affected by project activities will be treated as contact water.

The surface runoff diversion works for the management of non-contact water consist of diversion channels, perimeter channels, sediment ponds, crossing structures, water capture structures, water release structures, and freshwater ponds. These structures have an integrated functionality and have been sited according to the type of water control that is required.

As part of the drainage system for the access roads, longitudinal and transverse drainage has been built into the road design. Longitudinal drainage consists of perimeter channels, which capture surface runoff from the road platform and the basins they transect and direct it to the nearest discharge points; transverse drainage enables the downstream discharge of flows intercepted by the channels, or unimpeded flows in the large stream drainages. Transverse drainage consists of culverts and low-water crossings facilities.

18.10 Heap Leach Pad

Two leach pads will be developed, one for the on/off copper pad and one for the gold permanent pad.

The following section describes the development of the heap leach facilities (HLFs), which includes the leach pads and operations ponds, process plant, access roads and surface water.

18.10.1 Heap Leach Facility Siting Study

During the early stages of the prefeasibility study, Filo Mining requested Ausenco to perform a desktop siting study for the HLFs, excluding other mine facilities that were based on Ausenco's experience in projects with similar types of terrain. The aim was to provide Filo Mining with a better understanding of the HLFs location options to allow development of other facilities based on the siting of the leach pads. Ausenco drew on previous work and BGC Engineering's work on the cryosphere (glaciation, periglaciation, and permafrost) for the project. Facilities needed to avoid glaciers due to Argentinean regulations.

Based on the siting study and the requirement for two heap leach facilities, Ausenco identified the best locations based on proposed leaching operations. The copper on/off pad was located northeast of the open pit in a flat section of the upper end of the Mogotes River watershed and the permanent gold leach pad was location east of the on/off pad. The operations pond for each facility was located immediately down slope of each pad. The process plants (SX/EW and Merrill-Crowe) are located south of the on/off pad.

18.10.2 Heap Leach Process Plants and Ponds

The heap leach process plants and ponds are described in Section 17 Recovery Methods.

18.10.3 Heap Leach Pads

The heap leach pad consists of an underdrain (which doubles as the leak detection system), stormwater diversion channels, platforms, irrigation system, pad liner systems and solution collection systems to collect and convey the pregnant solution to the process plants, and the HLF ponds.

The pads will be located in a small watershed with mountainous terrain that has slopes ranging from 1 to 45%. The ultimate footprint of the on/off copper leach pad will be approximately 578,000 m^2 and the permanent gold leach pad will be 1,551,000 m^2. The following sections outline the general design features for each of the main components of the heap leach pads. Prefeasibility level drawings have been developed for the project to develop the material take-offs for the HLF.

18.10.4 Foundation

The development of the two leach pads requires the preparation of the foundation. Foundation preparation entails the stripping of loose surface soils. An average of 1 m excavation and replacement of soil was considered due to permafrost, and deeper in the toe of the gold pad and ponds for stability. Any unsuitable material will be transported to the waste dump facility.

Through earthworks for the two pads, a minimum drainage slope of 1 percent will be graded towards the low spot, solution exit point, of the pads. The on/off pad will be built from pre-stripping material, and the majority will be used as backfill. The backfill placement and compaction will be in thin layers to guarantee structural integrity. For the gold permanent pad, removal of permafrost is required in the platform area at the lowest point of the heap leach pad where the stability toe buttress is located. This is constructed from engineered fill material.

18.10.5 Heap Leach Liner Design

A composite liner system has been developed for the leach pads. The liner system consists of the following components:

- Overliner (38 mm minus with less than 10% fines content)
- The liner system consists of Low Linear Density Polyethylene (LLDPE) for the bottom of the heap leach pad and High-Density Polyethylene (HDPE) geomembrane on the UV exposed areas and ponds.
- To protect the geomembrane from puncturing due to exposed rock, a layer of geotextile will be applied to the underlining compacted surface, where required.

The LLDPE was selected for the main geomembrane liner systems for the heap leach pads as it has the following benefits (Lupo and Morrison, 2005):

- Generally higher interface friction values compared to other geomembrane materials;
- Good performance under high confining stresses (large heap height); and
- Higher allowable strain for projects where moderate settlement may become an issue.

Laboratory direct shear testing is recommended during the feasibility design to determine the interface shear strength of the liner materials and to confirm strengths are sufficient to provide long-term stability of the HLFs. Representative samples of the geomembrane materials should be used for testing and should be provided by the project supplier.

For ponds containing liquids with acid or cyanide, a double liner layer has been designed with a leak detection system.

18.10.6 Construction

The heap leach liner system for the on/off copper pad will be constructed in its entirety during mine construction because of the nature of on/off pad operations. The permanent gold pad will be constructed in stages, with liner expansions based on the ore stacking requirements. The liner system will be constructed with both the liner and natural low permeability soil extending to the confining limits for each stage to provide full containment. The geomembrane will be anchored in a trench in the ground and backfilled along the perimeter of the HLFs to ensure that ore loading does not compromise the liner's coverage of the leach pad footprint by dragging the liner into the pad. Along the staged expansions of the gold pad, all liners will be tied into their corresponding liner system along the foundation of the pad to provide a continuous sealed liner system along with connecting the solution collection system.

A small perimeter berm will also be constructed as part of the liner tie-in around the perimeter of both leach pad footprints to ensure that HLF solution is contained within the pad footprint and to also prevent surface runoff from the adjacent slopes entering the pads collection systems.

18.10.7 Overliner

A protective layer approximately 1.0 m thick of coarse crushed ore, screened waste rock or quarried rock will be placed over the entire liner system footprints to protect the liner's integrity from damage during ore placement (both pads) and offloading (on/off pad only). The overliner will also double as a drainage layer, promoting leachate solution drainage into the solution collection systems, therefore reducing phreatic head loading on the liner and maximizing solution recovery.

18.10.8 Solution Collection Systems

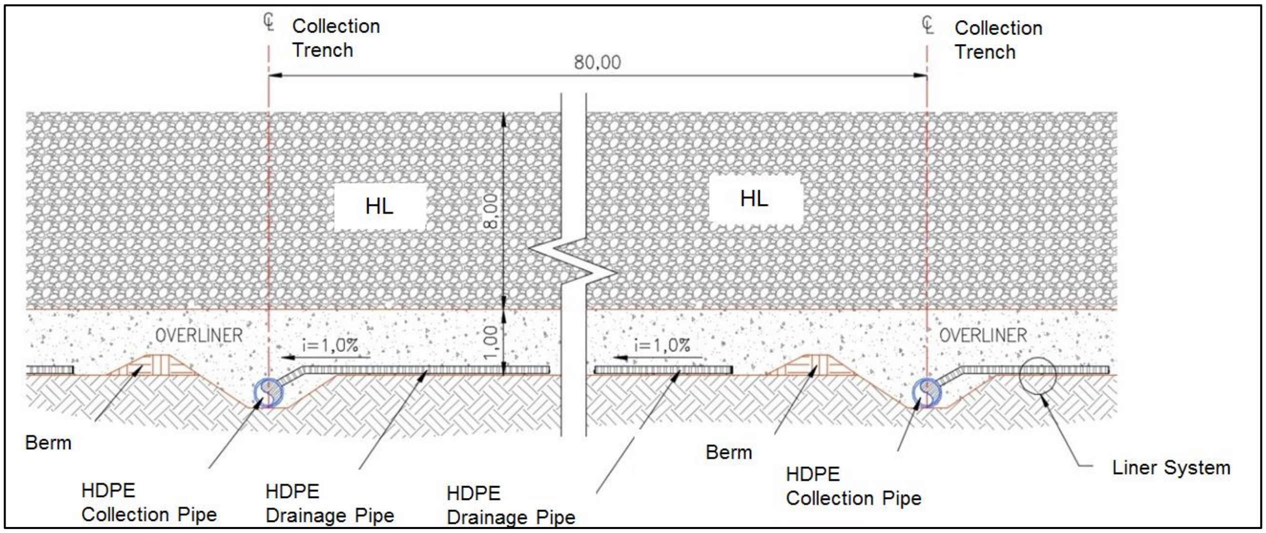
Solution collection and recovery of the pregnant solution in both pad will be undertaken by the solution collection system which will work in conjunction with the heap liner and overliner (refer to Figure 18-6 and Figure 18-7). The collection system will consist of the following pipe and sump components:

- Liner system
- Drainage pipes
- Collection pipes
- Pregnant Leach Solution (PLS) pond

The drainage and collection pipes were estimated using an irrigation rate of 12 L/h/m² considering a design safety factor of 15% and a maximum flow depth inside the pipes of 60% of the diameter. This was done to ensure solution transportation in the event of pipe flattening over time, due to the stack of ore on the pads.

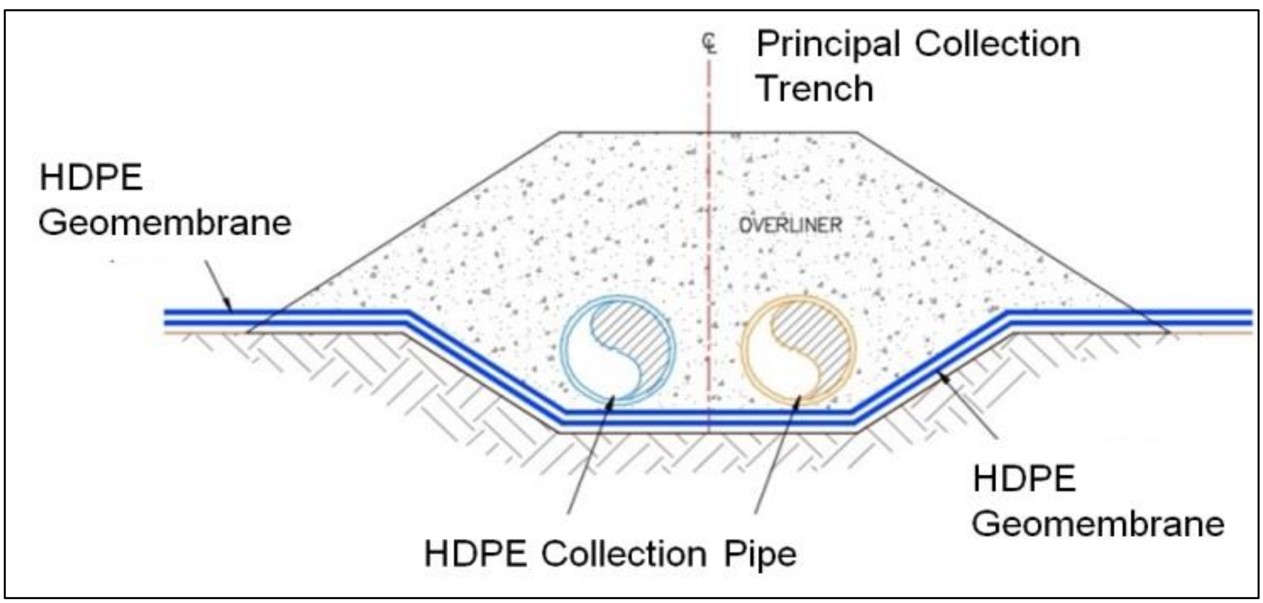
The drainage and collection pipes specified for this project are HDPE pipes; the HLF interior pipes will be corrugated double-wall and slotted pipes. The external pipes will be non-corrugated HDPE PE100 PN6 pipes. Solution captured by the solution collections systems for both pads will be conveyed to the appropriate ponds for conveyance to their respective process plants for metals extraction.

Figure 18-6: HLF Drainage System Detail



Source: Ausenco, 2019

Figure 18-7: HLF Collection System Detail



Source: Ausenco, 2019

The HLF external collection pipes will be covered with overliner material in consideration of thermal isolation and protection of the liner from the traffic of stacking equipment.

This study does not consider interlift liner or pipes. This assumption requires confirmation in the next engineering stage. The material take-off and capital cost estimates only consider liners and pipes at the base lift of each leach pad.

18.10.9 Leak Detection and Recovery System

The Leak Detection and Recovery System (LDRS) is part of the underdrain system and is designed to capture and convey any solution which leaks through the overlying geomembrane, low permeability soil layers and platform as part of the underdrain system that captures near surface groundwater.

The LDRS and underdrain is a network of drainage 'trenches' which contain perforated dual wall HDPE pipes surrounded by drainage gravel. The trenches are aligned underneath the low spots of the base of the valleys. The underdrain discharges into the underdrain pond. The water draining from the underdrain pipes will be monitored, if the water quality detects constituents of concern that exceed water quality standards, then the water will be diverted to the stormwater pond for reclaim. The water will continue to be tested, and if the results fall below water quality standards, then the flow will be discharged back into the environment.

19 MARKET STUDIES AND CONTRACTS

The principal planned products are copper cathode and gold/silver doré.

A small quantity of copper sulphide precipitate as generated from the SART process will also be produced.

19.1 Market Studies

No specific marketing study was conducted for the study. Copper cathode and gold/silver doré are readily traded commodities. Accordingly, for the purposes of the PFS, it is appropriate to assume that the products can be sold freely and at standard market rates.

19.2 Commodity Price Projections

Pricing of the products is shown in Table 19-1; these values were used in the economic analysis. These prices are in accordance with consensus market forecasts and are consistent with historic prices for these commodities (see Figure 19-1, Figure 19-2 and Figure 19-3). Ausenco also considers the prices used in this study to be consistent with the range of prices being used for other project studies.

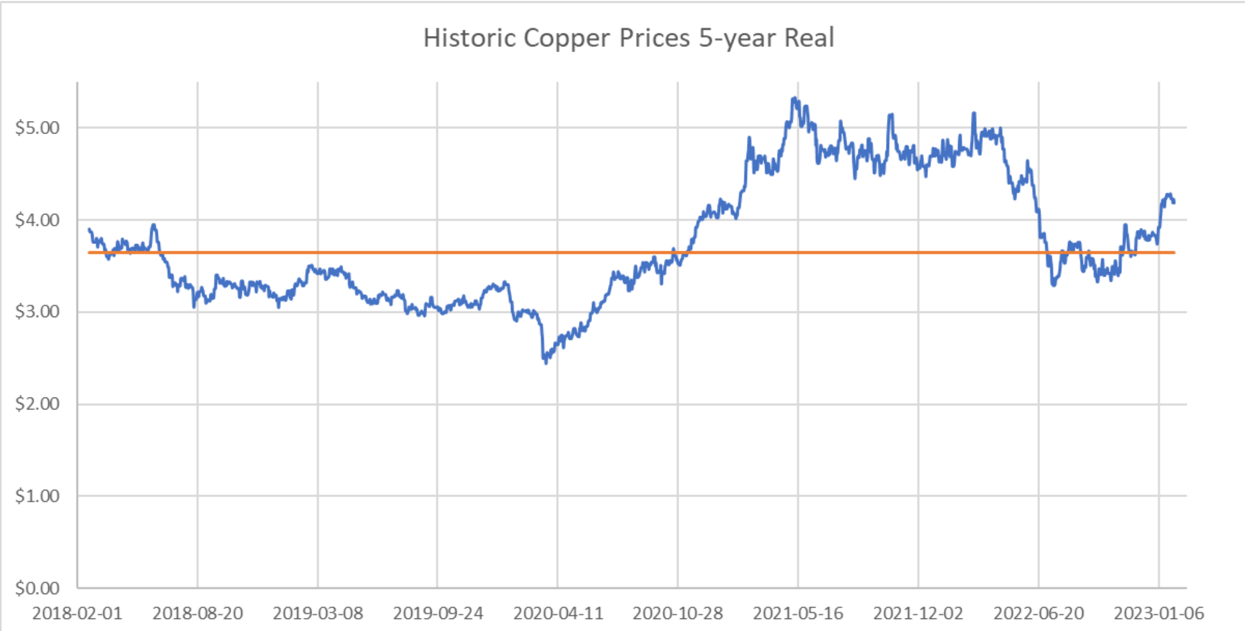
Note that the copper price excludes a 2.0% premium for cathode product. Cathode is expected to be LME Grade A, which conforms to the chemical composition of one of the following standards:

- BS EN 1978:1998 - Cu-CATH-1
- GB/T 467-2010 - Cu-CATH-1
- ASTM B115-10 - cathode Grade 1.

Table 19-1: Pricing Assumptions for Economic Analysis

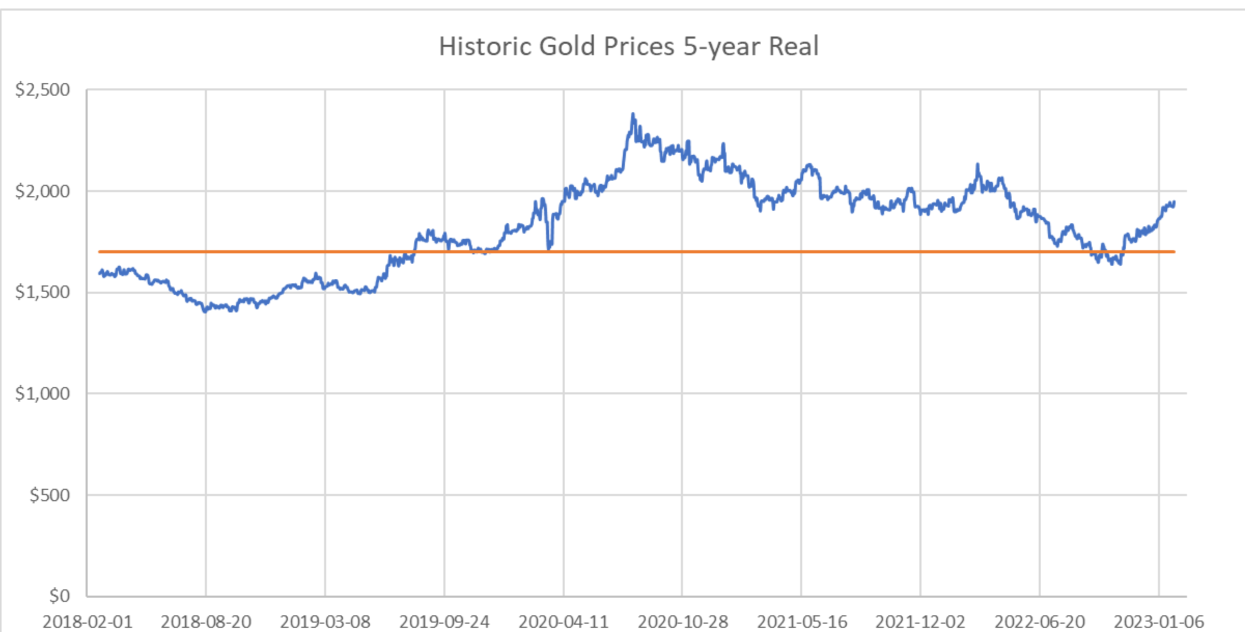
Commodity	Price
Copper (Cu)	$3.65 per pound (lb)
Gold (Au)	$1700 per ounce (oz)
Silver (Ag)	$21.00 per ounce (oz)

Figure 19-1: Historic Copper Prices



Source: SRK, 2023

Figure 19-2: Historic Gold Prices



Source: SRK, 2023

Figure 19-3: Historic Silver Prices

Source: SRK, 2023

19.2.1 Copper Precipitate

Copper is recovered in the SART process, as a high-grade copper sulphide precipitate. Key assumptions for the sale of the precipitate are similar to a traditional copper concentrate and are summarized in Table 19-2 below.

Table 19-2: Copper Concentrate Terms

	Units	Value
Copper grade	%	65
Moisture content	% w/w	8
Concentrate payability	% of contained	96.5
Freight charges	$/wmt	126.45
Treatment charges	$/dmt	75
Losses	%	0.25
Copper refining charges	$/lb Cu	0.075
Penalties	$/dmt	None Modelled

No deleterious elements are expected to be produced in quantities which would result in material selling penalties. The precipitate is to be trucked to a concentrate export port of Caldera on the Chilean coast and exported to smelters in Asia.

19.3 Comments on Market Studies and Contract

The QP is of the opinion that the marketing and commodity price information is suitable to be used in cashflow analysis to support the Filo del Sol Prefeasibility Study.

19.4 Contracts

The Company has no contracts in place.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

Filo Mining has made considerable efforts to undertake environmental studies and community engagement to advance the Filo del Sol Project (the Project). The following presents a brief summary of the environmental aspects, permitting and social or community impacts of the work program to date.

20.2 Permitting

The Project is substantially located within Argentina, however infrastructure including a portion of the pit, the explosives magazine, personnel camp, electrical transmission line, and transport corridor will be located in Chile. Accordingly, permits from both Argentina and Chile will be required.

20.2.1 Argentine Permitting Process

The legal framework for mine permitting in Argentina is derived mainly from the second section of the Mining Code and its supporting National Law No. 24.585, along with the General Environment Law 25.675. The institutional framework for the permitting process is driven by stipulations in Law No. 24.585, with technical Support of the National Mining Secretariat who is advised in turn by the National Unit of Environmental Management.

The Law dictates that an “Informe de Impacto Ambiental” or Environmental Impact Assessment (EIA) must be submitted prior to commencement of operations. Upon successful review of the EIA, authorities issue a “Declaración de Impacto Ambiental” (DIA), which serves as the overarching environmental license. Annex III of Law 24.585 establishes the minimum contents of an EIA, which must include:

- Description of the Environment (physical, biological, and socio-economic)
- Project Description
- Description of Environmental Impacts
- Environmental Management Plan (which includes measures and actions to prevent and mitigate environmental impact)
- Plan of Action on Environmental Contingencies
- Methodology Used

The complementary Law 6571 from San Juan Province has similar requirements, which are accommodated at the same time as the federal EIA.

An EIA and its subsequent DIA are required for the exploration phases of mineral development also. The Filo del Sol project has maintained all previous exploration activity permits in good standing, each of which required the submission of an EIA and receipt of a DIA. The most recent DIA was issued on 23 March 2022 and is valid for two years, whereupon it can be renewed.

In addition to the DIA, a number of permits, licenses and authorizations will be required to proceed with the construction and operation of the project. Most of these are similar to those already in possession of the project as part of exploration requirements; however, they will have to be expanded, renewed, and tied to the exploitation DIA.

Primary permits include:

- Certificate of Hazardous Waste management
- Registration as consumer of liquid fuels
- Certificate of Non-Existence of Archaeological and Paleontology Remains
- Registration as explosives user

20.2.2 Chilean Permitting Process

In Chile, mineral development triggers the requirement for an Environmental Impact Assessment (Estudio de Impacto Ambiental or EIA). The steps that are included in the process are listed below.

