NI 43-101 Technical Report, Pre-feasibility Study for the Filo del Sol Project

Prepared for

Filo Mining Corp.

Prepared by

Ausenco Engineering
22 February 2019

NI 43-101 Technical Report,
Pre-feasibility Study for the Filo del Sol Project

Prepared for

Filo Mining Corp.

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Effective Date: 13 January 2019

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Important Notice

This report was prepared as a National Instrument 43-101 Technical Report for Filo Mining Corp. by Ausenco Engineering Inc.as part of a team of consultants contracted by Filo Mining ("the Team"). The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Team's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Filo Mining subject to the terms and conditions of its contracts with Ausenco and the Team and to the relevant securities legislation. The contracts permit Filo Mining Corp. to file this report as a technical report with Canadian securities regulatory authorities, pursuant to National Instrument 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party are at that party's sole risk. The responsibility for this disclosure remains with Filo Mining. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new technical report has been issued.

Currency is expressed in U.S. dollars and metric units are used, unless otherwise stated. The Report uses Canadian English.

Acronyms and Abbreviations

Distance	
µm	micron (micrometre)
mm	millimetre
cm	centimetre
m	metre
km	kilometre
in	inch
ft	foot
Area	
m^2	square metre
km^2	square km
ac	acre
ha	hectare
Volume	
L	litre
m^3	cubic metre
ft^3	cubic foot
Mbcm	million banked cubic metres
Mass	
kg	kilogram
g	gram
t	metric tonne
kt	kilotonne
lb	pound
Mt	megatonne (million tonnes)
oz	ounce (troy)
wmt	wet metric tonne
dmt	dry metric tonne
Pressure	
psi	pounds per square inch
Pa	pascal
kPa	kilopascal
MPa	megapascal
Elements and Compounds	
Au	gold
Ag	silver
As	arsenic
Cu	copper
S	sulphur

Other	
°C	degree Celsius
cfm	cubic feet per minute
elev	elevation
hp	horsepower
hr	hour
kW	kilowatt
kWh	kilowatt hour
M	million or mega
mamsl	metres above mean sea level
mph	miles per hour
ppb	parts per billion
ppm	parts per million
s	second
V	volt
W	watt
kV	kilovolt
$k	thousand dollars
$M	million dollars
$B	Billion dollars
tph	tonnes per hour
tpd	tonnes per day
mtpa	million tonnes per annum
Ø	diameter
ARS	Argentine peso
Acronyms	
SRK	SRK Consulting (Canada) Inc.
CIM	Canadian Institute of Mining
NI 43-101	National Instrument 43-101
ABA	acid-base accounting
AP	acid potential
NP	neutralization potential
CONAGUA	Comisión Nacional del Agua
ML/ARD	metal leaching/ acid rock drainage
PAG	potentially acid generating
NAG	non-acid generating
RC	reverse circulation
IP	induced polarization
COG	cut-off grade
NSR	net smelter return
NPV	net present value
LOM	life of mine
Conversion Factors	
1 tonne	2,204.62 lb
1 oz (troy)	31.10348 g

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CERTIFICATE OF QUALIFIED PERSON

To accompany the 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 (the "Technical Report").

I, Bruno Borntraeger, P.Eng., do hereby certify that:

1. I am a currently employed as a Specialist Geotechnical Engineer | Associate with Knight Piésold (Vancouver) with an office at 1450-750 West Pender St., Vancouver, BC Canada.
2. I am a graduate of the University of British Columbia in Vancouver, Canada (Bachelor of Applied Science in Geological Engineering, 1990). I have practiced my profession continuously for 28 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.
3. I am a member in good standing of the Engineers and Geoscientists of British Columbia (License #20926).
4. I visited the property on March 22, 2018.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for sections 1.18, 20, parts of 25.1 of the Technical Report, as well as relevant parts in the Executive Summary, Reliance on Other Experts, References and Date and Signature of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have not had prior involvement with the subject property.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 22nd day of February, 2019 in Vancouver, B.C., Canada.

"original signed"

Bruno Borntraeger, P.Eng.
Specialist Geotechnical Engineer | Associate
Knight Piésold

CERTIFICATE OF QUALIFIED PERSON

To accompany the 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 (the "Technical Report").

I, Fionnuala Anna Marie Devine, P. Geo., do hereby certify that:

1. I am a geologist with Merlin Geosciences Inc. with an office at 178 – 6th Street, Atlin, BC, Canada, V0W 1A0, telephone +1 250-651-7569, email fdevine@merlingeo.com.
2. I graduated in Geological Sciences 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. I have practiced my profession continuously since 2005. I have been involved in mineral exploration for base and precious metals in a variety of deposit types in North and South America during that time.
3. I am a Professional Geoscientist registered with Engineers and Geoscientists BC, license # 40876.
4. I first visited the project site in January 2014.
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 fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for sections 1.2 to 1.10, 4 to 12, 23, and 24 of the Technical Report, as well as relevant parts in the Executive Summary, Reliance on Other Experts, Interpretation and Conclusions, Recommendations, References and Date and Signature of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have been involved in exploration of the property since 2014, including surface geological mapping and core reviews in 2015, 2016, and 2017.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 22nd day of February, 2019 in Enderby, B.C., Canada.

"original signed"

Fionnuala Anna Marie Devine, P. Geo.
Merlin Geosciences Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the 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 (the "Technical Report").

I, James N. Gray, P.Geo., do hereby certify that:

1. I am a President with Advantage Geoservices Limited with an office at 1051 Bullmoose Trail, Osoyoos, BC, Canada.
2. I am a graduate of the University of Waterloo in 1985 where I obtained a B.Sc in Geology. I have practiced my profession continuously since 1985. My relevant experience includes resource estimation work at operating mines as well as base and precious metal projects in North and South America, Europe, Asia and Africa.
3. I am a Professional Geoscientist registered with the Engineers and Geoscientists British Columbia, license # 27022.
4. I have not personally visited the project area.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for sections 1.12 and 14 of the Technical Report, as well as relevant parts in the Executive Summary, Reliance on Other Experts, Interpretation and Conclusions, Recommendations, References and Date and Signature of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have had prior involvement with the subject property, having 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;
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 22nd day of February 2019 in Osoyoos, BC, Canada.

"original signed"

James N. Gray, P.Geo.
President
Advantage Geoservices Limited

CERTIFICATE OF QUALIFIED PERSON

To accompany the 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 (the "Technical Report").

I, Jay Melnyk, P.Eng., do hereby certify that:

1. I am a Professional Engineer with AGP Mining Consultants Inc., with a business address at #246-132 Commerce Park Dr., Unit K, Barrie, ON, L4N 0Z7.
2. I am a graduate of the Montana Tech of the University of Montana with a Bachelor of Science degree in Mining Engineering in 1988 and from the British Columbia Institute of Technology with a Diploma in Mining Technology in 1984.
3. I am a member in good standing of the Engineers and Geoscientists of British Columbia (License #25975).
4. I have practiced my profession for 30 years. I have been directly involved in open pit mining operations, and design of open pit mining operations in Argentina, Canada, Chile, Eritrea, Indonesia, Iran, Mexico, Perú, and the United States.
5. I visited the property February 3, 2018.
6. 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.
7. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
8. I am a co-author of the Technical Report, responsible for sections 1.13, 1.14, 15 and 16.2 to 16.8 of the Technical Report, as well as relevant parts in the Executive Summary, Reliance on Other Experts, Interpretation and Conclusions, Recommendations, References and Date and Signature of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
9. I have not had prior involvement with the subject property.
10. As of the date of this certificate, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.
11. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 22nd day of February, 2019 in Vancouver, B.C., Canada.

"original signed"

Jay Melnyk, P.Eng.
AGP Mining Consultants Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the 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 (the "Technical Report").

I, Neil M. Winkelmann, FAusIMM, do hereby certify that:

1. I am a Principal Consultant with SRK Consulting (Canada) Inc., with an office at 2200-1066 W. Hastings St., Vancouver, BC, Canada.
2. I am a graduate of the University of New South Wales, Australia with a B.Eng. in Mining (1984). I am a graduate of the University of Oxford with an MBA in 2005. I have practiced my profession continuously since 1984 and I have 32 years' experience in mining. I have significant experience in the valuation of minerals-industry projects accrued over the past 14 years.
3. I am registered as a Fellow of The Australasian Institute of Mining and Metallurgy (AusIMM, #323673).
4. I visited the property February 2017.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for section 1.21, 1.22, and 22 of the Technical Report, as well as relevant parts in the Executive Summary, Reliance on Other Experts, Interpretation and Conclusions, Recommendations, References and Date and Signature of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have not had prior involvement with the subject property. I have had prior involvement with the subject property contributing to prior technical studies.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 22nd day of February 2019 in Vancouver, B.C., Canada.

"original signed"

Neil M. Winkelmann, FAusIMM
Principal Consultant (Mining)
SRK Consulting (Canada) Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the 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 (the "Technical Report").

I, Robin Kalanchey, P.Eng., do hereby certify that:

1. I am a Professional Engineer, employed as Director, Minerals and Metals – Western Canada with Ausenco Engineering Canada Inc. (Canada), with an office at 855 Homer Street, Vancouver, BC V6B 2W2.
2. I am a graduate of University of British Columbia with a Bachelor of Applied Science degree in metals and materials engineering in 1996.
3. I am a Professional Engineer registered with the Association of Professional Engineers and Geoscientists of Alberta, member number 61986.
4. I have practiced my profession continuously since 1996 and have been involved in: mineral processing and metallurgical testing, metallurgical process plant design and engineering, and metallurgical project evaluations for gold, nickel, cobalt, copper, zinc and molybdenum projects in numerous countries including Chile.
5. I visited the property February 3 and 4, 2018.
6. 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.
7. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
8. I am a co-author of the Technical Report, responsible for sections 1.1, 1.11, 1.15, 1.17, 1.19, 1.20, 1.23, 2, 3, 13, 17, 18.1, 18.3 to 18.7, 19, and 21 of the Technical Report, as well as relevant parts in the Executive Summary, Reliance on Other Experts, Interpretation and Conclusions, Recommendations, References and Date and Signature of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
9. I have not had prior involvement with the subject property.
10. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
11. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 22nd day of February, 2019 in Vancouver, B.C., Canada.

"original signed"

Robin Kalanchey, P.Eng.
Director, Minerals and Metals – Western Canada
Ausenco Engineering Canada Inc. (Canada)

CERTIFICATE OF QUALIFIED PERSON

To accompany the 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 (the “Technical Report”).

I, Scott Elfen, P.Eng., do hereby certify that:

1. I am the VP Global Lead for Geotechnical and Civil Services with Ausenco Engineering Canada Inc. (Canada), with an office at 855 Homer Street, Vancouver, BC V6B 2W2.
2. I am a graduate of the University of California, Davis with a Bachelor of Science degree in Civil Engineering (Geotechnical) in 1991. I have practiced my profession continuously for 26 years and have been involved in geotechnical, civil, hydrological, and environmental aspects for the development of mining projects; including feasibility studies on numerous underground and open pit base metal and precious metal deposits in North America, Central and South America, Africa and Australia.
3. I am a Registered Civil Engineer in the State of California (No. C56527) by exam since 1996 and am also a member of American Society of Civil Engineers (ASCE) Society for Mining, Metallurgy & Exploration (SME) that are all in good standing.
4. I visited the Project from February 3 and 4, 2018.
5. I have read the definition of “qualified person” set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (“NI 43-101”) and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for sections 1.16, 18.2, 18.8, 18.9, and 18.10 of the Technical Report, as well as relevant parts in the Executive Summary, Reliance on Other Experts, Interpretation and Conclusions, Recommendations, References and Date and Signature of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have had no prior involvement with the property that is the subject of the NI 43-101 Technical Report.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 22nd day of February 2019 in Vancouver, B.C., Canada.

“original signed”

Scott Elfen, P.Eng.
VP Global Lead for Geotechnical and Civil Services
Ausenco Engineering Canada Inc. (Canada)

CERTIFICATE OF QUALIFIED PERSON

To accompany the 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 (the "Technical Report").

I, Ian Stilwell, P.Eng., do hereby certify that:

1. I am a Professional Engineer, employed as Principal Geotechnical Engineer with BGC Engineering Inc., with an office at 234 St. Paul Street, Kamloops, British Columbia.
2. I am a graduate of the University of British Columbia with a Bachelor of Applied Science degree in Geological Engineering in 1995.
3. I am a member in good standing of the Engineers and Geoscientists of British Columbia (License #27316).
4. I have practiced my profession for 23 years. I have been directly involved in open pit mining operations, and design of open pit mining operations in Canada, the United States, Mexico, Chile, Peru, Ecuador, Argentina, Eritrea and Botswana.
5. I visited the property March 17, 2018.
6. 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.
7. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
8. I am a co-author of the Technical Report, responsible for section 16.1 of the Technical Report, as well as relevant parts in the Executive Summary, Reliance on Other Experts, References and Date and Signature of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
9. I have not had prior involvement with the subject property.
10. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
11. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 22nd day of February 2019 in Kamloops, B.C., Canada.

"original signed"

Ian Stilwell, P.Eng.
BGC Engineering Inc.

1 Summary

1.1 Introduction

Filo del Sol is an advanced stage copper-gold exploration project which straddles the border between Argentina and Chile.

In January 2018, Filo Mining Corp. contracted Ausenco Engineering Canada Inc., along with Merlin Geosciences Inc., Advantage Geoservices Ltd., and Knight Piésold Ltd. to conduct a pre-feasibility study (PFS) on the project.

This report, with an effective date of 13 January 2019, discloses the outcomes of the PFS and the first-time estimate of mineral reserves.

1.2 Property Description and Location

The Filo del Sol Project (the 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 28.49º S and 69.66º 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. For the purposes of this report, Filo Mining Corp. and the subsidiary companies are referred to interchangeably as “Filo Mining”.

In Argentina, Filo del Sol Exploración S.A. owns two exploration permits (Cateos) and nine exploitation permits (Manifestaciones). In Chile, Frontera Chile Limitada is the owner of sixteen exploration concessions (Manifestaciones), two exploitation mining concessions (Mensuras) in the process of being granted, and one unilateral and irrevocable option agreement to purchase 17 exploitation concessions (Mensuras). The total combined area of the Project is approximately 14,014 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 authorisation which allows for 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”. 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, although Copiapó is much closer and is approximately four hours driving time.

The climate is cold and windy, typical of the high Andes. The exploration field season runs from November to April. Field work is based out of the Batidero camp located approximately 20 km from Filo del Sol in Argentina. The Batidero camp can accommodate approximately 200 people. The site is remote and, other than road access, there is no infrastructure available.

The Project is in the Andes Mountains with elevations ranging from 3,800 m to 5,500 m above mean sea level (amsl). The mountains are generally not rugged and vehicle access is possible to most of the property. Vegetation is almost entirely absent in the area.

1.4 History

Cyprus-Amax was the first company to have done any serious 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 1998/99 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. Overlapping mineralizing events combined with weathering effects, including supergene enrichment, have created several different styles of mineralization, including structurally-controlled gold, stratiform high-grade silver (+/- copper) and high-grade supergene-enriched copper within a broader envelope of disseminated sulphide copper and gold mineralization. Mineralization is hosted in clastic rocks inferred to be Late Oligocene in age as well as in underlying rhyolitic volcanic rocks inferred to be part of the Permo-Triassic basement. It is located in the Andean Frontal Mountain Range, between the Maricunga gold porphyry trend to the north and the El Indio high-sulphidation epithermal trend to the south, both of Miocene age as is Filo del Sol.

The Filo del Sol deposit is comprised of two contiguous zones of mineralization, Filo del Sol to the north and Tamberias (previously called Flamenco or Filo Mining South) to the south. The two zones are interpreted to be separated by the Flamenco fault, and have been differentiated in metallurgical testwork, but have been aggregated in the mineral resource estimate.

In addition to the Filo del Sol deposit, several other exploration targets occur on the property.

1.6 Deposit Types

Mineralization in the Filo del Sol area shows affinities with both porphyry copper-gold-molybdenum and high-sulphidation gold-silver epithermal systems. The deposit defined by the Mineral Resource is best classified as epithermal, however adjacent mineralized zones, particularly to the south, appear to have characteristics of a copper-gold porphyry system. The mineralized system in its entirety is thought to represent a telescoped porphyry – epithermal system, with multiple intrusive and breccia centres, and so combines aspects of both these deposit types.

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 16 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, typically between November and April, and exploration seasons are described by the years which they bridge.

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.

1.8 Drilling

Drilling at Filo del Sol was initiated by Cyprus in 1998/99 and since then a total of 44,457 m of RC drilling in 184 holes and 6,790 m of diamond drilling (DD) in 31 holes has been completed on the property. All of these holes with the exception of 17 RC holes (3,693 m) were drilled in the Filo del Sol deposit.

1.9 Sampling Preparation, Analysis, and Security

Sampling procedures and protocols from drill programs have evolved over the last two decades not only at the Filo del Sol project, but throughout the industry. More than 68% of the current RC and DDH dataset had a rigorous QA/QC protocol with blanks, standards and laboratory duplicates. Another 10% has been checked with a second lab but does not have blank and standard controls. The remaining 22% of the dataset has only been verified (satisfactorily) 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 Indicated resource estimations.

1.10 Data Verification

To verify information provided by the Company, F. Devine visited the area of drilling and located a number of drillholes with a hand-held GPS. She was directly involved in the update of the geological model for the project area, including completing extensive surface geological mapping and core logging, data and interpretation review and discussion with Company personnel.

The results of these checks are considered a satisfactory confirmation of the results reported by Filo Mining.

A visit to the Copiapó office and support facilities was carried out by J. Gray, between 16th June 2014 and 21st June 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.

1.11 Mineral Processing and Metallurgical Testing

In 2016 and 2017, comprehensive metallurgical test programs were carried out at SGS Lakefield on selected samples from the Filo del Sol deposit. These focussed mostly on assessing the feasibility of using heap leaching to recover the copper, gold and silver from the various mineralization types identified.

To confirm and improve these results a sampling campaign was carried out in early 2018 to collect surface samples, RC chips and diamond drill core samples. A total of more than 3,500 kg of sample was shipped to the SGS facility in Lakefield, Ontario. Samples were submitted to various physical, chemical and detailed mineralogical characterisation tests.

Most of the metallurgical program was devoted to the leaching stage of the process, more particularly heap leaching. Heap leaching was simulated by completing column leaching tests on material 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 ore types (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 ore types (a total of 18 sequential column tests).

Variability and process optimization testing were 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.

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.

Final metal recoveries used in the financial model were 80% for copper, 70% for gold and 82% for silver.

1.12 Mineral Resource Estimates

The Filo del Sol Resource remains unchanged from the Mineral Resource estimate reported by the Company on August 8, 2018 and is based on a total of 44,600 metres of drilling in 188 holes, of which 158 holes are reverse circulation (RC) and 30 holes are core holes. The resource estimate presented below is the total Indicated and Inferred Resource, divided between oxide and sulphide mineralization.

The Mineral Resource estimate as of the effective date of June 11, 2018 is shown in the table below. The Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-1: Mineral Resource Estimate

Zone	Cutoff	Category	Tonnes (millions)	Cu (%)	Au (g/t)	Ag (g/t)	lbs Cu (millions)	Ounces Au (thousands)	Ounces Ag (thousands)
Oxide	* see notes	Indicated	349.6	0.34	0.32	12.6	2,656	3,623	141,364
		Inferred	103.9	0.26	0.32	8.7	585	1,083	29,067
Sulphide	0.30 % CuEq	Indicated	75.5	0.27	0.34	2.2	451	813	5,374
		Inferred	71.2	0.30	0.33	2.5	469	751	5,743
Total		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

Notes to accompany Filo del Sol Mineral Resource table:

1. Mineral Resources have an effective date of 11 July 2018;
2. The Qualified Person for the resource estimate is James N. Gray, P.Geo. of Advantage Geoservices Ltd.;
3. The Mineral Resources were estimated in accordance with the CIM Definition Standards for Mineral Resources and Reserves;
4. 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 the Sulphide mineralization, and metal prices of $3/lb copper, $1300/oz gold, $20/oz silver. The CuEq formula is:

CuEq=Cu+Ag*0.0089+Au*0.5266;

5. All figures are rounded to reflect the relative accuracy of the estimate;
6. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability;
7. The resource was constrained by a Whittle® pit shell using the following parameters: Cu $3/lb, Ag $20/oz, Au $1300/oz, slope of 45°, a mining cost of $2.50/t and an average process cost of $13.26/t;
8. 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 metallurgical recoveries of 82% for copper, 55% for gold and 71% for silver based on preliminary metallurgical testwork, and metal prices of $3/lb copper, $1300/oz gold, $20/oz silver. The CuEq formula is: CuEq=Cu+Ag*0.0084+Au*0.4239.

1.13 Mineral Reserve Estimates

The Initial Mineral Reserve estimate for Fil del Sol, shown below, is based on the Mineral Resource Statement with an effective date of June 11. The Mineral Resources are inclusive of Mineral Reserves.

Table 1-2: Mineral Reserve Estimate

Filo del Sol Mineral Reserve Statement (@ 0.01 $/t NVPT cut-off)								
	Tonnage	Grade				Contained Metal		
Category (all domains)	(Mt)	Cu (%)	Au (g/t)	Ag (g/t)	NVPT ($/t)	Cu (M lbs)	Au (k oz)	Ag (k oz)
Proven	-	-	-	-	-	-	-	-
Probable	259.1	0.39	0.33	15.1	25.30	2,226	2,764	126,028
Total Proven and Probable	**259.1**	**0.39**	**0.33**	**15.1**	**25.30**	**2,226**	**2,764**	**126,028**

Notes to accompany Filo del Sol Mineral Reserves table:

1. Mineral Reserves have an effective date of 13 January 2019. The Qualified Person for the estimate is Mr. Jay Melnyk, 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 $3.00/lb, Ag $20/oz, Au $1300/oz;
    - Mining cost of $2.00/t;
    - An average processing cost of $9.73/t;
    - General and administration cost of $2.02/t processed;
    - Pit slope angles varying from 29 to 45 degrees, inclusive of geotechnical berms and ramp allowances;
    - Process recoveries were based on rocktype. 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) breakeven cut-off considering metal prices, recoveries, royalties, process and G&A costs as per LG shell parameters stated above;
6. The life-of-mine (LOM) stripping ratio in tonnes is 1.52:1;
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-meter 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 14 years, including pre-stripping, with a life of mine strip ratio of 1.5:1. A maximum mining rate of approximately 65 Mt per year (including waste) is required to provide the nominal 60,000 tonnes per day of ore to the process facility. A total of 259 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. 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. Metal leaching is expected to span 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 portion of the barren leach solution, following zinc precipitation, will be treated to minimize the build-up of recirculating copper in the gold circuit solutions and return free cyanide to the gold heap leach. This treatment is based on the SART process (sulphidization, acidification, recycling and thickening) which produces a copper sulphide precipitate (which grades approximately 65% copper) and recovers cyanide for use in the heap leach.

The proposed production schedule and metal production profile are shown in the attached figures. Note that the project assumes a 24-month construction period.

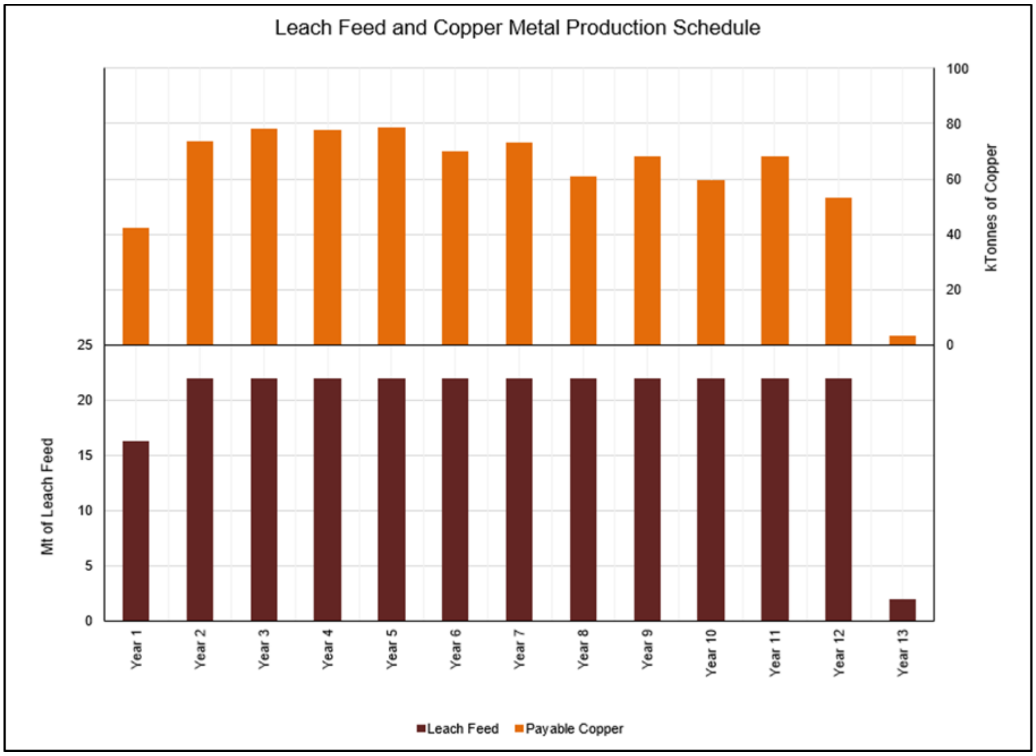


Figure 1-1: Leach Feed and Copper Metal Production Schedule

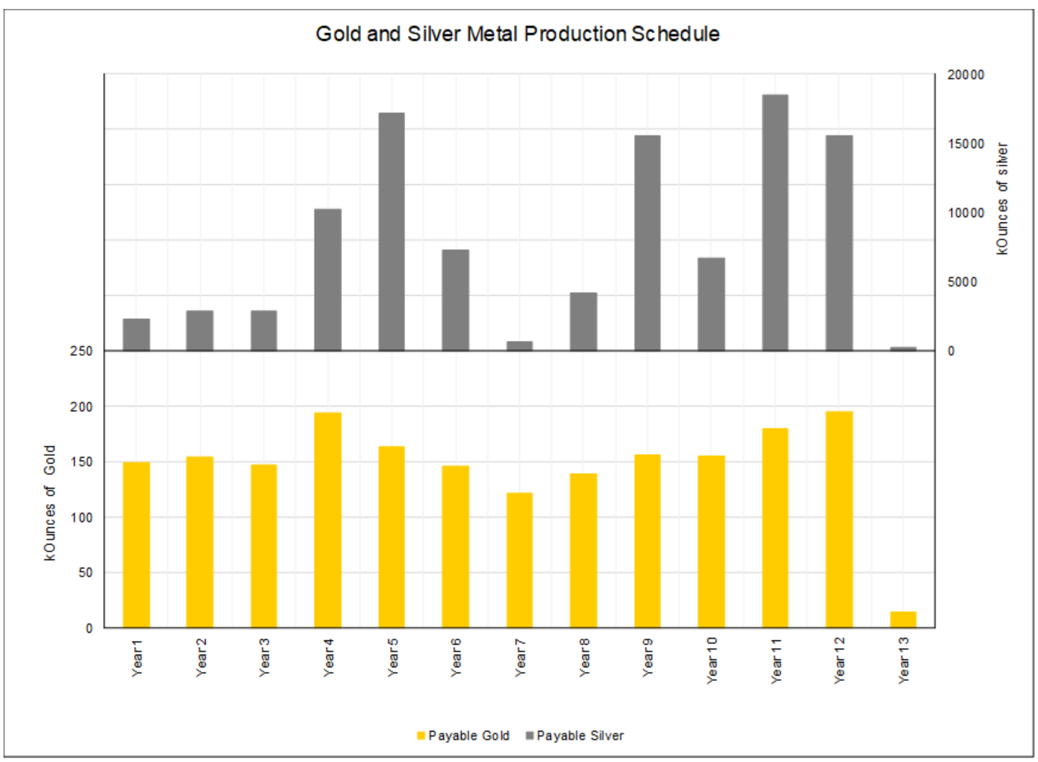


Figure 1-2: Gold and Silver Metal Production Schedule

1.16 Infrastructure

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 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. The majority of the mine site has permafrost located 0.5 to 1.0 meters below the surface. The design of structures took this into account.

The major infrastructure items considered and costed are listed below.

Water Supply: Water will be supplied from aquifers in Argentina, located near the proposed plant site. The industrial water make-up requirement is estimated to be 75 L/s and is expected to be fully supported by the aquifers.

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 52 MW. A price of $0.075/kWh was used for long-term power supply.

Product Transport: Copper cathode will be transported by truck to Puerto Caldera, a port near the city of Caldera which is located 77 km by road northwest of Copiapó. The approximate trucking distance from the plant site is 245 km, of which roughly 60 km of existing road will require upgrade to accommodate the truck traffic. Doré will be transported approximately 175 km to Aeropuerto Desierto de Atacama for ongoing airfreight.