- Develop the EIA, including numerical predictive modelling, as well as social assessment, management plans, and risk assessment, mitigation plans, emergency response plans, and summary tables.
- Community consultation is required for input to the social assessment, in the form of community meetings and open houses.
- Submission of the EIA to the Servicio de Evaluación Ambiental (SEA). This is done electronically, and it is assessed for completeness and admissibility prior to having a Summary posted online.
- Public Input is received during a 60 working day period within the review timeline. Open houses are conducted by the proponent as stipulated by the SEA during this period.
- The SEA usually emits a request for additional information called "Informe Consolidado de Solicitud de Aclaraciones, Rectificaciones y Ampliaciones (ICSARA)." The proponent must then undertake the necessary study and analysis to respond to the ICSARA.
- The SEA has 15 working days to evaluate the adequacy of the EIA addendum submitted in response to the ICSARA, and a further 15 working days to publish a summary of the addendum online, called an Informe Consolidado de Evaluación (ICE).
- The SEA may seek further information even after the addendum is submitted, through a second ICSARA.
- Upon completion of the review and cessation of further ICSARA, an authorization is issued called the Resolución Calificación Ambiental (RCA).

Subsequent to the RCA being issued, the proponent summarizes all the mitigation, compensation, and other relevant project commitments in a Table of Commitments (Matriz de Compromisos), which is posted online by the SEA.

Once the RCA is issued, the proponent can seek individual permits for construction and operation. The most significant of these are the water licences from the Dirección General de Aguas (DGA) and the mining license from the Servicio Nacional de Geología y Minería (SERNAGEOMIN). Each of these can be initiated during the EIA review period, however they cannot be granted until the EIA review concludes with a favourable decision.

Similar to requirements in Argentina, an RCA is also required for the exploration phase. The Filo del Sol project has maintained all previous exploration activity permits in good standing, the most recent RCA was issued on 22 August 2019 and is valid until 2023, whereupon it can be renewed.

20.3 Environmental Studies

A summary of the results of the environmental studies conducted to date is provided below.

20.3.1 Meteorology

Site-specific meteorological studies have been conducted for the project (Knight Piésold, 2018a, BGC, 2015a). A meteorological station was installed at the Filo del Sol project in January 2015, located at an elevation of 5,012m amsl. Additionally, two other climate stations were installed located close to the project, at the neighbouring Los Helados and Josemaría projects. The Los Helados climate station is located at an elevation of 4,974m amsl and was installed in late January 2015. The Josemaría climate station is located at an elevation of 4,448m amsl and was also installed in late January 2015.

All three stations collected air temperature, precipitation, wind speed and wind direction, relative humidity, snowpack depth, albedo, and solar radiation data. Information on snow cover conditions is also collected using an acoustic distance sensor. The assessment of meteorological conditions in the Project area is primarily derived from the three-year (2015 – 2017) record collected at the Filo del Sol climate station and is supported by data collected at the other two stations. In particular, climate data from the Los Helados station were used to fill in gaps of missing temperature and precipitation data at the Filo del Sol climate station.

There are several climate stations managed by Dirección General de Aguas (DGA) in Chile, as well as Servicio Meteorologico Nacional (SMN) and Instituto Nacional de Tecnología Agropecuaria (INTA) in Argentina, that either are operating or have operated in the vicinity of the Project area. All the regional stations are located at elevations at least 2,000 m lower than the project, and as such, record considerably different climate conditions. However, the regional climate data are well correlated with the Project data, and it is on this basis that long-term climate values were generated. Climate data from the Lautaro Embalse climate station operated by DGA were used to develop long-term synthetic estimates of temperature and precipitation for the Filo del Sol climate station. The Lautaro Embalse climate station is located approximately 65 km northwest of the Project at an elevation of 1,110m amsl.

20.3.1.1 Temperature

Mean, minimum, and maximum temperatures were available at the Filo del Sol station on an hourly basis from January 2015 to November 2021, except for period from December 2017 to January 2018. The mean annual temperature for the Project area was -5.19 °C for the period. For the same period, the maximum and minimum hourly air temperatures were 12.4 °C and -25.6 °C, respectively.

Mean, minimum, and maximum temperatures were available at the Filo del Sol station (Vicuña 4) located in Chile on an hourly and daily basis from March 2019 to March 2022. The mean annual temperature registered at this station for the Project area is -6.01 °C. For the same period, the maximum and minimum air temperatures were 11.5 °C (01/16/2022) and -26.04 °C (07/25/2020), respectively.

In order to develop a synthesized long-term temperature record for the Project, concurrent temperature data for the Filo del Sol climate station and the most representative regional climate stations were analysed to assess the suitability of the regional climate stations as predictors of climatic conditions site. The synthetic long-term monthly temperature series is summarized in Table 20-1. Based on this series, the long-term mean annual temperature is estimated to be 7.3 °C, with monthly mean temperatures ranging from a high of 3 °C in January 2015 to a low of -26.4 °C in June 1978.

Table 20-1: Monthly Mean and Annual Mean Temperature (°c) with Synthesized Data Set

Year	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
1973	-5.0	-5.8	-6.1	-9.4	-11.6	-15.5	-17.5	-12.6	-7.0	-5.8	-2.2	-2.5	-8.4
1974	-1.4	-3.0	-3.7	-7.1	-8.8	-14.0	-12.0	-10.3	-12.3	-7.6	-4.0	-4.0	-7.4
1975	-1.7	-2.2	-4.3	-6.3	-9.8	-10.1	-12.9	-13.3	-11.8	-7.4	-7.6	-4.1	-7.6
1976	-2.4	-3.7	-3.7	-7.4	-10.0	-15.2	-11.7	-13.7	-10.0	-6.8	-4.7	-1.1	-7.5
1977	-0.6	-7.9	-1.4	-5.6	-7.5	-9.4	-14.0	-11.2	-7.7	-7.0	-3.9	-0.8	-6.4
1978	-1.2	-3.0	-4.3	-7.0	-11.1	-26.4	-15.9	-12.2	-11.1	-5.0	-3.2	-2.1	-8.5
1979	-1.8	-3.1	-6.2	-7.2	-8.2	-11.7	-10.3	-5.7	-8.8	-3.7	-3.0	0.6	-5.7
1980	0.8	-2.0	-0.6	-5.7	-8.1	-11.8	-11.7	-9.4	-8.0	-8.7	-4.4	-1.4	-5.9
1981	-1.3	0.2	0.6	-6.4	-8.0	-9.9	-10.3	-8.1	-6.4	-5.7	-3.4	-0.6	-5.0
1982	-2.5	-2.5	-4.8	-6.8	-10.3	-12.0	-8.7	-6.6	-8.6	-5.3	-3.4	-2.9	-6.2
1983	-0.9	-0.6	-1.7	-4.1	-11.1	-16.9	-14.9	-11.8	-12.2	-4.8	-4.2	-2.7	-7.2
1984	-2.2	-2.0	-3.2	-6.2	-11.3	-15.1	-12.2	-12.9	-9.4	-6.4	-5.7	-3.4	-7.5
1985	-4.0	-2.4	-3.8	-6.5	-8.5	-8.6	-15.1	-12.2	-9.2	-5.5	-5.2	-3.8	-7.1
1986	-2.4	-3.3	-4.0	-6.1	-9.2	-11.7	-7.7	-9.3	-7.7	-5.6	-3.9	-1.2	-6.0
1987	-1.1	0.0	-3.2	-4.9	-13.2	-8.2	-16.3	-10.7	-8.5	-6.3	-3.7	-2.1	-6.5
1988	-2.1	-1.2	-2.2	-4.9	-8.8	-12.3	-14.5	-9.7	-12.5	-6.3	-4.8	-4.5	-7.0
1989	-3.0	-1.8	-4.6	-8.3	-9.5	-11.1	-13.0	-9.7	-12.4	-6.0	-5.1	-3.5	-7.3
1990	-1.7	-2.7	-3.6	-7.6	-8.4	-8.4	-12.0	-10.0	-9.1	-8.7	-5.0	-4.3	-6.8
1991	-3.7	-3.1	-3.7	-5.6	-7.3	-11.7	-11.8	-12.2	-8.0	-8.3	-4.3	-4.8	-7.1
1992	-1.5	-2.5	-4.0	-7.8	-10.3	-12.9	-11.5	-10.7	-9.3	-6.2	-5.0	-3.6	-7.1
1993	-2.5	-3.6	-4.1	-6.0	-10.3	-9.5	-13.2	-9.2	-11.3	-8.0	-5.9	-4.4	-7.3
1994	-3.2	-3.3	-4.5	-5.5	-8.8	-10.9	-13.4	-10.1	-6.0	-8.6	-6.6	-5.3	-7.2
1995	-5.4	-4.9	-5.6	-7.5	-8.7	-9.4	-14.9	-12.8	-10.1	-8.5	-6.7	-5.3	-8.3
1996	-5.7	-4.1	-5.8	-11.2	-11.5	-13.0	-10.7	-12.1	-9.1	-8.1	-5.3	-6.6	-8.6
1997	-3.2	-2.5	-4.4	-7.3	-9.2	-14.9	-10.1	-8.9	-7.4	-8.4	-41.1	-2.1	-9.9
1998	0.4	-1.3	-3.1	-8.0	-9.8	-11.8	-10.9	-12.7	-12.0	-7.6	-6.7	-5.1	-7.4
1999	-4.9	-2.1	-4.6	-8.1	-10.2	-12.5	-13.6	-10.5	-10.5	-9.5	-7.8	-5.7	-8.3
2000	-4.0	-3.7	-5.3	-9.2	-11.5	-14.2	-11.7	-11.0	-10.9	-7.0	-7.3	-4.1	-8.3
2001	-5.3	-3.5	-5.6	-8.3	-12.8	-13.4	-10.1	-9.4	-12.2	-7.3	-7.1	-3.5	-8.2
2002	-2.4	-1.5	-1.5	-7.3	-9.0	-11.7	-10.1	-7.7	-8.2	-6.7	-5.3	-5.3	-6.4
2003	-3.0	-2.1	-4.0	-7.5	-6.8	-10.4	-10.8	-8.3	-8.8	-6.3	-6.7	-5.7	-6.7
2004	-6.1	-6.1	-6.9	-10.1	-13.1	-10.4	-10.6	-12.7	-8.0	-9.2	-7.3	-6.7	-8.9
2005	-7.6	-7.0	-8.5	-8.1	-14.7	-8.6	-12.5	-7.0	-11.0	-8.7	-4.7	-4.1	-8.6
2006	-1.7	-1.0	-3.2	-6.2	-7.5	-8.6	-8.9	-8.0	-8.3	-7.7	-6.3	-4.7	-6.0
2007	-5.0	-6.0	-7.4	-8.2	-14.7	-17.1	-13.3	-18.1	-11.5	-10.6	-9.3	-7.2	-10.7
2008	-5.0	-5.9	-5.3	-9.4	-11.7	-13.3	-13.5	-11.5	-11.3	-10.1	-8.6	-4.0	-9.1
2009	-2.5	-1.5	-3.3	-5.3	-8.4	-10.2	-13.0	-8.5	-10.7	-6.6	-4.7	-3.4	-6.5
2010	-2.0	-2.4	-4.0	-6.5	-10.6	-13.5	-16.1	-9.6	-9.4	-8.6	-6.9	-6.1	-8.0
2011	-4.0	-2.9	-5.8	-7.9	-8.9	-14.9	-14.8	-13.2	-7.4	-11.6	-6.2	-4.5	-8.5
2012	-3.6	-2.0	-2.1	-8.0	-8.6	-11.1	-12.8	-13.1	-8.7	-8.9	-6.4	-4.5	-7.5
2013	-2.2	-2.6	-3.5	-7.7	-11.9	-11.6	-10.9	-9.7	-9.7	-7.7	-6.9	-4.7	-7.4
2014	-3.2	-3.2	-5.9	-7.7	-11.9	-13.7	-11.3	-8.5	-10.7	-4.9	-6.4	-4.5	-7.7
2015	3.0	-0.3	-0.5	-4.6	-9.6	-7.2	-11.0	-10.0	-9.3	-8.7	-7.3	-2.7	-5.7
2016	0.1	1.1	-1.4	-5.7	-8.9	-11.0	-10.7	-7.9	-4.9	-7.0	-4.5	-1.5	-5.2
2017	2.1	-2.0	-2.2	-5.0	-10.4	-9.7	-7.0	-9.9	-7.4	-7.5	-4.2	-0.9	-5.3
Mean	-2.6	-2.8	-3.9	-7.1	-10.0	-12.1	-12.2	-10.5	-9.4	-7.3	-6.3	-3.7	-7.3
Minimum	-7.6	-7.9	-8.5	-11.2	-14.7	-26.4	-16.3	-18.1	-12.5	-11.6	-41.1	-7.2	-10.7
Maximum	3.0	1.1	0.6	-4.1	-6.8	-7.2	-7.0	-5.7	-4.9	-3.7	-3.0	0.6	-5.0

20.3.1.2 Wind

The mean annual wind speed calculated from the three years of record at the site is 5.7 m/s. An average monthly low of 2.4 m/s was measured at Filo del Sol in January 2017, whereas an average monthly high of 8.5 m/s was measured in May 2017. The wind was calm (less than 1 m/s) for only 1.5% of the time, while wind speeds exceeded 10 m/s approximately 10% of the time. Winds are just as likely to occur at any time of day, and wind speeds are fairly consistent throughout the day. The prevailing wind direction throughout all seasons is northwest, with some strong gusts from the north-northwest. The site is consistently windy, both in terms of the frequency and the intensity of the wind. The maximum instantaneous wind speed measured was 19.2 m/s. Wind speeds are typically higher at the Filo del Sol climate station than the Los Helados and Josemaria climate stations, due to its greater elevation and exposure. The monthly mean wind speeds at the three stations are typically greatest during the coldest winter months (May to October) and lowest during the warmest summer months (December to March).

20.3.1.3 Evaporation

Monthly Potential Evapotranspiration (PET) values were estimated for Filo del Sol using three commonly applied empirical relationships, which are Hargreaves (Maidment, 1993), Thornthwaite (Thornthwaite, 1948), and Penman-Monteith (Smith et al., 1998). Values are provided in Table 20-2. The mean annual PET at the Project was calculated to be 242 mm, with an average monthly low of 4 mm during the month of July, and an average monthly high of 72 mm during the month of January.

Table 20-2: Estimated Mean Monthly Potential Evapotranspiration

Elevation	Method		Evapotranspiration (mm)												
		Year	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
5012 m	Hargreaves equation	2015	-	53	51	29	11	9	2	6	9	6	14	55	245
		2016	69	65	49	22	4	0	0	12	35	18	34	50	358
		2017	64	46	49	31	7	3	3	4	16	17	36	56	332
		Mean	66	55	49	27	8	4	2	7	20	14	28	53	334
	Thornthwaite equation	2015	-	0	0	0	0	0	0	0	0	0	0	0	0
		2016	85	163	0	0	0	0	0	0	0	0	0	0	248
		2017	160	0	0	0	0	0	0	0	0	0	0	0	160
		Mean	123	54	0	0	0	0	0	0	0	0	0	0	177
	Penman-Monteith equation	2015	-	22	21	16	12	13	10	11	13	16	20	26	179
		2016	28	24	22	14	10	8	10	14	19	18	22	26	214
		2017	28	21	21	16	15	10	13	13	16	19	22	-	195
		Mean	28	22	22	15	12	10	11	13	16	18	21	26	214
	Average of All 3 Methods		**72**	**44**	**24**	**14**	**7**	**5**	**4**	**7**	**12**	**11**	**16**	**27**	**242**

20.3.1.4 Precipitation

Precipitation at Filo del Sol is an infrequent occurrence, with little precipitation occurring in dry years, and only a few strong precipitation events providing the majority of the total rainfall in wet years. Precipitation data are available for the Josemaría, Filo del Sol, and Los Helados climate stations. The precipitation record for the Filo del Sol station demonstrates an average annual value of the three-year precipitation record of 338 mm.

The estimated long-term monthly precipitation series is presented in Table 20 3. Precipitation varies dramatically from year to year, with a mean annual value of 131 mm, annual values ranging from a low of 0 mm to a high of 738 mm, and monthly values ranging from 0 mm (many occurrences) to 381 mm (June 1997). These values are consistent with mean monthly and mean annual precipitation values recorded at the nearby Pascua Lama mine (Arcadis Geotecnica, 2004). Precipitation is generally greatest during the austral winter (May through to August) and very low for the rest of the year.

Table 20-3: Long-time Synthetic Monthly and Annual Total Precipitation (mm)

Year	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
1967	0	0	0	0	0	0	0	0	0	0	0	11	11
1968	0	0	0	0	0	0	0	0	0	0	0	0	0
1969	0	0	0	0	0	1	0	75	0	0	0	0	76
1970	0	0	0	0	2	0	0	0	0	0	0	0	2
1971	31	0	0	0	0	79	0	14	0	0	0	0	124
1972	0	0	0	0	0	0	0	0	0	0	0	0	0
1973	0	0	0	7	0	56	0	0	0	0	0	0	63
1974	0	0	0	0	0	0	0	0	29	0	0	0	29
1975	0	0	25	0	72	43	0	0	0	0	0	0	140
1976	0	0	0	0	57	0	0	16	4	0	0	0	77
1977	0	0	0	56	0	0	43	0	0	0	0	0	99
1978	0	0	0	0	0	0	0	0	0	0	0	0	0
1979	0	0	7	0	0	0	0	0	9	0	0	0	16
1980	0	0	0	79	0	0	68	25	31	14	5	0	223
1981	0	0	0	0	0	0	0	72	0	0	0	0	72
1982	0	0	0	0	0	18	0	0	5	0	0	0	23
1983	0	0	0	11	54	216	36	142	0	0	0	0	458
1984	0	0	47	0	0	27	246	0	0	0	0	0	320
1985	0	0	0	0	0	0	29	2	0	0	0	0	31
1986	0	0	0	0	36	0	9	54	0	2	0	0	101
1987	0	5	0	0	32	0	338	72	2	7	0	0	456
1988	0	0	0	0	11	0	0	0	9	0	0	0	20
1989	0	0	0	2	0	0	9	145	0	0	0	0	156
1990	0	0	0	0	0	0	38	0	0	0	0	0	38
1991	0	0	0	0	0	289	38	0	0	0	0	25	352
1992	0	0	31	29	163	99	0	0	0	0	0	0	321
1993	0	59	0	0	0	0	5	40	0	0	0	0	104
1994	0	13	0	0	0	0	0	0	0	0	0	0	13
1995	0	0	0	0	0	0	0	0	4	0	0	0	4
1996	0	0	0	0	0	0	0	5	0	0	0	0	5
1997	0	0	2	0	0	381	0	352	4	0	0	0	738
1998	0	0	0	0	0	20	4	0	0	0	0	0	23
1999	0	0	66	0	0	4	2	0	0	31	0	0	102
2000	0	0	0	2	86	174	20	0	0	0	0	0	282
2001	0	0	27	0	0	0	0	2	0	0	0	0	29
2002	0	0	0	43	72	0	101	93	0	0	0	0	309
2003	0	0	0	0	0	0	0	0	0	0	0	0	0
2004	0	0	16	0	0	0	93	0	0	0	0	0	110
2005	0	0	0	56	0	0	36	22	0	0	0	0	113
2006	0	0	0	0	0	4	0	0	0	0	0	0	4
2007	0	0	0	0	18	5	0	0	0	0	0	0	23
2008	0	0	0	0	16	2	47	0	22	0	0	0	86
2009	0	0	0	0	11	0	38	0	0	0	0	0	48
2010	0	0	0	0	163	0	0	5	41	0	0	0	210
2011	0	0	0	2	0	18	112	0	0	0	0	0	132
2012	0	0	0	54	0	0	0	0	0	0	0	0	54
2013	0	0	0	0	84	0	27	0	0	0	0	0	111
2014	0	0	0	0	84	4	0	0	22	0	0	0	110
2015	5	2	125	2	22	0	103	77	11	45	5	5	401
2016	5	5	4	90	94	4	0	0	0	0	0	18	219
2017	23	29	0	0	126	38	1	10	2	20	2	7	256
Mean	1	2	7	8	24	29	28	24	4	2	0	1	131
Minimum	0	0	0	0	0	0	0	0	0	0	0	0	0
Maximum	31	59	125	90	163	381	338	352	41	45	5	25	738

A set of 24-hour extreme precipitation estimates were completed. Values for the 24-hour extreme events having return periods of 10, 20, 50, 100, and 200 years were estimated as 76 mm, 94 mm, 119 mm, 137 mm, and 155 mm, respectively.

20.3.2 Noise and Vibration

Baseline noise and vibration measurements were carried out in February of 2014 (Métodos Consultores Asociados, 2014a, Métodos Consultores Asociados, 2014b). Ambient noise levels are generally low. Higher decibel readings of up to 53 dBA were associated with strong winds. Outside of the mineral exploration activity, there was no human-caused noise generation. In the baseline condition, ground vibrations were negligible.

20.3.2 Glaciology

The 2010 Federal Argentine Glacier Protection Law (Ley 26.639) is very broad in its definition of "glacier" and includes any perennial ice mass (covered or uncovered) and permafrost. It establishes a National Glacier Inventory, with the objective of protecting "strategic hydrologic reserves". Mining activity is prohibited where it negatively affects glaciers identified in the inventory.

In San Juan, the 2010 Provincial Glacier Protection Law (Ley 8144) provides similar definition of what types of ice masses are protected but does not explicitly prohibit mining activity. A provincial inventory is mandated as part of the law but is in progress and has not yet been published. Activities that destroy, reduce, or interfere in the advance of glaciers are prohibited. An Environmental Assessment is required to determine if a proposed activity will impact the glaciers or permafrost.

Chile does not have a specific glacier law, however general environmental legislation (Leyes 19.300 and 20.417) does require assessment of impact to glaciers for industrial developments, amongst many other environmental components. The Regulation SEIA DS 40/2013 further specifies the studies required for glaciers in an EIA, including their area, thickness, surface reflectance, ice-core characterization, movement assessment, and runoff calculations. The 2009 National Glacier Strategy offers additional considerations for study.

Chile does have a national inventory of glaciers as part of the Water Ministry's series of online mapping tools. There is also an Atacama Regional government's Inventory – the "Inventario de Glaciares, Ambiente Periglacial y otras Reservas Criosfericas de la III región de Atacama y Áreas Binacionales para Determinar nuevas Fuentes de Agua". The National and Atacama Region inventories are recognized as starting points for environmental assessments, with additional site-specific study required to support any given project.

To understand the cryosphere at Filo del Sol appropriately, Filo Mining contracted BGC Ingenieria Ltda. (BGC) to undertake annual glacial and periglacial studies, with the first investigations starting in 2013. Their work has produced a probabilistic permafrost distribution model, and the initiation of a cryosphere monitoring program, including analysis of satellite imagery and ground truthing of glacial and periglacial cryoforms. The cryosphere monitoring program consists of continuous monitoring of weather conditions, ground surface temperatures, ground thermal regimes, and stream flows, together with time-lapse photogrammetry of selected cryoforms.

Careful placement of infrastructure has been considered to avoid direct and indirect impacts to the inventoried glaciers. The EIA for the Project will require assessment of the potential impacts to glaciers in Chile and Argentina, which will incorporate the multi-year cryology study to engage with government and stakeholders.

20.3.3 Hydrology

The Project sits at the upper boundaries of both the Los Mogotes (Argentina) and the Upper Montoso River (Chile) watersheds. The Los Mogotes watershed flows into the Macho Muerto watershed, which ultimately feeds into the Blanco River watershed. The Upper Montoso River feeds into the Montoso River, which in turn feeds into the Pulido River, which is a tributary to the Copiapó River.

A summary of streamflow studies is provided in Knight Piésold (2018a). The mean unit runoff varies substantially in the region, but in general is low; typically below 5 L/s/km^2 for the November to June period. In many streams the maximum and minimum flows differ by as much as an order of magnitude, with high flows resulting from snowmelt due to periods of relatively warm temperatures and high incoming solar radiation, and very low flows occurring during freezing conditions. Streamflows in the project area are highly influenced by snowmelt, with the highest flows usually occurring after big snowfall events between February and May. Inter-annual variability in streamflow records can be largely attributed to El Niño Southern Oscillation (ENSO) climate events.

The Division de Hidrología (DH), a branch of the San Juan Government's Hydraulic Department, operated a streamflow station on the Blanco River, downstream of the Montoso River, from 2001 through to 2015. In addition, the Dirección General de Aguas (DGA) operated eleven streamflow monitoring stations along the Copiapó River and its tributary Pulido River, downstream of the Los Mogotes River, for varying periods over the past few decades. Of these eleven stations, seven are currently active. All of the regional stations mentioned above are in relatively large watersheds that are located at much lower elevations.

Streamflow data collection at the Project site is limited, however, it is reasonable to conclude that unit flows in Los Mogotes River, for the non-winter period of November to May, are likely in the order of 0.5 L/s/km^2 to 1.5 L/s/km^2 (period runoff depth of 16 mm to 48 mm). This low runoff depth is generally consistent with the low precipitation and relatively high evapotranspiration and sublimation conditions estimated for the Project area.

Additional in situ flow monitoring will be implemented during for the summer months of 2018 and 2019 in order to develop a high-resolution hydrograph for drainages local to the Project.

20.3.4 Geochemistry

20.3.4.1 Geochemical Test Program Sampling

In order to characterize the potential for acid rock drainage and metal leaching in the exposed pit wall and waste, a geochemical program was initiated in 2017 by phase Geochemistry under contract to Knight Piésold. Interim results of the ongoing program are provided in phase (2018), and summarized herein.

Initial sample selection consisted of a total of 180 samples (169 unique samples and 11 duplicates) representative of anticipated waste zones and low-grade ore zones. Sources utilized for sample selection included geological drill logs (lithology codes, alteration codes and mineral zonation codes) and assay data where available (specifically copper, copper equivalents and sulphur content if available). Spatial coverage was also considered to obtain samples in or close to the anticipated resource zones and from variable depths. A cutoff grade of 0.15% copper equivalent was used to screen waste rock samples from ore material.

Sample selection took into consideration lithology, alteration and mineralization zone. Those lithological units, alteration types and mineralization zones that represented 5% or more of the length-normalized drill logs were the focus of the initial sample program with proportions reflecting their relative abundance.

Sample selection aimed to select proportionate lithology, alteration and mineral zones in low grade and waste intervals while obtaining general spatial coverage. In most cases, 10 m intervals were selected which approximately represent a pit bench height. Occasionally, to obtain specific litho-alteration combinations, shorter intervals were required. Coarse rejects for the selected intervals were composited by Filo geologists to provide 2 kg of material for the analytical lab.

20.3.4.2 Analytical Program

20.3.4.2.1 Static Tests

Samples were sent to SGS Canada Inc. (SGS) in Burnaby, B.C., Canada for static testing. Static tests are one-time laboratory tests used to evaluate the acid-generation and short-term metal leaching potential of a sample. Static testing on the Filo samples was conducted in two phases, with the second phase of lab testing currently in progress, and included the following tests:

- Phase One:
  - Acid-Base Accounting (ABA),
  - Trace Element Analyses,
- Phase Two:
  - Shake Flask Extraction (SFE) Leach Tests (3:1 liquid to solid ratio),
  - Sequential Leach Extraction Tests,
  - Quantitative Evaluation of Minerals by Scanning Electron Microscope (QEMSCAN®), and
  - Humidity Cell Testing.

Modified ABA tests were conducted on all samples and included direct analysis of paste pH, total sulphur (Leco), sulphate sulphur by 25% HCl leach, sulphide sulphur by Sobek 1:7 nitric acid leach, fizz test, Modified Neutralization Potential (NP) and total inorganic carbon (TIC) with calculations of the insoluble sulphur, acid potential (AP), carbonate NP, net neutralizing potential (NNP) and neutralization potential ratio (NPR or NP/AP). These results determine the balance of acid producing potential and neutralization potential of a sample and allows for the classification of the sample with respect to acid generation potential.

Solid-phase trace element analyses on all samples were completed following an aqua-regia digestion with ICP-MS finish to quantify the metal content in the rock and identify potential parameters of environmental concern.

SFE leach tests (MEND, 2009) are a short-term water extraction leach test that provides an indication of what metals are soluble from the sample at the time of testwork. The SFE tests are conducted at a 3:1 liquid to solid ratio using distilled or de-ionized water as the leaching medium with analysis of leachate by ICP- MS. The Filo SFE tests are currently in progress on a subset of 23 samples. The sample selection represents the key lithology, alteration and mineralizing zones and a range of pH values, sulphide and sulphate contents and acid generating potentials.

Sequential leach extraction tests, via the Nevada Meteoric Water Mobility Procedure (MWMP) bottle roll method, are also in progress on a set of 20 composite samples. The composites represent varying pH values of key units on those samples that have a current pH value below 5. The objective of the test is to subject a larger sample size to a sequence of leach extraction tests while maintaining a constant solid to liquid ratio. Chemistry of each leach step and cumulative masses leached will provide an assessment of potential water quality from acidic samples and saturation limits (or maximum concentrations) through extended contact with the solids.

Mineralogical analyses are in progress on a subset of 18 samples via QEMSCAN®, a fully automated, high-definition mineralogical analysis including digital imaging and mineralogical and petrological analysis, to identify and quantify mineral phases in the rock samples with emphasis on carbonate and sulphide minerals, which are the primary sources of buffering and acidity. Of the 18 samples, 10 samples are composites being tested by sequential leach extraction tests and 8 samples tested in the humidity cells.

20.3.4.2.2 Kinetic Tests

Kinetic tests are long-term leach tests that provide insight into the weathering characteristics of materials over time including NP depletion, sulphide oxidation and metal leaching rates.

Kinetic testing on the Filo sample set consists of standard laboratory humidity cell tests on 8 samples with paste pH values above 5 as determined in the initial static test program. The sample set includes 2 samples that are classified as non-potentially acid generating, 2 that are uncertain and 4 that are classified as potentially acid generating but that have not yet developed acidic pH.

Humidity cell testing was initiated in October 2018 and is currently in progress at SGS Laboratory in Burnaby, Canada using the MEND (2009) procedure whereby the waste rock is flushed once per week with deionized water and the leachate is analyzed. Leachates are being analyzed for general parameters (pH, conductivity), anions (acidity, alkalinity, sulphate, chloride, fluoride), nutrients (ammonia, nitrate, nitrite, phosphorus) and dissolved metals by ICP-MS.

20.3.4.2.3 Results

Results are available for the Phase 1 static testing including ABA and trace element analyses. Phase 2 testing including SFE leach tests, sequential leach extractions, QEMSCAN and humidity cell tests are in progress and are expected for interpretation in 2019.

20.3.4.2.4 Acid Base Accounting

The ARD potential of a sample is a balance of the acid potential and the neutralization potential of the sample. Important sulphide minerals for acid production are predominantly the iron-bearing sulphides. While a variety of minerals can contribute to the neutralization potential, the most effective by far are the carbonate minerals calcite and dolomite. Other minerals such as feldspars and micas. can also buffer acidity when they dissolve, but they typically do not dissolve at a rate that can provide neutralization sufficient to affect the net acidity generated by pyrite oxidation.

Paste pH of the tested samples ranged from pH 0.6 to 7.4 (median pH 4.0), an indication of buffered and acidic samples in the dataset. Of the 169 samples tested, the majority (72% of samples) were acidic at the time of testing defined as those with a paste pH less than 5.0. The acidic paste pH samples were not confined to a single lithology, alteration type or mineralization zone. Only 28% of the sample set tested had paste pH values above 5.0, and only 3.5% had values above 6.0 including the wacke/sandstone, microdiorite and intrusive porphyry with feldspar, hornblende and biotite lithologies.

Total sulphur in the samples ranged from 0.05% S to 14.5% S, sulphate sulphur ranged from 0.03% S to 6.3% S and sulphide sulphur ranged from 0.01% S to 10.6%. Sulphur speciation indicates that sulphate sulphur is the predominant sulphur form in the sample set, and thus an indication that some sulphide oxidation has occurred in the samples and/or the predominance of primary sulphate minerals (e.g. alunite, jarosite). Sulphide sulphur, typically the active sulphur species for acid generation, was quite variable within each lithology. Sulphide acid potential ranged from 0.3 to 331 kg $CaCO_3$/t with a median of 3 kg $CaCO_3$/t.

An apparent relationship between total sulphur and paste pH was noted whereby samples with sulphur <0.1% had paste pH values above 5.0. Total sulphur versus depth shows high sulphur content both nearer to surface and at depth. With increasing depth, sulphur content was dominated by total sulphide.

Sulphur content appears to be slightly lower in the wacke/sandstone unit with a median total sulphur of 1.3% compared to a median total sulphur of 2.8% in the hornblende and biotite unit and median total sulphur between 4% and 5% in the other lithologies. Other sulphur trends include lower median total sulphur in the leached zone unit and the steam heated with residual opaline/silica alteration unit.

Neutralization potential in the data set showed that Modified-NP values were generally negative (-182 to 10.7 kg $CaCO_3$/t, median -17 kg $CaCO_3$/t). Negative NP values are indicative of already acid generating conditions and this is corroborated by the correspondingly low paste pH values. A relationship is evident between Modified-NP and paste pH whereby:

- Paste pH is maintained above pH ~6.5 when Modified-NP is above 0 kg $CaCO_3$/t;
- Paste pH rapidly declines from pH ~6.5 to approximately pH ~3.5, between Modified-NP 0 kg $CaCO_3$/t and -10 kg $CaCO_3$/t, respectively; and,
- Paste pH slowly declines from pH ~3.5 to pH ~0.5 between Modified-NP -20 kg $CaCO_3$/t and -180 kg $CaCO_3$/t, respectively.

All but six samples within the dataset show total inorganic carbon (TIC) values either at or below the method detection limit resulting in calculated Carbonate-NP (Ca-NP) values at the detection limit. Of the seven samples with TIC (and therefore Ca-NP) above the detection limit, four samples show Modified-NP > Ca-NP; two samples show approximately equivalent Modified and Ca-NP values; and one sample shows more Ca- than Modified-NP. This suggests that buffering from aluminosilicate minerals dominates what little neutralization potential exists in the Filo del Sol static dataset.

The ARD classification of a sample is based on the neutralization potential to sulphide acid potential ratio (NP/AP), or NPR. In this assessment, carbonate NP (Ca-NP) and thus Ca-NPR (Ca-NP/AP) was used to assess the ARD classification of the samples. Screening criteria as provided by the Global Acid Rock Drainage (GARD) Guide (INAP, 2009) and MEND (2009) guidelines have been adopted for classification in this assessment whereby a sample is considered:

- Potentially net acid generating (PAG) if NPR < 1,
- Not potentially net acid generating (non-PAG) if NPR > 2, and
- Uncertain (UC) if NPR is between 1 and 2.

It is standard practice that samples considered uncertain are generally managed as PAG rock unless a site-specific ratio can be demonstrated i.e. PAG if NPR<2.

The majority of samples in the dataset were acid generating (AG) upon receipt at the lab. Of those samples with a pH above 5, 11% would be considered non-potentially acid generating (non-PAG) and the remaining 17% would be expected to become acidic over time. Therefore, only a very small proportion of the sample set would classify as non-PAG. Those were represented by lithology samples with low sulphur content (approximately <0.3%).

20.3.4.2.5 Trace Elements

Multi-element ICP-MS analyses following an aqua regia digestion were conducted on all the samples to quantify the solid phase composition of the tested samples and provide an indication of what metals may be elevated in the waste rock. Metal values in excess of ten times the crustal average (as a whole) have been used to identify elevated or anomalous concentrations in the material as suggested by Price (1997).

Observations, based on median values, indicated:

- Variable metal content within each lithology and between lithologies;
- Generally lower copper, molybdenum and silver content, in the clastic rock units compared to the other lithologies;
- Higher arsenic levels in the rhyolite units;
- Higher copper and zinc content in the intrusive units i.e., intrusive porphyries and microdiorite;
- Lower levels of copper, molybdenum, silver and zinc in the steam-heated with residual/opaline silica alteration;
- Higher mercury in the steam-heated with residual/opaline silica alteration;
- Higher zinc levels in the quartz-illite alterations;
- Highest copper concentrations in the hypogene mineralization zones;
- Lower levels of arsenic, molybdenum, silver, zinc in the leached zone;
- Higher mercury levels in the leached and oxide zones; and
- Possibly higher zinc content (n=3) in the Hypogene Zone C (chalcopyrite-pyrite) unit.

The metal leaching potential of these metals will be examined in the Phase 2 test program (leach extractions and humidity cell tests) that are currently in progress.

Based on the geochemical program to date, the majority of tested lithologies are assumed PAG, so water management, waste rock handling, and heap runoff have been designed accordingly, as described in Section 18.8. As the geochemical program progresses, a higher resolution understanding of the potential acid generation or metal leaching of each waste lithology will evolve, which will allow for prescriptive handling and storage methods.

20.3.5 Water Quality and Aquatic Biota

A focused study program for the Filo del Sol project was conducted by Knight Piésold (2018b 2018c, and 2018d), which followed several previous regional studies. Sites throughout the area and in downstream catchments were sampled for water quality and for invertebrates and phytoplankton. Sample locations were located in the Los Mogotes River and Arroyo Pircas de Bueyes, in Argentina, and the Montoso River catchment in Chile, as shown on Figure 20 1.

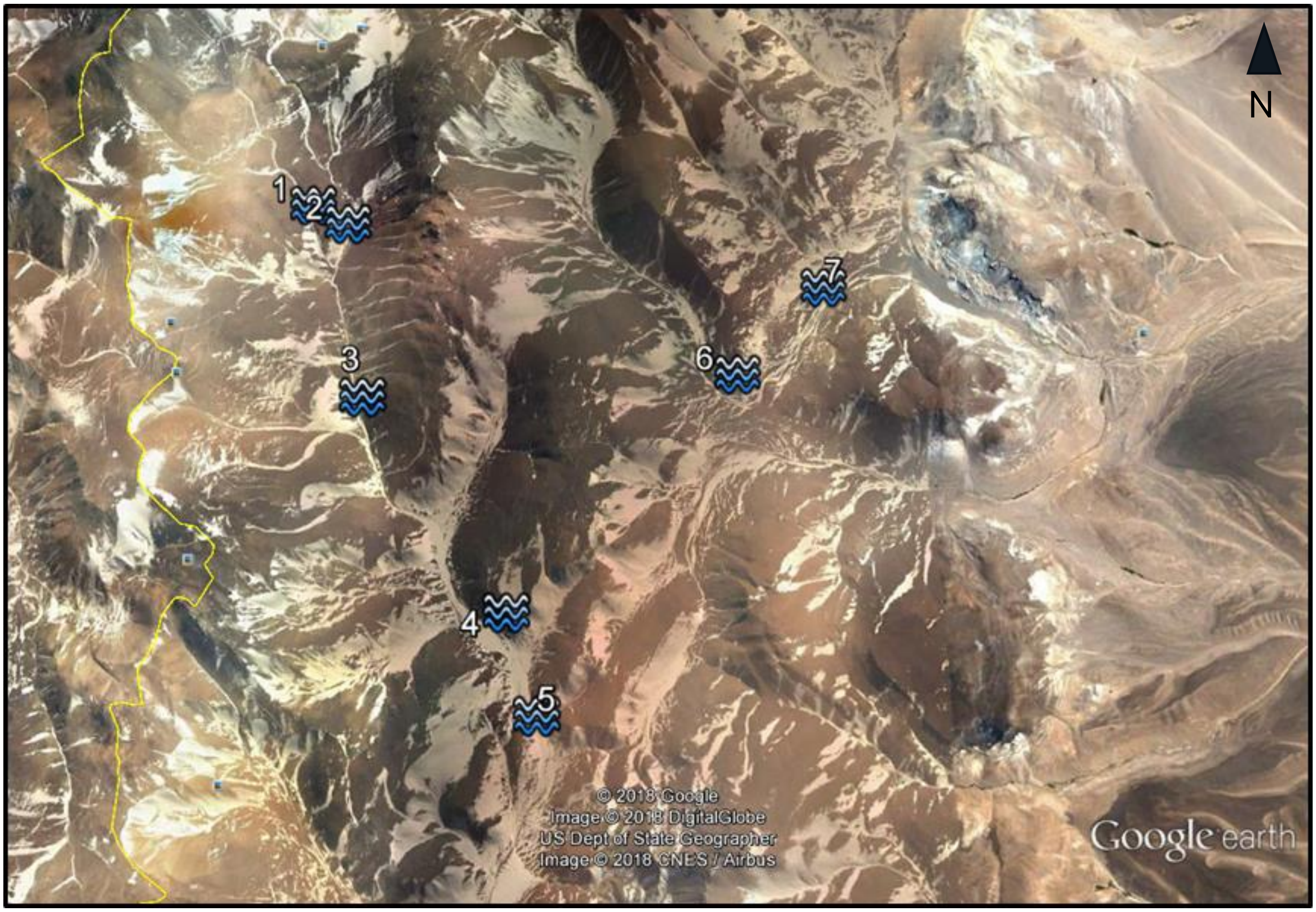
Results indicate that waters in the upper Los Mogotes are acidic, with pH values ranging between 3.67 and 4.5 at sites 1 and 2. The pH values increased at lower elevations, becoming neutral (up pH 7.05 to 7.8 in sites 3 to 5). Elevated metals were similarly found in the upper watershed, including aluminum, arsenic, barium, and copper. Metals concentrations decreased downstream.

Water samples from Arroyo Pircas de Bueyes have neutral pH, and generally low concentrations of metals, with the exception of arsenic and iron.

Samples from the upper Montoso River were only slightly acidic (pH 6.7 to 6.8), and became neutral at lower altitudes (pH 7.1 to 7.5). Similar elevated metals were found to those noted in to the findings from the Mogotes River, however there was no apparent decreasing downstream trend.

Species richness of invertebrates was very low in the upper Mogotes and Montoso Rivers. Invertebrates were found in much higher abundance in the Arroyo Pircas de Bueyes. No fish were identified in the Project area.

Figure 20-1: Water Quality and Aquatic Biota Sampling Locations



Source KP, 2018

20.3.6 Soils

A survey of the soil characteristics of the project area was conducted in 2015 (Pittaluga, M. 2015). All soils were all classified as entisols; young, with coarse texture, low organic content, very low fertility, and without defined edaphic horizons. All soils were classified under the USDA Natural Resources Conservation Service rating as “Class VIII”, which have limitations that preclude their use for commercial plant production and limit their use to recreation, wildlife, water supply, or for aesthetic purposes.

20.3.7 Flora and Fauna

Knight Piésold Consulting (2018e, 2018f, and 2018g) conducted surveys at the project for vegetation and wildlife, which complemented an earlier study from 2013, which included several adjoining mineral concessions, including the Filo de Sol project area (Molina, A. 2013).

The project is located within the High Andean Ecoregion, commonly referred to as páramo, or alpine desert. In general, the area is characterized by rocky terrain with entisolic soil, and a resultant scarcity of vegetation. The dominant vegetation is characterized by xerophytic grasses such as *Stipa spp*, dispersed in isolated clusters within the rocky or gravel matrix (Figure 20 2). Patches of low bush steppe vegetation dominated by *Adesmia spp* in the lower elevation areas of the project area are also present. No persistent vegetation or vertebrates were observed above 4,700 m amsl, where the majority of the Filo del Sol project footprint would be located. Wetlands or vegas are found in valley bottoms downstream from the Project where hydrologic conditions allow. Throughout the Ecoregion, vegas represent a small proportion of the area (approximately 1%); however, they have high productivity, and they provide sustenance to the diverse trophic levels within the ecosystem. Vegas were dominated by rushes and graminoids; primarily *Oxychloe castellanosii* (Figure 20 3), *Deyeuxia curvula, and Deyeuxia eminens*.

Figure 20-2: Typical open steppe habitat dominated by Stipa spp grasses

Source: KP, 2018

Figure 20-3: Vega in the riparian zones of the lower Mogotes River dominated by Oxychloe castellanosii

Source: KP, 2018

Faunal diversity was represented by 18 bird species, 3 mammal species (vicuña, guanaco, and gray fox), and 1 species of reptile (San Guillermo Lizard). The highest abundance of wildlife was associated with vega habitat downstream of the Project. This included several waterfowl species and passerine birds. Groups of vicuña were noted along the access road corridors.

Two species of plants in the project area associated with vegas are endemic and are monitored as part of the "PlanEAr" program (Plantas Endémicas de la Argentina) of the Argentinian Ministry of the Environment and Sustainable Development (Secretaría de Ambiente y Desarrollo Sustentable). Each of these species (Oxychloe castellanosii and Festuca argentiniensis) are considered abundant, although restricted in their distribution.

The Argentine Ministry of the Environment and Sustainable Development classifies faunal species of concern according to Law 22.421 Protection and Conservation of Wildlife (Protección y Conservación de la Fauna Silvestre), Resolution 1030/04. The lizard Liolaemus eleodori is classified as having "Insufficient Information", and Vicuña (Vicugna vicugna) is classified as "Vulnerable" under the Resolution.

Argentina is signatory to the Convention on International Trade in Endangered Species of Wild Fauna and Flora (CITES). The following species that were identified in the project area are classified under CITES as "Vulnerable" or "Threatened"; and therefore have restrictions on their transport and trade:

- Birds:
  - Caracara (*Polyborus megalopterus*)
  - Variable hawk (*Buteo polyosoma*)
  - Aplomado falcon (*Falco femoralis*)

  - Peregrine falcon (*Falco peregrinus*)
  - Darwin's Rhea (*Pterocnemia pennata garleppi*)
  - Andean Condor (*Vultur gryphus*)
- Mammals:
  - Vicuña (*Vicugna vicugna*)

These species are restricted in their trade and are a focus for protection.

No species of concern or protection were identified within the portion of study area in Chile.