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.

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

Knight Piésold completed the environmental baseline work for the Company in 2017 and 2018 in addition to reviewing the historical work from other independent consultants who assisted in the preparation of the environmental work. This work will be used to support the preparation of the respective Environmental Impact Assessments ("EIA").

Baseline studies to date include geosciences, air & water, terrestrial biota, the human environment, and natural & 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 land owners, and other interested parties is also ongoing.

1.19 Capital Cost Estimate

Capital costs were generated from a variety of sources including derivation from first principles, equipment quotationa and factoring from actual costs incurred in the execution of other, similar projects. Costs are estimated to an accuracy of +/- 25% which is equivalent to an AACE International, Class 4 Estimate.

All costs are reported in US dollars.

Table 1-3: Capital Cost Estimate

Cost Centre	$M
Mine Pre-strip	59
Mining	121
Crushing	67
Processing	325
On-Site Infrastructure	94
Off-Site Infrastructure	124
Total Direct Costs	**789**
Indirect Costs	132
Project Delivery	101
Owner's Costs	50
Contingency	194
TOTAL INITIAL CAPEX	**1,266**
LOM Sustaining Capital	217
Closure	51
TOTAL LIFE OF MINE CAPITAL	**1,534**

1.20 Operating Cost Estimate

Determination of the operating costs estimated that the C1 cash costs (co-product basis) over the life of mine will average $1.23/lb CuEq. C1 cash costs include at-mine cash operating costs, treatment and refining charges, royalties, selling costs, and transportation costs, and are reported on a $/equivalent payable unit of the primary metal.

Table 1-4: Operating Cost Estimate

Operating Costs	($/t processed)
Mining	3.86
Processing	8.90
Site G&A	1.44
TOTAL	**14.19**

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 and 8% discount rate is estimated to be $1.28 billion, with an IRR of 23%. The project financial summary is shown in Table 1-5.

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.

Table 1-5: Project Financial Summary

Project Metric	Units	Value
Pre-Tax NPV (8%)	$B	$1.86
Pre-tax IRR	%	27
After-Tax NPV (8%)	$B	$1.28
After-Tax IRR	%	23
Undiscounted After-Tax Cash Flow (LOM)	$B	$3.23
Average Operating Margin*	%	62
Payback Period from start of processing (undiscounted, after-tax cash flow)	years	3.4
Initial Capital Expenditures (rounded)	$B	$1.27
LOM Sustaining Capital Expenditure (excluding closure)	$M	$0.22
LOM C-1 Cash Costs (Co-Product)	$ per lb Cu.Eq.	$1.23
Nominal Process Capacity	tonnes per day	60,000
Mine Life (including pre-stripping)	years	14
Average Annual Copper Production**	tonnes Cu	67,000
Average Annual Gold Production**	ounces Au	159,000
Average Annual Silver Production**	ounces Ag	8,653,000
LOM Average Process Recovery – Copper***	%	80
LOM Average Process Recovery - Gold	%	70
LOM Average Process Recovery - Silver	%	82

* Operating Margin = Operating Cashflow/Net Revenue.

** Rounded and excluding final year of minimal leach operation.

*** Including 1% Cu recovery to concentrate for SART process.

1.22 Sensitivity Analysis

A cash flow valuation model for the project has been developed using a long-term copper price of $3.00/lb, gold price of $1,300/oz, and silver price of $20/oz. The following figure shows the sensitivity of estimated NPV for the Project's cash flow at various changes to metal prices at 8% discount rate.

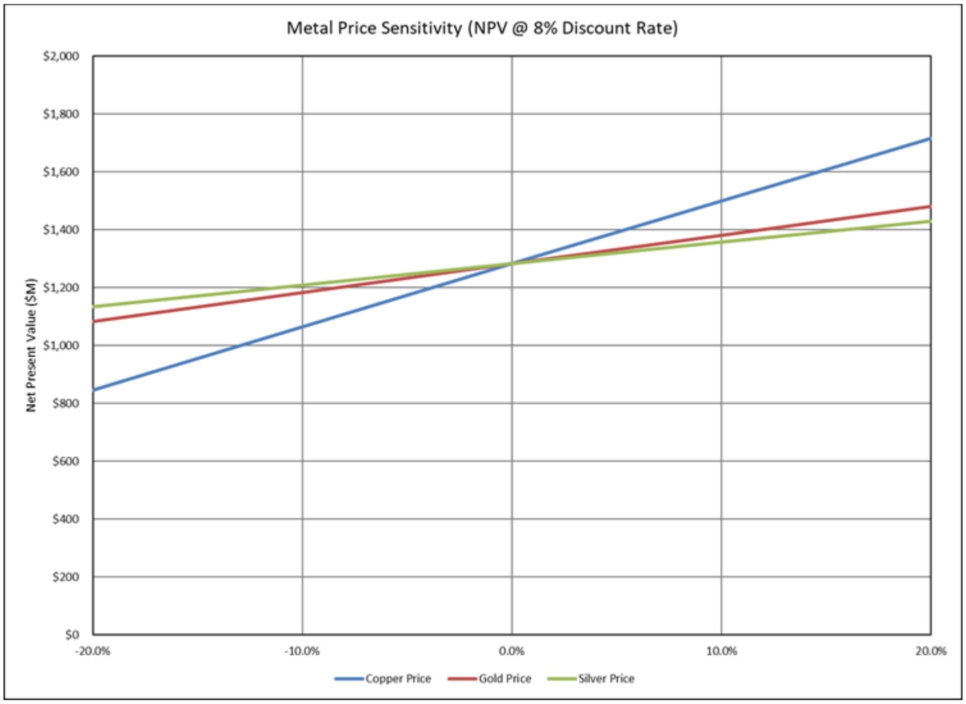


Figure 1-3: Metal Price Sensitivity

1.23 Interpretations and Conclusions

The work that has been completed in this PFS has indicated that the Filo del Sol Project has significant potential economic merit. Financial analysis has shown significant net present value and internal rate of return.

The Filo del Sol Project encompasses a very large zone of alteration and several mineralized showings 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 updated mineral reserve estimate presented in this report significantly expands an important copper-gold-silver deposit on the property.

The Filo del Sol Project is amenable to development by open pit mining methods. Ausenco 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.

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

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.

1.24 Recommendations

The Filo del Sol PFS has provided a technical and economic solution that is an excellent basis on which to further advance the project. The next phases of the project are to complete optimization studies and proceed to a Feasibility Study.

2 Introduction

2.1 Terms of Reference

The Filo del Sol Project is an early 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 technical and economic results of a Pre-Feasibility Study (PFS) which contemplates the mining and heap leach processing of the Filo del Sol deposit. This report follows the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1. The mineral resource and mineral reserve statements reported herein were prepared in conformity with generally accepted CIM "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines."

2.2 Qualified Persons

The following Qualified Persons 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 Elfen	Ausenco	1.16, 18.2, 18.8, 18.9, and 18.10 as well as relevant parts in the Conclusions and Recommendations
Robin Kalanchey	Ausenco	1.1, 1.11, 1.15, 1.17, 1.19, 1.20, 1.23, 2 (in its entirety), 3 (in its entirety), 13 (in its entirety), 17 (in its entirety), 18.1, 18.3 to 18.7, 19 (in its entirety), 21 (in its entirety), as well as relevant parts in the Conclusions and Recommendations
Bruno Borntraeger	Knight Piésold	1.18, 20 (in its entirety), parts of 25.1, as well as relevant parts in the Conclusions and Recommendations
Fionnuala Devine	Merlin Geoscience	1.2 to 1.10 , 4 (in its entirety), 5 (in its entirety), 6 (in its entirety), 7 (in its entirety), 8 (in its entirety), 9 (in its entirety), 10 (in its entirety), 11 (in its entirety), 12 (in its entirety), 23, and 24 as well as relevant parts in the Conclusions and Recommendations
Ian Stilwell	BGC Engineering Inc	Section 16.1 as well as relevant parts in the Executive Summary, Conclusions and Recommendations.
Neil Winkelmann	SRK	1.21, 1.22, 22 (in its entirety), as well as relevant parts in the Conclusions and Recommendations
Jim Gray	Advantage Geoservices Limited	1.12, 14 (in its entirety), as well as relevant parts in the Conclusions and Recommendations
Jay Melnyk	AGP Mining Consultants	1.13, 1.14, 15 (in its entirety), 16.2 to 16.8 as well as relevant parts in the Conclusions and Recommendations

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 2-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. Her most recent visit was in February 2017 for three days to visit the Tamberias 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.

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) w ere visited. Some drill sample handling was witnessed, but not inspected nor audited in detail.

2.3.3 2018 Site Visits

On 3rd February 2018 the following people visited the site: Scott Elfen, Robin Kalanchey and Jay Melnyk.

- 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. Kalanchey visited the project site, including proposed locations for infrastructure and process facilities, and reviewed the proposed access route and potential camp location. While at site he inspected drill core from the ongoing drill program.
- Mr. Melnyk inspected select drill core from the ongoing drill program and the terrain in the vicinity of the proposed pits and possible waste dump locations.

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

Mr. Ian Stilwell, P.Eng. visited the core storage facility in Copiapó, Chile to review the available drill core and select samples for testing on March 15, 16 and 18, 2018. Mr. Stilwell also visited the project site on March 17 to inspect the geotechnical drilling operation and review the geotechnical conditions of rock exposures in the vicinity of the proposed pit.

Table 2-2: QPs' Site Visits

Qualified Person	Company	Date(s) of Site Visit
Bruno Borntraeger	Knight Piésold	March 2018
Fionnuala Devine	Merlin Geoscience	January 2014, January 2015, February 2016 and February 2017
James N. Gray	Advantage Geoservices Limited	Did not visit the property.
Jay Melnyk	AGP Mining Consultants Inc.	February 2018
Neil Winkelmann	SRK Consulting Inc.	February 2017
Robin Kalanchey	Ausenco	February 2018
Scott Elfen	Ausenco	February 2018
Ian Stilwell	BGC Engineering Inc.	March 2018

2.4 Effective Dates

Mineral Resources have an effective date of 11 July 2018.

Mineral Reserves have an effective date of 13 January 2019.

The overall effective date of this PFS report is taken to be 13 January 2019.

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

Filo Mining previously commissioned SRK Consulting (Canada) Inc. to visit the property and conduct a Preliminary Economic Assessment (PEA) for the Filo del Sol Project. The services were rendered in 2017 leading to the preparation of the Technical Report that was disclosed publicly by Filo Mining in a news release on 28th November 2017. The PEA was released on 18th December 2017.

2.7 Reporting Standards

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

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 Nicholson y Cano Abogados addressed to Filo Mining Corp. November 12, 2018;
- Title Opinion letter from Bofill Mir & Alvarez Jana, Abogados addressed to Filo Mining Corp. December 19, 2018.

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 experts retained by Filo Mining for this information.

This information is used in Section 22 of the Report.

4 Property Description and Location

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 14,014 ha. This area does not match the sum of the individual claim areas 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.

4.1 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.

4.2 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 life time. 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 1 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. Filo del Sol Exploración S.A. has the properties detailed in Table 4-1 and Table 4-2 and shown on Figure 4-1.

Table 4-1: Exploration Cateos Owned – Argentina

Concession	File Number	Area (ha)
Cateo	339.215-C-92	3,807
Cateo	339.212-B-92	4,027

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. 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 4-2.

Table 4-2: 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 3	520-0101-B-98	1,491	15	48,000
Vicuña 5	425-247-B-00	1,500	15	48,000
Vicuña 6	414-145-C-04	1,500	15	48,000
Vicuña 7	1124-029-C-09	1,500	15	48,000
Vicuña 8	1124-286-F-14	1,488	15	48,000

* Area uncertain due to undefined National or Provincial boundary.

** Caballo II is comprised of vein deposit mining units.

The Vicuña 3 Concession is subject to an Asset Purchase Agreement between Filo Mining and NGEx Resources Inc. (“NGEx”) dated January 11, 2018. This agreement provides for the transfer of this concession from Filo Mining to NGEx within a period of 10 years subject to certain conditions.

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.

4.2.1 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.

4.3 Environmental Permits

- Caballo I and Caballo II: approved exploration EIR and evaluation of 1st update.
- Vicuña 1, Vicuña 2 and Vicuña 3: approved exploration EIR and evaluation of 1st update.
- Vicuña 5, Vicuña 6: the filing procedure has been complied with and it is ready for measurement, inclusion in the approved EIR may be requested.
- Vicuña 7 and permits 339215-C-92 and 339212-B-92: not yet eligible for inclusion in the approved EIR.
- Vicuña 8: eligible for inclusion in the approved EIR.

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.

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.

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.

Mensura

Within nine months of the approval of the “Request for Survey” by the court, the claim must be surveyed by a government licensed surveyor. 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 Tamberias Properties

Frontera Chile Limitada is the owner of 14 granted Exploration Mining Concessions, two Exploration Mining Concessions in the process of being granted, two 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 4-3: Exploration Mining Concessions Granted – Chile

Concession	National ID Number (ROL NACIONAL)	Status	Holder	Area (ha)
Tamberia Primera 1	03203-E615-5	Granted	Frontera	300
Tamberia Primera 2	03203-E632-5	Granted	Frontera	300
Tamberia Primera 3	03203-E604-K	Granted	Frontera	300
Tamberia Primera 4	03203-E631-7	Granted	Frontera	300
Tamberia Primera 5	03203-E616-3	Granted	Frontera	300
Tamberia Primera 6	03203-E641-4	Granted	Frontera	300
Tamberia Primera 7	03203-E605-8	Granted	Frontera	100
Tamberia Primera 8	03203-E621-K	Granted	Frontera	300
Tamberia Primera 9	03203-E617-1	Granted	Frontera	300
Tamberia Primera 10	03203-E642-2	Granted	Frontera	300
Tamberia Primera 11	03203-E606-6	Granted	Frontera	200
Tamberia Primera 12	03203-F072-1	Granted	Frontera	300
Tamberia Primera 13	03203-F073-K	Granted	Frontera	200
Tamberia Primera 14	03203-F074-8	Granted	Frontera	200
Tamberia Tercera 4	03203-F330-5	In process	Frontera	300
Tamberia Tercera 5	03203-F291-0	In process	Frontera	300

Table 4-4: Exploitation Mining Concession Granted

Concession	National ID Number (ROL NACIONAL)	Status	Holder	Area (ha)
Frontera IV 1/60	03203-7278-K	Granted	Frontera Chile Limitada	300
Frontera V 1/60	03203-7279-8	Granted	Frontera Chile Limitada	300

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	National ID Number (ROL NACIONAL)	Status	Holder	Area (ha)
Vicuña 14 1/30	03203-2889-6	Granted	Cía. Minera Tamberías SCM	300
Vicuña 13 1/30	03203-2888-8	Granted	Cía. Minera Tamberías SCM	300
Vicuña 12 1/30	03203-2882-9	Granted	Cía. Minera Tamberías SCM	300
Vicuña 11 1/30	03203-2887-K	Granted	Cía. Minera Tamberías SCM	300
Vicula 10 1/30	03203-2886-1	Granted	Cía. Minera Tamberías SCM	300
Vicuña 9 1/30	03203-2885-3	Granted	Cía. Minera Tamberías SCM	300
Vicuña 8 1/30	03203-2884-5	Granted	Cía. Minera Tamberías SCM	300
Vicuña 7 1/12	03203-2881-0	Granted	Cía. Minera Tamberías SCM	120
Tamberia 1 1/20	03203-4046-2	Granted	Cía. Minera Tamberías SCM	200
Tamberia 1 1/30	03203-4047-0	Granted	Cía. Minera Tamberías SCM	300
Tamberia 3 1/30	03203-4048-9	Granted	Cía. Minera Tamberías SCM	300
Tronco 1 1/41	03203-4145-0	Granted	Cía. Minera Tamberías SCM	41
Tronco 2 1/76	03203-4146-9	Granted	Cía. Minera Tamberías SCM	76
Tronco 3 1/50	03203-4147-7	Granted	Cía. Minera Tamberías SCM	50
Tronco 6 1/39	03203-4193-0	Granted	Cía. Minera Tamberías SCM	178
Anillo 10 1/81	03203-4351-8	Granted	Cía. Minera Tamberías SCM	81
Anillo 11 1/30	03203-4352-6	Granted	Cía. Minera Tamberías SCM	19

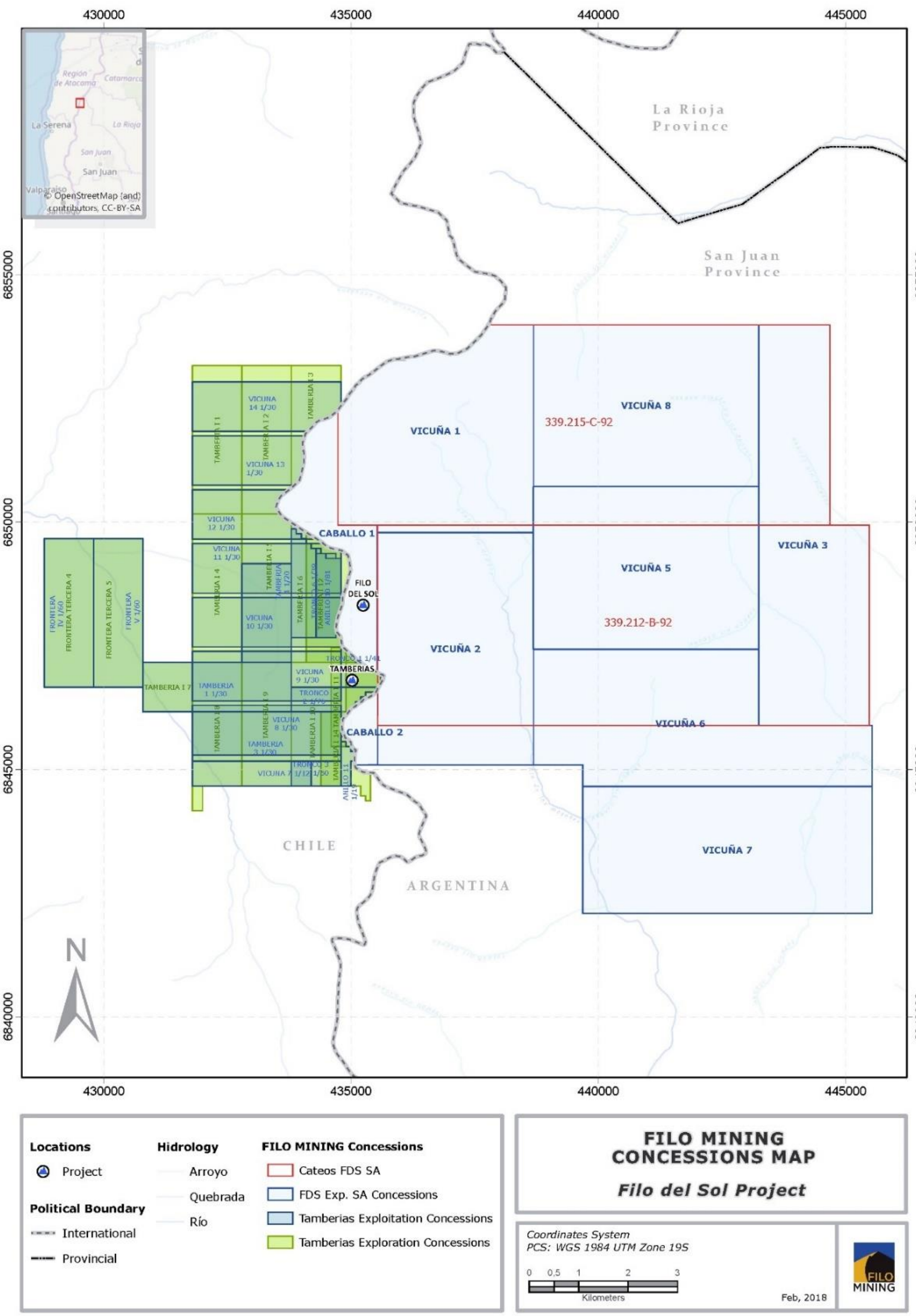
By public deed dated 27th July 2012 before the Santiago Notary Public of Antonieta Mendoza Escalas, Minera Frontera del Oro SpA assigned the Option Agreement to Frontera Chile Limitada. Frontera may exercise the Option Agreements within the period that ends on 30th June 2023. The purchase price of the 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, $ 3,200,000 of the total has been paid.

4.4.3 Surface Rights

Surface land rights in the area of the Tamberías Property are held by a local community, *"Comunidad* Civil Ex Estancia Pulido". Filo Mining has an agreement with the Pulido community to provide access to the project for a period of four years, beginning in November 2017.

4.4.4 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.



Source: Filo, 2019

Figure 4-1: Mineral Titles

5 Accessibility, Climate, Local Resources, Infrastructure, and Physiology

5.1 Access

The project is accessible by road from either Copiapó, Chile or San Juan, Argentina although Copiapó is closer. Access routes are shown in Figure 5-1.

Access from Copiapó (Chile) is via the C-35 sealed road in a southeasterly direction through the town of Tierra Amarilla and Punta del Cobre, along the Copiapó River valley through small villages Pabellon, Los Loros, La Guardia and Iglesia Colorada. After these small villages, the road becomes the C-453 and continues towards the El Potro bridge which is at about 130 km. Crossing the bridge, a gravel road leads the final 50 km to the drill sites. The total driving time is approximately four hours. Access by this route is generally possible from November to April.

An alternative but longer access is possible from the City of San Juan, Argentina. Travel starts northwards on 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.

5.2 Climate

The climate is frequently cold and windy, typical of the high Andes. The practical exploration field season runs from November to April and requires the presence of a bulldozer all season in case of sudden snowfalls. Year-round access to the project could be maintained if two to three bulldozers were deployed on road clearing and maintenance, but low temperatures make it difficult to work productively in the winter months.

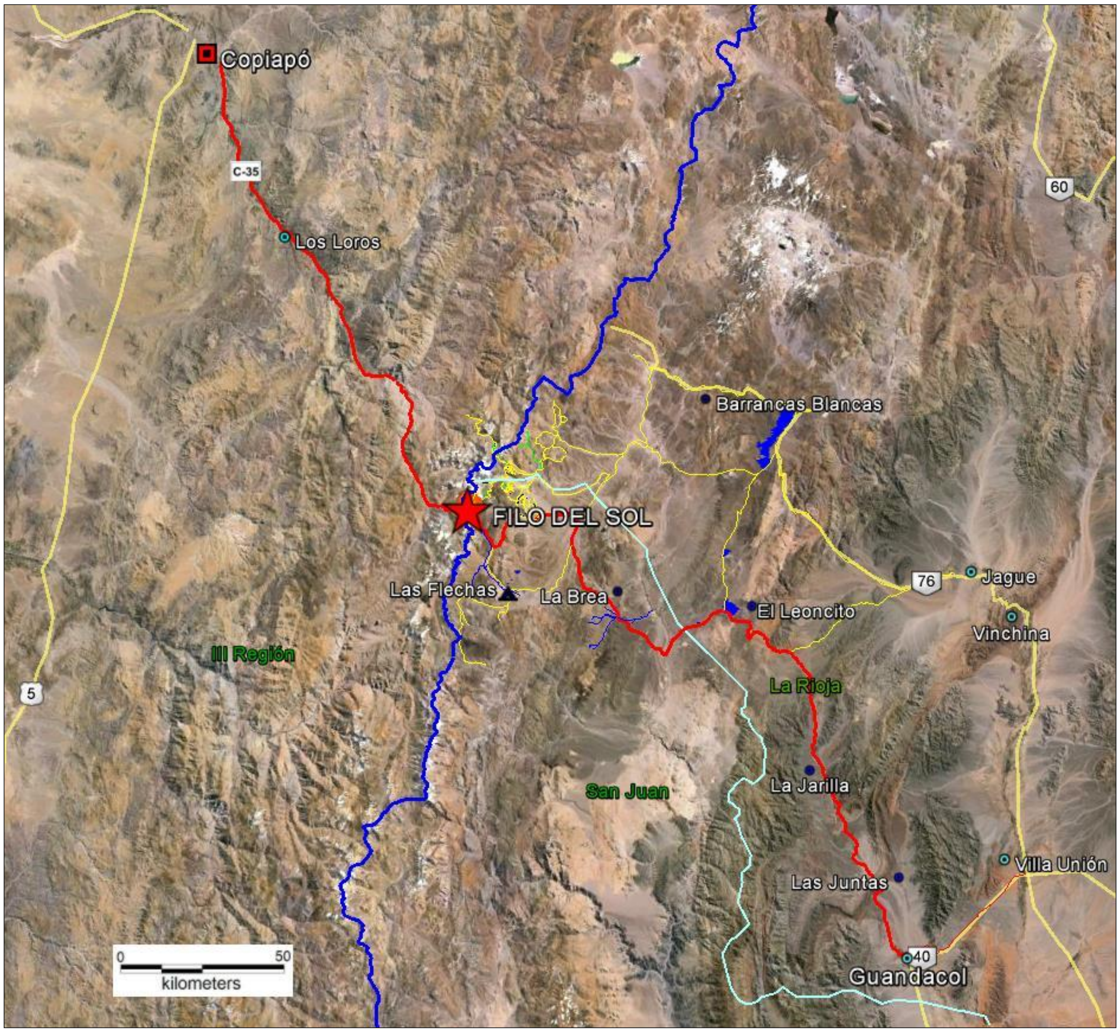
Should a project be developed into operation, mining conditions would be comparable to those at the El Indio, Veladero, and Refugio Mines.

5.3 Local Resources and Infrastructure

Field work is based out of the Batidero camp located approximately 20 km from the project in Argentina. The Batidero camp can accommodate approximately 200 people. The site is remote and, other than road access, there is no infrastructure available.

5.4 Physiography

Elevations on the property range from 3,800 m to 5,500 m amsl. The mountains are generally not rugged and vehicle access is possible to most of the property. Vegetation is almost entirely absent in the area.



Source: Devine et al., 2017

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

The site is situated in a glacial and periglacial belt that is characterized by permafrost and various cryoforms such as glaciers and rock glaciers. In addition, the majority of the rock glaciers are active with annual deformations between 0.1 m and 1.0 m.

6 History

Cyprus-Amax was the first company to carry out 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 Tamberias 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 Co. Vicuña, and a drill program of 2,519 m in 16 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 (Co. 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 on the property.

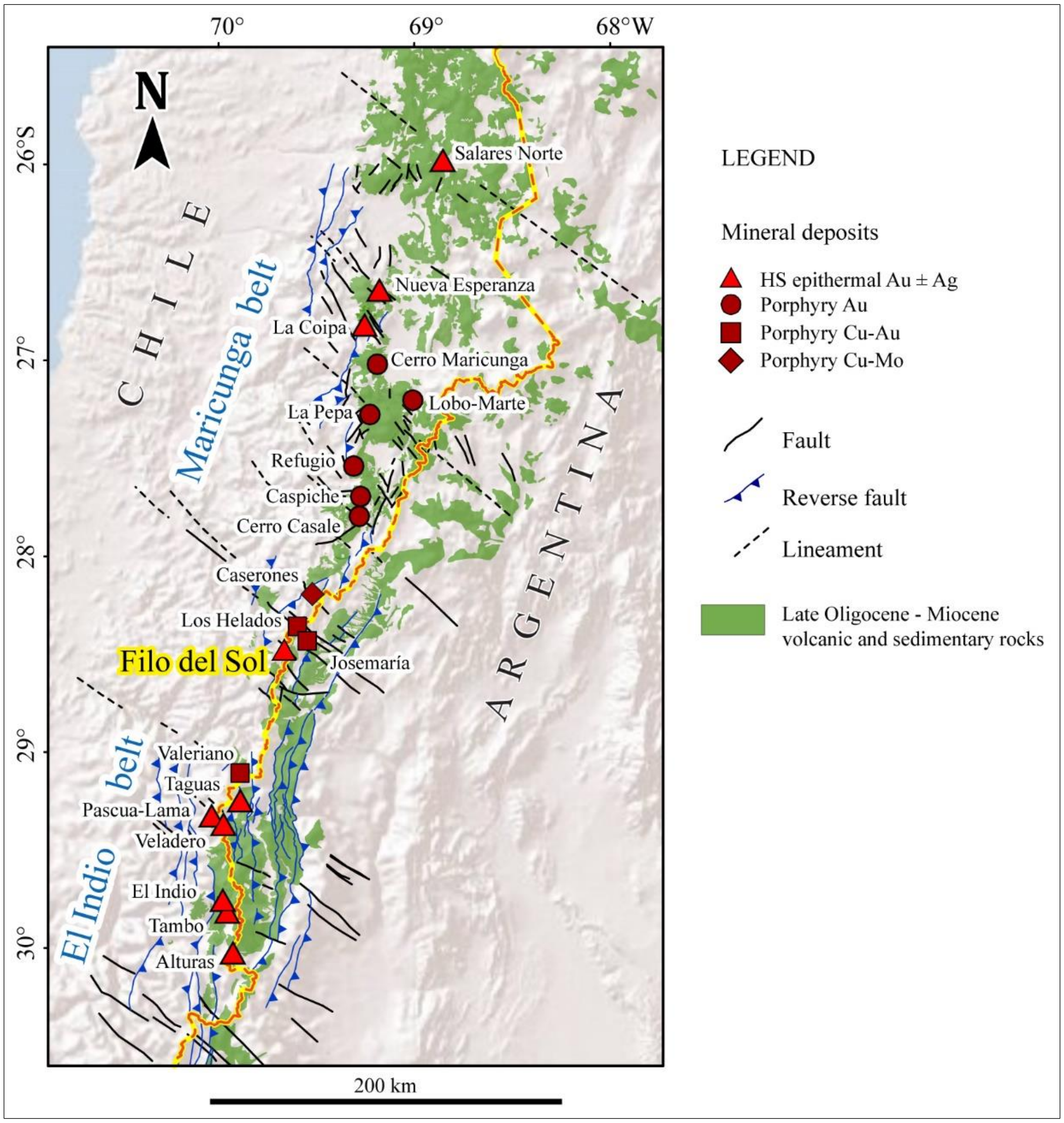
Filo Mining Corp. became involved in the project through its predecessor company, Tenke Mining Corp., which negotiated a purchase arrangement with Cyprus-Amax in August 1999. Tenke operated from 1999 to 2007 and subsequent field seasons were carried out by Filo Mining's predecessor companies, Suramina 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 Vicuña district is located at the crest of the Andes at 28.5° S latitude. The modern day geology is the result of east-directed subduction and volcanic arc development with associated volcanism and deformation along the Pacific margin. Basement rocks in the region include Late Paleozoic granites and rhyolites of the Choiyoi Group and equivalents that are overlain by Jurassic to Cretaceous sediments related to earliest arc development. Episodic contraction, beginning in middle Cretaceous time, has resulted in the uplift of the Andes. Periods of extension within the arc, such as is in the Paleocene-Eocene resulted in extensional faults and associated basins filled with terrigenous sediments. Eocene dioritic intrusive complexes are associated with this period. Compression from the Late Oligocene to present day, particularly associated with the development of the Miocene volcanic arc, has resulted in inversion of the Paleocene-Eocene extensional faults and related basins.