20.3.8 Archaeology

Several archaeological investigations have been conducted for the project, specifically in 2007, 2013, 2014, and 2018 (Durán, Lucero, Estrella, Castro and Yerba, 2014, Knight Piésold Consulting, 2018h). Additionally, San Juan province in Argentina has been the subject of many archaeological studies over several decades. Some sites in the province, associated with ancient hunter-gatherers, are thought to be more than 9,000 years old. Continuous, infrequent use of the area up to present times has been documented in the archaeological record. In Argentina, Law 7911/08 stipulates that artefacts older than 50 years are considered archaeological and are protected.

Four archaeological sites were identified within the Project area; one within the mineral concession close the Arroyo Pircas de Bueyes, and three approximately 3 km outside of the boundary along the margins of the lower Mogotes River. The sites were generally composed of rock formations (circles, semi-circles, or walls), with some associated with lithic material (Figure 20.4). All of the sites are located in the river basins associated with streams, valleys, bodies of water, wetlands and valleys, up to an approximate height of 4,400m amsl.

Figure 20-4: Archaeological site located in the Arroyo Pircas de Bueyes drainage

Source: KP, 2018

Project design will avoid direct impacts to archaeological sites where possible. Where impacts cannot be avoided, the identified site will be studies by a professional archaeologist and removed for archiving if appropriate. The presence of archaeological material in the project area is not considered a major impediment to exploitation of the resource.

20.4 Social Considerations

As part of the Lundin Group of companies, Filo Mining has relied on the Lundin Foundation to delineate the socio-economic environment of the project. The Lundin Foundation is a registered Canadian non-profit organization that works with corporate partners and stakeholders to improve the operations for the benefit of communities. The information below has relied upon their analysis, as provided to Knight Piésold.

20.4.1 Community Identification

In Argentina, the nearest settlements or homesteads are more than 100 km from the Project. The nearest town is Guandacol, which is approximately 150 km distant from the Project, accessed via remote mountain roads. Those few community members that live in this zone, either permanently or seasonally, have limited access to government resources or infrastructure. They are largely self-reliant, subsisting on small scale farming and ranching.

The principal access corridor for the Project is projected to traverse the border into Chile and follow the existing highway network in the Copiapó Province of the Atacama Region to the pacific port of Caldera. The largest population centre in the corridor is the city of Copiapó, and the towns of Paipote and Tierra Amarilla. According to the 2017 census, the area has 167,956 inhabitants. Tierra Amarilla is a city and commune located 15 km from Copiapó, and at 2017, it had a population of 14,019 inhabitants.

Mining is the dominant economic contributor to the Atacama Region and to Tierra Amarilla. It is responsible for nearly 90% of exports and 45% of the regional GDP. There is a well-established workforce and supply chain for mineral activity in this area.

Commercial agriculture in the Copiapó valley includes principally grape growing, but also olives, tomatoes, peppers, and other fruits and vegetables.

At higher elevations more proximate to the project, the predominant economic activity is livestock ranching (sheep and cattle), primarily sold locally, accompanied with small-scale farming.

20.4.2 Community Relations Plan

The Lundin Foundation has developed a Community Relations Plan for stakeholders along the transportation route who may be affected by the project. The plan utilizes dialogue and communication using diverse formats – meetings, field visits, local media, and website information. It is based on a platform of community participation and joint decision-making processes.

A formal Grievance Mechanism/Feedback Process is being implemented as part of the community engagement process. It includes internal guidance for staff and contractors of Filo Mining as to how to receive, log, and track grievances, feedback, suggestions, and comments from stakeholders. The mechanism assigns procedures and responsibilities to individuals to ensure the proper depth of response is provided.

Along the access route from the Argentine side, interactions have been limited to those populated areas near the town of Guandacol, located approximately 150 km from the project area, and have focused on road maintenance contracts and employment.

Increased interaction with the communities and implementation of formalized engagement is planned to be concomitant with feasibility level studies.

20.4.3 Indigenous Populations

No indigenous people have been identified in the Argentine Project area, including along the access corridors. There are identified communities and indigenous people of the Colla ethnic group in the region of Tierra Amarilla in Chile along the transportation corridor.

As part of the environmental permits for the Project exploration, an anthropological study was conducted in 2012 to ensure that impacts to the Colla del Torín Indigenous Community were minimized. Filo Mining has commissioned an update to that study as part of their ongoing activity in the area. These studies examine the ability of the Colla people at this community to carry out their way of life, including traditional customs, and access to culturally important sites. The planned update study will incorporate participatory methodology that incorporates criteria established by the Indigenous and Tribal Peoples Convention 169 of the International Labour Organization (ILO, 1989).

20.5 Waste Disposal

Waste dump designs were developed by AGP, which is more fulsomely described in Section 16 of this report.

20.6 Water Management

During the project life, water quantity and quality will be managed to maximize diversions and maintain "non-contact" water. The site water management plan is designed to "keep clean water clean" as much as possible, with the following primary objectives:

- Providing adequate protection to internal infrastructure and personnel from the uncontrolled effects of surface water runoff during storm events
- Maximizing the internal recycle of contact and process waters in ore processing on the heap leach pads, thereby minimizing the use of external water sources
- Preventing sediment entry toward facilities and erosion at discharge points
- Achieve environmental compliance

Diversion ditches will be installed around the waste rock dump, pit, and heap leach facilities to convey clean or non-contact freshwater around these disturbed areas, where it is physically practical. Water that accumulates on project infrastructure will be collected for settling and testing prior to any discharge. No water will be discharged to the environment that would have adverse environmental impact.

20.7 Mine Closure

No financial bonding for closure is required for the project to the government of Argentina. In Chile, Law 20.551 requires that a closure plan and accompanying cost estimate is submitted to and approved by the National Geology and Mining Service (Servicio Nacional de Geologéa y Minería or SERNAGEOMIN). Guidance on closure costing and bonding under Law 20.551 was updated in 2018. The SERNAGEOMIN approval of a closure plan and cost follows both the successful resolution of the EIA and sectorial permit processes but precedes the initiation of construction.

A provisional closure plan will be included with the Mine EIA submission for both countries. The closure plan will be designed to ensure long term stability of both physical and chemical properties of the site, and to blend with the high-altitude, rocky environment. Specific closure items will include:

- Reagents and supplies will be removed and will be returned to suppliers, sold to other operations, disposed of in approved waste facilities, or transported to a certified company for disposal.
- Equipment, conductors and other above ground facilities for the electrical supply will be dismantled or demolished.
- All foundations will be demolished and covered to approximate as closely as possible the pre-mining landscape topography.
- Where excavations or construction of berms and walls were required, these will also be regraded to approximate pre-construction land contours. If soil contamination is detected around any facility, remediation alternatives will be evaluated and applied.

- Access to areas such as the open pit, waste rock facilities and the heap leach facilities will be restricted with the use of berms, road closures, and warning signs to restrict access of personnel and vehicles.
- The pit will be allowed to fill to the phreatic level
- Spent ore on the heaps will be rinsed until it can be demonstrated that it does not contain levels of contaminants that are likely to become mobile and degrade downstream waters
- Heaps will be covered to isolate spent ore, limit influx of atmospheric water and oxygen, and control upward movement oxidation products
- Removal and re-grading of all access roads, ditches and borrow areas not required beyond mine closure
- Long-term stabilization of all exposed erodible materials.

Active closure is expected to take two years, with a further five years of monitoring for a total 7-year closure period.

A detailed closure cost will be developed to support the Mine EIA submission, supported with feasibility level design. Based on the foregoing, a preliminary estimate of approximately $68.5M has been developed and incorporated to project costing as illustrated in Table 20-4.

Table 20-4: Preliminary Closure Cost Estimate

Closure Aspect	Unit	Quantity	Unit Cost ($)	Total ($M)
Dismantling of equipment and structures, demolition of structure and foundations	ha	15	675,000	10.1
Access control / safety berm around pit	m	5,000	270	1.4
Retraining waste dump diversion ditches to the pit	km	2	472,500	0.9
Rinsing of Heap Leach Pads	year	4	270,000	1.1
Re-profiling and placement of evaporative earthen cover	ha	282	40,500	11.4
Scarification and contouring of the footprint	ha	27	270,000	7.3
Scarification and contouring of the internal access roads	km	80	8,100	0.6
Dismantling of electrical transmission line	km	140	3,375	0.5
Detailed closure engineering and planning	Study	1	1,080,000	1.1
Active closure monitoring	year	2	256,500	0.5
Post closure monitoring	year	5	222,750	1.1
Misc. (Waste management and disposal, specialist contracts, etc.)	Lump Sum	1	1,620,000	1.6
			Subtotal of Direct Costs	**37.7**
Indirect Costs				
Contractor Fees (25% of Direct Costs)				9.4
Administration (15% of Direct Costs)				5.6
			Subtotal of Indirect Costs	**15.1**
Subtotal				**52.7**
Contingency at 30%				15.8
Total Closure Costs (USD)				**68.5**

Note: Numbers may not add due to rounding.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

All capital and operational cost estimates are presented in United States dollars (US$), with no escalation.

21.2 Capital Cost Estimates

21.2.1 Summary

Table 21-1 summarizes the LOM capital cost estimate including initial capital, sustaining capital and closure costs. The estimate included costs for mining, process plant, site preparation, tailings facility, on-site and off-site infrastructure.

As outlined in Table 21-1, the overall LOM capital cost of the project is US$2,013 M, comprised of the following:

- Initial capital cost includes the costs required to construct all the surface facilities, and open pit development to commence a 21.9 Mt/a operation. The initial capital cost is estimated to be US$1,805 M.
- Sustaining capital costs: includes all the costs required to sustain operations, with the most significant being mine development. Sustaining capital costs total US$140 M over the LOM
- Closure costs total US$69 M as defined in Section 20.7.

Table 21-1: Capital Cost Summary

WBS	Description	Initial (US$M)
1000	Mine	230
3000	Processing	610
4000	On Site Infrastructure	117
5000	Off-Site Infrastructure	188
	Subtotal Direct Costs	**1,145**
6000	Indirect costs	185
7000	EPCM Services	149
8000	Owner's Costs	50
9000	Provisions	275
	Subtotal Indirect Costs	**660**
	Initial Capital Cost - Total	**1,805**
	LOM Sustaining Capital	140
	Closure Costs	69
	LOM Total	**2,013**

Note: Numbers may not add due to rounding.

21.2.2 Initial Capital Cost

The capital cost estimate was developed in $US in Q1 2023. The capital cost estimate has been prepared in accordance with the recommended practices of the American Association of Cost Engineers (AACE) and is classified as an AACE Class 4 Prefeasibility Study estimate with an accuracy range of +30/-20%. The typical purpose of the estimate will be for budgetary, viability purposes, to determine validity of a business case or option validation and assessment.

The estimate is based on an EPCM approach for the process/infrastructure areas as outlined in Section 24. The following parameters and qualifications were considered:

- No allowance has been made for exchange rate fluctuations
- A growth and contingency allowance was included
- There is no escalation added to the estimate
- Data for the estimates have been obtained from numerous sources, including
- Mine schedules
- Prefeasibility level engineering design
- Topographical information obtained from the site survey
- Geotechnical investigations
- Vendor equipment and material supply costs;
- Budgetary unit costs from contractors for civil, concrete, steel, electrical, piping and mechanical works
- Data from similar recently completed studies and projects.

Initial capital costs of US$1,805 M are shown in various formats in the following tables:

- by Level 1 major areas (Table 21-2)
- by major discipline (Table 21-3)
- by Level 2 summary (Table 21-4).

Table 21-2: Initial Capital Estimate Summary Level 1 Major Area

Cost Type	WBS LVL 1	LVL 1 Description	Total (US$M)
Direct	1000	Mine	230
	3000	Process Plant	610
	4000	On-Site Infrastructure	117
	5000	Off-Site Infrastructure	188
		Direct Subtotal	**1,145**
Indirect	6000	Indirects	185
	7000	EPCM Services (Project Delivery)	149
	8000	Owners Costs	50
	9000	Provisions (Contingency)	275
		Indirect Total	**660**
TOTAL			**1,805**

Note: Numbers may not add due to rounding.

Table 21-3: Initial Capital Estimate by Major Discipline

Disc.	WBS Description	Total (US$M)
A	Site Development	102
B	Earthworks	86
C	Concrete	52
D	Structural Steel	24
E	Architectural	21
F	Platework	11
G	Mechanical Equipment	339
H	Mobile Equipment	6
I	Painting And Coatings	2
J	Piping	75
K	Electrical Equipment	163
L	Electrical Bulks	34
M	Instrumentation	12
R	Third Party Estimates (Mining)	218
	Subtotal Direct Costs	**1,145**
S	Field Indirects	131
T	Spares & First Fills	37
U	Vendors	18
V	EPCM Services (Project Delivery)	149
W	Owner's Costs	50
y	Provisions	275
	Subtotal Indirect Costs	**660**
TOTAL		**1,805**

Note: Numbers may not add due to rounding.

Table 21-4: Initial Capital Estimate Summary (Level 2)

Cost Type	WBS LVL 2	LVL 2 Description	Total (US$M)
Direct	1100	Mine Development Surface	168
	1200	Dewatering	0
	1300	Mining Equipment	50
	1400	Ancillary Services	11
	1500	Mine Explosives Magazine	0
	3100	Crushing	95
	3200	Copper On/Off Circuit	155
	3400	Copper Processing (SX-EW)	194
	3500	Gold Circuit	123
	3600	Gold Processing (Merrill-Crowe)	43
	3700	SART (Future)	0
	4100	Site Development	34
	4200	Power Supply and Distribution	24
	4300	Utilities	35
	4400	General Buildings	6
	4500	Plant Buildings	13
	4600	Mobile Equipment	6
	5100	Off-Site Roads	43
	5200	Power Supply	98
	5300	Water Supply	42
	5400	Permanent Accommodation	5
		Subtotal Direct Costs	**1,145**
Indirect	6100	Field Indirects	39
	6200	Heavy Lift Cranes	4
	6300	Accommodation & Messing	88
	6400	Vendor Representatives	18
	6600	Spares And First Fills	37
	7100	Project Delivery (EPCM Services)	142
	7300	Project Delivery (EPCM Expenses)	7
	8100	Owner's Costs	2
	8200	Permitting, Social and Environmental	6
	8500	Land	2
	8600	Pre-Production Costs	37
	8700	Financing	3
	9100	Contingency	275
	9200	Client Contingency (Risk)	Excluded
	9300	Forex	Excluded
	9400	Escalation	Excluded
		Subtotal Indirect Costs	**660**
TOTAL			**1,805**

Note: Numbers may not add due to rounding.

21.2.3 Sustaining Capital Costs

The Sustaining capital cost estimate has been summarized at the levels indicated by the following Table 21-5 and stated in United States Dollars (USD) with a base date of Q1 2023 and with no provision for forward escalation.

Table 21-5: Sustaining Capital by Major Area

Cost Type	WBS LVL 1	LVL 1 Description	Total (US$M)
Direct	1000	Mine	9
	3000	Process	131
	4000	On-Site Infrastructure	0
	5000	Off-Site Infrastructure	0
TOTAL			**140**

Note: Numbers may not add due to rounding.

21.3 Capital Cost Basis of Estimate

The following basic information pertains to the estimate:

- the estimate base date is Q1 2023
- the estimate is expressed in United States dollars
- metric units of measure are used throughout the estimate
- actual estimate accuracy is defined by the stated maturity of the information available

21.3.1 Definition of Costs

The capital cost estimate includes direct and indirect initial capital and sustaining capital.

Initial capital is the capital expenditure required to start up a business to a standard where it is ready for initial production.

Direct costs are those costs that pertain to the permanent equipment, materials and labour associated with the physical construction of the process facility, infrastructure, utilities, buildings, etc. Contractor's indirect costs are contained within each discipline's all-in labour rates.

Indirect costs include all costs associated with implementation of the plant and incurred by the owner, engineer or consultants in the design, procurement, construction, and commissioning of the project.

Sustaining capital is the capital cost associated with the periodic addition of new plant, equipment or services that are required to maintain production and operations at their existing levels.

21.3.2 Methodology General

The estimate is developed based on material take-offs and factored quantities and costs, semi-detailed unit costs and defined work packages for major equipment supply.

The structure of the estimate is a build-up of the direct and indirect cost of the current quantities; this includes the installation/construction hours, unit labour rates and contractor distributable costs, bulk and miscellaneous material and equipment costs, any subcontractor costs, freight and growth.

Major cost categories (permanent equipment, material purchase, installation, subcontracts, indirect costs and owner's cost) were identified and analysed. Percentage of contingency was allocated to each of these categories on a line-term basis based on the accuracy of the data. An overall contingency amount was derived in this fashion.

The methodology applied, and source data used to develop the estimate is as follows:

- define the scope of work;
- quantified the work in accordance with standard commodities;
- structure the estimate in accordance with an agreed WBS;
- calculated "all in" labour rates for construction work by major trade groups;
- determine the purchase cost of equipment and bulk materials;
- determine the installation cost for equipment and bulks;
- determine the cost for temporary facilities required at site during the construction period;
- established requirements for freight;
- determine the costs to carry out detailed engineering design and project management;
- determined foreign exchange content and exchange rates;
- determined growth allowances for each estimate line item;
- determined the estimate contingency value;
- undertake internal peer review; and
- finalized the estimate, and estimate basis.

21.3.3 Source Data

- equipment lists
- scope of work
- process design criteria
- general arrangement drawings
- drawings and sketches
- process flow diagrams
- material take-offs
- equipment and bulks pricing
- contractor installation (labour rates, historical data)
- vendor equipment and material supply costs
- third party estimates
- historical data
- project schedule

21.3.4 Exchange Rates and Foreign Content

The exchange rates used in the estimate are shown in Table 21-6 and have been determined from the XE website as of November 15th, 2022 and are applied to foreign currency data.

Table 21-6: Estimate Exchange Rates

Code	Currency	Exchange Rate
USD	US Dollar	1 USD = 1.00 USD
CAD	Canadian Dollar	1 USD = 1.33 CAD
EURO	Euro	1 USD = 0.96 EUR
GPB	Great Britain Pound	1 USD = 0.84 GPB
ARS	Argentine Peso	1 USD = 162.34 ARS
AUD	Australian Dollar	1 USD = 1.47 AUD
CLP	Chilean Peso	1 USD = 886.52 CLP

The following Table 21-7 identifies the foreign priced content and US$ priced content.

Table 21-7: Foreign and US$ Priced Content

Country	Initial CAPEX (US$M) (excl contingency)	% of Costs (excl contingency)
United States priced content	1,576	94.4%
Canadian priced content	13	0.78%
European priced content	0	0%
Great Britain priced content	0	0%
Argentine priced content	48	2.84%
Australian priced content	1	0.04%
Chilean priced content	33	1.98%
Total – Directs and Indirects (less contingency)	**1,671**	**100%**

Note: Numbers may not add due to rounding.

21.3.5 Market Availability

The pricing and delivery information for quoted equipment, material and services was provided by suppliers based on the market conditions and expectations applicable at the time of developing the estimate.

The market conditions are susceptible to the impact of demand and availability at the time of purchase and could result in variations in the supply conditions. The estimate in this report is based on information provided by suppliers and assumes there are no problems associated with the supply and availability of equipment and services during the execution phase.

21.3.6 Exclusions

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

- land acquisitions
- taxes
- scope changes and project schedule changes and the associated costs
- any facilities/structures not mentioned in the project summary description
- costs to advance the project from PFS to FS
- geotechnical unknowns/risks
- any costs for demolition or decontamination for the current site
- third party costs.

21.4 Operating Cost Estimates

21.4.1 Summary

Operating costs for the project include those related to mining, processing, tailings disposal, and general administration activities. The operating cost estimate has an accuracy of +30/-20% due to the approach employed to create the capital estimate and the conceptual level of engineering definition.

Table 21-8 below summarizes the operating costs, including mining, processing and G&A with an average cost of US$18.01/t ore processed.

Table 21-8: Operating Cost Estimate Summary

Operating Costs	US$/t Processed	US$M/a	Life of Mine (US$M)
Mining	6.63	132	1,720
Processing	9.72	213	2,523
Site G&A	1.67	37	434
TOTAL	**18.01**	**382**	**4,677**

Note: Numbers may not add due to rounding.

21.4.2 Basis of Estimate

The following key assumptions were made to estimate the operating costs for the project:

- Cost estimates are based in Q1 2023.
- Costs are expressed in US Dollars (US$).
- Filo Mining provide costs for fuel, electricity, miscellaneous expenses, which were reviewed and validated against Ausenco's database.
- Diesel price at US$1.05 /L based on Filo Mining's long range diesel price forecast
- A throughput of 60,000 t/d was used for the processing plant.
- Processing plant availabilities and operating costs were benchmarked against similar or comparable plants.
- Plant crusher availability is assumed to be 72%, heap leach area availability 98% and plant availability of 95%.
- Metal recoveries are based on metallurgical testwork results described in Section 13.
- Material and equipment are purchased as new.
- Reagent consumption rates are based on metallurgical testwork results as described in Section 13.
- Mobile equipment costs provided for fuel and maintenance.
- Labour rates and rotation schedule were established based on local Ausenco offices databases.
- Operating consumables are based on benchmarks from similar operations and from vendor information, and reagent usages are calculated based on SGS 2018 test results.
- Contingency was not included in the operating cost estimate.

21.4.3 Mining

Mining costs were estimated by building up the cost estimate over the LOM and presenting an average annual operating cost during the first 10 years of US$156 M, and LOM average annual cost of US$132 M, which is equivalent to US$6.63/t processed.

The build-up is summarized in Table 21-9, which shows the cost components relevant to each cost centre. Costs were developed for each year of operation.

Mining operating costs include:

- Salaries and wages: based on an estimate of staff and labour numbers and using labour rates current for Argentina. Total mine staff is 55 to 56, and mine labour varies by the year and ranges between 115 and 248, averaging approximately 238 in years 3 to 7.

- Fuel and power: based on a listing of required equipment and vendor suggested consumption rates.
- Consumables: includes tires, replacement parts and rebuilds, lubricants, and ground engagement tools.
- Additional costs have been included for 'down the hole' contract blasting, road/rock/ stemming crushing, ore control sampling, dewatering, and operation of the autonomous haulage system.
- Financing costs for most mine mobile equipment which are costed assuming a 20% down payment, a 6% interest rate, and payment terms varying from 24-60 months.

Costing is based on the following inputs:

- Diesel price at US$1.05 /L based on Filo Mining's long range diesel price forecast.
- Electricity price at US$0.081 /kWh based on Filo Mining's long range power price forecast.

Table 21-9: Mining Operating Costs (LOM)

Centre	Salaries and Wages (US$M)	Fuel and Power (US$M)	Consumables and Services (US$M)	Life of Mine (US$M)
General	62	0	6	67
Drilling	18	36	147	201
Blasting	0	0	260	260
Loading	18	89	108	215
Hauling	34	243	340	617
Support	18	41	69	128
Sundry costs			60	60
Financing			173	173
TOTAL	**149**	**409**	**1,163**	**1,720**

Note: Numbers may not add due to rounding.

Total mining cost for the LOM is US$1,720 M excluding pre-production capitalized stripping. Total ore crushed is 260 million tonnes.

21.4.4 Processing

Processing costs consist of costs for power, consumables maintenance and labour, as summarized in Table 21-10.

Table 21-10: Processing Costs

Processing Cost Item	Annual Cost (US$M)	Unit Cost (US$/t processed)
Power	40	1.82
Consumables	139	6.35
Maintenance	26	1.18
Labour	8	0.37
TOTAL	**213**	**9.72**

Note: Numbers may not add due to rounding.

The processing operating cost estimate includes:

- Labour for supervision, management and reporting of on-site organizational and technical activities directly associated with processing plant and water supply;
- Labour for operating and maintaining processing plant and water supply including mobile equipment and light vehicles;
- Fuel, reagents, consumables and maintenance materials for and water supply;
- Fuel, lubricants, tyres and maintenance materials used in operating and maintaining the mobile equipment and light vehicles;
- Power supplied to the site from the main site substation;
- Raw water supply; and
- Power and contractor operating costs for sample preparation, assay and metallurgical laboratory.

Table 21-11: Data Sources for Processing Costs

Cost Category	Source of Cost Data
Processing labour	Salaries, wages and labour roster for processing were provided by both Filo Mining and Ausenco Argentina office.
Reagents	Unit costs provided by benchmark operations and the Chilean ministry of mines publications on reagents for the mining industry, with estimated freight costs as percentage of the unit cost. Consumption rates based on SGS 2018 test work.
Consumables	Unit prices provided by suppliers. Crushers liner consumption rates were estimated based on testwork results, benchmarks from similar operations and from vendor information.
Power	All other costs were calculated using load factors, operating hours per year and installed equipment power taken directly from the Mechanical Equipment List. The grid power cost of US$0.081/kWh was supplied by Filo Mining.
Maintenance spares and consumables	Estimated at 4% of the total equipment cost for infrastructure and ancillary and 8% of the total equipment cost for each plant area. Mechanical, electrical and instrumentation costs were taken from the capital cost estimate.
Sample preparation, assaying and metallurgical testing	Laboratory costs were assumed based on similar projects.

Cost Category	Source of Cost Data
Light vehicle and mobile equipment	Fuel consumption rates were estimated from experience or using the Caterpillar Handbook. Annual hours of use were estimated from relevant personnel labour rosters.

21.4.4.1 Power

Costs for power for the processing plant were estimated by calculating annual power consumption for each WBS area, derived from installed power as shown in the plant equipment list together with equipment utilization and load factors.

Annual power consumption is shown in Table 21-12 at a total annual consumption of 490,815 MWh which costs US$40 M/a and US$1.82/t processed. The average power demand is 56 MW.

Table 21-12: Operating Costs – Power

WBS	Area Description	Installed Power (kW)	Consumed Power (kW)	Annual Power Consumed (kWh)
3100	Crushing	6,962	5,541	48,542,664
3200	Copper On/Off Circuit	8,413	6,356	55,682,502
3400	Copper Processing (SX-EW)	37,025	29,493	258,361,429
3500	Gold Circuit	15,444	10,905	95,528,413
3600	Gold Processing (Merrill-Crowe)	3,183	1,815	15,904,376
3700	Reagents & Water Services	621	326	2,855,760
4000	On-Site Infrastructure	35	27	241,776
5000	Off-Site Infrastructure	2,264	906	7,936,560
1400	Ancillary Services	250	200	1,752,000
future	SART	600	500	4,009,756
	TOTAL	**74,796**	**56,071**	**490,815,236**

Note: Numbers may not add due to rounding.

Note that the costs for the future installation of the SART plant have been included and averaged over the life-of-mine.

21.4.4.2 Consumables and Reagents

Processing reagent and consumable costs were estimated based on the throughput. The costs were based on calculated consumption rates and unit costs supplied by vendors. Reagents costs include estimated transport to site. The consumption of reagents was calculated based on SGS 2018 test work.

Crusher liners, and screen deck consumption rates were estimated based on vendor information and benchmarking similar plants.

Costs for consumables and reagents are summarized in Table 21-13 below, which shows individual costs for consumables, reagents and SART operating costs. The table also shows US$16 M for transport of consumables and reagents, taken as 12% of supply cost. Total annual cost is US$139 M, which is equivalent to US$6.35/t processed.

Table 21-13: Operating Costs – Consumables and Reagents

	Annual Costs (US$M)
Consumables (including diesel)	6
Reagents	109
SART	8
Transport	16
TOTAL	**139**

Note: Numbers may not add due to rounding.

Table 21-14: Consumables

Consumables	Annual Consumption	Annual Cost (US$M)
Primary Crusher Bowl/Mantles/etc.	1 set	0.6
Coarse Screen Top deck	8 decks	0.3
Coarse Screen Bottom deck	8 decks	0.3
Secondary Crusher Bowl/Mantles/etc.	2 sets	0.9
Lime Slaker - Consumable Parts	1 set	0.0
Lime Slaker - Balls	34 t/a	0.0
Anodes	361 units	0.2
Cathodes (incl. edge strips)	505 units	0.2
Diesel (Hot Water Cathode Wash)	639 kL	0.7
Mist Suppression (Beads)	53 m^3	0.7
Mobile Equipment fuel	270 kL/month	2.0
Lab consumables	(allowance)	0.2
TOTAL		**6.1**

Note: Numbers may not add due to rounding.

Table 21-15: Reagents

Reagent	Annual Consumption (t/a)	Annual Cost (US$M)
Sulphuric acid	5,595	1.1
Cement	87,600	16.2
Sodium cyanide	30,240	63.6
Lime	34,930	12.2
Extractant	134	1.4
Diluent	1,411	2.6
Clay	43	0.1
Anthracite	23	0.0
Garnet Sand	59	0.0
Filter Sand	59	0.0
Smoothing Agent	13	0.0
Cobalt Sulphate	1	0.0
Salt	5	0.0
Antiscalant (m3/a)	110	0.4
Zinc Dust	1,643	10.1
Lead Nitrate	164	0.5
Gold room reagents	233	0.5
TOTAL		**108.7**

Note: Numbers may not add due to rounding.

Table 21-16: SART Operating Costs

Reagent	Annual Consumption (t/a)	Annual Cost (US$M)
Sodium hydrosulphide	2,915	3.3
Lime (hydrated)	6,200	2.2
Sulphuric Acid	12,770	2.4
TOTAL		**7.9**

Note: Numbers may not add due to rounding.

Costs for the SART plant were derived from reported values.

21.4.4.3 Maintenance

Costs for maintenance are summarized in Table 21-17 below, where the total annual maintenance cost is US$26M/a, which is equivalent to a unit cost of US$1.18/t processed.

Annual maintenance spares and consumables costs were estimated at 8% of the total installed mechanical equipment, plate work, electrical and instrumentation equipment cost for the processing plant and infrastructure. Maintenance spares and consumables include:

- Mechanical equipment replacement parts
- Pipes, valves and fittings
- Electrical, instrumentation and control equipment, cable and replacement parts
- Bulk materials such as steel plate and general liners, miscellaneous structural steel

The plant maintenance spares and consumables exclude:

- Maintenance labour, which is included under labour costs.
- Special wear part and liners for the crushers and mills, which are included in consumable unit costs.

Building maintenance and power supply maintenance costs were based on an allowance of 4% of the total installed mechanical equipment, plate work, electrical and instrumentation equipment cost per year.

Maintenance costs for mobile vehicles were estimated from the number of vehicles, estimates of daily operating hours and unit rates specific to the type of vehicle. The costs for the future installation of the SART plant have been included, averaged over the life-of-mine.

Table 21-17: Operating Costs – Maintenance

WBS	Area Description	Total Equipment Cost (US$M)	Factor of Equipment Cost	Annual Maintenance (US$M)
3100	Crushing	33	8%	3
3200	Copper On/Off Circuit	44	8%	3
3400	Copper Processing (SX-EW)	95	8%	8
3500	Gold Circuit	66	8%	5
3600	Gold Processing (Merrill-Crowe)	12	8%	1
4300	Reagents & Services	8	8%	1
4000	On-Site Infrastructure	1	4%	0
5000	Off-Site Infrastructure	0	4%	0
1400	Ancillary Services	0	4%	0
Future	SART	66	8%	5
	Mobile Equipment Maintenance	1		1
TOTAL				**26**

Note: Numbers may not add due to rounding.

21.4.4.4 Labour

Costs for labour are summarized in Table 21-18 below, where the total annual labour cost is US$8M/a, which is equivalent to a unit cost of US$0.37/t processed.

Labour costs include all processing and maintenance costs and are based on salaries and labour rosters provided by Filo Mining and validated by Ausenco.

- On-costs: (payroll burdens) 32.5% was selected.
- All personnel are on shift schedule working 12 hours per day, 2 weeks on and 2 weeks off.
- Regular pay for the first 2,080 hours per year with no need to overtime payments.
- No annual leave and public holidays are accounted other than 2 weeks off every 4 weeks, as per the shift roster.

Transportation, recruitment and training costs are included as separate cost items in the G&A costs.

A breakdown of processing labour schedules and costs are summarized in Table 21-18 below.

Table 21-18: Operating Costs – Labour

Cost Centre	Number of People	Annual Cost (US$M)
Plant management	28	2
Shift crew	96	3
Laboratory and refinery	38	1
Maintenance	36	1
Mobile equipment operators	4	0
TOTAL	**202**	**8**

Note: Numbers may not add due to rounding.

21.4.5 Site G&A

Operating cost estimates for G&A were prepared by Ausenco and are summarized in Table 21-19, where the total annual G&A is US$37M/a, which is equivalent to a unit cost of US$1.67/t processed. The G&A costs include camp operations, G&A personnel, off-site offices as well as miscellaneous project costs.

Table 21-19: G&A Cost Summary

Cost Centre	Annual Cost (US$M)	Unit Cost (US$/t processed)
Labour	4	0.20
Processing and operations	1	0.03
Administration and other costs	9	0.41
Contracts	22	1.02
Mobile equipment maintenance	0	0.01
TOTAL	**37**	**1.67**

Note: Numbers may not add due to rounding.

Most G&A costs are based on benchmarked data from similar projects in South America. The following sections describe the build-up of the G&A area.

21.4.5.1 Labour

The annual labour costs for G&A were estimated at US$4 M/a, which is the equivalent of US$0.20/t processed as summarised in Table 21-20. This is based on 86 personnel and includes 32.5% on-costs.

Table 21-20: G&A Costs – Labour

Cost Centre	Number of People	Annual Cost (US$M)	Annual Cost (US$/t processed)
Admin including General Manager	7	1.2	0.05
Supply and clerks	22	0.8	0.04
IT support	4	0.1	0.00
Emergency and first aid	5	0.2	0.01
Camp manager	1	0.1	0.00
Transportation	19	1.0	0.05
HR	3	0.1	0.00
Environmental management	5	0.1	0.00
Environmental and Sustainability	20	0.8	0.04
TOTAL	**86**	**4.4**	**0.20**

Note: Numbers may not add due to rounding.

21.4.5.2 Processing and Operations

The Processing/Operations department cost is made up of the test work, training, safety equipment, and laboratory equipment maintenance costs.

Training cost is calculated as 2% of labour cost and US$200/person/year is allowed for safety equipment.

The cost items are summarized in Table 21-21. The annual processing and operations costs for G&A were estimated at US$670,000/a, which is the equivalent of US$0.03/t processed.

Table 21-21: G&A Costs – Processing and Operations

Cost Centre	Annual Cost (US$M)	Annual Cost (US$/t processed)
Metallurgical test work	0.10	0.00
Training	0.41	0.02
Safety equipment	0.06	0.00
Environmental test work	0.05	0.00
Laboratory equipment maintenance	0.05	0.00
TOTAL	**0.67**	**0.03**

Note: Numbers may not add due to rounding.

21.4.5.3 Administration and Other Costs

An allowance of US$750k per year was made for corporate travel. This allowance includes all travel and conferences for senior personnel.

Recruitment allowance of US$2,500/person was made for 10% estimated turnover rate.

Costs for camp lodging and catering were US$70 per person per day which was benchmarked against other projects.

The administration and other cost items are summarized in Table 21-22. The annual administration and other costs for G&A were estimated at US$8.9M/a, which is the equivalent of US$0.41/t processed.

Table 21-22: G&A Costs – Administration and Other

Cost Centre	Annual Cost (US$M)	Annual Cost (US$/t processed)
Travel expenses	0.8	0.04
Recruitment	0.1	0.00
Camp	5.4	0.25
Insurance	1.0	0.05
Allowances	1.7	0.08
TOTAL	**8.9**	**0.41**

Note: Numbers may not add due to rounding.

Other G&A costs include IT, mobile phones, couriers/post, legal and other feeds, government charges, in-house conferences cost, community relations, community development, local education/scholarships, office supplies, office furniture, external consultants, software, medical equipment/consumables for on-going drug and alcohol tests, lab consumables including reagents and chemicals, and recreational costs. These allowances were estimated separately and are presented here as a single sum.

21.4.5.4 Contracts

Road maintenance costs were calculated based on an allowance of 2% of capital costs for 500 km of gravel roads for shift workers transportation to the nearest city. Labour, equipment and material costs are included in road maintenance cost; but reconstruction and improvements are excluded.

Security, cleaning service, maintenance contractors, and effluent handling / garbage removal allowances were estimated based on Fil Mining's guidance. Security contract costs includes security personnel, management, camp lodging and catering as well as transport of security personnel to site.

Heap leach development is calculated at US$0.96/t based on annual tonnage placed on the heap.

Employee transport cost is estimated at US$30 per person per week over 1/3 year each, based on a total of 448 people.

The contracts cost items are summarized in Table 21-23. The annual contracts costs for G&A were estimated at US$22.4M/a, which is the equivalent of US$1.02/t processed.

Table 21-23: G&A Costs – Contracts

Cost Centre	Annual Cost (US$M)	Annual Cost (US$/t processed)
Road maintenance	0.9	0.04
Security	0.1	0.00
Cleaning service	0.1	0.00
Effluent handling / garbage removal	0.1	0.00
Heap leach development	21.0	0.96
Employee transport	0.2	0.01
TOTAL	**22.4**	**1.02**

Note: Numbers may not add due to rounding.

21.4.5.5 Mobile Vehicle Maintenance

Maintenance costs for mobile vehicles were estimated from the number of vehicles, estimates of daily operating hours and unit rates specific to the type of vehicle.

The mobile vehicle maintenance cost items are summarized in Table 21-24. The annual mobile vehicle maintenance costs for G&A were estimated at US$0.17M/a, which is the equivalent of US$0.01/t processed.

Table 21-24: G&A Costs – Mobile Vehicle Maintenance

Cost Centre	Annual Cost (US$M)	Annual Cost (US$/t processed)
Warehouse Forklift	0.08	0.00
Fire Truck	0.01	0.00
Crew Bus	0.05	0.00
Crew Van	0.02	0.00
Hazardous Response Vehicle	0.00	0.00
Mobile Gen-Sets	0.01	0.00
TOTAL	**0.17**	**0.01**

Note: Numbers may not add due to rounding.

22 ECONOMIC ANALYSIS

22.1 Cautionary Statement

Certain information and statements contained in this section 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 present value (NPV); 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 signification 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.

22.2 Methodology Used

Economic analysis was undertaken using a discounted cashflow model that was constructed in MS Excel®. The model used constant (real) 2023 USD and modelled the project cashflows in annual periods.

The model assumes a 24-month physical construction period, and a production duration of 13 years, including the final year where leaching is assumed to continue although little mining is taking place.

The model does not place the project within an estimated calendar timeline and is intended only as an indication of the economic potential of the project to assist in investment decisions. Between the date of this report and the commencement of construction, a period of time sufficient for the feasibility study work program to be executed must be allowed.

Important Note: The economic model considered only cashflows from the beginning of actual construction forward. Schedule and expenditure for the feasibility study, including technical and economic studies, engineering studies, cost estimating, resource delineation and infill drilling, pit slope geotechnical characterization, metallurgical sampling and test-work, associated exploration, strategic optimization, mine, plant and infrastructure design, permitting and other pre-construction activities were not modelled.

Attention is drawn to Section 26 where the work plan and costs for the continued development of the project are summarized.

22.3 Economic Results

Table 22-1 shows a summary of key project parameters and project economics. LOM project annual cash flow is shown in Table 22-5.

Table 22-1: Project Economic Summary

Project Metric	Units	Value
Pre-Tax NPV (8%)	US$M	2,040
Pre-tax IRR	%	24%
Post-Tax NPV (8%)	US$M	1,310
Post-Tax IRR	%	20%
Undiscounted Post-Tax Cash Flow (Life of Mine)	US$M	3,560
Average Operating Margin*	%	60%
Payback Period from Start of Processing (Undiscounted, Post-Tax Cash Flow)	years	3.4
Initial Capital Expenditures	US$M	1,805
LOM Sustaining Capital Expenditure (Excluding Closure)	US$M	140
LOM C-1 Cash Costs (Co-Product)	US$/lb CuEq	1.54
Nominal Process Capacity	t/d	60,000
Mine Life (including pre-stripping)	years	13
Average Annual Copper Production**	tonnes	66,000
Average Annual Gold Production**	oz	168,000
Average Annual Silver Production**	oz	9,256,000
LOM Recovery – Copper***	%	78%
LOM Recovery – Gold	%	70%
LOM Recovery – Silver	%	83%

Notes: * Operating Margin = Operating Cashflow/Net Revenue. ** Rounded and excluding final year of minimal leach operation. *** Excluding 1% Cu recovery to concentrate for SART process.

22.4 Financial Model Parameters

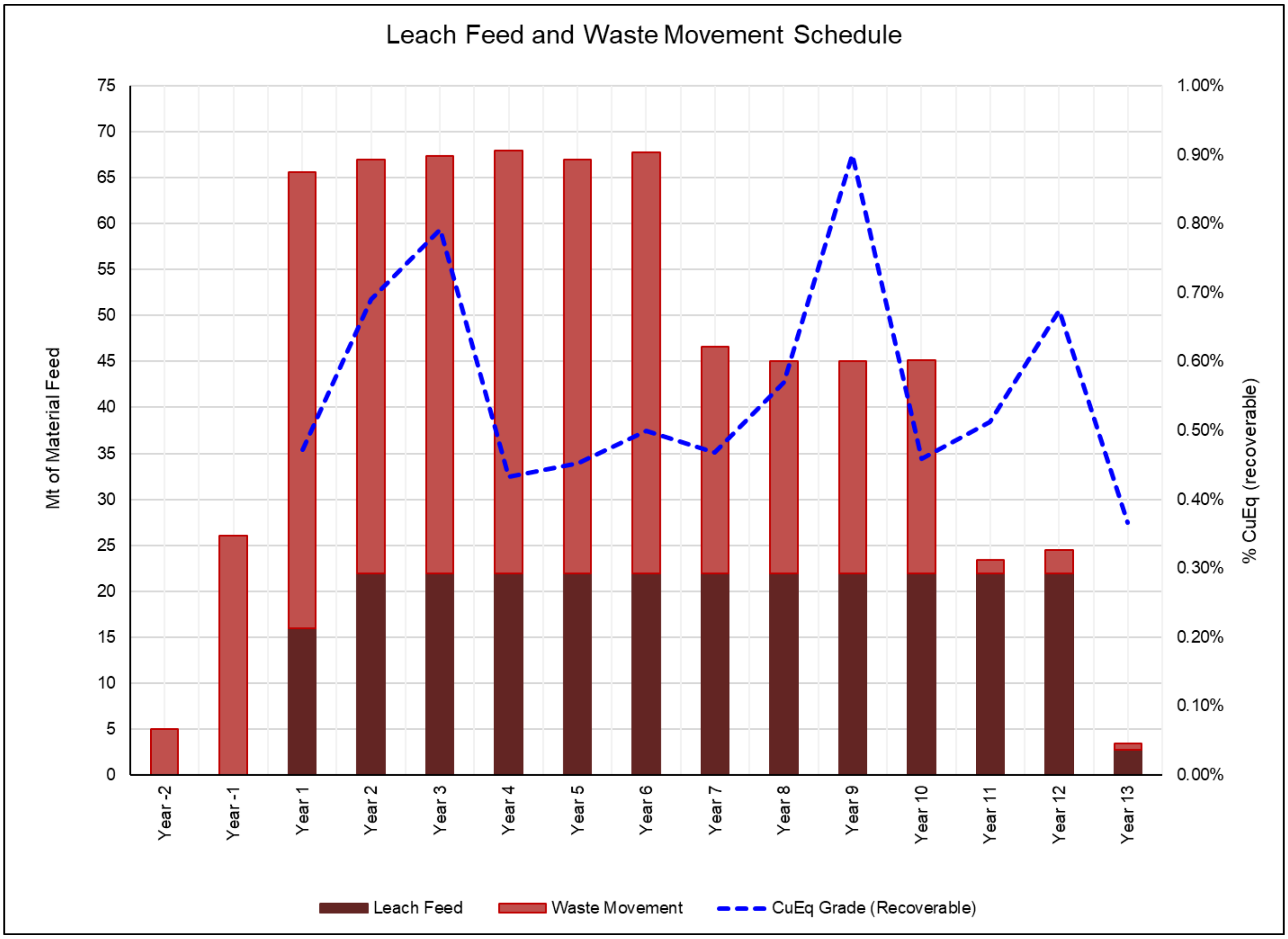
22.4.1 Mineral Resources, Mineral Reserve, and Mine Life

The mine plan evaluated for the purposes of the analysis also represents the Mineral Reserves for the Project. No inferred material is included in the material scheduled for processing. This was achieved by assigning it zero grade in the mine planning process.

Figure 22-1, Figure 22-2 and Figure 22-3 illustrate a summary of mine physicals and metal production on an annual basis.