Late Oligocene to Miocene intrusions and associated volcanic rocks form several belts in the central Andes, prospective for porphyry and epithermal Au-Cu systems (including the Maricunga belt) and high-sulphidation epithermal systems (including the El-Indio – Pascua district). The Maricunga belt includes mineralization from Late Oligocene as well as Miocene events (Vila and Sillitoe, 1991), while the more southerly El Indio-Pascua district high-sulphidation alteration and mineralization is predominantly Middle to Late Miocene (Bissig, et al., 2002). Until the late 1990's, the Maricunga and El Indio belts were the focus of exploration in the region due to the interest generated by earlier discoveries; however, it was recognized (Tenke Mining Corp. and early explorers; Mpodozis and Kay, 2003) that the area between these two districts was prospective for similar mineralized systems. Subsequent work has shown this to be true, with the discovery of the Los Helados, Josemaría and Filo del Sol deposits with Late Oligocene to Late Miocene ages. In addition, intrusions in the region with associated hydrothermal alteration and some similarity to the Maricunga-style Au-porphyries have been dated as Middle Miocene and there is local evidence of magmatic rocks synchronous with the Late Miocene El Indio mineralization event (Mpodozis and Kay, 2003).



Source: Modified after Sillitoe et al., in preparation

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

Figure 7-1 shows the distribution of Late Oligocene to Miocene volcanic rocks in the region and associated porphyry and epithermal deposits.

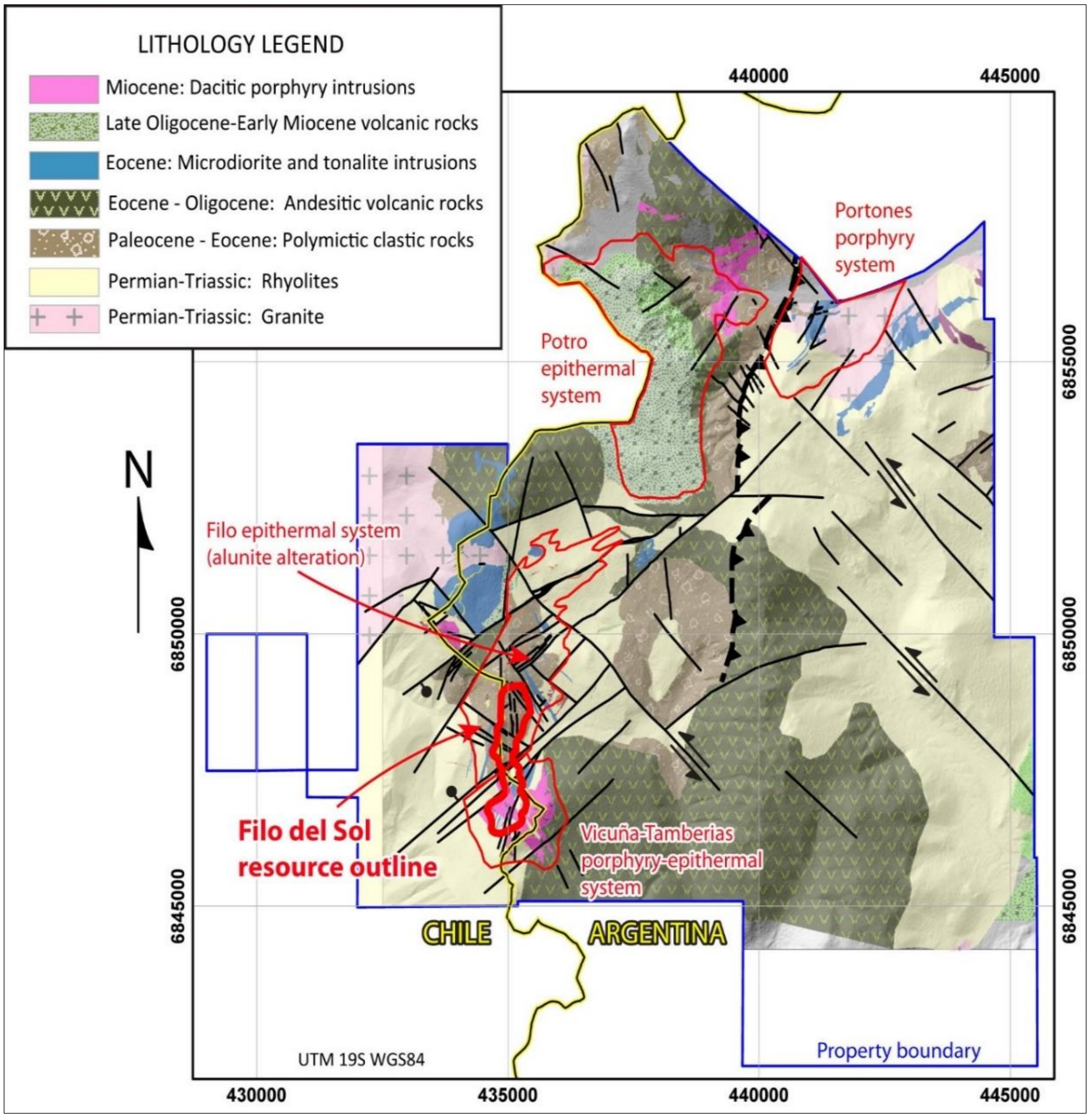
7.2 Project Geology

The Filo del Sol region hosts both porphyry Cu-Au and high-sulphidation epithermal Au-Cu-Ag mineralization. Several Miocene porphyry centres have been mapped, in many cases with preserved high-sulphidation epithermal systems topographically above, or structurally-offset from, the deeper-level porphyry domain.

In the Filo del Sol area, the mineralized system is associated with Middle Miocene porphyry intrusions and overprinting, telescoped high-sulphidation epithermal alteration within Permo-Triassic rhyolite basement rocks and overlying conglomeratic sediments thought to be either Eocene of Late Oligocene, in age. Porphyry intrusions and related high-sulphidation epithermal mineralization have been defined along a north-south trend which may broadly be controlled by a reactivated, pre-Miocene, north-south extensional fault. Cu-Au porphyry mineralization is mapped around several porphyry centres, with overprinting epithermal alteration and mineralization. Miocene uplift and telescoping within the system is evident with different intrusions having been emplaced at different levels and with early potassic alteration overprinted by high-sulphidation epithermal-related assemblages.

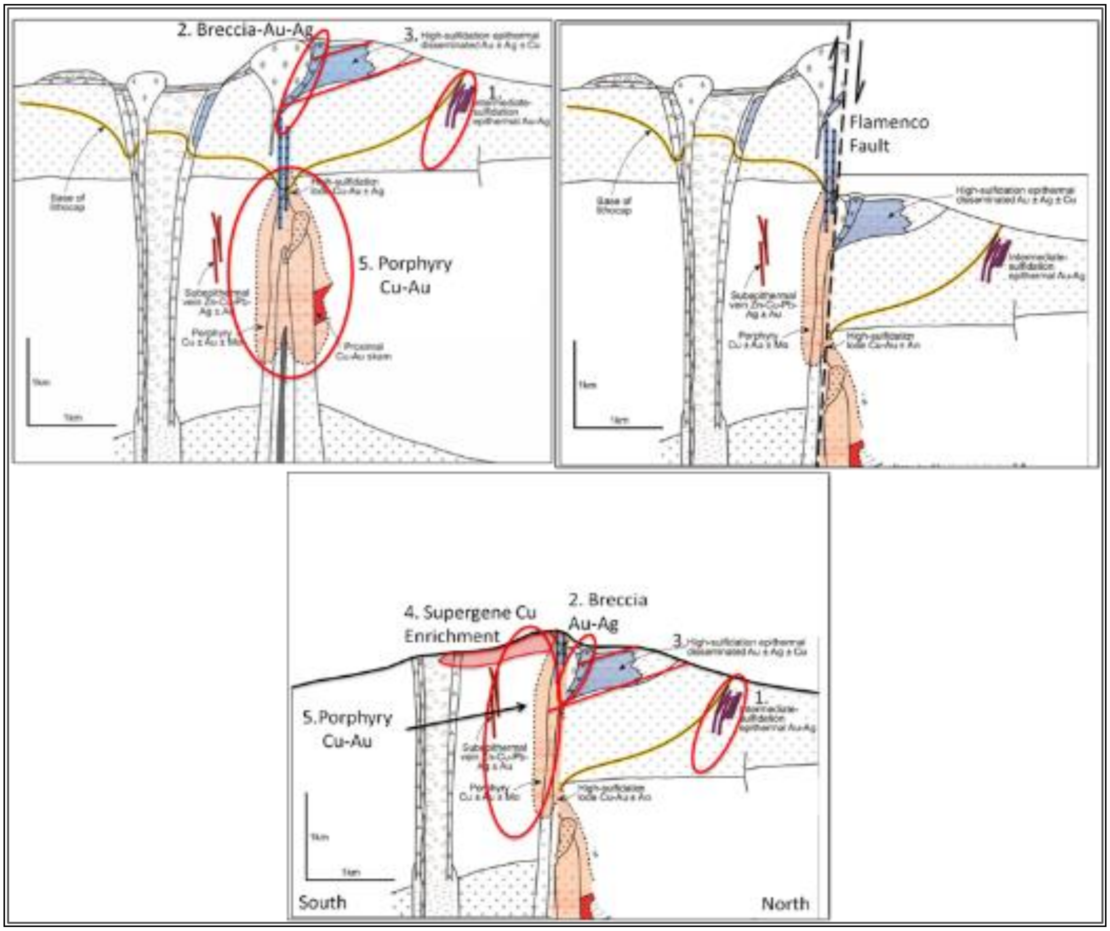
Through a combination of telescoping, as a result of unroofing of the system during emplacement, and post-mineral faulting, different parts of the magmatic-hydrothermal system have been exposed at surface. The Filo del Sol resource spans two domains that share similar geology but are offset across the post-mineral northeast-trending Flamenco fault resulting in erosion to different levels. The domains are: a shallowly eroded high-sulphidation epithermal system in the north (at Filo del Sol, Figure 7-3 and Figure 7-4) and an outcropping porphyry copper-gold/epithermal system in the south at Tamberias (previously referred to as Cerro Vicuña-Flamenco or Filo Mining South). The Filo del Sol deposit presented in the current mineral resource estimate spans both domains.

The largest part of the known Filo del Sol deposit, in terms of contained metal, encompasses the epithermal system preserved under the lithocap domain along the topographic ridge crest at Filo del Sol. Lithological control plays a significant role in the focus and concentration of metals in this domain, with a high-grade silver zone developed at the base of the conglomerate unit which is interpreted to have controlled fluid flow during development of the system. Upper-level gold and copper mineralization have been affected by supergene processes that leave gold distribution unaffected, but have redistributed copper. The result is a high-grade soluble copper zone near the top of the system below the uppermost, locally gold-rich leached cap. Part of what may be the offset lateral continuation of the Filo del Sol epithermal system continues to the south of the Flamenco fault in the Tamberias area. Here the epithermal alteration and mineralization overlies, and is juxtaposed against, an eroded porphyry system with Cu-Au mineralization associated with Middle Miocene dacitic porphyry intrusions.



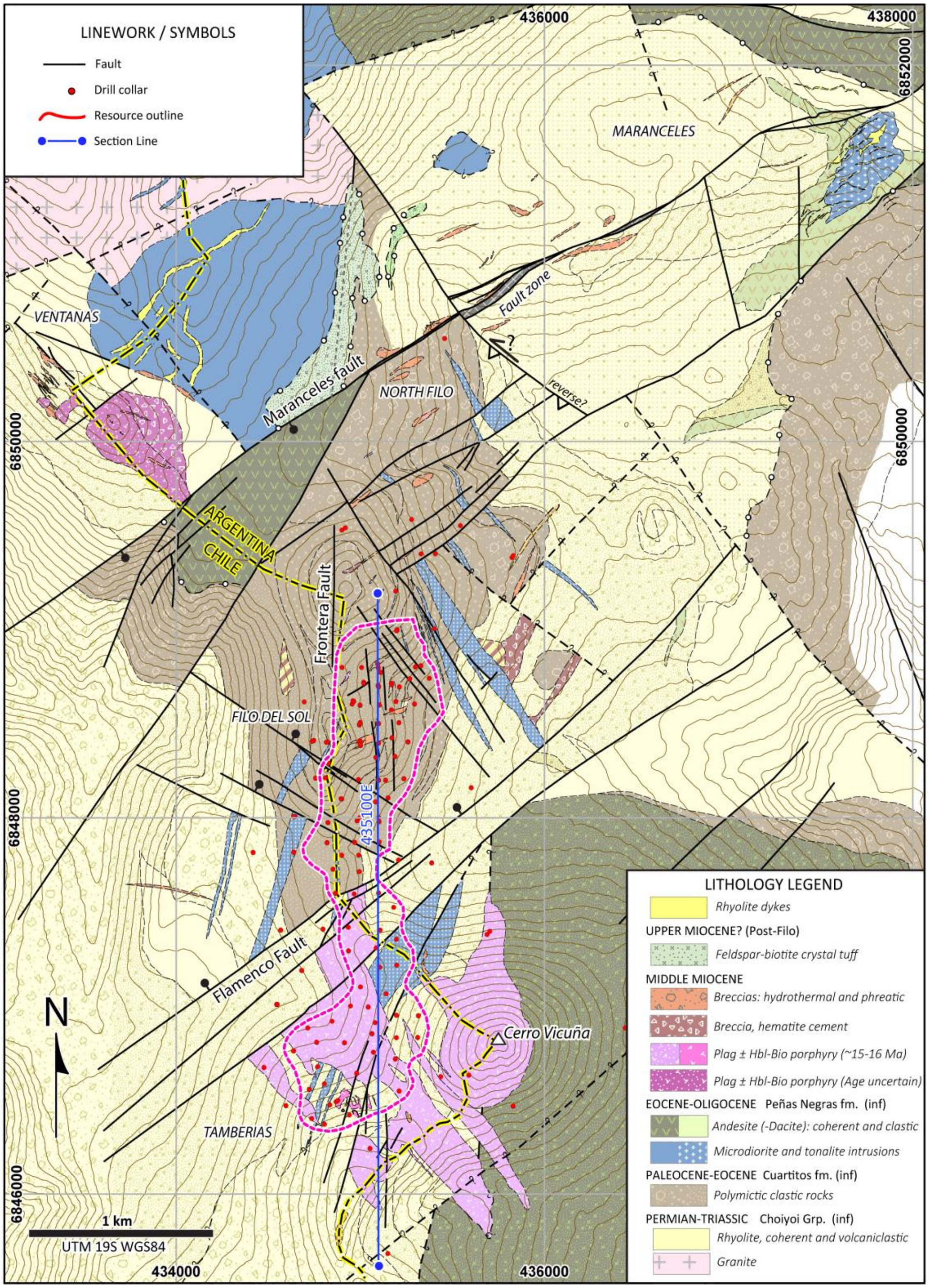
Source: Devine, 2017

Figure 7-2: Property Geology



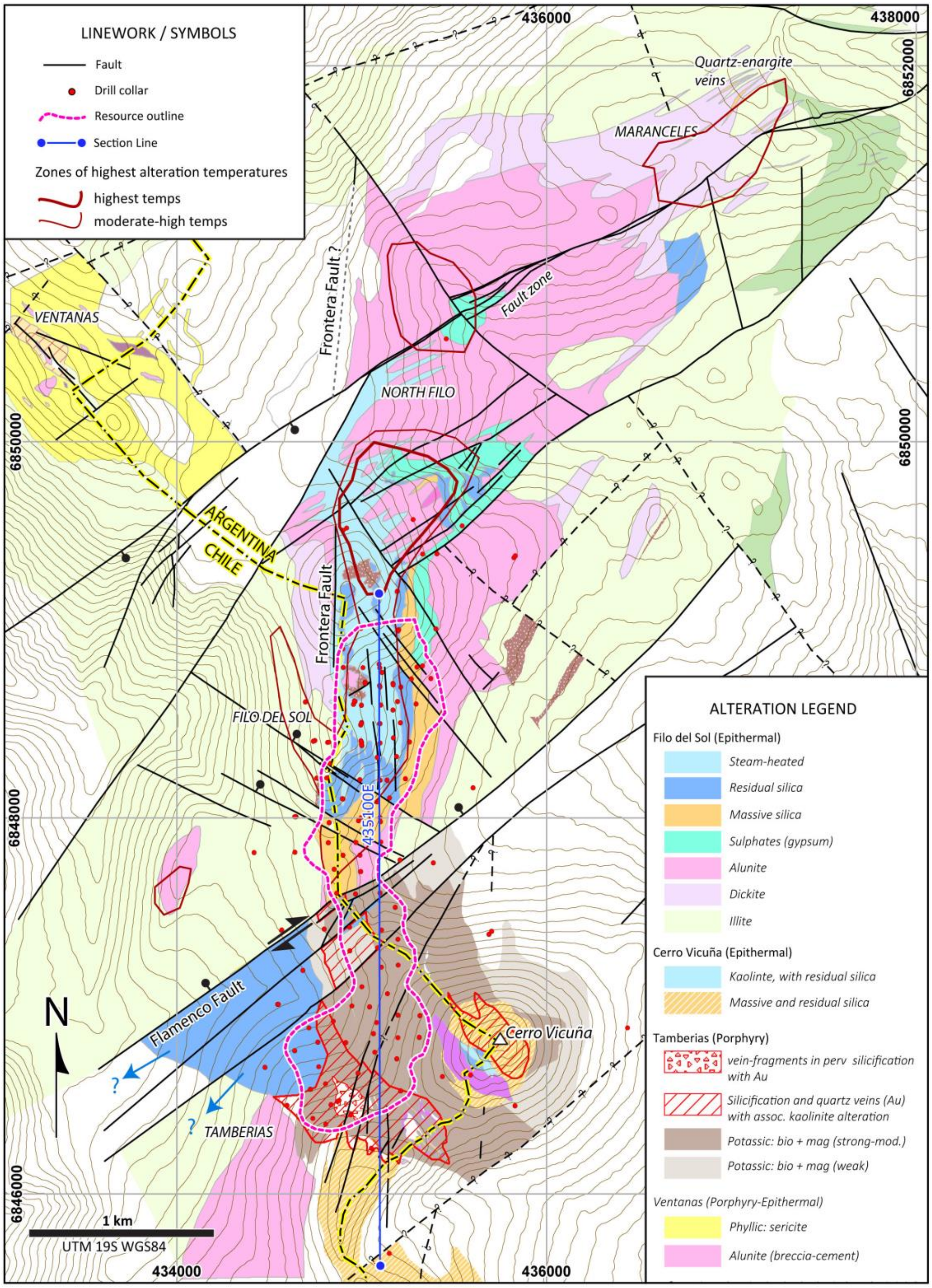
Source: Carmichael, 2015 - modified from Sillitoe, 2010
Note: E-W Vertical Section

Figure 7-3: Schematic Showing Juxtaposition of Deep vs. Shallow Levels Across Flamenco Fault using a structural interpretation. At Filo, telescoping has also played a role in the juxtaposition.



Source: Devine et al. 2017

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



Source: After Devine, et al., 2017; alteration temperature interpretation after Heberlein and Heberlein, 2015

Figure 7-5: Surface Alteration Map – Filo del Sol Deposit Area

7.2.1 Lithology

Host Rocks

Much of the Filo del Sol area is underlain by a rhyolitic volcanic package including quartz-feldspar phyric flow-banded rhyolites, and rhyolitic tuffaceous rocks. Some granite occurs to the north of the deposit area. The rhyolite and granite are both considered to be Permo-Triassic in age, part of the Choiyoi Group and equivalents that commonly form the basement in this region.

A conglomeratic unit unconformably overlies the rhyolitic units. The conglomerates are distinctly red to maroon weathering where fresh and unaffected by Miocene alteration. They range from medium-grained, bedded sandstones to granule, pebble, and cobble conglomerates that are comprised of a variety of rounded, dominantly rhyolitic clasts, in a granular, clastic matrix. The conglomeratic clastic rocks overlie the Permo-Triassi rhyolitic units across a significant regional unconformity. They have been inferred to belong to the Paleocene-Eocene Cuartitos (Chapitas) formation extensional basin-fill deposits, however recent work suggests that a relationship to the Late Oligocene compressional event in the region cannot yet be ruled out.

A dacitic to andesitic volcaniclastic unit overlies the conglomerate. It includes bedded lapilli-tuff on the eastern flank of Cerro Vicuña, but ranges to more texturally variable volcaniclastic units in other parts of the property, including andesitic-clast breccias and conglomerates.

Throughout the Filo del Sol area a suite of microdiorite dykes, 3 m to 100 m wide, cut across the rhyolite and conglomeratic units. The dykes are pre-mineral (relative to Filo del Sol), but their age is uncertain. They may be an intrusive equivalent to the andesitic volcaniclastic rocks overlying the conglomerate (Sanguinetti, pers. comm., 2017).

Miocene Porphyry Intrusions, Breccias, and Volcanic Rocks

Miocene porphyry intrusions occur in several areas along the trend of the Filo del Sol system. They are dacitic in composition and feldspar ± hornblende - biotite phyric. Three main domains are mapped: in the Cerro Vicuña-Tamberias area, along Filo del Sol, and in the Ventanas area.

The Cerro Vicuña-Tamberias porphyries are the most voluminous (at surface) and include several bodies comprised of at least two distinct varieties of intrusion based on phenocryst morphology and content: a crowded feldspar-phyric variety, and a plagioclase-hornblende-biotite-phyric variety. The main northwest-trending dyke-like body is predominantly composed of the plagioclase-hornblende-biotite phase, while the central part of the Tamberias area includes a feldspar-phyric phase. An important feature of the Tamberias area is that different porphyry bodies have been intruded to different depths, indicating unroofing and erosion during porphyry emplacement. The highest-elevation Vicuña body (~16.1 Ma; Devine and Friedman, 2015) and the main northwest body called the Flamenco intrusion (~15.4 Ma, Devine and Friedman, 2015) are more deeply eroded than the small central Tamberias body, which is inferred to be the youngest intrusion with its cupola domain is preserved.

In Filo del Sol, high-level feldspar-phyric porphyry intrusions are small (less than 20 m across) and have round to north-easterly elongate ellipsoidal shapes. They are all strongly leached and phenocryst content other than feldspar is not distinguishable. U-Pb geochronology shows that they are age-equivalent to the within the younger part of the Cerro

Vicuña - Tamberias suite with one returning an age of ~15.48 Ma (Devine and Friedman, 2015).

Intrusions in the Ventanas area include plagioclase-hornblende-biotite phyric dactic porphyry intrusions and fine-grained equigranular tonalite. There is some evidence of brecciation along the margin of the body with porphyry clasts in an intrusive cement. U-Pb zircon geochronology on one of these bodies returned an age of ~16.4 Ma, equivalent to the older intrusions around Cerro Vicuña.

The only place where Miocene volcanic rocks have been mapped in the Filo del Sol area is along the Filo trend to the northwest of North Filo. Unaltered biotite crystal tuff overlies clastic rocks affected by the Filo del Sol hydrothermal system. This biotite-phyric tuff is interpreted to be the remnant of Late Miocene volcanism that capped the Filo system.

Hydrothermal and phreatic breccias occur in several parts of the Filo system. They are most abundant in the area of North Filo, where alunite-quartz-cemented hydrothermal and phreatic breccias occur over an area 2 km long by 1 km wide that is roughly centred at the northern end of Filo ridge.

The breccias include several varieties. They are divided based on clast content (polymictic vs. monomictic), clast-shape, matrix material, and types of mineral cement and include:

1. Polymictic, subrounded- to angular-clast, matrix-supported (with a >50% milled rock matrix), alunite-cemented breccias.
2. Monomictic (to weakly polymictic), jigsaw-fit to rotated-clast, alunite-cemented breccias.
3. Polymictic, rounded- to subangular-clast, clast- and matrix-supported breccias with a specular hematite + quartz ± tourmaline(?) cement.
4. Phreatic breccias.

The first two types have a close spatial relationship, occurring within the same area with phreatic breccias generally at higher elevations. The specular hematite-cemented breccias are not as common in the Filo area, but are mapped on the east of Filo del Sol at lower elevations.

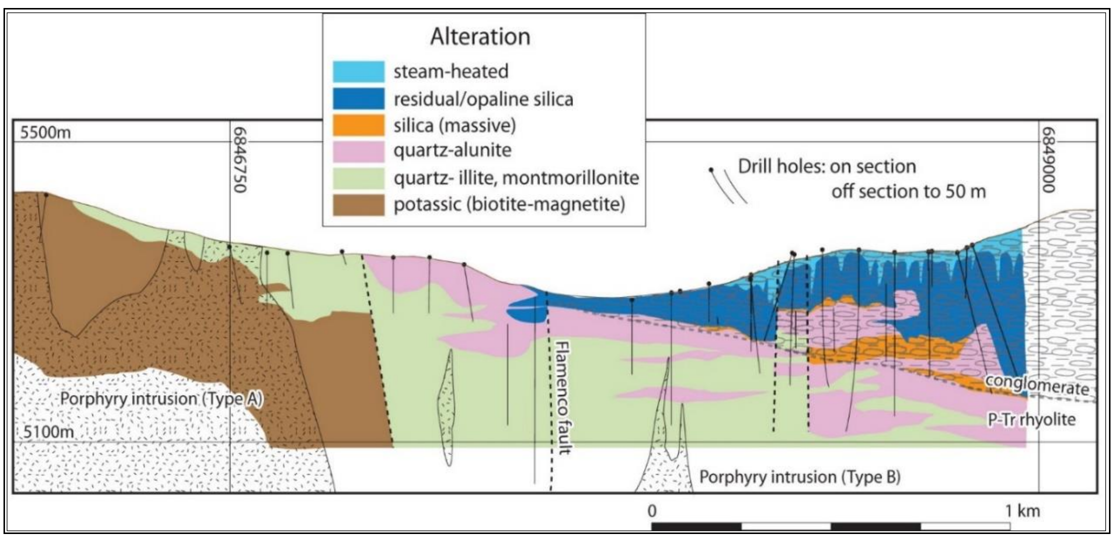
7.2.2 Alteration

The Filo del Sol property is characterized by a large hydrothermal alteration zone extending for approximately 18 km^2, from south of Cerro Vicuña to the Maranceles and Potro areas in the north (Figure 7-5). The alteration types and zonation across the area allow for a series of porphyry intrusive centres with associated higher-level high-sulphidation epithermal systems that have been structurally-offset and exposed to different depths. Alteration in the south around Cerro Vicuña-Tamberias includes both remnant potassic porphyry-style alteration, with overprinting epithermal alteration; while Filo del Sol up to Maranceles presents high-sulphidation epithermal mineralogy. The system at Ventanas has not yet been extensively evaluated but appears to also have both porphyry and epithermal features.

Within the epithermal domain from Filo del Sol to Maranceles, alteration mineral spectra allow for recognition of relative temperature gradients across the area, and several zones of relatively high temperatures characterized by quartz + Na-alunite are mapped (Heberlein and Heberlein, 2015). These areas of higher temperatures may represent up-flow zones within the overall epithermal system. Outward from these quartz + Na-alunite domains alteration changes progressively, although with overprinting assemblages, through K-alunite, high-crystallinity illite, illite, and dickite. Pyrophyllite is noted only locally within the more central part of the system. The distribution of alteration is strongly controlled by pre-mineral faults, both a northeast-trending set, and the north-trending Frontera fault on the west side of Filo del Sol. Alteration extends in a corridor to the northeast, from Filo del Sol towards Maranceles, covering a distance of over 3 km with various zones of quartz-alunite alteration and relatively high alteration temperatures. Quartz-enargite veins are mapped at the northerly end of this trend in the Maranceles area.

Filo del Sol

In the immediate Filo del Sol deposit area, alteration is consistent with a high-sulphidation epithermal system. The deposit occurs within a lithocap domain formed to the south of a possible up-flow zone mapped through alteration mineralogy. A shallowly west- and north-dipping quartz-alunite alteration domain is controlled, at least in-part, by the basal contact of the conglomeratic rocks. Below the quartz-alunite zone, alteration changes rapidly to argillic assemblages that include illite and downwards into montmorillonite. Immediately overlying the quartz-alunite zone are alteration domains resulting from fluid interactions at a receding paleo-water surface, including a domain of massive silicification, followed upwards generally by residual silica with several perched lenses of opaline silica, and capped by steam-heated alteration formed in the vadose zone above the progressively descending paleo-water surface. The distribution of these domains is shown in Figure 7-5 and Figure 7-6. Locally overprinting the quartz-alunite alteration, particularly in the northern part of Filo, is a sulphate-dominant assemblage consisting primarily of gypsum with associated pyrite, although other Fe-sulphates reported in the Filo deposit area (Heberlein, 2015) may also be included.



Source: Sanguinetti, 2014

Figure 7-6: N-S Vertical Section (looking west) 435100E Showing Alteration Zonation

Tamberias

In the Tamberias area the early, more deeply eroded porphyry intrusions and their host rock are potassically-altered with fine-grained biotite and disseminated magnetite (now hematite). Biotite alteration with associated secondary copper mineralization at surface is particularly well-developed within the microdiorite dyke host rock that transects the area.

Potassic alteration is overprinted by kaolinite alteration spatially associated with the latest, more shallowly eroded 'Tamberias' intrusion. A distinctive bright green clay, possibly kaolinite, occurs within 100 m of the porphyry intrusion in veins and feldspar-phenocryst replacement. Quartz veining and a quartz-cemented intrusive breccia with surface values up to 2.3 g/t Au are associated with this domain.

Alunite in replaced feldspar phenocrysts is common throughout the Tamberias area and is inferred to relate to the lower parts of the now mostly eroded Vicuña epithermal system that is preserved as a leached cap at the top of Cerro Vicuña and on the ridge extending south from Tamberias. This epithermal system is preserved topographically higher than the Filo del Sol part of the system and is interpreted to represent an older, higher-level, epithermal domain within the overall magmatic-hydrothermal system.

7.2.3 Structure

Structures within the Filo del Sol area are highly reactivated both pre- and post-mineral. The 'thick-skinned' style of Andean deformation (Godoy and Davidson, 1976; Moscoso and Mpodozis, 1988) involves the arc basement and inherits older features of basement structure, as well as reactivating extensional faults formed in the Jurassic to Eocene. These reactivated extensional faults are mapped in this region as north- to northeast trending high-angle reverse faults.

Structures in the area have focused the ascent of magmatic bodies and also the emplacement of hydrothermal alteration and mineralization systems, as is the case in Filo del Sol. The N-S trending Frontera fault that is interpreted to control the western side of Filo del Sol may be a pre-mineral reverse fault that controlled the later flow of hydrothermal fluids within the Filo system. (Figure 7-5). The offset continuation of the Frontera fault is inferred to the north of Filo where it is covered by post-mineral Late Miocene volcanic rocks.

Northeast-oriented structures also appear to have some syn-mineral control on the hydrothermal system, and the structural regime during development of the Filo system may have included some extensional movement across these faults. The interpreted high-temperature alteration mineral up-flow zone in North Filo coincides with an intersection of the Frontera fault with this northeast-trending structural zone. It is a focus for the some of the most well-developed hydrothermal and phreatic quartz-alunite-cemented breccias, north-east-trending porphyry intrusions, and pervasive quartz-alunite alteration of the host conglomeratic rocks (the gold breccia exploration target – see Section 24.1 for additional discussion of exploration potential). This interpreted centre remains largely undrilled.

Early ideas for the structural evolution within the system required significant motion across the Flamenco to account for the southern porphyry domain and the northern Filo domain being juxtaposed at surface. However, while some syn-mineral structural movement and fluid control within the system is evident, the juxtaposition may also be achieved through telescoping of a younger epithermal system (Filo del Sol) over the slightly older Vicuña porphyry system. Recent geochronology is helping to define the degree of structural displacement within the system.

7.3 Deposit Description

The Filo del Sol deposit encompasses both the Filo del Sol high-sulphidation epithermal system and part of the Tamberias area which is interpreted as a fault-offset part of the same hydrothermal system. The deposit occurs within a porphyry Cu-Au and high-sulphidation epithermal Cu-Au-Ag system that formed during rapid uplift and erosion in the Middle

Miocene. Porphyry intrusions and associated Cu-Au mineralization that are associated with an early, and largely eroded, epithermal system have been overprinted by at least one subsequent stage of porphyry intrusion and epithermal system development. The Filo del Sol system is largely associated with this second, later part of the progressive development of the system.

At Filo del Sol, mineralization is hosted within a high-sulphidation lithocap domain that was formed within conglomeratic host rocks above a regionally-significant unconformity. Steam heated and residual silica alteration along the ridge at Filo mark the uppermost part of the system, with gold mineralized domains formed at a receding paleowater table. Alteration extends downward to massive silica, quartz-alunite and argillic alteration. The basal conglomerate contact controls an important high-grade silver horizon (M zone) within the lithocap domain that has a distinctive metal signature (Ag, Sb, Cu) and forms a key part of the deposit. The conglomerate contact is slightly northwest-dipping as is the M zone formed along it. The top of the M zone is irregular but generally coincides with the lowest extent of the silica alteration. The western side of the Filo section of the deposit is bound by the pre-mineral Frontera fault, a high-angle structure that may have controlled fluid upflow within the system. IP geophysics clearly defines this western boundary. To the north of the Filo section of the deposit, post-mineral faults complicate the geology and the extension of the M zone becomes more irregular, although the deposit remains open in this direction.

Leaching of the uppermost parts of the system has resulted in the development of a gold-only oxide zone (AuOx) that is underlain by supergene Cu enrichment in an oxide zone composed largely of Cu-sulphate minerals that form a bright blue blanket across the surface and extend to depth (CuAuOx). The intersection of the upper leached zone and the M zone forms the thickest, highest-grade copper mineralization in the deposit. The M high-grade silver zone marks the base of the highest soluble copper grades, but hypogene copper and gold sulphide mineralization continues to depth and has not been extensively drilled. Recent work indicates that the presence of blind porphyry intrusions in the hypogene zone correspond to areas of higher copper and gold grades.

The Tamberias section of the deposit is offset from the Filo section across the Flamenco fault. In Tamberias, 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 and are largely blind to the surface, and are associated with copper sulphide mineralization and elevated gold values. The western side of Tamberias hosts epithermal alteration that overprints the earlier potassic domain; this is interpreted to be the lateral continuation of the Filo system, with gold, silver and antimony mineralization. Alteration in the epithermal zone is largely composed of residual silica which corresponds to a leached domain with no copper. No high-grade silver domain occurs in Tamberias; the leached zone is underlain by an oxide zone with Cu-sulphates, that progresses down to a copper-gold hypogene sulphide domain that has only been drill tested in only a very limited area.

7.3.1 Mineral Zoning

Water table fluctuations and the extremely arid conditions prevalent in the area, coupled with marked tectonism and faulting at regional and local scales, have generated a unique weathering profile with the presence of clearly identified zones based on the presence of sulphides vs. oxides as well as metal content (Figure 7-7).

The leached zone (LIX) is represented by vuggy and highly-fractured residual quartz. This zone has a close relationship with the steam heated and residual silica alteration within the epithermal lithocap that has been enhanced by surface weathering and supergene enrichment. Well-developed leaching occurs in the northern part of the deposit with thicknesses exceeding 150 m.

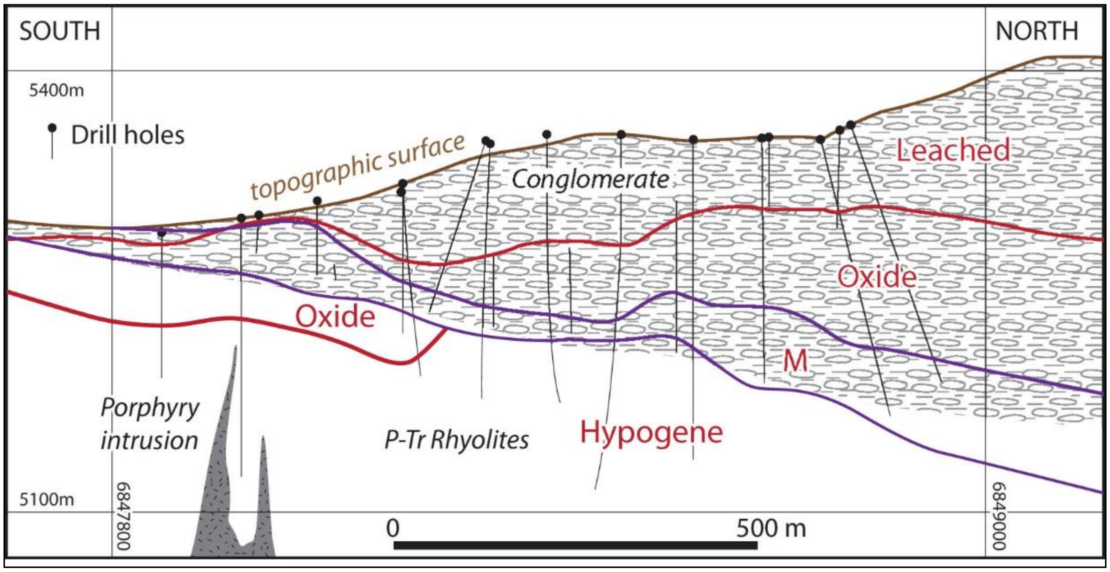
A significant gold oxide zone (AuOx) is hosted within this leached material. This zone is shown in Table 14-7 and is characterized by drill intersections in holes VRC097 (84 m @ 1.36 g/t Au) and VRC099 (78 m @ 1.02 g/t Au).

Immediately below the leached zone, there is an oxide zone (OX) characterized by the presence of Fe, Fe-Cu and Cu oxides and hydroxides. The thickness of the oxide zone ranges from 40 m to 300 m.

The oxide zone hosts the important soluble copper mineralization comprising hydrated sulphate minerals (chalcanthite, copiapite, cuprocopiapite) (CuAuOx Zone). This zone is shown in Table 14-18 and 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. All precipitation in the area is in the form of snow, which, once accumulated, is dissipated by a combination of sublimation and melting. The melting, active only during the short spring and summer season, provides the water required for the sulphide oxidation to proceed. If more water were to be available, the sulphates would dissolve and their copper content would descend in solution to produce chalcocite enrichment at the top of the underlying sulphide zone.

The M zone corresponds to the high-grade silver zone focused along the basal conglomerate contact. This is an important mineral zone within the deposit and is described further below and shown in Table 14-20. It is overprinted by the oxide zone.

The hypogene zone (HIPO) is characterized by the presence of sulphides and the absence of oxide minerals. Three different zones have been logged based on sulphide mineralogy: enargite-pyrite (Zone A); chalcocite-pyrite (Zone B) and chalcopyrite-pyrite (Zone C). Even though it was possible to identify the different hypogene zones in drill cuttings, the interpretation of sections proved challenging due to the irregular nature of the zones, especially hypogene Zones A and B. The thickness of the hypogene zone is unknown as all holes drilled to date terminate within it, however a minimum thickness at Filo del Sol between 200 m to 300 m is apparent. This zone is shown in Table 14-21. In the Tamberias area, the porphyry-related hypogene sulphide mineralization has not yet been extensively explored.



Source: Devine, 2017

Figure 7-7: N-S Vertical Section 435100E (view west) Showing the Mineral Zonation Profile

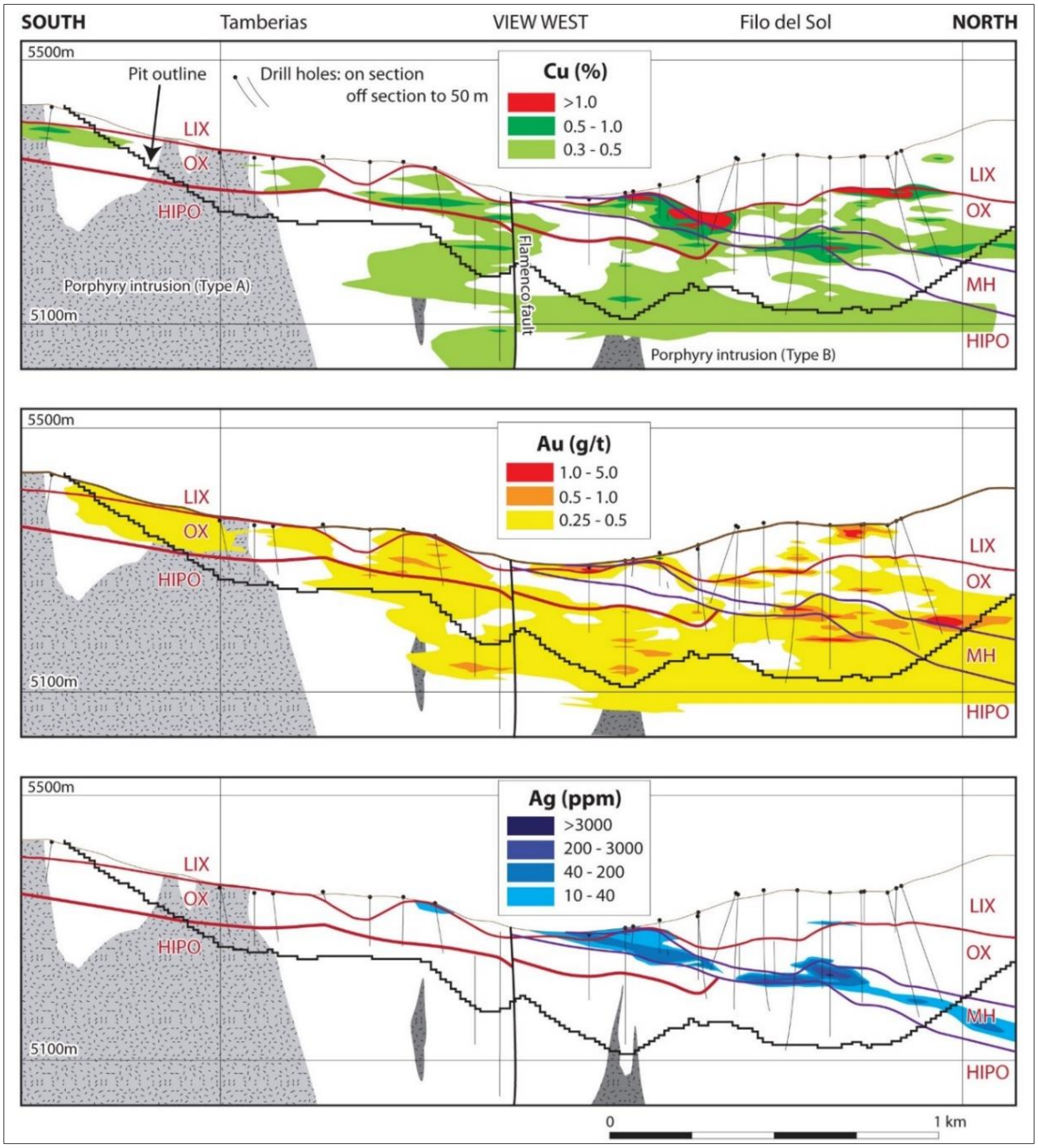
7.3.2 Metal Distribution

The distribution of copper, silver and gold reflects several types of hypogene hydrothermal mineralization and at least some supergene remobilization, resulting in a relatively complex zonation. The highest copper grades correspond to the soluble copper in the oxide zone. The distribution of copper values can be seen in Figure 7-8, Cu distribution includes both sulphide and oxide mineralization.

In the hypogene zone, copper values tend to decrease, averaging 0.28 % Cu, with maximum values to 2.1% Cu.

The distribution of gold is more irregular (Figure 7-8) and it occurs throughout all zones of the deposit. Maximum values reach 21.7 g/t over a 2 m RC sample length. The gold zones coincide, at least in part, with domains of massive and opaline silica that formed at the paleo water surface during progressive recession of the paleo water table. There is a general correlation of some of the better gold grades in Filo and Tamberias with the lower section of logged silica alteration within the epithermal system. Some values may also occur within steep structures; however, these structures have been difficult to identify conclusively in vertical RC drill holes. Gold values also occur associated with copper in the hypogene sulphide zone.

Silver mineralization occurs within a relatively restricted domain at Filo del Sol between 10 m and 50 m thick that is inferred to correspond to the base of the clastic conglomerate unit. It occurs with the mineral zone defined as ‘M’. Its origin has debated alternately as supergene or hypogene, as described below in Section 7.3.3. Maximum values above 3000 g/t Au over a 2 m sample length are reported in intercepts of the M zone. A lower grade silver domain has also been defined at Tamberias, with similar metal associations (including antimony) as at Filo del Sol, in an area that is being explored as the potential offset lateral equivalent to the Filo epithermal domain.



Source: Devine, 2017

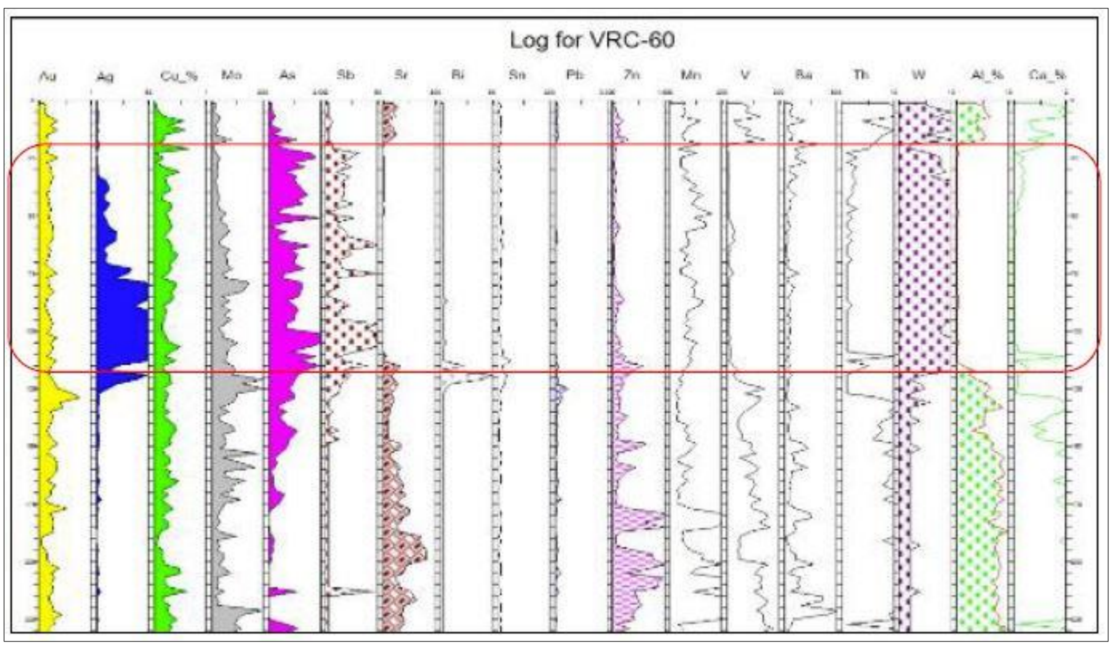
Figure 7-8: Section 435100 Along Filo del Sol and Tamberias, View to the West

7.3.3 High-grade Silver Zone (M Zone)

A high-grade silver zone is a key part of the Filo del Sol deposit, occurring as a shallowly north (20°) and west-dipping (10° to 15°) zone 20 m to 50 m thick and extending at least 1,200 m N-S and 400 m to 600 m E-W. It includes gold mineralization, generally increasing from south to north (Figure 7-8). It usually appears in drill cuttings as unconsolidated grayish to black sandy mud, commonly with associated soluble copper mineralization as Cu-sulphates. Silver mineralization in this zone is mostly composed of chlorargyrite (AgCl) and Ag and Cu sulfosalts 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) (Figure

7-9). The mineral resource for this zone is shown in Table 14-19 and it remains open to the north and east. It is truncated to the south by the Flamenco fault (approximately coincident with its surface outcrop) and to the west by the NS-trending Frontera fault.

Two working models for the origin of this zone have been explored. Its general spatial association with the base of the mixed zone, particularly in the northern part of the deposit, led to the suggestion that it is a supergene enrichment zone resulting from extreme leaching of the upper parts of the high-sulphidation system. However, several observations now suggest that it is instead hypogene in origin, the result of fluid mixing during outflow and oxidization of hydrothermal fluids interacting with descending acid-sulphate fluids. Its attitude and orientation correlate with the mapped and inferred dip of the base of the conglomerate host rock unit, which is being explored as a fluid conduit that focused out-flow of hydrothermal fluids from the northern part of the system (Sanguinetti, pers.comm., 2014; Devine et al., 2015). The coincidence of the upper (southern) part of the high-grade silver zone and the mixed and oxide zones descending from surface have generated a mineral assemblage that may have both hypogene (silver minerals) as well as supergene (Cu-sulphates) features.



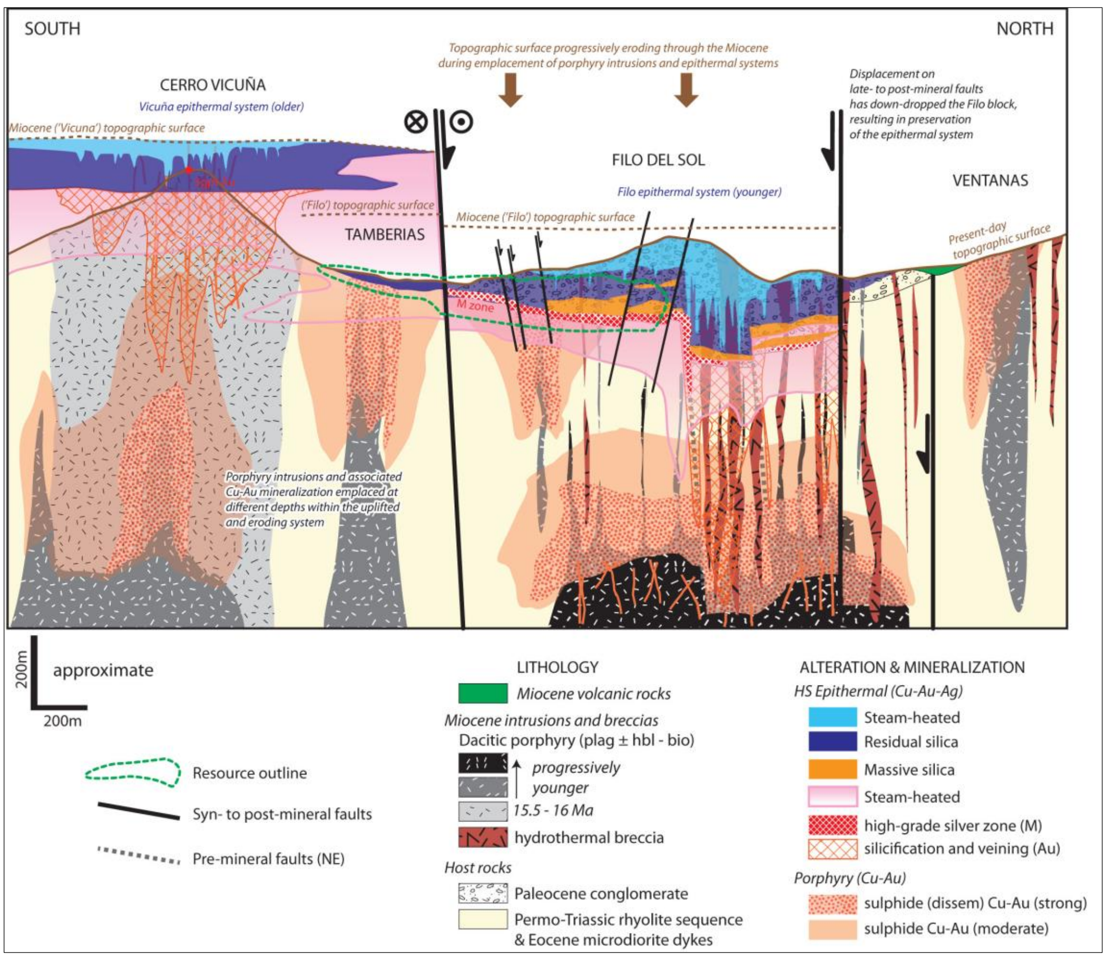
Source: Sanguinetti, 2014

Figure 7-9: Geochemical Profile of Hole VRC-60 Showing Distinct Signature of the Silver zone

8 Deposit Types

Mineralization in the Filo del Sol area shows affinities with both porphyry copper-gold-molybdenum and high-sulphidation gold-silver epithermal systems. The deposit defined by the mineral resource is best classified as epithermal in the north (Filo del Sol) and copper-gold porphyry plus epithermal in the south (Tamberias). The mineralized system in its entirety is thought to represent a telescoped porphyry – epithermal system, with multiple intrusive and breccia centres, and so combines aspects of both these deposit types. The Flamenco fault is an important structure that separates the exposed porphyry Au-Cu system in the south at Cerro Vicuña - Tamberias, from the relatively down-dropped block to the north with the preserved high-sulphidation system at Filo del Sol.

Figure 8-1 shows a conceptualized diagram that summarizes the proposed and possible relationships between these two areas.



Source: Devine, 2017

Note: Broadly to scale and view to the west

Figure 8-1: Conceptual Cross Section of the Cerro Vicuña to North Filo

8.1 Maricunga Style Cu-Au-Mo Porphyry Systems

The Maricunga Belt of northern Chile contains a number of alteration zones hosting Au-Cu(-Mo) porphyry systems and associated high-sulphidation gold deposits. This is a district-specific specialization of the much larger family of Cu±Au±Mo porphyry systems found throughout the Pacific Rim. Porphyry-related deposit models in the Maricunga belt have been studied and documented in considerable detail over the last few decades (Vila and Sillitoe, 1991, Muntean and Einaudi, 2000 and Sillitoe, 2010).

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 other Maricunga Belt systems 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. Porphyry-style Au-Cu mineralization appears to be located in the southern part of the Filo del Sol system, in a fault block preserving a deeper part of the district-scale system. A similar porphyry Au-Cu target may exist below the Filo del Sol area (

Figure 8-1).

8.2 High-Sulphidation Epithermal

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 at 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 sixteen 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, 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

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, Tamberias 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 carried out in the main mineralized areas. Sampling was much more restricted in area than the talus fine sampling, covering mainly the Filo del Sol and Tamberias 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 and 34 m at 1.75% Cu and 0.52 g/t Au. These samples are all in the Cerro Vicuña area, south of the Flamenco fault.