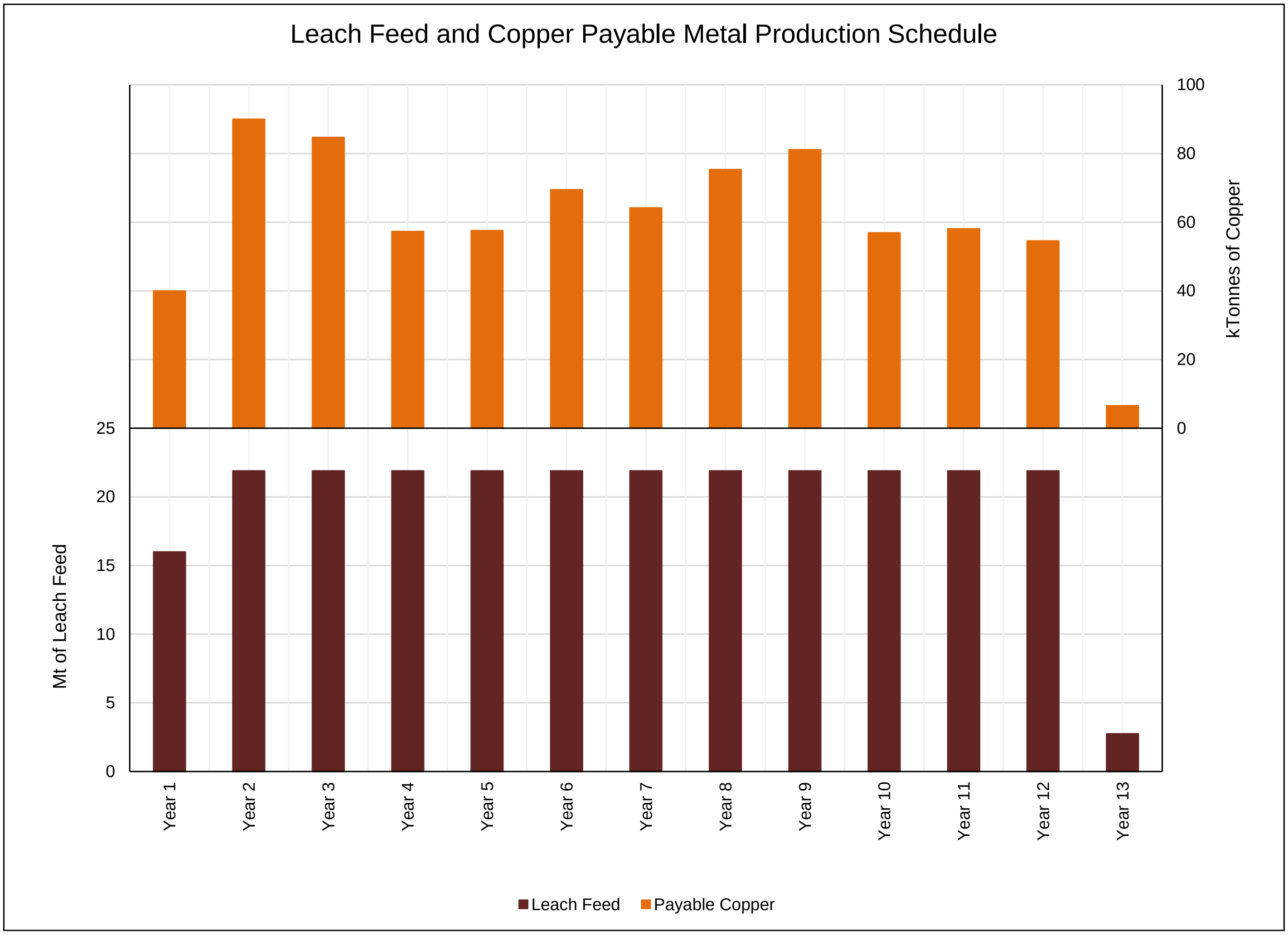
Table 22-2, Table 22-3, and Table 22-4 summarize the mine production and net revenue calculations by year.

Figure 22-1: Leach Feed and Waste Movement



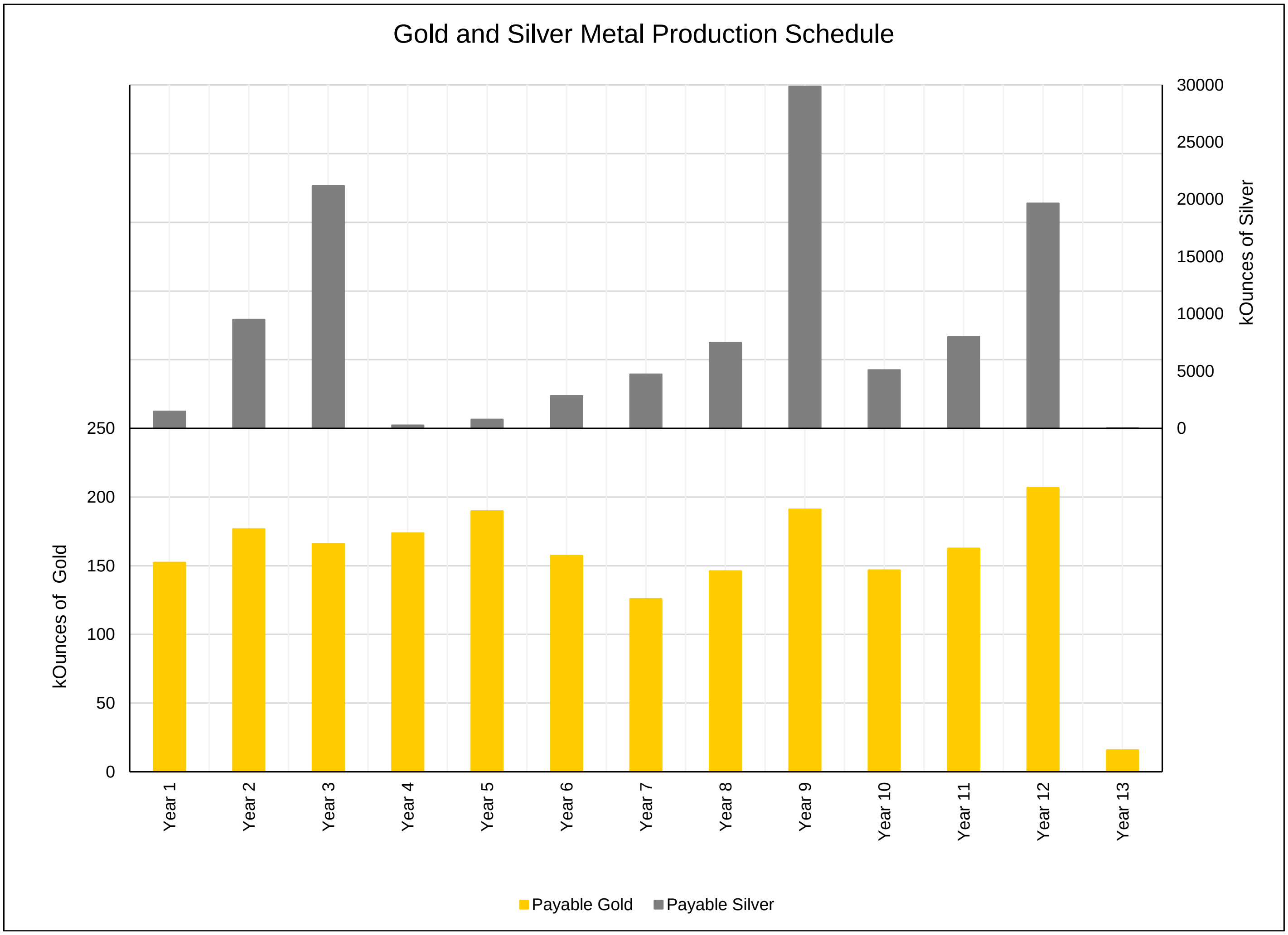
Source: SRK, 2023

Figure 22-2: Leach Feed and Payable Copper



Source: SRK, 2023

Figure 22-3: Payable Gold and Silver



Source: SRK, 2023

Table 22-2: Production Summary

Production Summary	Units	LOM Total	1	2	3	4	5	6	7	8	9	10	11	12	13
Leach Feed	Mt	**259.6**	16.0	21.9	21.9	21.9	21.9	21.9	21.9	21.9	21.9	21.9	21.9	21.9	2.7
Waste Movement	Mt	**407.1**	49.6	45.1	45.5	46.1	45.1	45.8	24.7	23.1	23.1	23.2	1.5	2.6	0.7
Payable Gold	000 oz	**2,012.0**	152.4	176.7	166.1	173.9	189.9	157.5	125.9	146.2	191.2	146.9	162.8	206.8	15.8
Payable Silver	000 oz	**111,066**	1,504	9,526	21,194	284	798	2,854	4,726	7,502	29,868	5,097	8,004	19,665	44
Payable Copper	Kt	**795.9**	40.0	90.0	84.7	57.3	57.6	69.5	64.2	75.4	81.1	56.9	58.1	54.5	6.5

Table 22-3: Recovered Metals by Country

			Period												
Metal Production by country			1	2	3	4	5	6	7	8	9	10	11	12	13
Commodity Prices															
Gold Price	US$/oz		1,700	1,700	1,700	1,700	1,700	1,700	1,700	1,700	1,700	1,700	1,700	1,700	1,700
Silver Price	US$/oz		21.00	21.00	21.00	21.00	21.00	21.00	21.00	21.00	21.00	21.00	21.00	21.00	21.00
Copper Price	US$/lb		3.72	3.72	3.72	3.72	3.72	3.72	3.72	3.72	3.72	3.72	3.72	3.72	$3.72
Recovered Metals															
Argentina															
Gold	koz	**1,745**	123.7	158.8	122.4	83.0	182.2	149.5	124.7	140.4	188.0	115.9	155.6	198.0	2.7
Silver	koz	**100,781**	1,292	8,586	17,061	59	754	2,815	4,707	6,319	29,638	4,455	6,399	18,690	4
Copper	kt	**664**	32.1	79.2	68.0	6.5	53.7	66.4	62.4	71.4	78.6	39.5	55.0	51.1	0.6
Chile															
Gold	koz	**267**	28.7	18.0	43.7	90.8	7.7	8.0	1.2	5.8	3.2	31.1	7.2	8.9	13.1
Silver	koz	**10,285**	211	940	4,133	225	44	39	19	1,183	230	642	1,605	975	40
Copper	kt	**123**	7.5	9.9	15.8	50.7	3.2	2.3	1.1	3.1	1.5	16.9	2.4	2.7	6.0

Table 22-4: Net Revenue Detail

Net Revenues				Period												
				1	2	3	4	5	6	7	8	9	10	11	12	13
Leach Revenue Argentina		**Total**	**NPV**													
Gross Revenue																
Gold	$M	**2,966.0**	**1,620.1**	210.2	269.9	208.0	141.2	309.7	254.1	211.9	238.6	319.6	197.0	264.5	336.5	4.7
Silver	$M	**2,116.4**	**1,078.7**	27.1	180.3	358.3	1.2	15.8	59.1	98.9	132.7	622.4	93.6	134.4	392.5	0.1
Copper	$M	**5,453.6**	**3,009.0**	263.4	650.0	558.5	53.5	440.9	544.7	511.8	586.2	645.0	324.4	451.0	419.2	4.8
Total Gross Revenue	$M	**10,535.9**	**5,707.7**	500.8	1,100.2	1,124.8	195.9	766.5	857.9	822.6	957.6	1,587.0	615.0	849.9	1,148.3	9.5
Total Payable Revenue	$M	**10,511.8**	**5,695.3**	500.3	1,098.1	1,121.0	195.8	766.0	857.0	821.4	956.0	1,580.4	613.8	848.3	1,144.0	9.5
Total Refining Charges	$M	**35.9**	**18.3**	0.5	3.1	6.0	0.0	0.3	1.0	1.7	2.3	10.4	1.6	2.3	6.6	0.0
Total Freight and Insurance	$M	**75.6**	**41.1**	3.2	8.4	8.7	0.7	5.0	6.4	6.3	7.4	11.5	4.3	6.0	7.5	0.2
Mine Head and Export duty	$M	**296.0**	**161.2**	13.8	33.2	33.6	4.1	20.4	24.3	22.4	27.5	46.4	16.1	22.8	31.3	0.2
Total Argentina Leach Net Revenue	**$M**	**10,104.3**	**5,474.7**	**482.8**	**1,053.4**	**1,072.6**	**191.0**	**740.3**	**825.3**	**790.9**	**918.9**	**1,512.1**	**591.8**	**817.3**	**1,098.7**	**9.1**
Leach Revenue Chile																
Gross Revenue																
Gold	$M	**454.5**	**280.2**	48.8	30.5	74.3	154.4	13.1	13.7	2.1	9.9	5.4	52.8	12.2	15.1	22.3
Silver	$M	**216.0**	**124.5**	4.4	19.7	86.8	4.7	0.9	0.8	0.4	24.8	4.8	13.5	33.7	20.5	0.8
Copper	$M	**1,009.2**	**614.4**	61.5	81.4	129.8	416.2	26.0	18.9	8.7	25.2	12.3	138.5	19.8	22.1	48.9
Total Gross Revenue	$M	**1,679.7**	**1,019.1**	**114.7**	**131.7**	**290.9**	**575.4**	**40.0**	**33.4**	**11.2**	**60.0**	**22.5**	**204.8**	**65.7**	**57.6**	**72.0**
Total Payable Revenue	$M	**1,677.1**	**1,017.5**	**114.6**	**131.5**	**290.0**	**575.2**	**40.0**	**33.3**	**11.2**	**59.7**	**22.5**	**204.6**	**65.3**	**57.4**	**71.9**
Total Refining Charges	$M	**3.7**	**0.0**	0.1	0.3	1.5	0.1	0.0	0.0	0.0	0.4	0.1	0.2	0.6	0.3	0.0
Total Freight and Insurance	$M	**12.7**	**7.7**	0.7	1.0	2.0	4.5	0.3	0.2	0.1	0.5	0.2	1.6	0.5	0.4	0.6
Private NSR Royalty	$M	**17.0**	**0.0**	0.0	0.0	0.0	8.6	0.6	0.5	0.2	0.9	0.3	3.0	1.0	0.8	1.1
Total Chile Leach Net Revenue	**$M**	**1,643.8**	**998.6**	**113.8**	**130.1**	**286.5**	**562.0**	**39.1**	**32.6**	**10.9**	**57.9**	**21.9**	**199.7**	**63.3**	**55.8**	**70.3**
SART Revenue																
Payable Revenue - SART Argentina	**$M**	**67.5**	**36.6**	**2.9**	**7.1**	**6.6**	**0.7**	**5.4**	**6.7**	**6.2**	**7.2**	**8.1**	**4.3**	**6.0**	**6.2**	**0.1**
Total TCRC Freight	$M	**4.3**	**2.3**	0.2	0.5	0.4	0.0	0.3	0.4	0.4	0.5	0.5	0.3	0.4	0.4	0.0
Net Revenue - SART Argentina	**$M**	**63.2**	**34.2**	**2.7**	**6.7**	**6.2**	**0.6**	**5.0**	**6.3**	**5.8**	**6.7**	**7.6**	**4.0**	**5.6**	**5.8**	**0.1**
Payable Revenue - SART Chile	**$M**	**12.3**	**7.5**	**0.7**	**1.0**	**1.5**	**5.1**	**0.3**	**0.3**	**0.1**	**0.3**	**0.2**	**1.7**	**0.2**	**0.3**	**0.6**
Total TCRC Freight	$M	**0.8**	**0.5**	0.0	0.1	0.1	0.3	0.0	0.0	0.0	0.0	0.0	0.1	0.0	0.0	0.0
Net Revenue - SART Chile	**$M**	**11.5**	**7.0**	**0.7**	**0.9**	**1.4**	**4.8**	**0.3**	**0.3**	**0.1**	**0.3**	**0.1**	**1.6**	**0.2**	**0.3**	**0.6**
Total SART Net Revenue	**$M**	**74.7**	**41.2**	**3.4**	**7.6**	**7.6**	**5.4**	**5.4**	**6.6**	**5.9**	**7.0**	**7.7**	**5.6**	**5.8**	**6.1**	**0.7**

Table 22-5: Net Revenue Summary

				Period												
Total Net Revenue		**Total**	**NPV**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**
Total Argentina Leach Net Revenue	**$M**	**10,104.3**	**5,474.7**	482.8	1,053.4	1,072.6	191.0	740.3	825.3	790.9	918.9	1,512.1	591.8	817.3	1,098.7	9.1
Total Chile Leach Net Revenue	**$M**	**1,643.8**	**998.6**	113.8	130.1	286.5	562.0	39.1	32.6	10.9	57.9	21.9	199.7	63.3	55.8	70.3
Total SART Net Revenue	**$M**	**74.7**	**41.2**	3.4	7.6	7.6	5.4	5.4	6.6	5.9	7.0	7.7	5.6	5.8	6.1	0.7
Grand Total Minesite Net Revenue	**$M**	**11,822.8**	**6,514.5**	600.0	1,191.1	1,366.7	758.3	784.7	864.5	807.7	983.8	1,541.7	797.1	886.4	1,160.5	80.1

22.4.2 Metallurgical Recoveries

Metallurgical recoveries, as described in Section 13, were applied in accordance with advice from Ausenco in the economic model. The ROM grades delivered to the heap leach pads were the basis for the recovery calculations. The algorithms were applied to the annual average grades. In reality, variability of the grades within the annual period will result in slightly different outcomes, but this is not believed to be material at a PFS level. The recovery algorithms employed are detailed Section 13, and the LOM recoveries are provided in Table 22-6.

Table 22-6: LOM Recoveries

Metal	Value
Copper (excluding 1% from SART)	78%
Gold	70%
Silver	83%

22.4.3 Freight, Smelting and Refining Terms

Three products are contemplated. A copper cathode, a gold & silver doré and a copper precipitate from the SART circuit. The terms assumed for these are set out in Section 19. The freight, treatment and refining terms for the copper precipitate are based on industry standard terms. Given the relatively small volume of precipitate at Filo del Sol, a separate study was not considered to be warranted. The precipitate was estimated to be very high grade (relative to flotation products) at 65% contained copper. SRK understands that this is typical for the precipitate produced by SART circuits, which consists primarily of precipitated chalcocite.

22.4.4 Metal Prices

Flat real prices were assumed for the life of the project. Table 22-7 shows the price assumptions used. SRK considers these prices to be reasonable for a PFS study such as this, and within the range of consensus forecasts of which SRK has knowledge.

Table 22-7: Pricing Assumptions for Economic Analysis

Commodity Market Prices	Units	Price
Copper Price excl. 2.0% Cathode Premium	US$/lb	3.65
Gold Price	US$/oz	1,700
Silver Price	US$/oz	21.00

A 2.0% premium associated with selling cathode copper from the SX-EW plant was assumed, increasing the received price to US$3.72 per pound.

22.4.5 Operating Costs

The operating costs modelled are detailed in Section 22.

Table 22-8 summarizes the overall unit costs resulting from the incorporation into the economic model.

Table 22-8: Operating and Sustaining Costs

Operating & Sustaining Costs	LOM (US$M)	Unit Opex (US$/t)	Unit Opex (US$/lb Payable Cu.Eq)
Mine Operating Cost	**1,720**	**6.63**	**$0.52**
Process Operating Costs			
Power	471	1.82	0.14
Consumables	1,647	6.35	0.50
Maintenance	309	1.19	0.09
Labour	96	0.37	0.03
Total Processing Costs	2,523	9.72	0.76
General and Administrative	434	1.67	0.13
Total Operating Costs	**4,677**	**18.01**	**1.42**
Offsite Unit Costs			
Total Payable Deduction	30	**0.11**	0.01
Total TCRC Freight and Insurance	133	**0.51**	0.04
Total Royalties	254	**0.98**	0.08
Total Offsite Unit Costs	417	1.61	0.13
Sustaining Capex	140	0.54	0.04
Closure	69	0.26	0.02
All-in Sustaining Costs	**5,303**	**20.42**	**1.61**

The mine operating costs were US$2.58 per tonne of total material moved.

22.4.6 Capital Costs

Capital costs used for the economic evaluation are summarized in Table 22-9 and shown by period in Table 22-12. Initial capital costs were scheduled to be spent equally across the two-year timeline for project construction, except for mining costs which were explicitly scheduled to match the expected spend during the two years of project construction.

Sustaining capital costs for mining were scheduled to match the expected spend profile developed as part of the mining cost estimation process and are matched to the production and waste movement profile.

In the case of plant and infrastructure, the total sustaining capital spend estimated was scheduled by pro-rating it to tonnes processed in each period. This proxy is reasonable for a PFS.

Table 22-9: Capital Cost Summary

Capital Expenditure	Initial (US$M)	Sustaining (US$M)	LOM (US$M)
Mine	230	9	238
Process Plant	610	131	741
On-site Infrastructure	117	140	258
Off-site Infrastructure	188	0	188
Subtotal Direct Costs	**1,145**	**140**	**1,285**
Indirects	185	0	185
EPCM Services	149	0	149
Owner's Costs	50	0	50
Provisions	275	0	275
Subtotal Indirect Costs	**660**	**0**	**660**
Project Total	**1,805**	**140**	**1,945**

22.4.7 Royalties

Royalties were applied in both Argentina and Chile. The mine plan was produced with ore and waste volumes being attributed to the country of origin in accordance with the in-situ location. For the purposes of estimating revenue, the payable metal and offsite costs were attributed by country or origin. For operating costs (where relevant) the costs were assumed to be prorated according to the proportion of total material mined in each country (for mining costs) and the proportion of heap leach material placed according to country of origin (for processing and G&A costs). SRK considers that this is a reasonable approach for the PFS. A true accounting model that matched costs and revenue is possible, but the complexity is not warranted at this stage.

22.4.7.1 Argentinian Royalties

Argentinian royalties were estimated at 3% of “mine head revenue” which is defined as net revenue minus all operating costs other than mining costs.

22.4.7.2 Chilean Royalties

Chilean royalties were estimated based on a private 1.5% NSR royalty applicable after recovery of costs by the owner. This cost recovery was estimated to take 3 years of production (estimated on a whole-of project basis), and the royalty was applied thereafter.

22.4.8 Argentinian Export Duty

A duty was applied to copper revenue derived from the Argentinian-hosted material. This duty is calculated on a sliding scale that is a function of the realised copper price and a “decreed” price. For the cases presented here, the rate is estimated to be 1.08%.

22.4.9 Working Capital

Working capital was estimated based on revenue for accounts receivable, and on operating costs for accounts payable and stores stock movements. A contraction discount to account for stores losses and obsolescence of 5% of stores value per year was also applied. Table 22-10 summarizes the assumption made.

Table 22-10: Working Capital Assumptions for Economic Analysis

Working Capital	Units	Value
Receivables outstanding	days	15
Payables outstanding	days	30
Annual operating costs in stores	% of annual opex	12%
Contraction discount (stores value lost per year)	% of stores balance	5%

22.4.10 Taxes

Two tax models were created, splitting the revenues and costs by country, and estimating taxable income for each. Argentina-sourced income taxes were calculated using a sliding scale of between 25% and 35%. The overall effective tax rate (taxes-paid divided by taxable-income) was approximately 30%. Chilean sourced income was taxed at a flat rate of 27%.