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 Flamenco 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: Tamberias 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

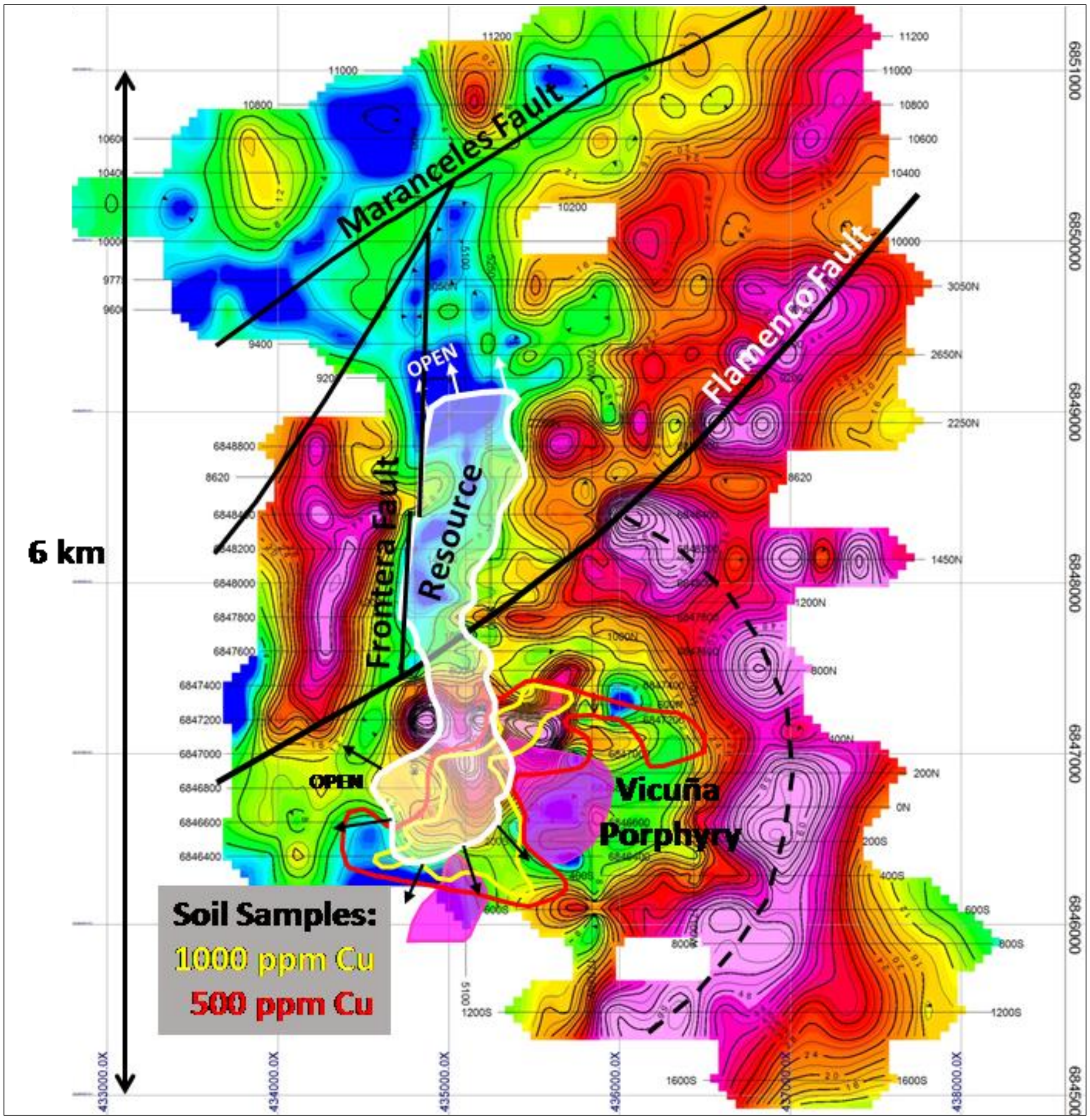
IP geophysical surveys have been very useful in defining geological features at Filo del Sol. Several generations of surveys have been completed, notably in the 1999/2000, 2004/2005, 2005/2006, 2006/2007, 2007/2008, 2011/2012, 2014/2015 and 2015/2016 seasons. Surveys were completed by Zonge Ingenieria y Geofisica (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).

The Filo del Sol resource correlates well with a low-chargeability resistive feature north of the Flamenco fault. This feature extends over a kilometre to the north of the resource, which has not been closed off in this direction. The surrounding phyllic-altered rocks and predominance of disseminated pyrite are characterized by high-chargeability zones.

In east-west geophysical cross-sections, the presence of the Frontera fault and the wedge-shaped epithermal system are apparent. Plan chargeability sections show the main geological features including the epithermal zone, the Flamenco fault and the Cerro Vicuña porphyry system. The latter is highlighted by a circular high-chargeability “ring” surrounding a lower chargeability zone centred on the Cerro Vicuña porphyry (Table 9-1). This ring feature is clearly truncated by the Flamenco fault.

The Maranceles fault correlates well with a north-easterly trending low-chargeability feature that was mapped out by the northern extension to the IP grid completed in the 2015/2016 season.



Source: Quantec, 2012
Note: Plan at 200 m depth

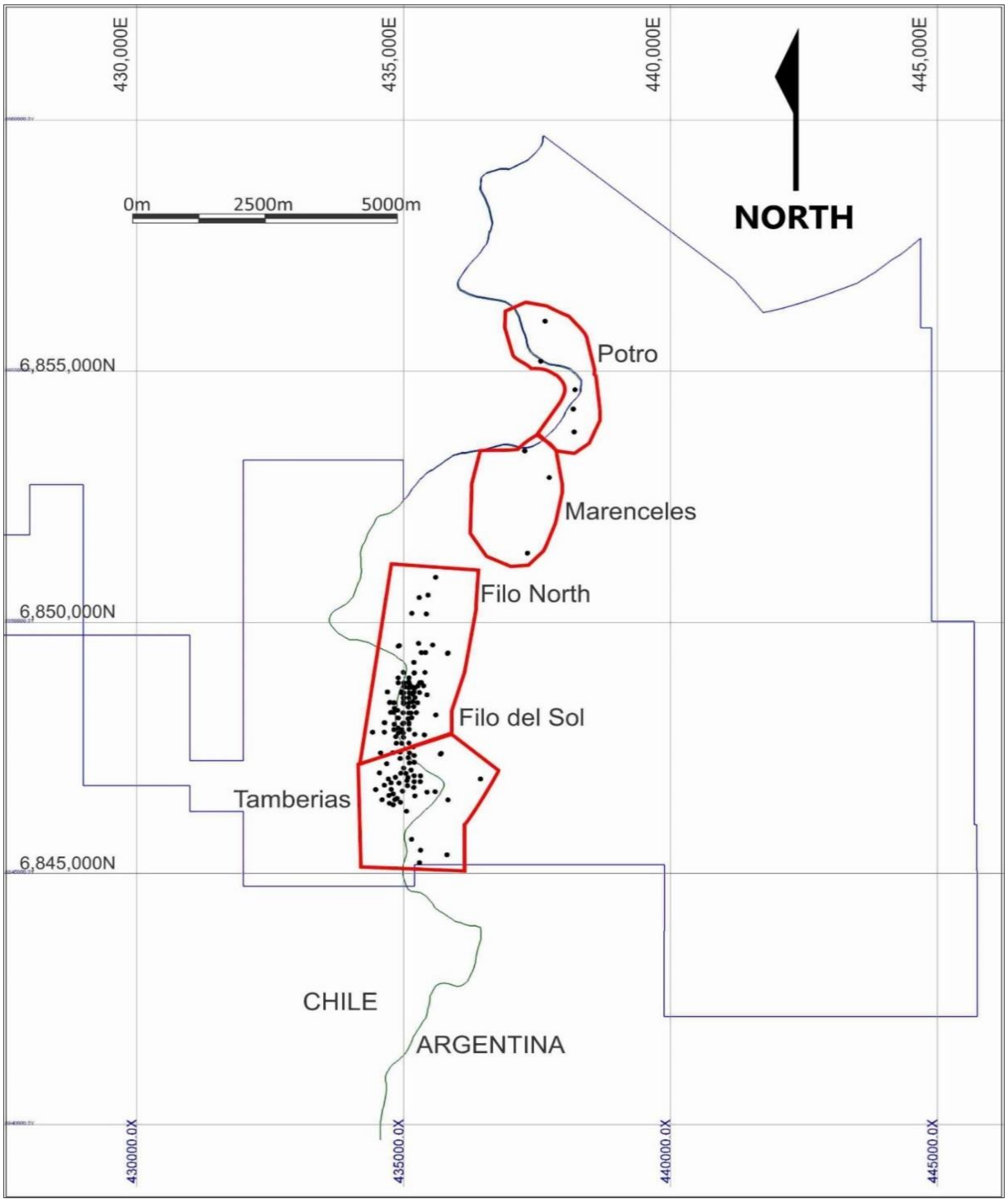
Figure 9-1: Chargeability in relation to Filo Mining Resource and Talus Fine Cu Anomaly

In addition to IP, surface magnetic surveys were completed in 2000/2001, 2004/2005, 2005/2006 and 20072008. The results of these surveys help to define the major structures.

10 Drilling and Trenching

Drilling at Filo del Sol was initiated by Cyprus in 1998/99 and since then a total of 44,457 m of RC drilling in 184 holes and 6,790 m of diamond drilling (DD) in 31 holes has been completed on the property. All of these holes with the exception of 17 RC holes (3,693 m) were drilled in the Filo del Sol deposit.

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



Source: Carmichael, 2017

Figure 10-1: Drill Hole Collar Locations

10.1 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 in particular has been used sparingly due the water-soluble nature of the copper mineralization and to difficulties in completing holes and cost related to lost equipment. Most of the drilling has been done by RC methods due to its lower cost and higher productivity.

An increased emphasis was put on diamond core drilling for the 2017/2018 season in order to better understand the geology and to 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.

10.2 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 about 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 8616 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 88%, with recoveries during the 2017/2018 campaign averaging 92%.

Data analysis shows no correlation between recovery and grade.

10.3 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 Potro were surveyed by hand-held GPS. The drill platforms are easily visible on the orthorectified GEOEYE satellite images and provide good confirmation of the accuracy of the collar surveys.

10.4 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. 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. Azimuths tend to be quite consistent.

10.5 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.

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.

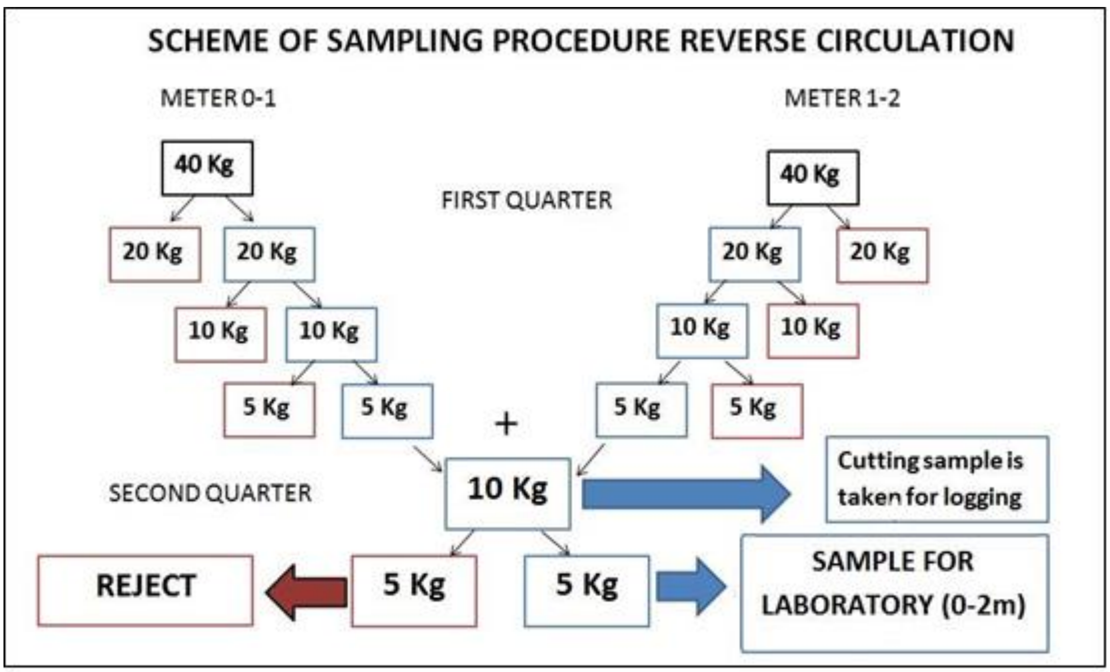
ACME procedures included 35-element four-acid or aqua regia digestion ICP-AES, Au fire assay Atomic Absorption finish and trace Hg by cold vapour/Atomic Absorption.

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.



Source: Charchaflié and Gray, 2014

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

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 was 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. 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. In 2017/2018, only core from the CuAuOx and M zones was split this way, other zones with no soluble copper were cut with a rock saw in order to better preserve the core.

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. 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 2015 samples were delivered directly to ACME's preparation facilities in Copiapó by company personnel, considerably reducing turn-around times.

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 date) or ACME Laboratories Chile (since 2010/2011 up to 2015).

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 latest campaign were stored in vacuum-sealed bags in order 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 33 (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.

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.

Cyprus Drilling, RCV-02 to RCV-17; 1998/99 Program

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 (R^2> 0.81) whereas Cu duplicates display moderate correlation (R^2 = 0.61), a result most likely associated to a single sample pair that assays 1452 and 9668 ppm. These results seem acceptable for all elements.

VRC01 to VRC21; 2000/01 Program

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

VRC25 to VRC55; 2003 to 2008 Programs

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.

VRC56 to VRC79; 2013/14 Program

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 Cut-off)
- 2 Standard 3 (high – over expected Cut-off)
- 2 Field duplicate (second half core)
- 4 Blank (coarse material)
- 2 Preparation duplicate (make second pulp)
- 2 Assay duplicate (second assay)

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 R^2: 0.999 and Au R^2: 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 (R^2> 0.934) between ALS and ACME analyses. No bias between laboratories is observed, and results provided by both companies appear to be similar.

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

The quality control protocol implemented in 2013 was continued in the 2014/2015 season.

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 R^2 >0.991 whereas preparation duplicate's R^2 >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 R^2: 0.993, Au R^2: 0.838 and Ag R^2: 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.

VRC100 to VRC134; 2016/17 Program

The quality control protocol implemented in 2013 was continued in the 2016/2017 season.

Blank material inserted during the 2016/2017 campaign consisted of white quartz fragments obtained from a pegmatite quarry in Valle Fertil, San Juan. The blanks are considered un-mineralized as copper concentration is generally below 20 ppm Cu and Au is commonly below the 0.01 ppm Au detection limit.

Field duplicates were obtained taking a second split of the sample to be analyzed independently. Both preparation and assays duplicates were made by the laboratory and assigned a specific number in the sequence. The preparation duplicate consisted of a second pulp from the original sample whereas the assay duplicate was a subsample made from the original pulp.

In late 2014, three standards (STD01, 02 and 03) were prepared by Filo Mining using selected coarse rejects from Filo del Sol drill holes. Selected coarse rejects were submitted to Vigalab Laboratories in Copiapó for crushing, pulverization, homogenization and splitting. Vigalab produced small envelopes containing 80 g to 90 g of material. Five analytical laboratories located within the region were used to perform a round robin test of results: ACME, ActLabs, Andes Analytical Assay, ALS and Vigalab. Five envelopes of each standard were sent to each of these laboratories. Based on the round robin results, the standards have been assigned averages and accepted ranges. Accepted grades and 2 and 3 standard deviations for each standard are shown in Table 11-1:

Table 11-1: 2014 Standard Reference Material Specifications

	Cu Avg	Cu 2 Std	Cu 3 Std	Au Avg	Au 2 Std	Au 3 Std	Ag Avg	Ag 2 Std	Ag 3 Std
	%			**ppm**			**ppm**		
STD1	**0.238**	0.023	0.034	**0.210**	0.031	0.046	**2.0**	2.201	3.302
STD2	**0.543**	0.049	0.074	**0.518**	0.045	0.067	**148.9**	8.977	13.465
STD3	**0.925**	0.063	0.095	**0.607**	0.061	0.092	**45.2**	5.268	7.901

Quality control of samples submitted during the 2016/2017 campaign is discussed below.

QC Rationale

The following logic was used to define failed QC samples.

- Lab results that were more than the "Warning Limit" for Cu or Au are failures. The "Warning Limit" was defined as 5x the average grade of the quartz used as blank (WL = 150 ppm Cu and 0.05 ppm Au).
- Preparation or assay duplicates with a relative difference of ± 10% in Cu and Au or absolute differences + 0.05% Cu and + 0.1 ppm Au.
- Standards for Cu and Au beyond the ± 3 std limits.

Blank Results

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

Duplicate Results

No failures were recognized in preparation and assay duplicates in 109 pairs of samples. Assay duplicates have Cu, Au and Ag R^2 >0.981 whereas preparation duplicate's R^2 >0.985.

Field duplicates have good correlation factors (Cu R^2: 0.993, Au R^2: 0.838 and Ag R^2: 0.980) and absolute differences expected in natural systems.

Standard Results

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.

More than 68% of current RC and DDH dataset had a rigorous follow up with blanks, standards and laboratory duplicates. Another 10% has been checked with a second lab but does not have blank and standard controls. The remaining 22% of the dataset has only been verified (satisfactorily) with duplicates. No sample appears to be misplaced or intentionally deleted from the database. In our opinion, the current drillhole dataset for the Filo del Sol Project is consistent and has adequate quality to be used for Indicated resource estimation.

12 Data Verification

J. Gray visited the core logging facility 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, including review and discussion with Company personnel and extensive surface geological mapping and drill core and chip logging.

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.

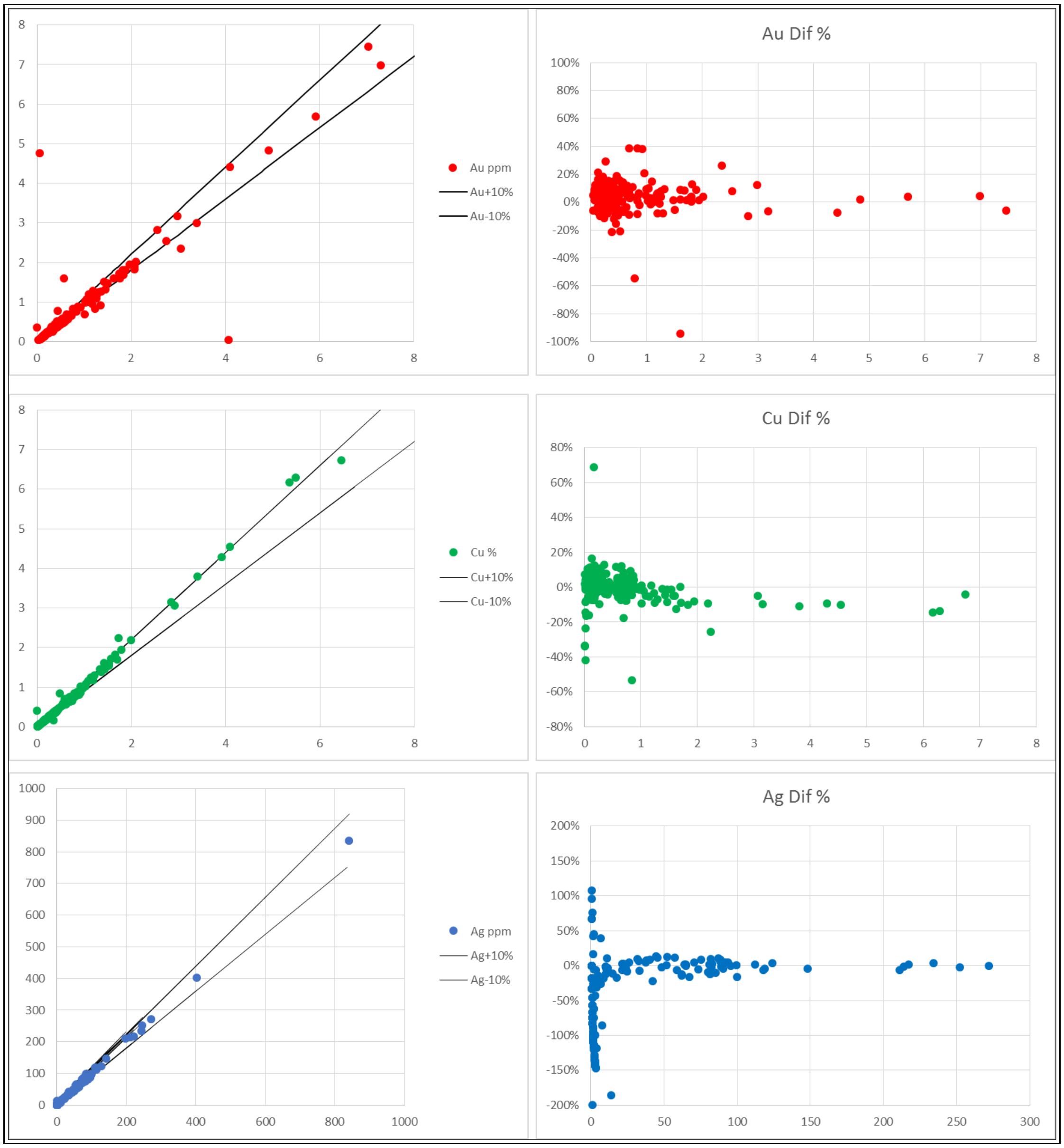
In the opinion of the QPs, the data contained in this report is adequate to estimate an Indicated Mineral Resource.

12.1 Verification of Au, Ag, and Cu Analyses

A total of 206 pulps from RC holes drilled between 2000 and 2008 were selected to perform an independent geochemical study 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 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 12-1 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.



Source: Charchaflié, 2014

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

A visit to the Copiapó office and support facilities was carried out by J. Gray, 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 12-1 results agreed closely with the original values.

Table 12-1: Results of Six Independent Samples

Hole-ID	From (m)	To (m)	Gold Assay (g/t Au) Orig.	Gold Assay (g/t Au) Indep.	Silver Assay (g/t Ag) Orig.	Silver Assay (g/t Ag) Indep.	Moly Assay (ppm Mo) Orig.	Moly Assay (ppm Mo) Indep.	Arsenic Assay (ppm As) Orig.	Arsenic Assay (ppm As) 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) Orig.**	**Copper Assay (% Cu) Indep.**	**Acid Sol Cu Assay (% Cu) Orig .**	**Acid Sol Cu Assay (% Cu) Indep.**	**CN Sol. Cu Assay (% Cu) Orig.**	**CN Sol. Cu Assay (% Cu) Indep.**	**Resid. Cu Assay (% Cu) Orig.**	**Resid. Cu Assay (% Cu) 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: June 19, 2014 (Gray, 2014)

12.2 Verification of Collar Locations

In the main area of drilling, seven drill hole sites were visited 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 12-2. 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. For present purposes and within the uncertainties of hand-held GPS measurements it is accepted that the drill holes are as represented by the differential GPS data. It should also be noted that most of the drill platforms are visible on the GEOEYE satellite images (0.5 m resolution) acquired by Filo Mining, and plotting of surveyed drill collar locations on the satellite images further confirms the accuracy of the drill hole locations.

Table 12-2: 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 (Tamberias gold oxide (TMB AuOx), Filo del Sol gold oxide (FDS AuOx), Tamberias 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 4 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.

Table 13-1: Novatech 2001 – 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

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.

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 gpt Au; 0.44% Cu	95.1% Cu, 87% Au	0	-
Oxide Gold (AuOx)	0.49 gpt Au; 0.02% Cu	93.2% Au	-	0.67
Silver (M)	0.34 gpt 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 4 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 Tamberias (TMB AuOx)); a copper-gold oxide zone (CuAuOx) (two areas: Filo del Sol (FDS CuAuOx) and Tamberias (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 Tamberias, five RC samples from Filo del Sol and five RC samples from Tamberias were collected. For the CuAuOx zone, two surface samples from Filo del Sol, two surface samples from Tamberias and four RC samples from Filo del Sol were collected. Sample locations are given in Table 13-4 and Table 13-5.

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

Zone	Hole ID	From	To
	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	
Name	**Zone**	**East**	**North**	**East**	**North**
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 characterisation and host geology. Samples were differentiated based both on mineralization type and location (Filo vs

Tamberias), 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 Tamberias;
- Copper-Gold Oxide (CuAuOx) – Filo and Tamberias;
- Silver-rich Mineralization (M-Ag) – Filo only (this mineralization does not exist in Tamberias 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 make up 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 4 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 3 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 4 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 5 drill core samples were collected for FDS AuOx, 11 for FDS CuAuOx and 4 for M-Ag. No drill core was available for Tamberias 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 3 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 13-6: 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

13.2.2 Head Sample Characterization

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

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)
			H_2SO_4	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

Sample Name	Mineralization Type	Element Cu (%)	Sequential Copper (%)			Au (g/t)	Ag (g/t)	Fe (%)	As (%)	Al (%)	Hg (g/t)
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

Sample Name	Mineralization Type	Element Cu (%)	Sequential Copper (%)			Au (g/t)	Ag (g/t)	Fe (%)	As (%)	Al (%)	Hg (g/t)
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

13.2.3 Physical Characterization

Various samples from the Filo del Sol and Tamberias 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 13-4 compare 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

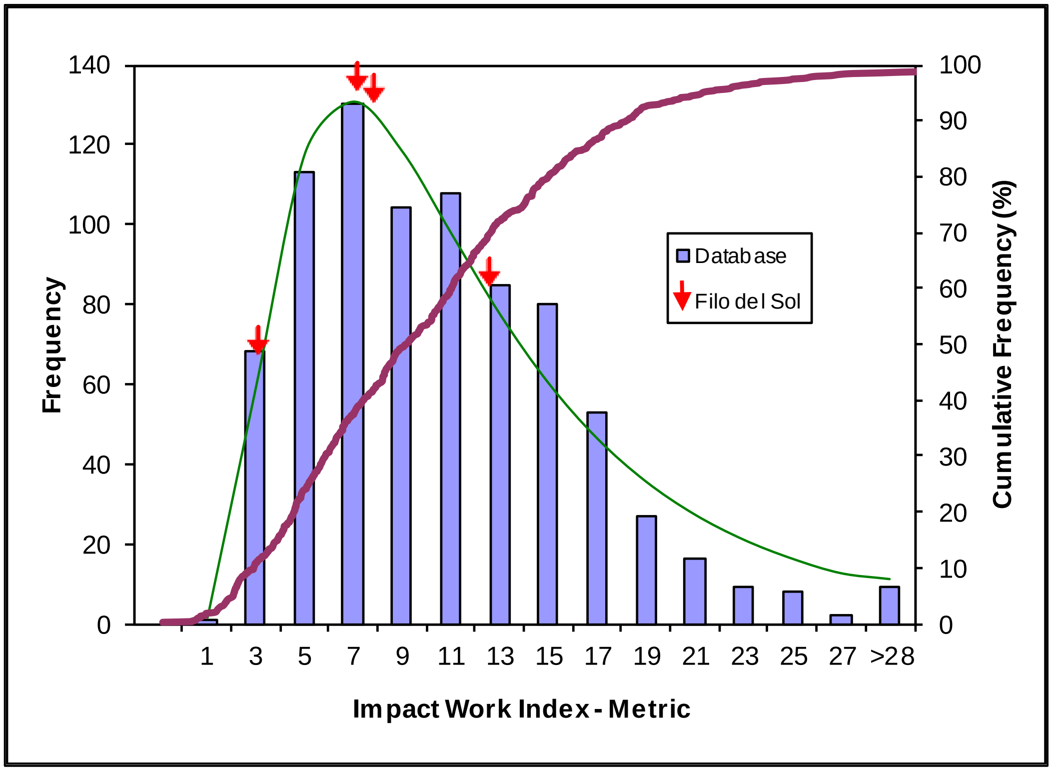


Figure 13-1: Bond Low Energy Impact Test Comparison

Table 13-9: SMC Testing (Summary)

Sample Name	A	b	A x b	Hardness Percentile	T_a[1]	DWI (kWh/m^3)	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 T_a 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

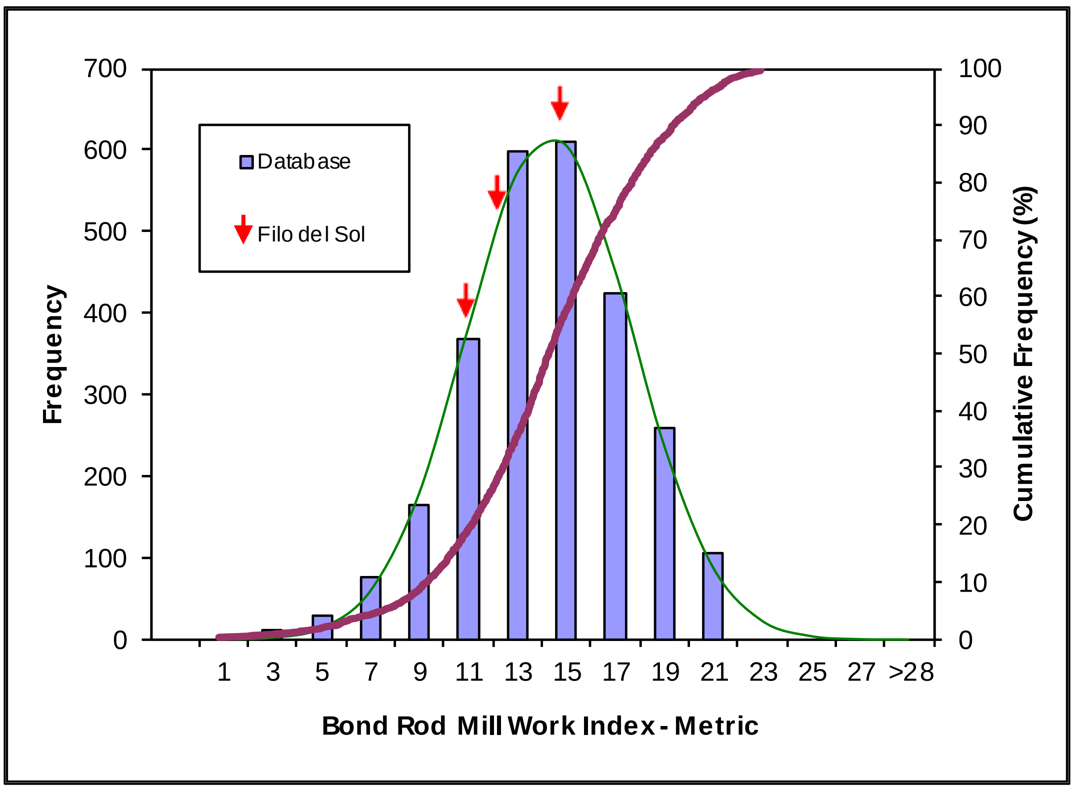


Figure 13-2: Bond Rod Mill Work Index Comparison

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

Sample Name	Mesh of Grind	F_{80} (µm)	P_{80} (µm)	Gram per Revolution	Work Index (kWh/t)	Hardness Percentile
T18 Cu Comp	100	2,195	111	1.12	19.2	89

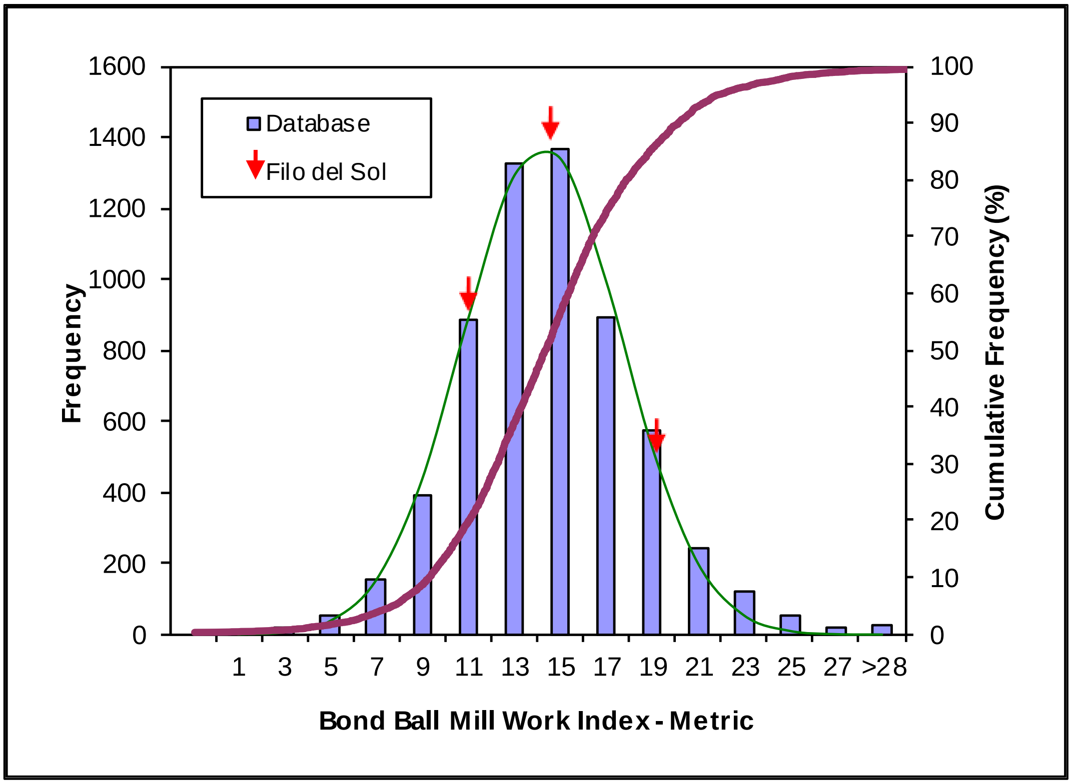


Figure 13-3: Bond Ball Mill Work Index Comparison

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

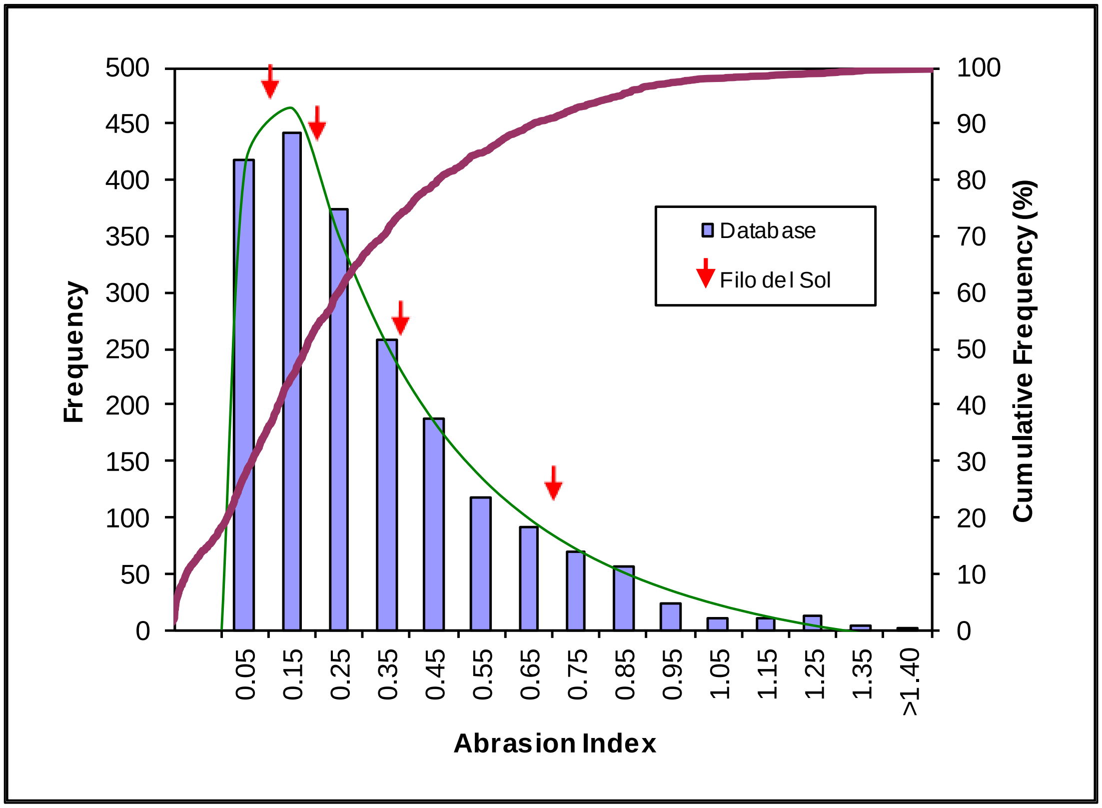


Figure 13-4: Bond Abrasion Index Comparison

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 Tamberias 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 Tamberias deposit over the life of mine.

Table 13-13: Preparation of T18G-Composite Sample

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

Bottle Roll Tests

A series of 6 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 Table 13-14.

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

Year	Test No.	Sample	Head Assay		NaCN	Extraction		Reagent Consumption	
			Au (g/t)	Ag (g/t)	(g/L)	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.

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/hr/m^2. The results are summarized in Table 13-15.

Table 13-15: Tamberias 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.

After 56 days, column 13CN was rested for 4 days (i.e. no solution was applied to it), and the column was then restarted for a few additional weeks, but it did not significantly improve gold extraction.

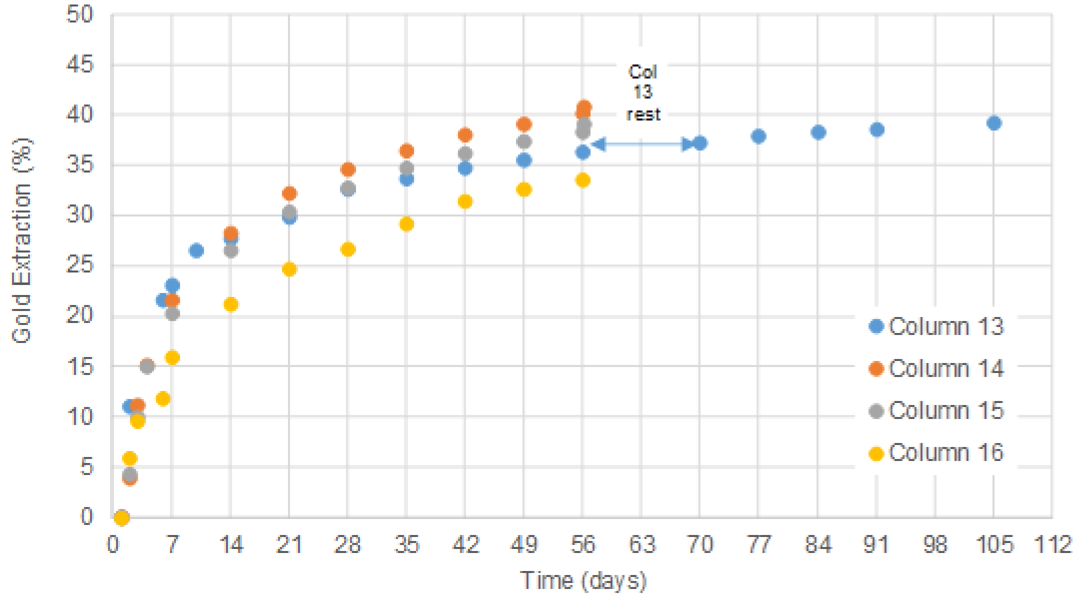


Figure 13-5: Tamberias Gold T18G Comp. - Kinetics of Column Leaching

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

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
	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.

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/hr/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	0^{Δ}	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
$^{\Delta}$ Column stopped due to poor solution flow

The F18G Compe 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 silver.

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 13-6.

They indicate rapid kinetics with extraction nearing completion after only 4 weeks.

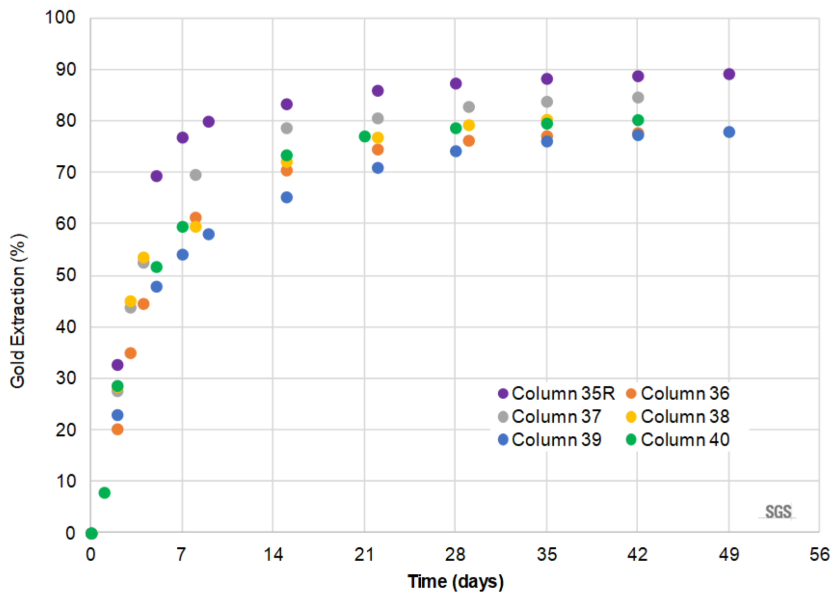


Figure 13-6: Filo Gold Oxide Composite – Kinetics of Column Leaching

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

Two surface sample (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 about 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).

Bottle Roll Tests

In 2018, a total of twelve 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: 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
2017	LC-3	FDS CuAuOx-VRC64	0.33	0.34	1.0	8.2	93.0	88.7	48.2	0	1.4	4.3
2017	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
2017	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
2017	LC-6	FDS CuAuOx-VRC76	0.88	1.51	0.80	19.3	98.8	98.1	46.1	0	0.7	1.8
2017		**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
2018	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
2018	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
2018	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
2018	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
2018	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
2018	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
2018	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
2018	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
2018	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
2018	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
2018	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
2018	**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**
2018	**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.

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/hr/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 13-22below.

Table 13-21: F18 Cu Comp – 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 Comp – 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

Year	Test #	Sample	% Recovery			Reagent Consumption		
	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 H_2SO_4 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 13-7. The kinetics were very rapid, with leach completion (for 1.5 inch crush size) achieved after only 4 weeks.

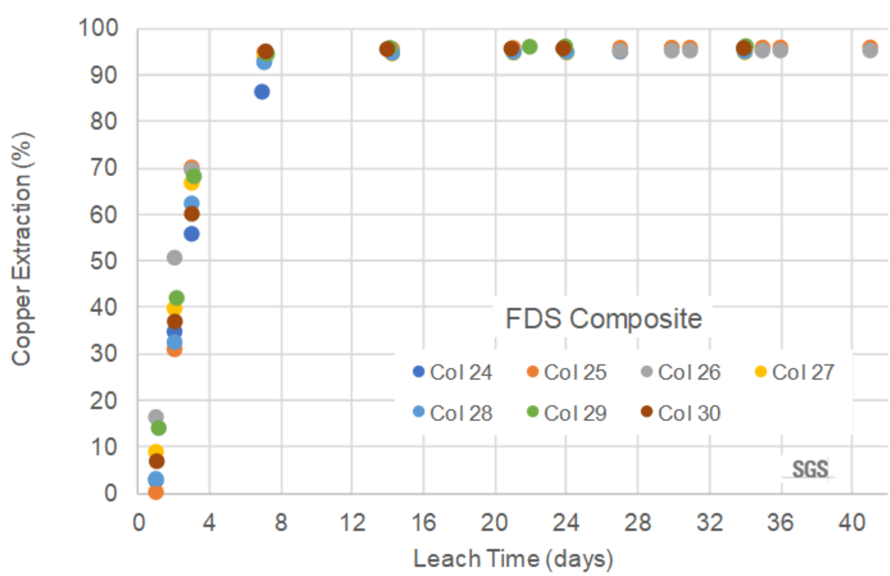


Figure 13-7: Copper Extraction for FDS CuAuOx

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

Leach kinetics for gold and silver are shownin Figure 13.8 overleaf.

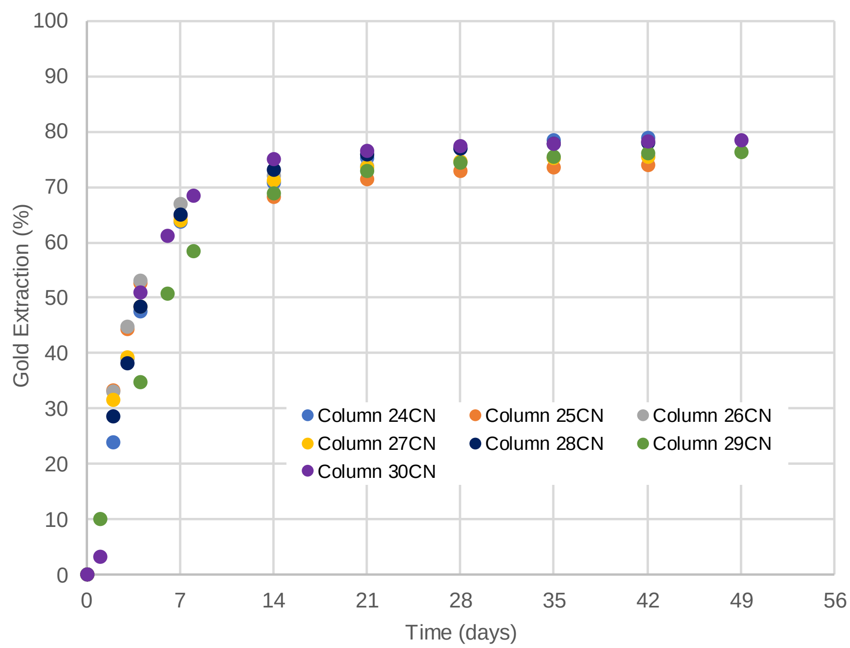


Figure 13-8: Gold Extraction for FDS CuAuOx

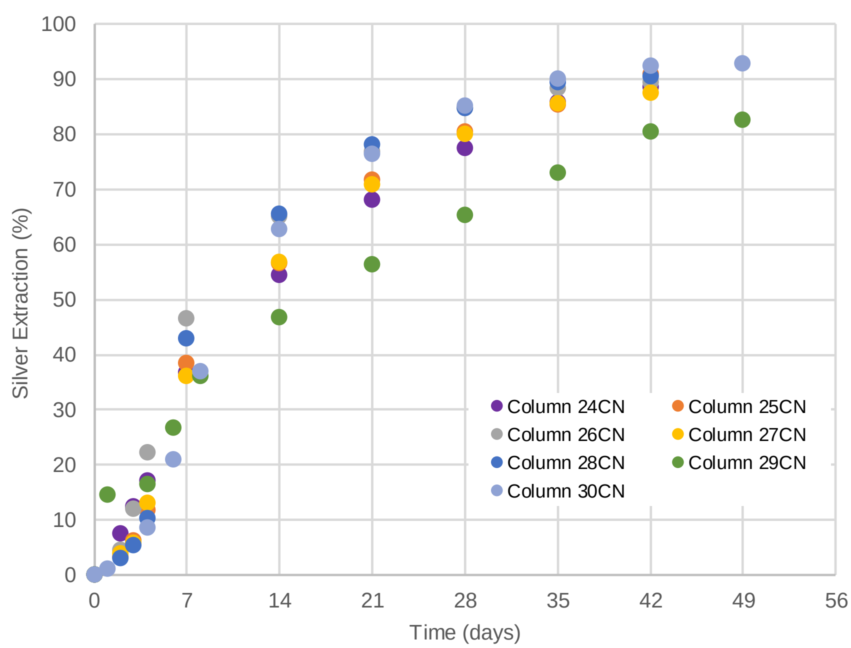


Figure 13-9: Silver Extraction for FDS CuAuOx

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.

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 fourteen 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 Comp 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)
		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
2018	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 Components					16.8	83.6	81.7	69.1	-23.6	5.24	1.29
		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
2018	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.

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; 10L/hr/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 Tamberias Copper Gold Oxide Zone (TMB CuAuOx)

Six surface samples from the TMB CuAuOx zone were selected and Combined to prepare anoverall composite sample, representative of the TMB CuAuOx composition over the life of mine. Table 13-26 below 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 about 75% acid soluble copper.

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, pH ~10.5 (cyanide leach).

Results are summarized in Table 13-27.

Table 13-27: T18 Cu Comp. – 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%.

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/hr/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 Figure 13-10, Figure 13-11 and Figure 13-12 respectively

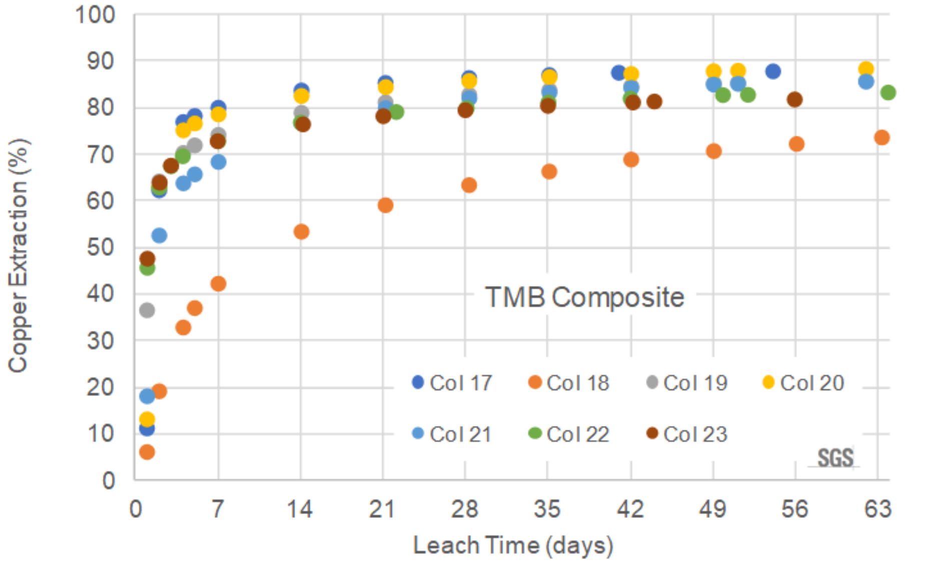


Figure 13-10: Copper Extraction for TMB CuAuOx Composite

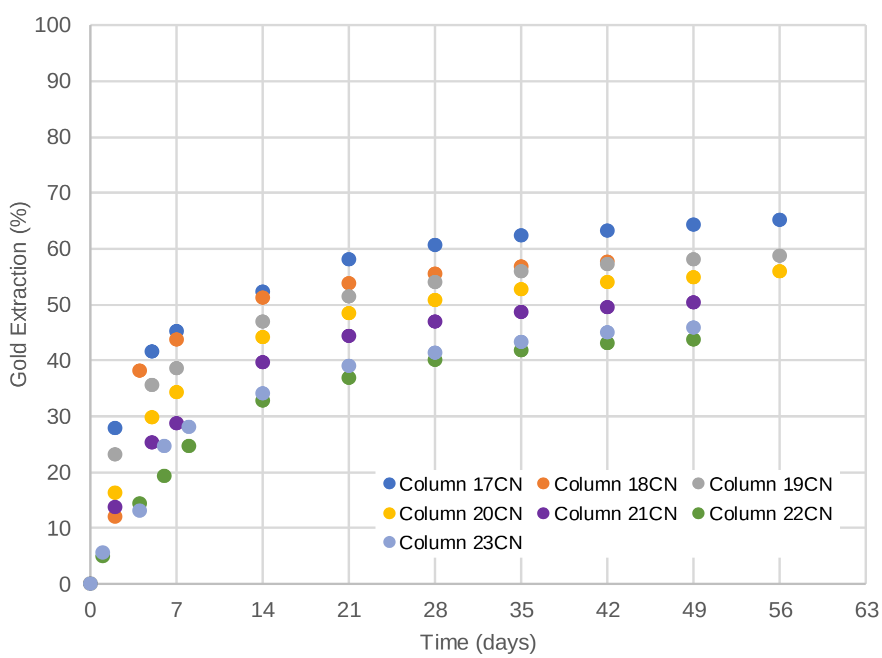


Figure 13-11: Gold Extraction for TMB CuAuOx Composite

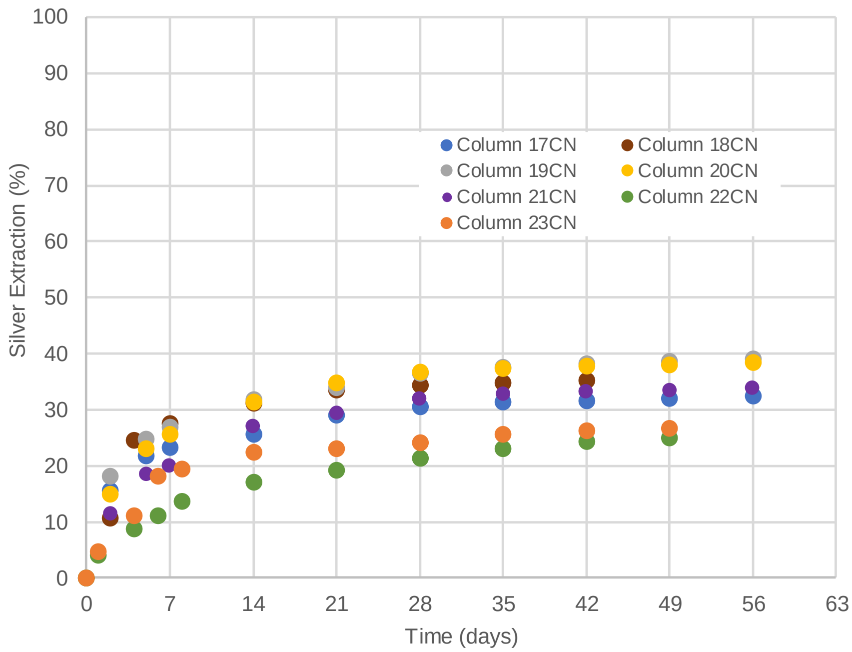


Figure 13-12: Silver Extraction for TMB CuAuOx Composite

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

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 mineralisation type in the Filo del Sol project, as it was understood in 2018.

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 asF18 Cu Comp and F18 M-Ag making up65% of Copper Blend #2 as compared to 35% in Copper Blend #1.

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/hr/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 2 to 3 weeks.

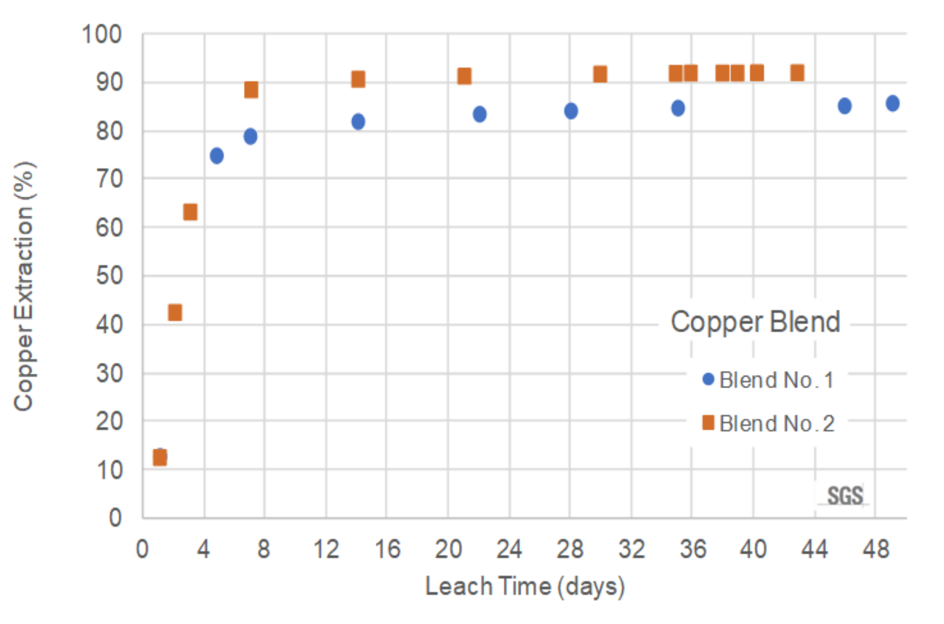


Figure 13-13: Copper Extraction for Copper Blends

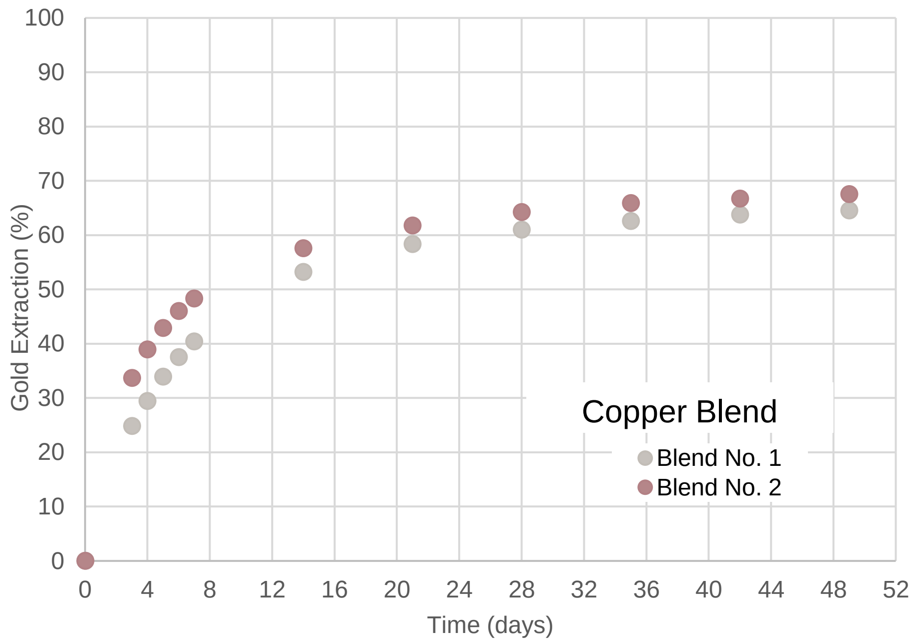


Figure 13-14: Gold Extraction for Copper Blends

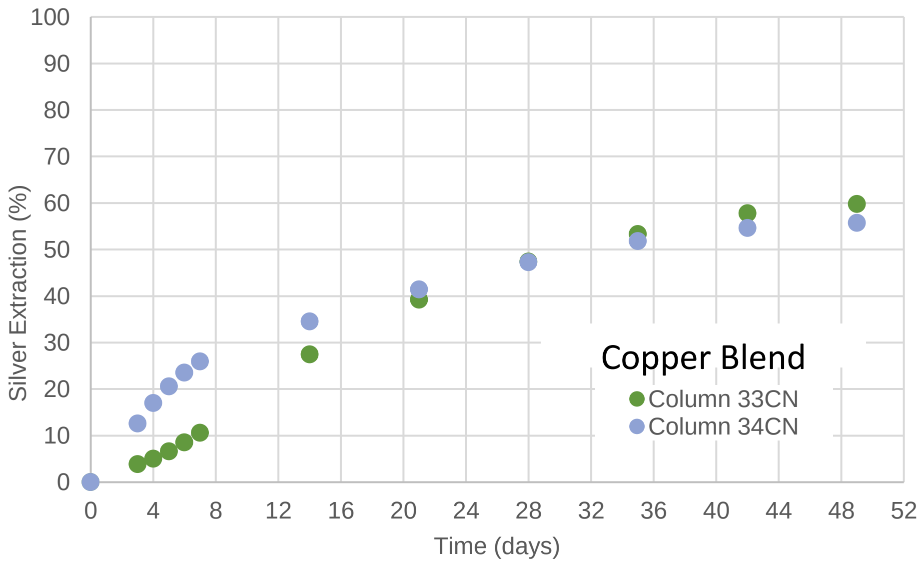


Figure 13-15: Silver Extraction for Copper Blends

13.4 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.