For the proportion of production and estimated profits attributable to Chile, an additional Mining Tax applies. This is based on a sliding scale of rates that vary according to a calculated margin. The tax rate ranges from 5% to 14% of taxable income. For the purposes of estimating this tax, no loss carry-forward was modelled. The effective average rate of 7.8% was applied only to years of positive taxable income, resulting in a payment of US$65.2 M over the life of the project. SRK considers it possible that more detailed analysis, and consideration of how periods of loss are utilized, could result in a reduction in this effective rate under base case conditions.

Tax depreciation was estimated using simplified assumptions. The financial analysis was undertaken using a non-accounting model where revenues and costs were not explicitly matched for true tax accounting purposes. Taxes payable should be considered as high-level estimates only.

22.4.11 Closure Costs and Salvage Value

An allowance of US$68.5 M was made for closure, based on an estimate developed by Knight Piesold and supplied via Ausenco. The spending was scheduled to occur across the three years following the cessation of production. No provision or accrual for closure was made (cash or otherwise) for the purposes of the economic evaluation. A requirement to undertake progressive closure, or to post a cash bond, would affect the timing of these cashflows.

No net-positive salvage value was assumed for any items. For the plant and infrastructure, salvage value was assumed to be netted-off within the closure cost estimate. In the case of the mining fleet, optimization of sustaining capital expenditure was assumed, rendering salvage value as zero.

22.4.12 Financing

No consideration of financing was made. The model considers the cashflow only at an asset level and assumes 100% equity ownership.

22.4.13 Inflation

The modelling was primarily undertaken in real 2023 USD with no inflation applied to either commodity prices or costs. An assumption of USD accounting was made and nominal dollar modelling (using an assumed inflation rate of 1.80%) was used where carry-forward balances were present. This was restricted to depreciation balance carry-forwards and accounts payable (AP) and accounts receivable (AR) working capital estimates. These cashflows were then converted back to real USD values before being re-incorporated in the cash flow calculations.

22.5 Financial Results

Analysis of the project demonstrates that the mine plan has positive economics under the assumptions used. The project post-tax NPV at an 8% discount rate is estimated to be US$1,310 M, with an IRR of 20%. The project financial summary is shown in Table 22-11. The year-by-year cashflows are summarized in Table 22-13.

Table 22-11: Project Economic Summary

Project Metric	Units	Value
Pre-Tax NPV (8%)	US$M	2,040
Pre-tax IRR	%	24%
Post-Tax NPV (8%)	US$M	1,310
Post-Tax IRR	%	20%
Undiscounted Post-Tax Cash Flow (Life of Mine)	US$M	3,560
Average Operating Margin*	%	60%
Payback Period from Start of Processing (Undiscounted, Post-Tax Cash Flow)	years	3.4
Initial Capital Expenditures	US$M	1,805
LOM Sustaining Capital Expenditure (Excluding Closure)	US$M	140
LOM C-1 Cash Costs (Co-Product)	US$/lb CuEq	1.54
Nominal Process Capacity	t/d	60,000
Mine Life (including pre-stripping)	years	13
Average Annual Copper Production**	tonnes	66,000
Average Annual Gold Production**	oz	168,000
Average Annual Silver Production**	oz	9,256,000
LOM Recovery – Copper***	%	78%
LOM Recovery – Gold	%	70%
LOM Recovery – Silver	%	83%

Notes: * Operating Margin = Operating Cashflow/Net Revenue. ** Rounded and excluding final year of minimal leach operation. *** Excluding 1% Cu recovery to concentrate for SART process.

Table 22-12: Capital Expenditure by Period

Capital Cost Summary		Total	NPV	Period -2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Mine	$M	230	213	114.8	114.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Process Plant	$M	610	565	305.0	305.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
On-site Infrastructure	$M	117	109	58.7	58.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Off-site Infrastructure	$M	188	174	93.9	93.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Subtotal Direct Costs	**$M**	**1,145**	**1,061**	**572.4**	**572.4**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**
Indirects	$M	185	172	92.6	92.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
EPCM Services	$M	149	138	74.5	74.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Owner's Costs	$M	50	46	24.9	24.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Provisions	$M	275	255	137.7	137.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Subtotal Indirect Costs	**$M**	**660**	**611**	**329.8**	**329.8**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**
Project Total	**$M**	**1,805**	**1,672**	**902.3**	**902.3**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**
Project Total Sustaining Capex	$M	140	78	0.0	0.0	8.4	11.8	11.7	11.6	12.6	12.5	13.3	12.0	11.8	11.1	11.1	11.1	1.4	0.0	0.0	0.0	0.0
Closure Capex	$M	69	19	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	22.8	22.8	22.8	0.0
Grand Total Project Capex	**$M**	**2,013**	**1,769**	**902.3**	**902.3**	**8.4**	**11.8**	**11.7**	**11.6**	**12.6**	**12.5**	**13.3**	**12.0**	**11.8**	**11.1**	**11.1**	**11.1**	**1.4**	**22.8**	**22.8**	**22.8**	**0.0**

Table 22-13: Summary Cashflow by Period

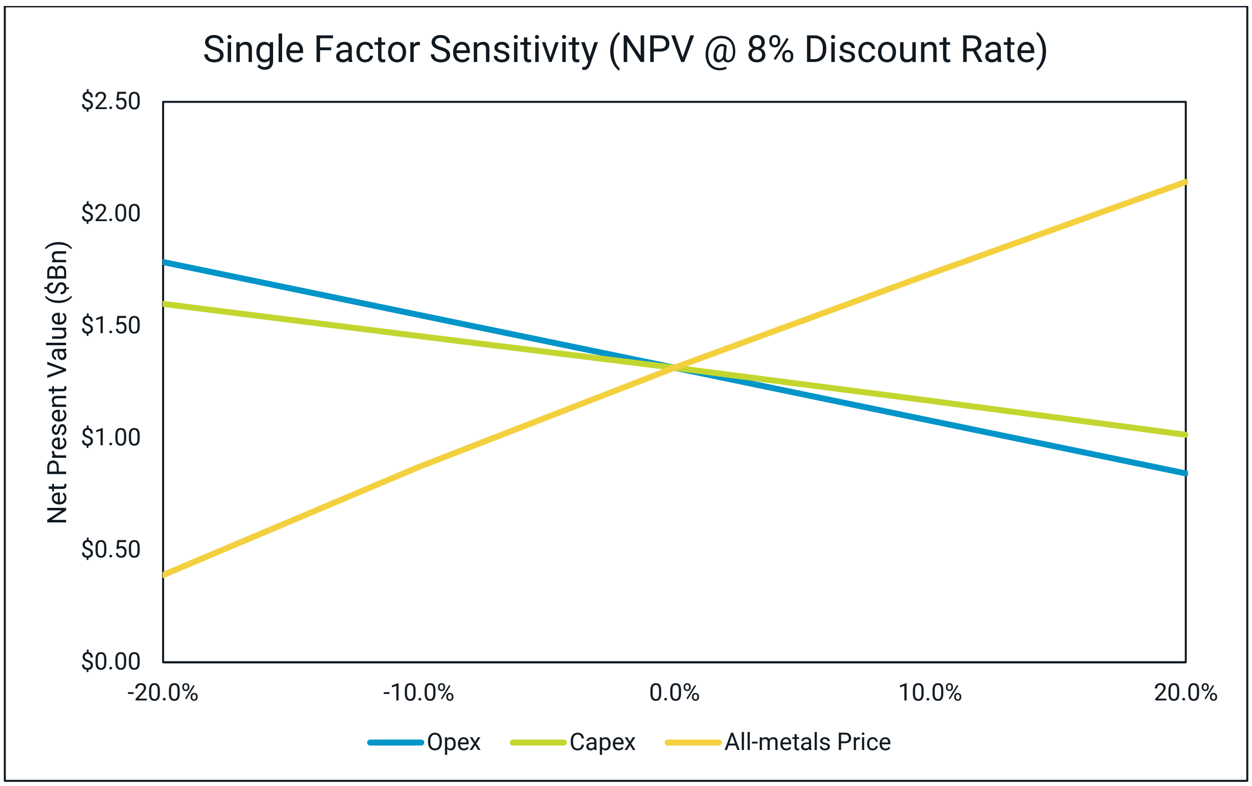
Summary Cashflow		Total	NPV	Period -2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Grand Total Minesite Net Revenue	**$M**	**11,823**	**6,515**	**0.0**	**0.0**	**600.0**	**1,191.1**	**1,366.7**	**758.3**	**784.7**	**864.5**	**807.7**	**983.8**	**1,541.7**	**797.1**	**886.4**	**1,160.5**	**80.1**	**0.0**	**0.0**	**0.0**	**0.0**
Operating Costs																						
Mine	$M	1,720	1,032	0.0	0.0	199.6	199.1	191.7	173.6	152.8	163.0	120.7	124.5	117.3	121.2	70.7	75.2	10.8	0.0	0.0	0.0	0.0
Processing and Infrastructure	$M	2,523	1,387	0.0	0.0	148.3	215.9	212.3	204.0	215.3	212.9	216.0	215.7	214.4	211.9	216.0	215.6	25.1	0.0	0.0	0.0	0.0
General and Administrative	$M	434	239	0.0	0.0	26.7	36.6	36.6	36.6	36.6	36.6	36.6	36.6	36.6	36.6	36.6	36.6	4.6	0.0	0.0	0.0	0.0
Total Operating Costs	**$M**	**4,677**	**2,658**	**0.0**	**0.0**	**374.5**	**451.6**	**440.6**	**414.2**	**404.7**	**412.4**	**373.3**	**376.8**	**368.3**	**369.6**	**323.2**	**327.3**	**40.5**	**0.0**	**0.0**	**0.0**	**0.0**
Operating Cashflow	**$M**	**7,146**	**3,856**	**0.0**	**0.0**	**225.5**	**739.5**	**926.1**	**344.1**	**380.1**	**452.1**	**434.5**	**607.0**	**1,173.4**	**427.5**	**563.3**	**833.2**	**39.6**	**0.0**	**0.0**	**0.0**	**0.0**
Grand Total Project Capex	**$M**	**2,013**	**1,769**	**902.3**	**902.3**	**8.4**	**11.8**	**11.7**	**11.6**	**12.6**	**12.5**	**13.3**	**12.0**	**11.8**	**11.1**	**11.1**	**11.1**	**1.4**	**22.8**	**22.8**	**22.8**	**0.0**
Working Capital	$M	30	44	0.0	0.0	41.1	29.8	9.7	-23.2	3.1	6.0	-1.5	9.7	25.0	-27.7	3.9	13.6	-54.6	-4.8	0.0	0.0	0.0
Pre-Tax Cashflow	**$M**	**5,103**	**2,043**	**-902.3**	**-902.3**	**176.0**	**697.9**	**904.8**	**355.7**	**364.4**	**433.5**	**422.7**	**585.4**	**1,136.6**	**444.2**	**548.3**	**808.6**	**92.9**	**-18.0**	**-22.8**	**-22.8**	**0.0**
Total Tax	$M	1,541	729	0.0	0.0	0.0	0.0	56.1	63.0	112.8	136.0	129.9	169.3	344.4	123.5	166.8	246.4	11.0	-3.7	-4.4	-5.7	-2.7
Post-Tax Net Cash Flow (Real)	**$M**	**3,562**	**1,314**	**-902.3**	**-902.3**	**176.0**	**697.9**	**848.7**	**292.7**	**251.6**	**297.6**	**292.8**	**416.2**	**792.2**	**320.7**	**381.5**	**562.2**	**81.9**	**-14.3**	**-18.4**	**-17.1**	**2.7**

22.6 Sensitivity Analysis

Sensitivity analysis was conducted to estimate the response of the project NPV to changes in assumptions on key inputs of metals prices, capital costs and operating costs. The results across a range of +/-20% relative to base-case assumptions are shown in Figure 22-4 and Figure 22-5. The project maintains a positive NPV across the range tested.

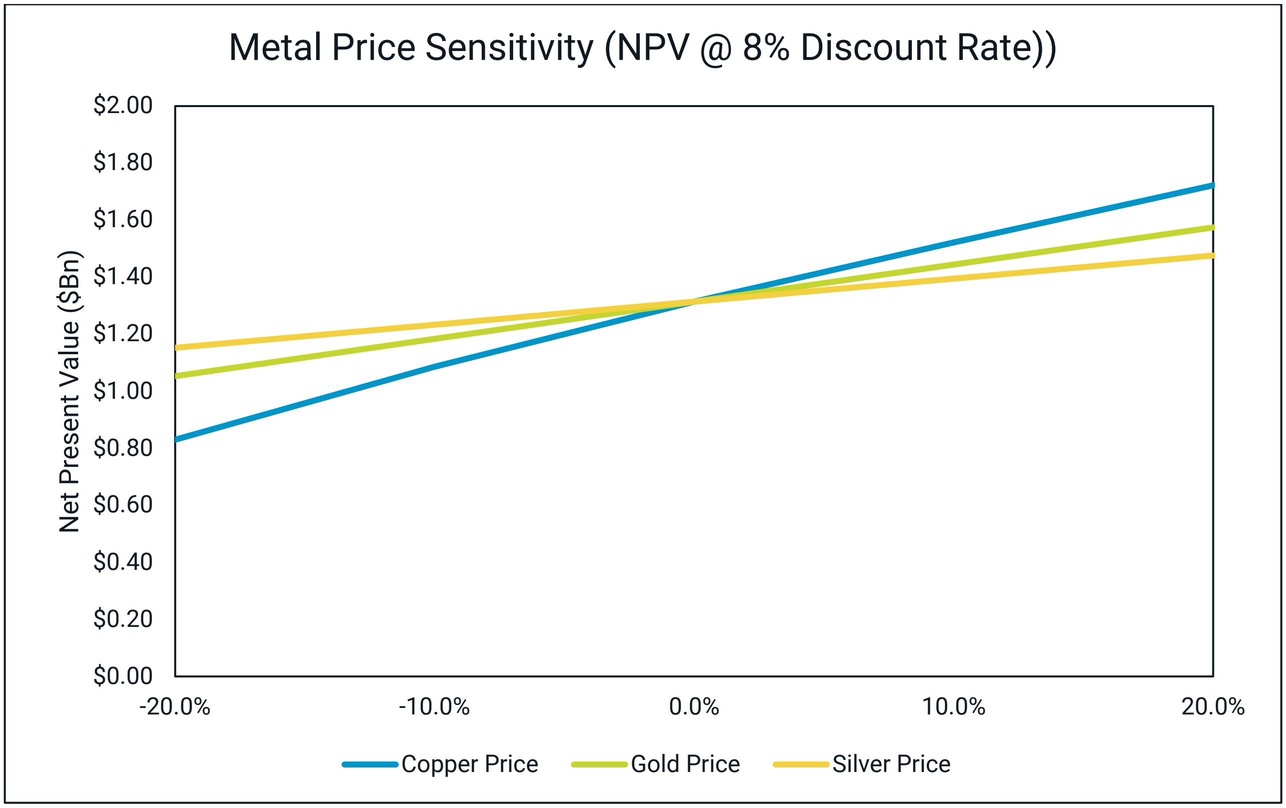
The results of this simplified testing do not reflect the embedded optionality within the project. In reality, significant changes in key economic parameters would lead to a response in the way the project would be operated. In general, mitigation of downside outcomes, and enhancement of upside opportunities may lead to better outcomes than what the sensitivity analysis suggests.

Figure 22-4: Single-Factor Sensitivity Chart



Source: SRK, 2023

Figure 22-5: Metal Price Sensitivity Chart



Source: SRK, 2023

Further analysis of key risks was undertaken by varying certain parameters and is summarized in Figure 22-6.

22.6.1.1 Note on Leach Revenue Timing

The base-case analysis assumed no significant delay between mining and the receipt of revenue from the leached metal. It can be seen from the tornado diagram in Figure 22-6 that this assumption is likely not material from an overall project perspective, but SRK recommends explicit modelling of revenue timing based on the modelled leach kinematics for the Feasibility Study.

22.6.2 Sensitivity Tables

Table 22-14to Table 22-17show the sensitivity of project NPV to variations in various key factors. They are generally presented as two-factor tables so that the total effect of combinations of assumptions can be seen.

Table 22-14: Opex and Capex Sensitivity – NPV @ 8% (US$B)

		Operating Costs				
		-20.0%	-10.0%	0.0%	10.0%	20.0%
Capital Costs	-20.0%	$2.07	$1.83	$1.60	$1.36	$1.13
	-10.0%	$1.93	$1.69	$1.46	$1.22	$0.98
	0.0%	$1.79	$1.55	**$1.31**	$1.08	$0.84
	10.0%	$1.64	$1.40	$1.17	$0.93	$0.69
	20.0%	$1.49	$1.25	$1.01	$0.78	$0.54

Table 22-15: Metal Price and Discount Rate Sensitivity – NPV @ 8% (US$B)

		Discount Rate				
		6.0%	7.0%	8.0%	9.0%	10.0%
Metal Prices	-20.0%	$0.66	$0.52	$0.39	$0.27	$0.17
	-10.0%	$1.21	$1.03	$0.87	$0.72	$0.59
	0.0%	$1.72	$1.51	**$1.31**	$1.14	$0.98
	10.0%	$2.20	$1.95	$1.73	$1.53	$1.35
	20.0%	$2.67	$2.39	$2.14	$1.92	$1.71

Table 22-16: Metal Price and Capex Sensitivity – NPV @ 8% (US$B)

		Metal Prices				
		-20.0%	-10.0%	0.0%	10.0%	20.0%
Capex	-20.0%	$0.87	$1.34	$1.79	$2.20	$2.61
	-10.0%	$0.63	$1.11	$1.55	$1.97	$2.38
	0.0%	$0.39	$0.87	**$1.31**	$1.73	$2.14
	10.0%	$0.15	$0.63	$1.08	$1.50	$1.91
	20.0%	($0.10)	$0.39	$0.84	$1.26	$1.67

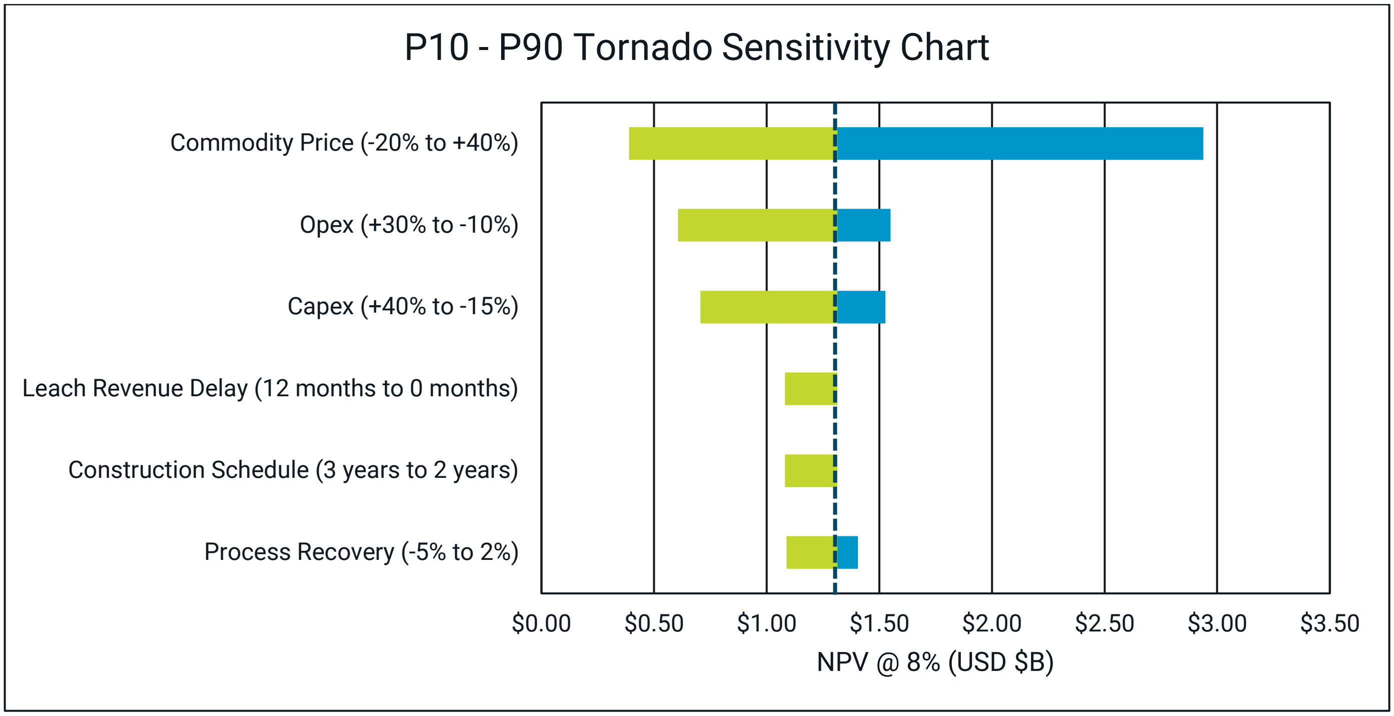
Table 22-17: Metal Price and Opex Sensitivity – NPV @ 8% (US$B)

		Metal Prices				
		-20.0%	-10.0%	0.0%	10.0%	20.0%
Opex	-20.0%	$0.87	$1.34	$1.79	$2.20	$2.61
	-10.0%	$0.63	$1.11	$1.55	$1.97	$2.38
	0.0%	$0.39	$0.87	**$1.31**	$1.73	$2.14
	10.0%	$0.15	$0.63	$1.08	$1.50	$1.91
	20.0%	($0.10)	$0.39	$0.84	$1.26	$1.67

22.6.2.1 Tornado Chart

The tornado diagram presented in Figure 22-6 allows comparison of key risks using upside and downside parameters selected to reflect an approximately equivalent level of likelihood.

Figure 22-6: Key Risk Tornado Diagram



Source: SRK, 2023

23 ADJACENT PROPERTIES

There are no relevant adjacent properties for the purposes of this report.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Exploration Potential

Drilling since 2019 has established Filo del Sol as a major deposit of copper, gold and silver. Geologically it is recognized as a highly telescoped, high-sulphidation epithermal/porphyry deposit. This style of deposit forms some of the largest copper-gold deposits known to date. The deposit remains open in almost all directions and will require significant drill testing and associated fieldwork to delineate the extents of the deposit and the mineral endowment contained within. Several high-potential target areas exist for the discovery of new mineralized centres, and it remains to be determined if these will prove to be separate deposits themselves, or different parts of one very large deposit contiguous with what has already been discovered.

25 INTERPRETATION AND CONCLUSIONS

25.1 Conclusions

25.1.1 Mineral Tenure, Surface Rights, and Royalties

Filo Mining, through its Argentinian and Chilean subsidiaries, Filo del Sol Exploración S.A and Frontera Chile Limitada, respectively holds numerous exploration and exploitation concessions which cover the Filo del Sol project area in its entirety. Legal opinions have been obtained to demonstrate that the concessions covering the Filo del Sol deposit and relevant infrastructure areas are in good standing and owned or controlled by Filo Mining. The project is included within the "Vicuña Additional Protocol" under the Mining Integration and Complementation Treaty between Chile and Argentina which allows for people and equipment to freely cross the border of both countries in support of exploration and prospecting activities. Development of the Filo del Sol project is contemplated under the Treaty.