13.4.1 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.

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 (ºC)	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

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 preferreable for Filo del Sol ores.

13.4.2 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 Au Ox.

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.

13.5 Preliminary Evaluaiton 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.

13.5.1 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 Extraciton
			0 g/L Fe^{3+}	10 g/L Fe^{3+}	%
FDS CuAuOx	FSDH022 (106-116)	55 86	46.6 -	- 67.3	+20.7
	F18 CuCN Comp.	58 67	70.9 -	- 71.9	+1.0
	VRC101 (242-252)	19 83	59.3 -	- 73.2	+13.9
FDS M-Ag	F18 M-Ag Comp.	53 66	91.8 -	- 92.0	+0.2
	VRC062 (270-286)	29 84	60.5 -	- 60.4	-0.1
	VRC062 (286-296)	30 85	48.7 -	- 46.6	-2.1
TMB CuAuOx	T18 Cu Comp.	37 65	81.0 -	- 80.5	-0.5
Copper Blend #2	Copper Blend #2	69 71	90.9 -	- 91.8	+0.9

Significantly improved copper extractions (Δ Cu Extraction of 14 to 21%) were obtained for the 2 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.

13.5.2 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, Ag_2S) 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, a 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), 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.

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

*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.

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.

13.5.3 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 (Cu_2S) 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.

Table 13-42: Test Conditions of the SART Tests

Year	Test #	Temp (ºC)	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

*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/m^3)		
		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

Year	Test #	Feed Analysis			Metal Precipitated		CN_{WAD} Regenerated	Precipitate Assay		Reagent Consumption (kg/m^3)		
2018	SART 2	80	28.9	0.08	99.0	~100	>99	51.4*	18.8*	0.05	2.22	4.21

*Values calculated from solution assays (In/Out) and weight of Cu_2S precipitate (too little weight for direct assays).

SART results were excellent with almost complete regeneration of the CN_{WAD} 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.

13.6 Miscellaneous Test Programs

13.6.1 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	g/L Cu	g/L Fe	pH	(ºC)	(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.

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-IsoI	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**

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.

13.6.2 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

*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.

13.6.3 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.

13.6.4 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 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.

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.

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: Tamberias 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 Tamberias 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.

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% (Tamberias) 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).

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.

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 CN_{WAD} 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.

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

13.7 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.

13.7.1 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 8 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 13-16 illustrate 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%.

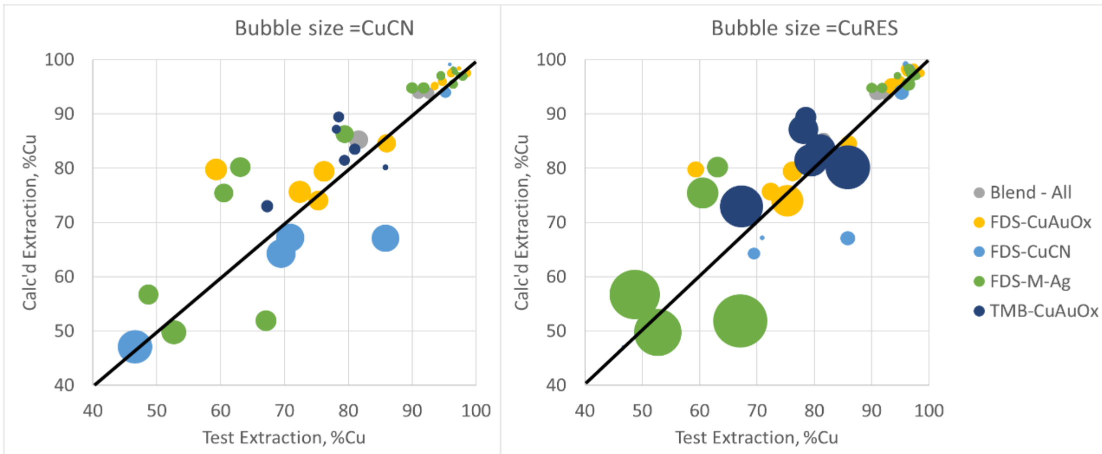


Figure 13-16: FDS and TMB CuAuOx Cu extraction calculated vs BRT result

13.7.2 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 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.

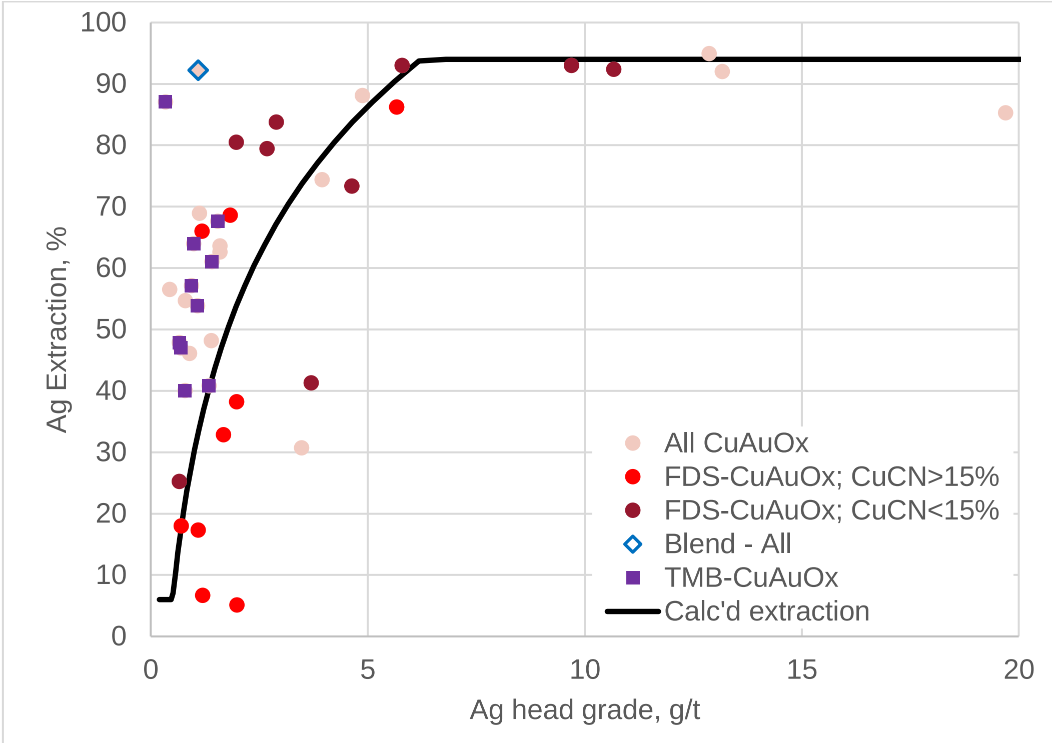


Figure 13-17: FDS CuAuOx Ag extraction equation and BRT result

Figure 13-18 Illustrates the correlation between calculated silver extraction and actual test results.

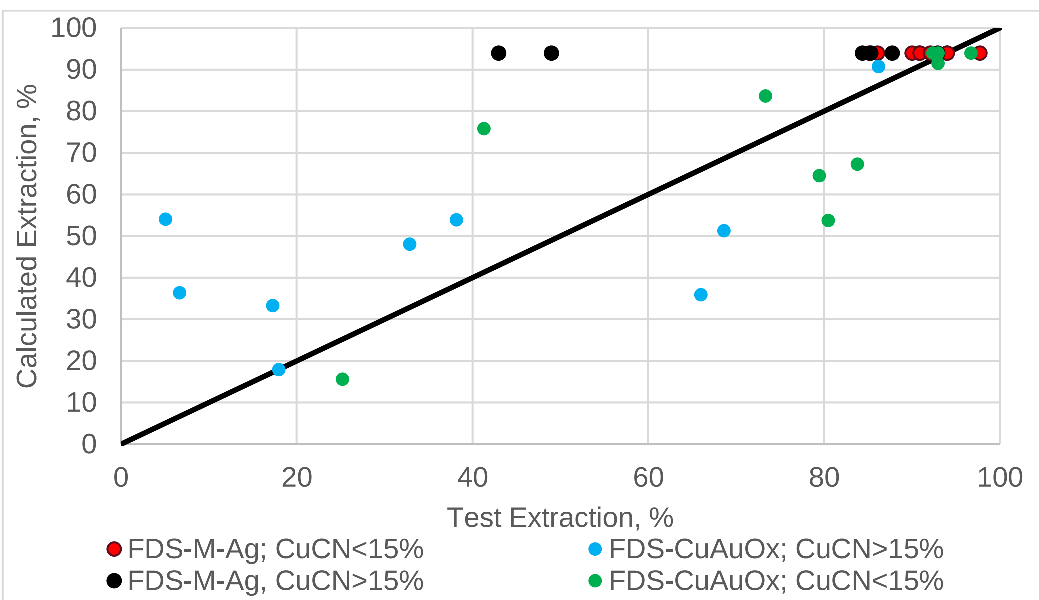


Figure 13-18: FDS CuAuOx Ag extraction calculated vs. BRT result

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

13.7.3 Overall Metal Recovery

Overall metal recovery, from ore to cathode and SART precipitate for copper or dore for gold and silver was calculated on the basis of extractions achieved 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% factor to copper extraction and 3% factor to gold and silver extractions, as summarised in Table 13-54, to determine overall recovery to product. 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

The recoveries were calculated per ore type, per year according to the mine production schedule and Table 13-55 presents the life of mine recoveries for the project, including the additional 1% Cu recovery from the SART plant.

The additional 1% Cu recovery is expected as a SART copper sulphide precipitate. Copper recirculating in the gold leach solution was estimated based on cyanide soluble copper assays in the mine plan. The SART plant will treat a 500 m^3/h bleed of barren leach solution. Of that bleed 90% of the copper is estimated to precipitate in concentrate using sodium hydrosulphide. This is based on a review of SART plants either in operation or were operating. Should the cyanide soluble copper levels reduce below a threshold, then the SART plant would be shutdown.

Table 13-55: LOM Recoveries

	LOM Recovery from Ore
Cu	80%
Au	70%
Ag	82%

14 Mineral Resource Estimates

14.1 Introduction

This resource update replaces the resource estimate released on August 21, 2017. This updated resource includes the results of 45 new holes completed during the drill program completed in March 2018.

The impact of new drilling was modest in terms of change to global resource numbers. The increased drill density did, however, have a material impact on geologic model and overall resource confidence. This confidence is reflected in the conversion of a significant tonnage from Inferred to Indicated Mineral Resource. Ongoing metallurgical and engineering studies have continued to support the breakdown of resource reporting by four mineralisation types each with a specific cut-off grade.

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 2017 Mineral Resource; 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 optimised pit ensures reasonable prospects of eventual economic extraction. Table 14-1 summarises the Mineral Resources at Filo del Sol.

Table 14-1: Filo del Sol Deposit

Min. Type	Cut-off	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	49.9	0.04	0.42	3.0	45	679	4,810
		Inferred	20.8	0.08	0.34	2.4	35	226	1,580
CuAuOx	0.15 % CuEq	Indicated	259.2	0.38	0.29	2.7	2,166	2,385	22,500
		Inferred	74.3	0.29	0.31	2.1	481	735	5,040
Ag	20 g/t Ag	Indicated	40.5	0.50	0.43	87.6	446	562	114,180
		Inferred	8.8	0.36	0.43	79.3	70	121	22,400
Sulphide	0.30 % CuEq	Indicated	75.5	0.27	0.34	2.2	451	813	5,370
		Inferred	71.2	0.30	0.33	2.5	470	750	5,740
Total		Indicated	425.1	0.33	0.32	10.7	3,108	4,439	146,860
		Inferred	175.1	0.27	0.33	6.2	1,056	1,832	34,760

14.2 Available Drill Data and Model Set Up

This Filo del Sol resource update is based on assay data available as of May 8, 2018. The update is based on a total of 44,600 metres of drilling in 188 holes including an additional 6,390 metres of reverse circulation drilling in 33 new holes and 2,533 metres of diamond drilling in 12 new holes from the drill program completed in March 2018. Figure 14.1 illustrates drill hole locations as well as block model outlines, the crest of the resource pit and the national border; holes drilled since the 2017 resource estimate are shown in red. For comparative purposes, the limit of the 2017 resource pit is shown in light blue; the pit outline has changed very little since the 2017 version.

The block model setup was changed slightly since the 2017 resource. The bench height was increase from 10 to 12 metres based on the anticipated production rate. Block model configuration details are provided in Table 14-2.

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			

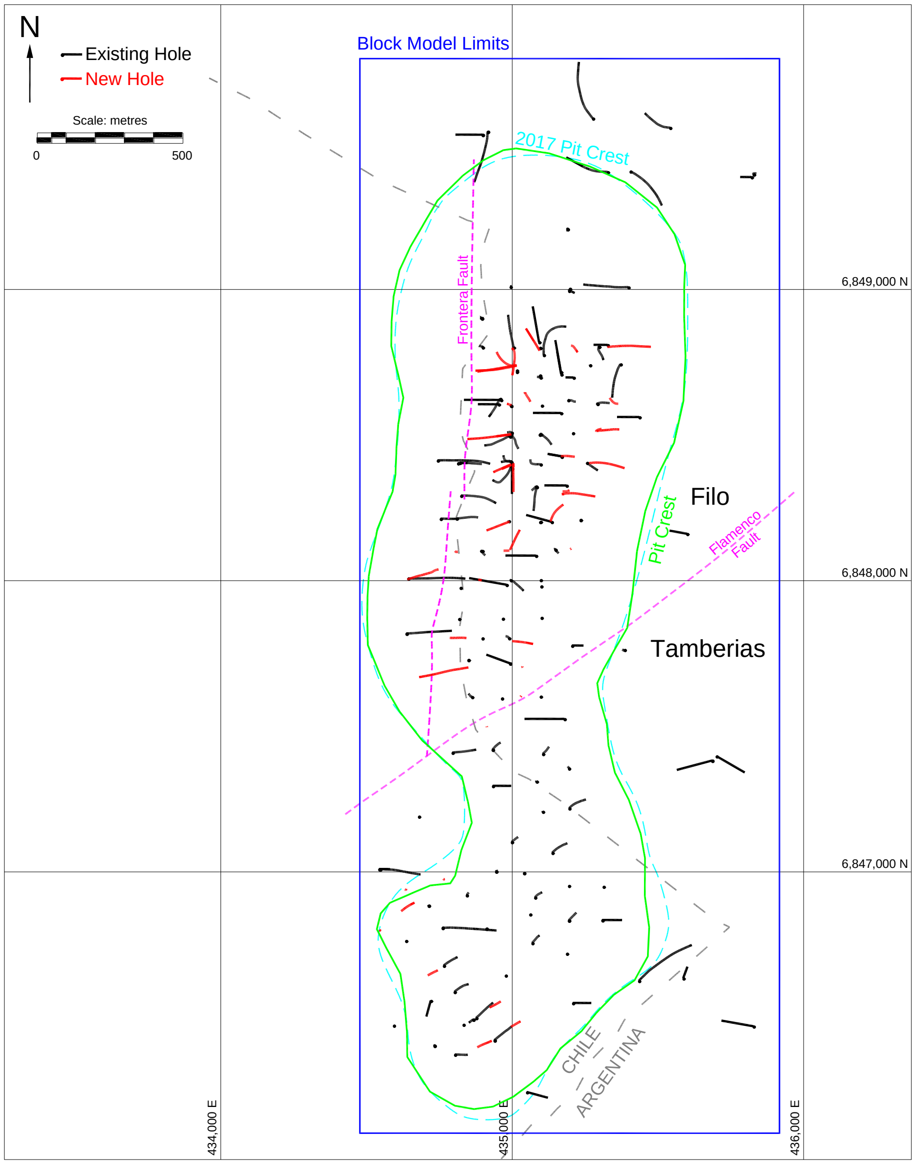


Figure 14-1: Filo del Sol Exploration Drilling and Block Model Limits

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, by the SW-NE Flamenco Fault, into a north - Filo Mining Area (FDS) and a south - Tamberias Area (TMB). FDS mineralisation is bounded to the west by the near vertical north-trending Frontera fault (see Figure 14-1). Units are consistent with those used for grade estimation in 2017.

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 Tamberias area, a zone of silica alteration correlates with elevated gold grades.

14.4 Assay Compositing

Assays were composited to a constant length of two metres within intervals of intersection with the geologic model. Virtually all (99.6%) of the 21,000 assays in the resource area were sampled at two metres making the choice of composite interval obvious. 123 composites of less than half the standard length (1 m) were removed from the dataset prior to grade estimation. Composites were tagged with units of the geologic model to control their use in grade interpolation.

A total of 20,966 Cu, Au and Ag composites were used for 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.

14.5 Grade Capping

Grade capping is used to control the impact of extreme, outlier high-grade samples on the overall resource estimate. For this estimate, composite grades were 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 mineralised 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 cut-off. Metal removed by capping is generally low reflecting the fact that relatively few composite grades were capped (see composite statistics below). Metal removed through capping amounts to: 0.7% Cu, 2.7% Au and 6.0% Ag.

Table 14-3: Grade Composite Capping Levels

Min Zone		Cu (%)	Au (g/t)	Ag (g/t)
FDS	1 - Lix	1	7	40
	3 - Oxide	10	5	100
	4 - Hypogene	1	3	50
	11 - Ag Zone	*uncap*	4	1500
TMB	31 - Lix	0.5	1	10
	33 - Oxide	*uncap*	2	17
	34 - Hypogene	2.3	4	20
	23 - Silica Alteration	0.6	2	*uncap*

Table 14-4: 2m Composite Statistics – Copper

	Min Zone	Count	Cu(%) mean	Cu(%) max	Cu(%) CV	CuCap(%) # Cap	CuCap(%) mean	CuCap(%) max	CuCap(%) CV
FDS	1 - Lix	5,842	0.03	4.45	3.2	6	0.03	1.00	1.8
	3 - Oxide	6,838	0.38	16.60	1.9	8	0.37	10.00	1.8
	4 - Hypogene	1,512	0.27	1.92	0.6	7	0.26	1.00	0.6
	11 - Ag Zone	1,471	0.48	3.46	0.8	0	0.48	3.46	0.8
	FDS Total:	*15,663*	*0.25*			*21*	*0.24*		
TMB	31 - Lix	420	0.07	0.94	1.4	4	0.07	0.50	1.2
	33 - Oxide	2,393	0.29	3.77	1.1	0	0.29	3.77	1.1
	34 - Hypogene	1,635	0.21	3.53	1.0	6	0.21	2.25	0.8
	23 - Silica Alt'n	855	0.06	0.90	1.6	3	0.06	0.60	1.6
	TMB Total:	*5,303*	*0.21*			*13*	*0.21*		

Table 14-5: 2m Composite Statistics Gold

	Min Zone	Count	Au(%) mean	Au(%) max	Au(%) CV	AuCap(%) # Cap	AuCap(%) mean	AuCap(%) max	AuCap(%) CV
FDS	1 - Lix	5,842	0.21	19.10	2.8	9	0.20	7.00	2.3
	3 - Oxide	6,838	0.26	13.70	1.7	11	0.25	5.00	1.2
	4 - Hypogene	1,512	0.33	11.29	1.5	10	0.31	3.00	0.9
	11 - Ag Zone	1,471	0.44	17.88	1.6	14	0.42	4.00	1.0
	FDS Total:	*15,663*	*0.26*			*44*	*0.26*		
TMB	31 - Lix	420	0.26	10.00	2.0	4	0.24	1.00	0.7
	33 - Oxide	2,393	0.29	3.32	0.7	6	0.28	1.76	0.6
	34 - Hypogene	1,635	0.21	2.59	0.8	0	0.21	2.59	0.8
	23 - Silica Alt'n	855	0.33	4.06	0.9	2	0.33	2.00	0.7
	TMB Total:	*5,303*	*0.27*			*12*	*0.26*		

Table 14-6: 2m Composite Statistics – Silver

	Min Zone	Count	Ag(g/t) mean	Ag(g/t) max	Ag(g/t) CV	AgCap(g/t) # Cap	AgCap(g/t) mean	AgCap(g/t) max	AgCap(g/t) CV
FDS	1 - Lix	5,842	1.8	131.0	2.5	17	1.7	40.0	2.0
	3 - Oxide	6,838	3.1	563.0	3.7	14	2.9	100.0	2.3
	4 - Hypogene	1,512	3.1	132.8	2.1	6	3.0	50.0	1.6
	11 - Ag Zone	1,471	103.1	6,967.1	2.9	15	93.2	1,500.0	1.9
	FDS Total:	*15,663*	*12.0*			*52*	*10.9*		
TMB	31 - Lix	420	1.9	15.0	1.0	2	1.8	10.0	0.9
	33 - Oxide	2,393	1.4	48.7	1.5	8	1.4	17.0	1.2
	34 - Hypogene	1,635	1.3	32.9	1.4	6	1.3	18.0	1.1
	23 - Silica Alt'n	855	4.7	45.0	1.2	0	4.7	45.0	1.2
	TMB Total:	*5,303*	*2.0*			*16*	*1.9*		

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		Spherical Component 2	
				Sill	Range(m)	Sill	Range(m)
1. Lix	X	0/355	0.10	0.41	60	0.49	375
	Y	0/265			105		220
	Z	90/000			15		170
3. Oxide (flattened)	X	58/011	0.28	0.49	135	0.23	230
	Y	18/249			50		275
	Z	25/150			45		250
4. Hypo	X	-5/101	0.23	0.27	135	0.50	275
	Y	-14/009			65		145
	Z	75/030			15		125
11. Ag Zone	X	0/050	0.11	0.35	65	0.54	160
	Y	-15/320			45		80
	Z	75/320			10		45

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 (flattened)	X	0/165	0.28	0.43	45	0.29	100
	Y	-85/255			45		80
	Z	-5/075			45		70
34. Hypo	X	-5/330	0.23	0.29	130	0.48	195
	Y	-9/240			60		145
	Z	80/270			20		350
23. Sil. Alt'n	X	0/070	0.06	0.37	50	0.57	130
	Y	0/340			170		270
	Z	90/000			20		185

Table 14-8: Gold Variogram Models

FDS Domain	Axis	Direction (dip/azimuth)	Nugget Effect	Spherical Component 1 Sill	Spherical Component 1 Range(m)	Spherical Component 2 Sill	Spherical Component 2 Range(m)
1. Lix	X	29/002	0.10	0.42	60	0.48	180
	Y	36/247			80		105
	Z	40/120			10		80
3. Oxide	X	49/319	0.30	0.36	20	0.34	200
	Y	26/196			30		230
	Z	30/090			30		155
4. Hypo	X	0/205	0.34	0.41	10	0.25	160
	Y	0/295			180		280
	Z	-90/000			15		215
11. Ag Zone	X	0/040	0.31	0.40	160	0.29	255
	Y	0/310			75		270
	Z	90/000			15		95

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	0/185	0.24	0.34	35	0.42	165
	Y	0/275			15		230
	Z	-90/000			25		140
34. Hypo	X	0/240	0.15	0.17	10	0.68	220
	Y	0/330			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	0/330	0.33	0.36	80	0.31	180
	Y	10/240			10		210
	Z	80/060			10		130
3. Oxide	X	-10/205	0.40	0.44	60	0.16	100
	Y	-80/025			30		95
	Z	0/115			20		120
4. Hypo	X	9/130	0.37	0.33	35	0.30	135
	Y	-59/054			55		70
	Z	30/035			120		180
11. Ag Zone	X	75/280	0.14	0.54	10	0.32	40
	Y	0/190			35		95
	Z	15/100			35		70

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	-2/229	0.33	0.39	115	0.28	265
	Y	30/141			55		190
	Z	60/315			15		110
34. Hypo	X	0/020	0.39	0.35	85	0.26	185
	Y	0/110			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 and a maximum of six samples per hole. Check models were estimated by inverse distance weighting and by nearest neighbor. Search orientations and distances are listed in Table 14-10. Search directions were chosen to best-fit the orientation of the different mineralized zones.

In order 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 optimisation and reporting.

Table 14-10: Estimation Search Parameters

Min Zone			Search Direction (dip/azimuth) X	Search Direction (dip/azimuth) Y	Search Direction (dip/azimuth) Z	Axis Radii (m) X	Axis Radii (m) Y	Axis Radii (m) Z
FDS	1	- Lix	0/031	-4/301	86/301	150	150	75
	3	- Oxide (Au, Ag)	0/059	-4/329	86/329	150	150	75
		- Oxide (Cu)	0/090	0/00	90/00	150	150	75
	4	- Hypogene	0/078	-5/348	85/348	150	150	75
	11	- Ag Zone	0/062	-11/332	79/332	150	150	75
TMB	31	- Lix	0/039	-19/309	71/309	150	150	75
	33	- Oxide (Au, Ag)	0/030	-21/300	69/300	150	150	75
		- Oxide (Cu)	0/090	0/00	90/00	150	150	75
	34	- Hypogene	0/030	-10/300	80/300	150	150	75
	23	- Silica Alt'n	0/090	0/00	90/00	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

A significant number of additional density measurements were made during the 2017/2018 drill campaign. In total, 1,369 wax-dip water immersion density measurements have been made on Filo del Sol core samples. These samples were coded with a “Min Zone” based on the revised geologic model and examined statistically. Thirty-nine samples were judged to have spurious values and two samples were located outside the Min Zone volumes; these results were removed from the dataset used to derive averages per geologic division. The mean values listed in Table 14-12, except as noted in the table, were assigned to blocks based on their Min Zone value and used for resource tonnage calculation.

Table 14-12: Average Bulk Density

Min Zone		Count	Bulk Density (t/m^3)			Comments
			Mean	Min	Max	
FDS	1 - Lix	409	2.08	1.44	2.65	
	3 - Oxide	477	2.19	1.61	2.69	
	4 - Hypogene	97	2.60	2.26	2.89	
	11 - Ag Zone	115	2.24	1.62	2.72	
TMB	31 - Lix	1	2.35	2.35	2.35	use 2.08 based on Filo
	33 - Oxide	40	2.53	2.28	2.68	
	34 - Hypogene	189	2.50	2.18	2.76	
	23 - Silica Alteration	0	2.41			average of non-Lix
Total:		1,328	2.36			avg. used west of Frontera Fault

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 14-2 to Figure 14-4 respectively. There is good visual correlation between composite and estimated block grades for all modelled elements.