25.1.2 Exploration

Exploration activities by Filo Mining at Filo del Sol have been appropriate for the deposit type and have resulted in the discovery of a significant deposit of copper, gold and silver as quantified by the Mineral Resource and Mineral Reserve statements.

The Filo del Sol alteration and mineralization system extends well beyond the known resource, and excellent potential remains to increase the size of the deposit through continued exploration.

25.1.3 Geology and Mineralization

The Filo Mining exploration team has developed a comprehensive geological model through geological mapping and drill hole logging. This model provides an understanding of the geological processes which developed the current distribution of metals within the deposit and the controls on that distribution. The geological model provides a level of understanding sufficient for the declaration of an indicated mineral resource. Additional work is required to continue to refine the model, particularly at the smaller geographic scale necessary for effective control of grades during the mining operation.

25.1.4 Drilling

Drilling was initiated in the Filo del Sol area 24 years ago, but there has been a switch over the past 5 years from drilling exclusively with RC to predominantly diamond drilling core. While drilling through some of the upper-level rock conditions has been challenging, conditions improve with depth and drilling to depths of over 1400 metres is now being achieved. There has been a significantly sharp increase in the understanding of the deposit geology and improved targeting as a result.

25.1.5 Sampling and Assay

Sampling and assaying procedures are well-documented over the life of the project, and sufficient QA/QC work has been done to confirm that the results represent an accurate representation of the distribution of grades within the deposit volume, at the level of detail provided by the sample (drillhole) spacing. Multi-element analyses provide a comprehensive understanding of the levels of various elements within the deposit.

25.1.6 Data Verification

Data verification activities by the QP's are adequate to confirm the quality and accuracy of the exploration data used to develop the geological model and mineral resource declaration.

25.1.7 Mineral Resources Estimation

Mineral resource estimates presented in this report represent the global mineral resources located at Filo del Sol as of January 18, 2023. However, several factors such as additional drilling and sampling may affect the geological interpretation or the conceptual pit shells. Other factors that may have an impact, positive or negative, on the estimated mineral resources include the following:

- Changes in interpretations of mineralization geometry and continuity of mineralization zones
- Input parameters used in the pit optimization process, including:
    - Metallurgical and mining recoveries
    - Operating and capital cost assumptions
    - Metal price and exchange rate assumptions
- Confidence in the modifying factors, including assumptions that surface rights to allow infrastructure such as leach pads and SXEW plants to be constructed will be forthcoming.
- Delays or other issues in reaching agreements with local or regulatory authorities and stakeholders.
- Changes in land tenure requirements or in permitting requirements from those discussed in this Report.

25.1.8 Mineral Reserve and Mining

Estimations of Mineral Reserves for the Project conform to industry best practices and meet the CIM Definition Standards for Mineral Resources and Reserves (2014). Reviews of the environmental, permitting, legal, title, taxation, socio-economic, marketing, and political factors and constraints for the operation support the declaration of Mineral Reserves using the set of assumptions outlined.

Factors that may affect the Mineral Reserves estimate include dilution; metal prices; metallurgical recoveries and geotechnical characteristics of the rock mass; capital and operating cost estimates; and effectiveness of surface and groundwater management.

The Filo del Sol deposit is a large near surface, bulk mineable deposit that is well suited for extraction by conventional open pit methods. The mine plan is based on reasonable long-term metal prices, supported by an indicated mineral resource, and PFS level geotechnical and metallurgical inputs.

AGP assessed and included autonomous haulage as part of the overall mine plan. The Filo del Sol project, being a remote camp at high elevation with a harsh working environment, is considered an ideal candidate for implementing this technology. While there is a certain degree of risk associated with any relatively new technology, AGP are of the opinion that autonomous haulage is sufficiently proven in operations to be used to support a mineral reserves disclosure.

25.1.9 Metallurgical Testing and Recovery Methods

Application of conventional crushing, sequential acid and cyanide heap leaching, solvent extraction-electrowinning (for copper), and Merrill-Crowe processing (for gold) to the Filo del Sol project is technically feasible at the production scale contemplated in this study (60,000 t/d ore) and brings with it a well-established understanding of the recovery process and the factors critical to maintaining or improving metal recovery. Metallurgical testwork on samples representing the major lithological and spatial zones of the deposit as well as on samples representing life of mine average ore compositions sufficiently supports the selection of the processing method carried in the PFS and the recovery correlations and values used as the basis of the financial evaluation.

25.1.10 Infrastructure

The proximity of the deposit to the mining hub of Copiapó and its highly developed infrastructure for transport, utilities, resources and materials, provides opportunities for the project to improve existing infrastructure, such as the site access road, minimizing the need for construction of entirely new infrastructure. The availability of unvegetated space (for process facilities, tailings and waste rock sites) adjacent to the deposit minimizes the need for site development and reduces the amount of additional infrastructure required. Further, the opportunity to locate the construction and operations camp facilities in Chile, at a conveniently located site within a short distance of the mine and process plant, allow for the habitation of personnel at a significantly lower altitude and minimizes exposure risk.

The city of Copiapó is a source of skilled mining labour and the region will benefit from employment opportunities and the flow-on effect of investment in the project and surrounding communities. The position of the project in this region, which is seeking investment and holds a favourable attitude to mining, promises the opportunity to continue to develop mutually beneficial relationships with the local communities, regional authorities, and the state government.

Waste rock generated during extraction of ore from open pit operations will be permanently stored immediately east of the Filo del Sol pit.

Water will be supplied from aquifers in Argentina, located near the proposed plant site.

The processing plant includes two heap leach facilities: an on/off copper pad and a permanent gold pad. No tailings as such as produced; all ore processed remains on the permanent gold pad.

25.1.11 Marketing

No specific marketing study was conducted. Copper cathode and gold/silver doré are readily traded commodities and it is appropriate to assume that the products can be sold freely and at standard market rates. A small quantity of copper precipitate as generated from the SART process will be produced and additional testwork is required to confirm the marketability of this precipitate.

25.1.12 Environmental, Permitting, and Social Licence

Filo Mining has conducted environmental studies in the project area using qualified consultants for a number of years, which provides a defensible baseline. An experienced team from the Lundin Foundation is leading meaningful social engagement programs to support appropriate Corporate Social Responsibility.

Current exploration activity is fully permitted and in good standing. Mine development will require the successful conclusion of an Environmental Impact Assessment and permitting under both Argentinian and Chilean jurisdictions. Each are recognized processes with successful precedent in the San Juan province of Argentina and in Region III of Chile. There are no known environmental issues that could materially impact the ability of Filo Mining to extract the mineral resources at the Filo del Sol project.

25.1.13 Cost Estimating

The cost estimates have been prepared in accordance with the recommended practices of the American Association of Cost Engineers (AACE) and is classified as an AACE and Class 4 Prefeasibility Study estimate with an accuracy range of +30/-20%, and as such is within the accuracy level expected of a prefeasibility study. Several potential cost savings opportunities were identified and will be further investigated in the next phases of the project. These opportunities are expected to have a positive impact on the project economics.

Filo Mining is confident that this Filo del Sol Project Technical Report has been completed in accordance with the requirements of National Instrument 43-101 Standards for Disclosure for Mineral Projects, and that there are no significant unidentified risks or uncertainties that could affect the results or conclusions presented herein.

25.1.14 Financial Evaluation

Analysis of the project demonstrates that the mine plan has positive economics under the assumptions used. The project post-tax NPV at 8% discount rate is estimated to be $1.31 billion, with an IRR of 20%. The positive economics remain valid across a wide range of assumptions for key inputs.

Important Note: The economic model considered only cashflows from the beginning of actual construction forward. Schedule and expenditure for the feasibility study, including technical and economic studies, engineering studies, cost estimating, resource delineation and infill drilling, pit slope geotechnical characterization, metallurgical sampling and test-work, associated exploration, strategic optimization, mine, plant and infrastructure design, permitting and other pre-construction activities were not modelled.

25.2 Risks

25.2.1 Influence on Project Activities on Regional Glaciers

Operation of mining projects can generate measurable airborne particulates and given the proximity of Filo del Sol to regional glaciers, there is a risk that the project activities could adversely affect the size and stability of existing glacial features at or near the project site.

Filo Mining has undertaken a comprehensive glacier survey as part of project development and as such will be able to accurately assess the influence of the project on the glaciers on a continuous basis. Additionally, the project site has been designed in respect of the buffer zone requirements between the existing glacial structures and the project boundary, and with inclusion of modern dust mitigation equipment to limit any potential adverse effects.

25.2.2 Delayed Metal Recoveries and Build-up of Process Inventory

The functionality of heap leaching processes is contingent principally upon solution flow through the heap; as heaps increase in height or decrease in structural integrity, a decrease in solution flux will result in accumulation of dissolved metal values within the heap as process inventory. There is an inherent risk in heap leaching operations that metal values will be delayed in the heap itself and have a negative, delayed effect on project revenue. This effect can be exacerbated in harsh climates, where permeability of the heap may be affected by freezing conditions. Additionally, the heterogeneity of grade distribution and minerology may make average recoveries difficult to predict. The effects of a delayed revenue stream on the project economics have not been evaluated as part of the current study.

Filo Mining is presently evaluating the development of a dynamic project simulation to evaluate the potential influence of process inventory accumulation on the project economics by determining the time influence of metal recovery over the life of mine.

25.2.3 Hydrogeological Considerations

Slope design criteria assume fully depressurized conditions in the proposed open pit slopes, as they are primarily above the regional groundwater table. Observations during drilling and in open holes from previous programs indicate that water is greater than 150 m below ground surface. No hydrogeological testing data were collected for this study. If groundwater is encountered in future studies, the recommended slope design criteria may need to be revised.

25.2.4 Commodity Prices and Supply Costs

The key financial risk is in relation to commodity prices. There is always a large degree of uncertainty regarding long-term commodity prices. There is also a risk that the input costs that underpin the project costs estimates may be different to the PFS assumptions. Input costs and commodity prices historically are somewhat correlated, and this can have the effect of lessening the effects of variation. Put another way, high input costs tend to be associated with high commodity prices, and vice versa.

Input costs and/or exchange rates may be less favourable than modelled, resulting in a decrease in project cashflows and value.

The analysis has assumed that revenues from leaching are received at the time of mining and placement. Significantly longer-than-anticipated leach times will delay revenue and result in a lower NPV for the project. Sensitivity analysis indicates that assumptions of leach times of several months do not materially affect the NPV, however.

The analysis is undertaken considering only cashflows incurred from the decision to construct the project. Timelines for permitting and financing the project are uncertain. The combination of additional costs associated with those processes, as well as the effect of deferring the project cashflows are likely to reduce the project Net Present Value in comparison to the analysis presented here.

25.2.5 Operability and Functionality of Sequential Leaching

The Filo del Sol PFS considers a high altitude, relatively complicated heap leach flowsheet in an extreme climate. The flowsheets consist of a cyanide heap leach for gold ores and an on-off leach for copper-gold ores which contained an economic copper content such that copper could be recovered prior to gold heap leaching. Although accomplished at other locations in South America, there is considerable operational risk inherent in a heap leach at altitude, in an aggressive climate. In particular, the added complexity of an on-off pad and multiple ore handling steps may be more economically significant than can be readily quantified. Additionally, metallurgical testing showed a percentage significant of weight loss occurs in some of the ore types following acid leaching; the impact of which may cause materials handling difficulties which are difficult to quantify.

Filo Mining is currently contemplating a definitive evaluation of alternative process options which will support a decision on the preferred process to be used as the basis for future development of the project.

25.2.6 Permitting and Project Implementation

Like all minerals industry projects, there are risks with respect to permitting. For the purposes of this PFS, it is assumed that the project can be permitted in the configuration conceived. No timeline has been assumed for permitting, and the project analysis considers only cashflow from the construction commencement. In general, the project will be required to demonstrate that the project plans result in acceptable environmental impact in the context of the economic benefits that accrue from the project.

Risks relating to permitting include, but are not limited to:

- Permits to allow access to water for operation of the project may be difficult to obtain. Water is a relatively scarce resource in the region of the project and limited water rights may be available. Downstream water users may oppose the granting of water rights to the project.
- The long-term impact of deleterious elements remaining in the heap (e.g., Mercury) will need to be studied in more detail to ensure that the closure plan adequately ensures that contaminants will not become mobile and degrade downstream waters.
- The existence of unfavourable sub-surface conditions such as permafrost may increase the risk associated with demonstrating an effective environmental management plan.
- Despite the binational treaty which exists to facilitate project development, the location of the project on the border between Argentina and Chile will result in additional complexity with respect to permitting.

Filo Mining is continuing environmental baseline and glaciology work to ensure these items are addressed to support future Mine EIA development.

25.2.7 Mineral Reserve and Mining

The mine schedule requires approximately one and a half years of pre-stripping to liberate sufficient ore to support the processing ramp-up period and then full production. This pre-stripping period involves significant pioneering efforts on steep slopes, requiring early-works road construction and mining of irregular benches. If this pre-stripping period were delayed it would impact the ability of the mine to supply sufficient ore for the ramp up period. A detailed schedule and cost model for early works is required in further studies.

The mine production and cost model assume efficient operating and maintenance practices, supporting a productive mining operation. If inefficiencies are realized in fleet performance metrics or staffing levels it could have an adverse impact on unit costs and mine schedule adherence. The implementation of autonomous haulage also introduces risks to mining fleet performance.

Currently no consideration is provided for management of different waste types in the mine optimization approach and schedule, though it is understood that some of the waste may be potentially acid generating and may contain other deleterious minerals. It is possible that further geochemical and effluent management studies or government regulations will necessitate additional waste management considerations beyond those contemplated in this study.

25.2.8 Geotechnical

The PFS geotechnical field investigation showed intermittent permafrost in the area of mine infrastructure. Due to the limited geotechnical program, the actual limits could vary and increase or decrease earthworks or engineering costs which would be required to mitigate potential issues related to permafrost below structures.

25.3 Opportunities

25.3.1 Mineral Reserves

Only Measured and Indicated oxide Mineral Resources were considered for processing. Inferred Mineral Resources were treated as waste. Increasing Mineral Reserves through the incorporation of both conversion of Inferred Mineral Resources and/or the incorporation of mixed/sulphide resources into the mine plan has the potential to increase mine life and reduce stripping ratios.

An opportunity exists to increase the Mineral Reserve with further study if desired by increasing the processing (heap leach) capacity available for the project. In the current mine schedule, the pit design is smaller than what could be supported, and inside this pit the cutoff grade has been elevated to ensure that sufficient capacity exists on the leach pad.

Opportunities to bring forward the production profile also exist in the mine schedule if processing bottlenecks can be alleviated either through a dynamic cutoff grade or enabling long-term stockpiling of lower-grade ores.

Exploration at Filo del Sol has resulted in new mineral discoveries with every season, and several areas of the deposit remain open. The area is large, with a trend of over 7 km long that has been affected by Miocene hydrothermal alteration. The project remains dynamic with the excitement of growing the deposit with every drill season and new exploration in peripheral areas of mineralization that have never been drill tested. The advances in understanding the geological controls on mineralization continue to generate new conceptual targets that guide exploration in this complex system that hosts several stages of mineralization.

25.3.2 Commodity Prices and Supply Costs

Commodity prices may be higher than assumed, leading to enhanced project economics. The ability of the mine plan to adapt to higher commodity prices by altering cutoff policies and other strategic assumptions could potentially add value.

Input costs and/or exchange rates may be more favourable than modelled, resulting in an increase in project cashflows and value.

When considering the effect of risks on project financial outcomes, the value of management options should not be overlooked. In the case that the project economic environment is different to the that assumed, management has the option to alter the operating strategy to mitigate downside outcomes, and to enhance upside outcomes. The benefits of this can be substantial and are not included in the deterministic economic analysis used for this PFS.

25.3.3 Improved Metal Recoveries

Testwork indicated that intensive cyanidation and longer leach cycle times have the potential to improve upon modelled metal recoveries and positively impact project economics. Additional metallurgical work may lead to supporting improved metal recovery estimates and improve project revenues.

In addition to the preliminary flotation tests, the flotation cleaner tailings were subjected to intensive cyanide leaching tests in an effort to recover additional precious metal values. The results indicated that an additional 10-16% of the gold and 10-26% of the silver could potentially be recovered. Using a concentrator plus tails leach flowsheet, approximately 88% of the copper and 80% of the gold was recovered from the first sample, and 90% of the copper and 75% of the gold from the second sample.

There are potential economic opportunities in sulphide zone that can be evaluated in future studies. Preliminary metallurgical testwork was conducted on three composite samples of sulphide material from drill core originating from the 2018-2019 and 2019-2020 drilling campaigns on material that is not included in the resource model. The metallurgical testwork was also completed at SGS Minerals (Lakefield) during 2020, 2021 and 2022. The material tested was not intended to reflect the elemental grade(s) of the oxide resource nor the proportion of each type of material that may occur within the oxide resource. The focus of the preliminary testing was to provide insight and direction for future testing requirements for the hypogene sulphide portion of the deposit. Preliminary scoping flotation tests were performed on three composite samples, including rougher kinetic and batch cleaner tests. Two locked cycle flotation tests were performed, although the flowsheet and reagent scheme were not fully optimized, one concentrate contained 22% Cu, 18 g/t Au, 37 g/t Ag and 880 g/t As and the second contained 26% Cu, 14 g/t Au, 106 g/t Ag and 52,400 g/t (5.24%) As.

25.3.4 Power Supply

The transmission line route from Los Loros to Filo, referenced as Transmission Line 2, was selected due to two reasons: lower capital cost and potentially easier to permit.

A third route along the mine access road up the Montoso Valley may be a viable alternative, with possible reduced capital costs.

25.3.5 Crushing Circuit Optimization

The PFS flowsheet has a closed-circuit secondary crushing circuit, modelled to give the required sizing of crushed ore to the heap leach. A more conventional approach, commonly used in South America, is an open-circuit crushing circuit, which could reduce operating costs. This opportunity should be considered, including developing the model for the crushing circuit and evaluating the effect of slightly different crush size at the leaches.

25.3.6 Emerging Technologies

As the project is in a high elevation operating environment, where labour productivity and equipment utilization can be impacted, the PFS currently plans for the adoption of autonomous haulage. This emerging technology has begun to demonstrate its value at mining operations across the globe. While there is a certain element of risk associated with the adoption of new technology, autonomous haulage has sufficiently matured to be considered for Filo del Sol. Additional autonomous technologies, such as autonomous blasthole drilling, may be considered for the project to further reduce costs or increase cutoff grades.

25.3.7 Alternate Ore Processing

Metallurgical testwork completed in support of the PFS identified the opportunity for readily soluble copper to be extracted from the ore by way of a water or mild acid leach, nominally an ore washing process, which for selected ores resulted in very rapid recovery of up to 70% of the copper before any heap leaching. During the PFS an evaluation of selected process options was completed to evaluate the viability and potential economic benefits of an ore washing process. The evaluation highlighted that at traditional milling, copper atmospheric leach and gold cyanide in leach (CIL) was economically competitive with the contemplated, at that time, three heap leach process.

The evaluation of selected process options showed that the originally contemplated three-heap leach option was comparable to the milling/leaching flowsheet with respect to capital costs and highlighted the sensitivity of the economic model to metal recoveries and to a lesser extent operating cost. In summary, the financial evaluation of the two options was not sufficiently compelling for a preferred process option and given the sensitivity was highly variable pending the assumptions used.

The potential to apply a tank leaching (or similar) based flowsheet can mitigate some of the perceived operating and maintenance risks inherent in a heap leach in a challenging environment. Thus, there is opportunity in developing a financial case for the alternative process (tank leaching) option to the same level of confidence to enable a definitive decision on the preferred process option for advancing the Project.

26 RECOMMENDATIONS

Although this PFS outlines a compelling economic case for additional studies and eventual development of the Filo del Sol oxide resource, the extent and tenor of the significant sulphide mineralization discovered by drilling since 2019 indicates that the focus should continue to be on outlining and defining the full potential of the Filo Del Sol property. Once a more comprehensive understanding of the entirety of the mineralization has been developed, options on how to best progress the development of the deposit will be assessed.

Recent drilling has intersected long intervals (>1km) of high-grade mineralization beneath, and to the north of, the current mineral resource. Although this additional mineralization has not been fully defined and remains open to expansion it is already significant enough to change the entire scope of the project. This zone of mineralization beneath and north of the resource has been named the Aurora Zone.

In addition to the Aurora Zone, more widely spaced drilling has encountered significant mineralization in areas distal to the resource, namely the Bonita zone, the Flamenco Zone and the Gemelos Zone.

To continue to define the mineralized potential of Filo Del Sol, an initial program of 35,000m of diamond drilling is recommended in order to accomplish 3 main objectives:

- Infill and short-range expansion drilling of the Aurora Zone
- Medium-range (1 – 2km) step out drilling to expand the Bonita Zone and determine if it, and other apparently satellite zones, are contiguous with the Aurora Zone, and
- Long-range (>2km) exploration drilling to test new target areas indicated by geology and surface sampling, primarily the Gemelos and Flamenco Zones
- This work is not contingent on any other work programs.

Data collected from this drilling should be used to create a comprehensive geological model incorporating lithology, alteration and mineral zonation which can be used to develop an updated mineral resource estimation with a goal of adding the sulphide material to the current oxide resource.

One of the key discoveries since 2019 is a zone of very high-grade material which occurs between 700 m and 1,000 m below surface. Grade variability within this zone indicates that it will likely need to be drilled at close spaced centres in order to be fully delineated and defined.

Given the technical challenges with completing this drilling from surface, an assessment of the viability of an underground drill drift should be completed which would allow this, and other areas of the Aurora zone, to be drilled from underground. As the project advances, underground access would also facilitate the recovery of bulk samples for metallurgical testwork.

The mineralization discovered by drilling since 2019 is primarily hypogene sulphide mineralization and will require processing by a crush/grind/float process rather than a leach process as described in the current study. Additional geometallurgical studies and metallurgical testwork are recommended in order to better understand the mineralogical distribution of ore minerals and develop a better understanding of the number, size and distribution of geometallurgical zones within the deposit.

Environmental base line studies and data collection should also continue to ensure a comprehensive and continuous record of data collection.

Depending on the results from this initial diamond drill program, subsequent drill programs may be required to achieve the level of understanding of the entirety of the mineralization required for evaluation of future development options.

Table 26-1: Filo del Sol Recommended Work Program Cost Estimate

Program Component	Cost Estimate (US$M)
Environment, Social and Governance	3.8
Land Holding Cost	1.2
Resource Drilling and Support	69.0
Project support logistics	7.1
Metallurgical and Engineering Studies	3.5
Total	**84.6**

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