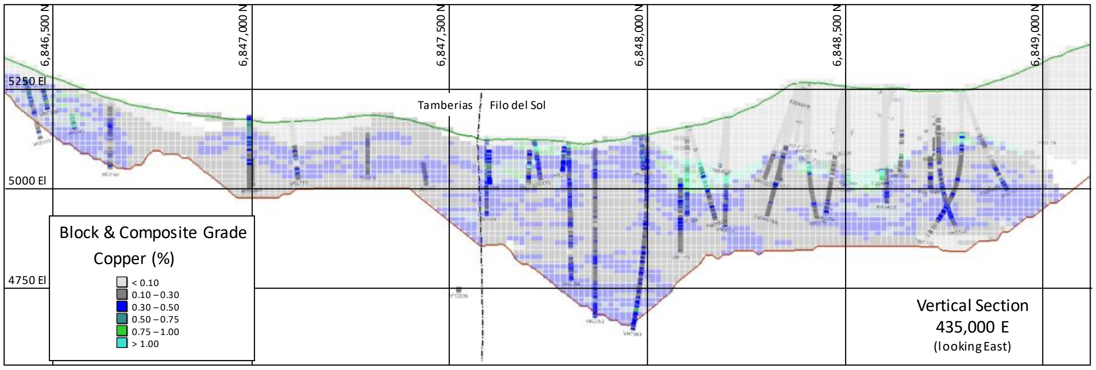


Figure 14-2: Copper Block and Composite Grades

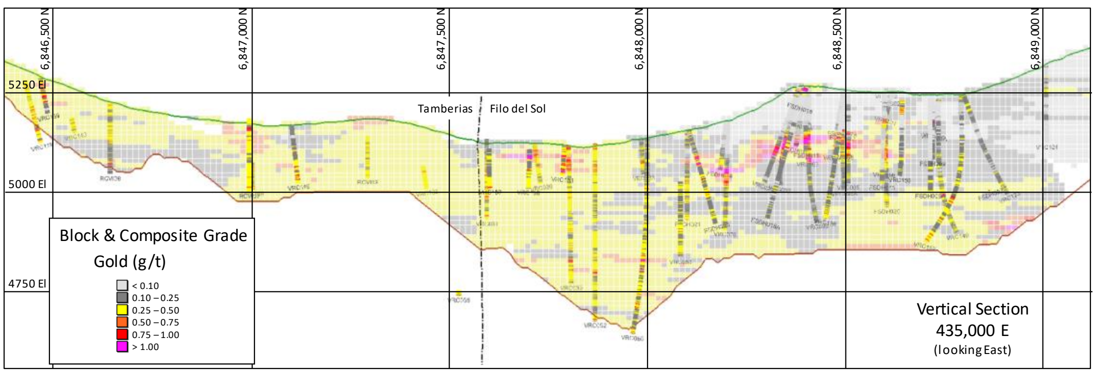


Figure 14-3: Gold Block and Composite Grades

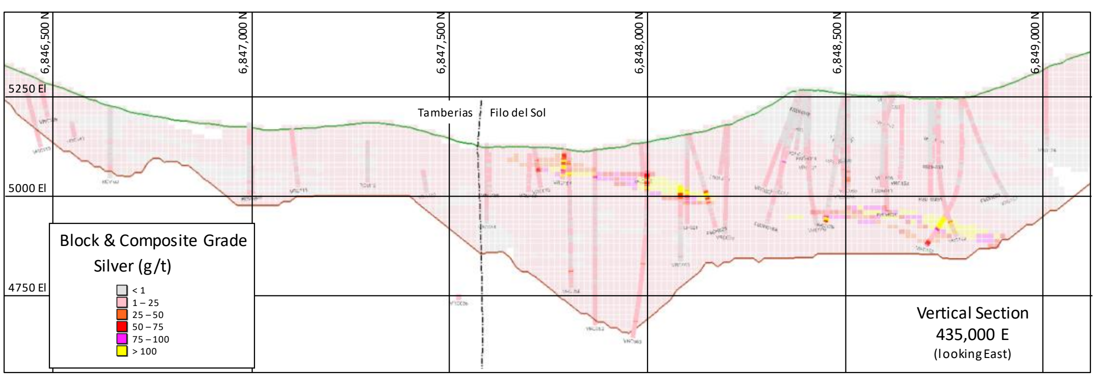


Figure 14-4: Silver Block and Composite Grades

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 in Figure 14-5, Figure 14-6 and Figure 14-7 respectively. The OK estimates are appropriately smooth in comparison to the nearest neighbor models. Globally, model average grades above zero cut-off compare very closely indicating no bias; mean grades at zero cut-off 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 Count	Cu (%)			Au (g/t)			Ag (g/t)		
			OK	NN	ID²	OK	NN	ID³	OK	NN	ID²
FDS	1 - Lix	24,890	0.03	0.03	0.03	0.20	0.20	0.20	1.8	1.9	1.8
	3 - Oxide	36,691	0.38	0.38	0.38	0.28	0.27	0.27	2.9	2.8	2.8
	4 - Hypogene	16,823	0.28	0.28	0.28	0.33	0.33	0.33	2.6	2.5	2.6
	11 - Ag Zone	9,018	0.47	0.46	0.46	0.42	0.41	0.42	79.2	80.3	80.6
	FDS Total:	*87,422*	*0.27*	*0.27*	*0.27*	*0.28*	*0.28*	*0.28*	*10.4*	*10.5*	*10.5*
TMB	31 - Lix	3,578	0.06	0.06	0.06	0.26	0.27	0.25	1.7	1.9	1.7
	33 - Oxide	16,698	0.32	0.30	0.31	0.31	0.31	0.31	1.5	1.5	1.5
	34 - Hypogene	4,823	0.28	0.27	0.27	0.33	0.33	0.33	1.5	1.5	1.5
	23 - Silica Alt'n	2,873	0.08	0.08	0.07	0.41	0.41	0.41	4.1	4.1	4.2
	TMB Total:	*27,972*	*0.25*	*0.24*	*0.25*	*0.32*	*0.32*	*0.32*	*1.8*	*1.8*	*1.8*

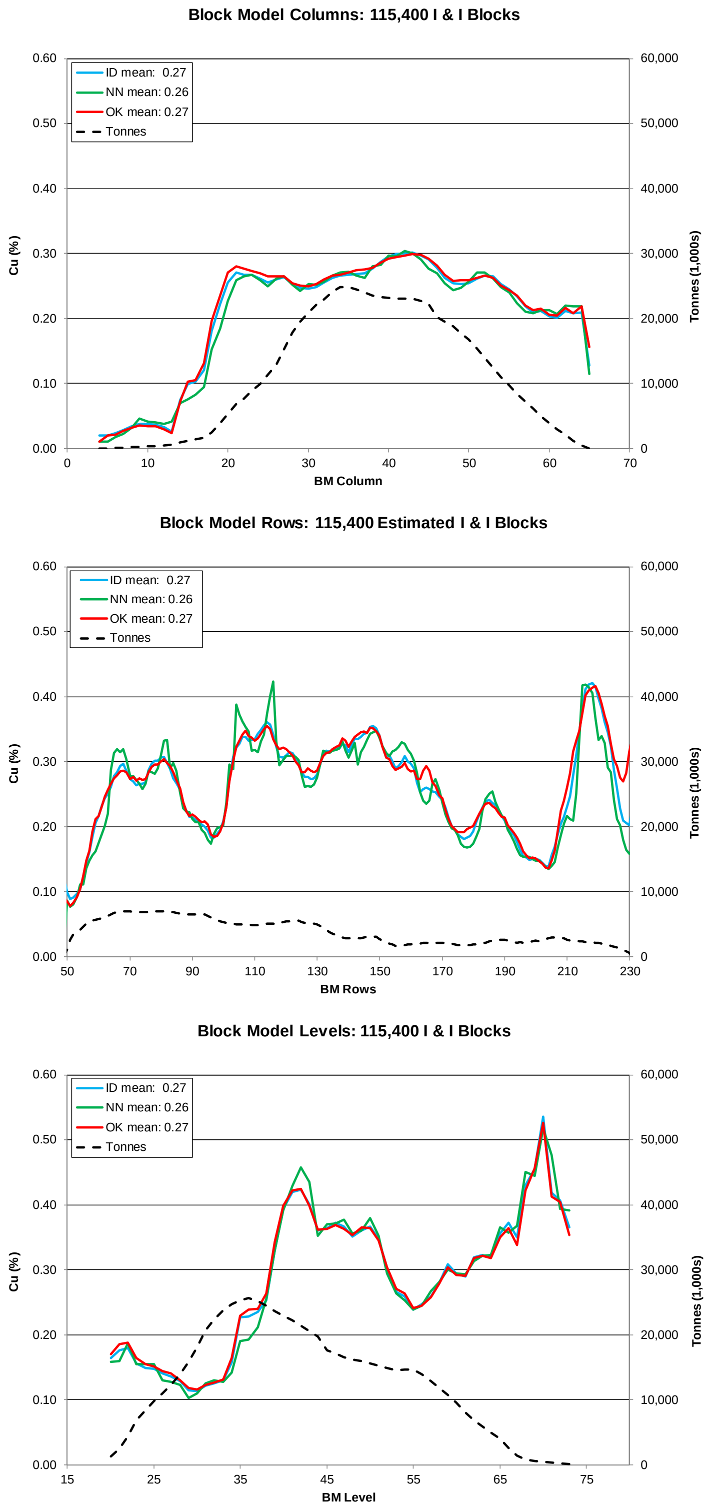
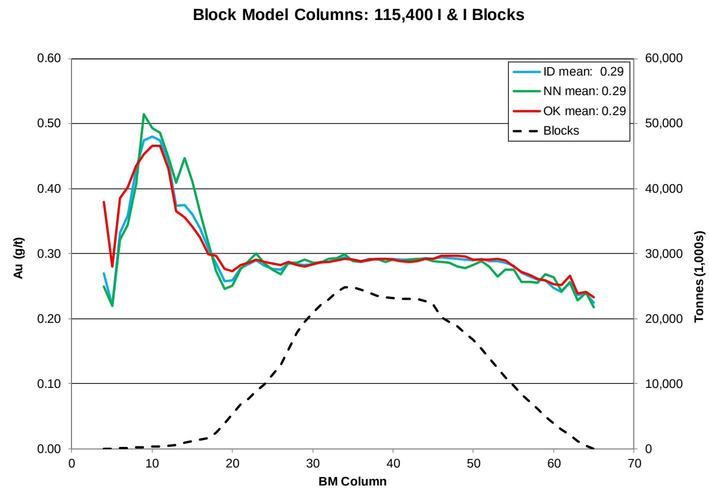
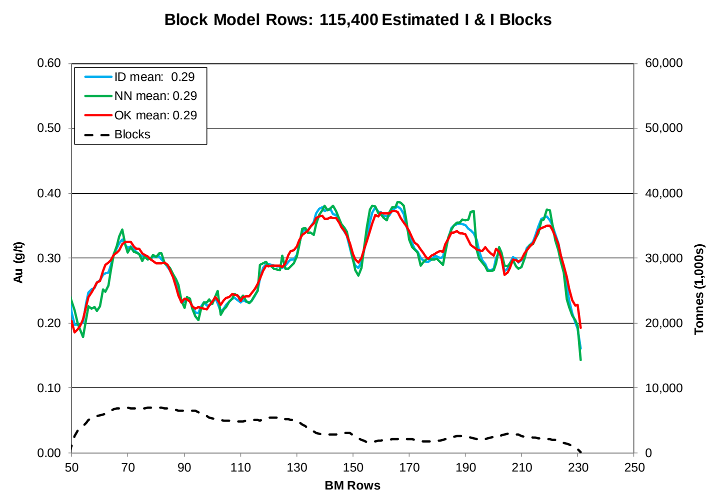


Figure 14-5: Copper Grade Swath Plots Comparing OK, NN, and ID Estimates





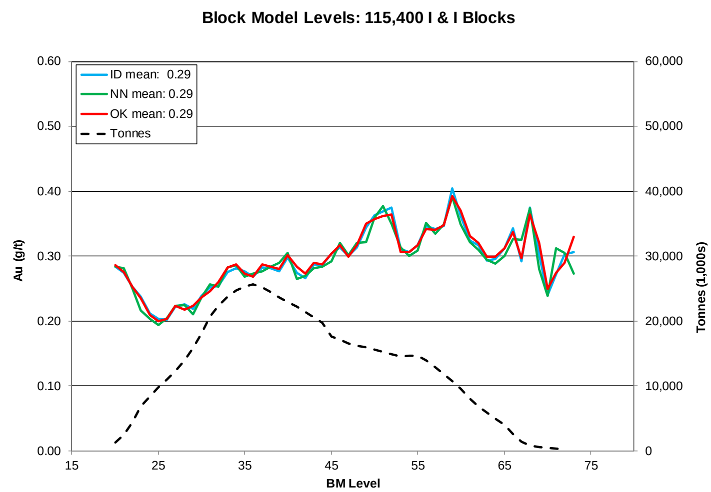
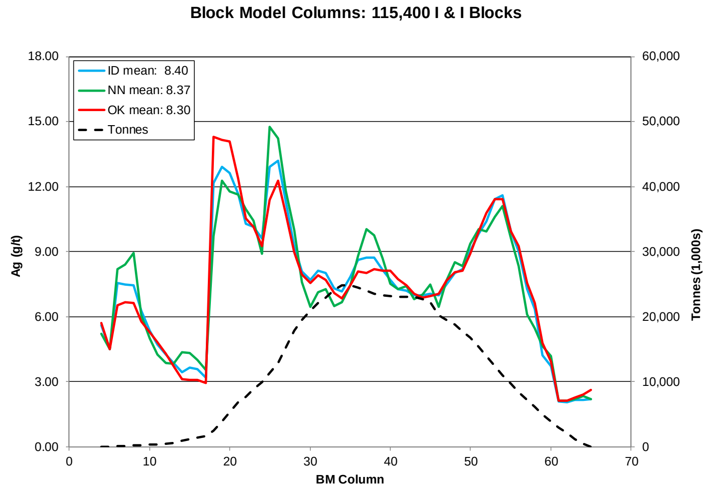
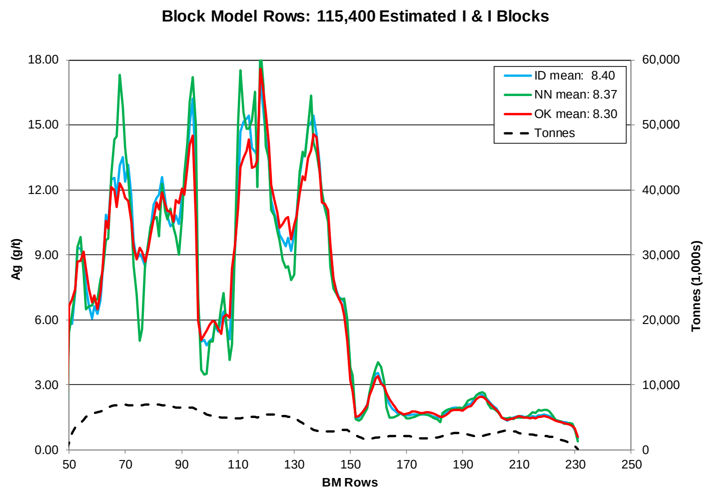


Figure 14-6: Gold Grade Swath Plots Comparing OK, NN, and ID Estimates





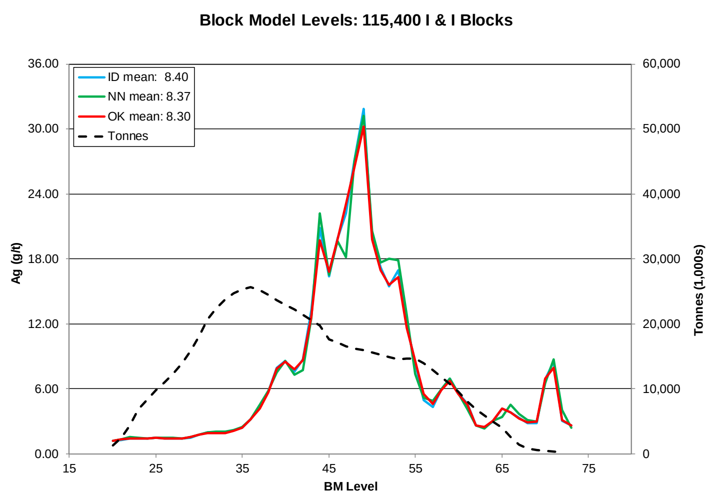


Figure 14-7: Silver Grade Swatch Plots Comparing OK, NN, and ID Estimates

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 optimised pit. To ensure appropriate classification of contiguous blocks, blocks were classified inside a solid volume. As utilised 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

Measures were taken to ensure the resource meets the condition of "reasonable prospects of eventual economic extraction" as required under NI 43-101. An optimised pit shell was generated using Whittle® software and only blocks within the pit volume are included in the Mineral Resource. Pit optimisation was carried out by AGP Mining Consultants using parameters listed in Table 14-14. Metal recoveries and process costs were variable by mineralisation type; averages are included in Table 14-16.

Table 14-14: Pit Optimisation Parameters

Metal	Metal Price	Av.Recovery
Cu	US$ 3.0/lb	75%
Au	US$ 1300/oz	68%
Ag	US$ 20/oz	82%
Mining Cost:	$ 2.50 / tonne	
Av. Process Cost:	$13.26 / tonne, including G & A	
Pit slope:	45°	

The Filo del Sol Mineral Resource Estimate is tabled by mineralisation type based on metallurgical testwork to date. Four mineralisation types are anticipated; their correspondence to the Min Zones used for estimation is listed in Table 14-15. Mineralisation types were assigned per block and used in final resource tabulation.

Table 14-15: Mineralisation Type Assignment

Min Zone				Min. Type
FDS	1	- Lix		AuOx
	3	- Oxide		CuAuOx
	4	- Hypogene		Hypogene
	11	- Ag Zone	≥ 20 g/t Ag	Ag
			< 20 g/t Ag	CuAuOx
TMB	31	- Lix		AuOx
	33	- Oxide		CuAuOx
	34	- Hypogene		Hypogene
	23	- Silica Alteration		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 for the two mineralisation types 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)	3	1300	20	82	55	71	Cu+Ag*0.0084+Au*0.4239
Hypogene (Sulphide)	3	1300	20	84	70	77	Cu+Ag*0.0089+Au*0.5266

Cut-off grades were chosen based on preliminary expected mining and processing costs per mineralisation type. Cut-offs are specified with the updated Mineral Resource in Table 14-17. The four mineralisation types are tabled at a range of cut-off grades in Table 14-18 to Table 14-21.

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

Min. Type	Cut-off	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	49.9	0.04	0.42	3.0	45	679	4,810
		Inferred	20.8	0.08	0.34	2.4	35	226	1,580
CuAuOx	0.15 % CuEq	Indicated	259.2	0.38	0.29	2.7	2,166	2,385	22,500
		Inferred	74.3	0.29	0.31	2.1	481	735	5,040
Ag	20 g/t Ag	Indicated	40.5	0.50	0.43	87.6	446	562	114,180
		Inferred	8.8	0.36	0.43	79.3	70	121	22,400
Sulphide	0.30 % CuEq	Indicated	75.5	0.27	0.34	2.2	451	813	5,370
		Inferred	71.2	0.30	0.33	2.5	470	750	5,740
Total		Indicated	425.1	0.33	0.32	10.7	3,108	4,439	146,860
		Inferred	175.1	0.27	0.33	6.2	1,056	1,832	34,760

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Table 14-18: Gold Oxide Zone by Gold Cut-off

Min. Type	Cut-off	Category	Tonnes (millions)	Cu (%)	Au (g/t)	Ag (g/t)	lbs Cu (millions)	Ounces Au (thousands)	Ounces Ag (thousands)
AuOx	0.10 g/t Au	Indicated	76.5	0.04	0.33	2.5	64	799	6,140
		Inferred	31.4	0.06	0.27	2.2	41	276	2,200
	0.20 g/t Au	***Indicated***	***49.9***	***0.04***	***0.42***	***3.0***	***45***	***679***	***4,810***
		Inferred	***20.8***	***0.08***	***0.34***	***2.4***	***35***	***226***	***1,580***
	0.40 g/t Au	Indicated	22.4	0.04	0.60	3.6	22	427	2,560
		Inferred	4.9	0.09	0.49	3.5	10	77	540
	0.50 g/t Au	Indicated	13.1	0.04	0.70	3.6	13	295	1,500
		Inferred	1.7	0.09	0.60	3.9	4	33	210

Table 14-19: Copper Gold Oxide Zone by Copper Equivalent Cut-off

Min. Type	Cut-off	Category	Tonnes (millions)	Cu (%)	Au (g/t)	Ag (g/t)	lbs Cu (millions)	Ounces Au (thousands)	Ounces Ag (thousands)
CuAuOx	***0.15 % CuEq***	***Indicated***	***259.2***	***0.38***	***0.29***	***2.7***	***2,166***	***2,385***	***22,500***
		Inferred	***74.3***	***0.29***	***0.31***	***2.1***	***481***	***735***	***5,040***
	0.30 % CuEq	Indicated	233.6	0.40	0.30	2.8	2,071	2,230	21,260
		Inferred	63.6	0.32	0.33	2.2	445	665	4,550
	0.50 % CuEq	Indicated	93.5	0.61	0.35	3.7	1,252	1,037	11,070
		Inferred	20.9	0.46	0.39	2.8	210	259	1,850
	0.70 % CuEq	Indicated	38.0	0.89	0.36	4.4	750	442	5,430
		Inferred	4.4	0.67	0.50	2.5	64	71	350

Table 14-20: Silver Zone by Silver Cut-off

Min. Type	Cut-off	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***	***40.5***	***0.50***	***0.43***	***87.6***	***446***	***562***	***114,180***
		Inferred	***8.8***	***0.36***	***0.43***	***79.3***	***70***	***121***	***22,400***
	50 g/t Ag	Indicated	27.4	0.50	0.42	113.1	303	371	99,780
		Inferred	5.6	0.39	0.42	105.3	48	76	18,970
	60 g/t Ag	Indicated	23.9	0.51	0.42	121.8	266	324	93,480
		Inferred	4.8	0.41	0.43	114.1	43	66	17,480
	80 g/t Ag	Indicated	17.3	0.51	0.42	141.8	196	233	78,730
		Inferred	3.2	0.43	0.44	135.7	31	45	14,040

Table 14-21: Hypogene Zone by Copper Equivalent Cut-off

Min. Type	Cut-off	Category	Tonnes (millions)	Cu (%)	Au (g/t)	Ag (g/t)	lbs Cu (millions)	Ounces Au (thousands)	Ounces Ag (thousands)
Hypogene	***0.30 % CuEq***	***Indicated***	***75.5***	***0.27***	***0.34***	***2.2***	***451***	***813***	***5,370***
		Inferred	***71.2***	***0.30***	***0.33***	***2.5***	***470***	***750***	***5,740***
	0.40 % CuEq	Indicated	56.1	0.30	0.36	2.4	365	644	4,340
		Inferred	59.7	0.32	0.34	2.6	419	649	5,060
	0.50 % CuEq	Indicated	24.4	0.34	0.40	2.9	183	313	2,270
		Inferred	29.3	0.37	0.36	3.4	238	340	3,190
	0.60 % CuEq	Indicated	6.9	0.39	0.49	3.9	58	107	850
		Inferred	9.4	0.43	0.41	5.1	89	124	1,540

Table 14-22 compares this resource estimate to that completed in 2017. The impact of new drilling was modest in terms of change to global resource numbers. The increased drill density did, however, have a material impact on geologic model and overall resource confidence. This confidence is reflected in the conversion of a significant tonnage from Inferred to Indicated Mineral Resource.

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)
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
2017	Indicated	372.9	0.34	0.33	9.2	2,774	3,954	109,880
	Inferred	238.9	0.27	0.33	7.8	1,442	2,510	59,990
Difference	Indicated	+14.0%	-2.5%	-1.6%	+16.7%	+12.0%	+12.2%	+33.5%
	Inferred	-26.7%	+1.1%	-1.3%	-20.7%	-26.9%	-26.9%	-42.0%

15 Mineral Reserve Estimates

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 were based on the project Mineral Resource estimate with an effective date of June 11, 2018, which is discussed in Section 14. The work was performed using metal prices of Cu $3.00/lb, Ag $20/oz, Au $1300/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) values and cut-off 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 at using a mixing zone approach, where the volumes of dilution gain and ore loss would ‘wash out’, resulting in diluted grades that were lower than the in-situ resource grades, but tonnage remained the same. A mixing zone extending three meters 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 rocktype, and variable process costs by rocktype, 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 Rocktype 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	3.00	1300.00	20.00
Selling Cost	0.20	0.50	0.50
Net Price	2.80	1299.50	19.50

As of the effective date of the Mineral Reserves, (January 13, 2019), the metal prices used are all higher than current spot prices ($2.66/lb Cu (LME), $1289/oz Au (COMEX) and $15.61/oz Ag (COMEX)), and higher than the three-year trailing averages ($2.66/lb Cu (LME), $1260/oz Au (COMEX) and $16.61/oz Ag (COMEX)).

The selling costs were based on values used in the PEA study 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 below:

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 affect on the pit shapes and quantity of material above cut-off.

15.3.3 Operating Costs

The process costs developed for the 60,000 tpd throughput rate, are variable by domain groups, as shown in Table 15-3 below.

Table 15-3: Process Operating Costs

Domains	Min Zone	Process Cost ($/t)
FDS Domains	1, 3, & 11	9.67
TMB Domains	23, 31, & 33	10.08

The G&A cost used was $2.02/t processed. The mining cost was $2.00/t mined, which included a $0.15/t sustaining capex allowance.

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.

15.3.4 Cut-Offs

Mine planning was performed at a marginal breakeven cut-off. Elevated cut-offs and long-term low-grade stockpiling were not considered due to the risk of mobilization / loss of fast leaching copper from stockpiled ores. As the NVPT is inclusive of process and G&A costs, the breakeven cut-off is $0.01/t NVPT. Mining costs are not included in the breakeven cut-off analysis.

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 ™, (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 0.2 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	Total Crusher Feed					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_0.2	547	45.05	0.57	0.50	21.6	58	605	0.11	25
RF_0.225	1,444	42.98	0.56	0.48	20.1	144	1,588	0.10	59
RF_0.25	4,136	39.34	0.49	0.46	20.9	335	4,471	0.08	154
RF_0.275	32,389	43.80	0.55	0.37	26.3	49,591	81,980	1.53	1,251
RF_0.30	56,254	39.71	0.48	0.35	27.5	71,782	128,036	1.28	1,972
RF_0.325	58,811	39.58	0.48	0.35	27.2	74,176	132,988	1.26	2,055
RF_0.35	61,901	39.41	0.48	0.35	26.9	78,781	140,682	1.27	2,151

Revenue	Total Crusher Feed					Waste	Total Material	SR	Pre-Capex Undiscounted Cash Flow
RF_0.375	135,280	31.67	0.42	0.33	21.6	175,454	310,734	1.30	3,648
RF_0.40	146,760	31.22	0.42	0.33	20.7	187,468	334,228	1.28	3,896
RF_0.425	169,484	30.79	0.42	0.34	19.7	230,375	399,859	1.36	4,399
RF_0.45	176,725	30.56	0.42	0.34	19.4	241,126	417,851	1.36	4,544
RF_0.475	185,701	30.35	0.42	0.34	19.3	260,725	446,425	1.40	4,721
RF_0.50	188,814	30.18	0.42	0.34	19.1	263,392	452,206	1.39	4,772
RF_0.525	193,613	30.00	0.42	0.34	19.0	273,184	466,798	1.41	4,852
RF_0.55	200,820	29.63	0.42	0.34	18.7	284,418	485,238	1.42	4,955
RF_0.575	204,333	29.39	0.42	0.34	18.4	287,614	491,947	1.41	4,998
RF_0.60	211,922	29.00	0.41	0.34	18.2	302,700	514,621	1.43	5,092
RF_0.625	216,015	28.71	0.41	0.34	17.9	305,892	521,907	1.42	5,131
RF_0.65	221,665	28.29	0.41	0.33	17.6	310,455	532,119	1.40	5,181
RF_0.675	228,613	27.79	0.41	0.33	17.1	316,410	545,024	1.38	5,236
RF_0.70	240,240	27.06	0.40	0.33	16.5	334,229	574,470	1.39	5,323
RF_0.725	248,662	26.56	0.40	0.33	16.2	347,797	596,459	1.40	5,381
RF_0.75	255,412	26.10	0.39	0.33	15.8	353,343	608,756	1.38	5,419
RF_0.775	**264,386**	**25.51**	**0.39**	**0.33**	**15.4**	**360,032**	**624,418**	**1.36**	**5,464**
RF_0.80	268,733	25.22	0.39	0.33	15.2	362,715	631,448	1.35	5,483
RF_0.825	273,716	24.91	0.39	0.33	15.0	368,499	642,214	1.35	5,503
RF_0.85	278,359	24.62	0.38	0.33	14.8	372,450	650,808	1.34	5,518
RF_0.875	283,703	24.27	0.38	0.33	14.5	376,740	660,443	1.33	5,531
RF_0.90	287,170	24.09	0.38	0.33	14.4	384,464	671,635	1.34	5,540
RF_0.925	290,529	23.88	0.38	0.33	14.3	388,247	678,776	1.34	5,545
RF_0.95	293,040	23.72	0.38	0.32	14.2	390,809	683,848	1.33	5,548
RF_0.975	296,025	23.54	0.38	0.32	14.0	394,975	691,001	1.33	5,551
RF_1.00	298,261	23.40	0.38	0.32	14.0	398,062	696,323	1.33	5,551

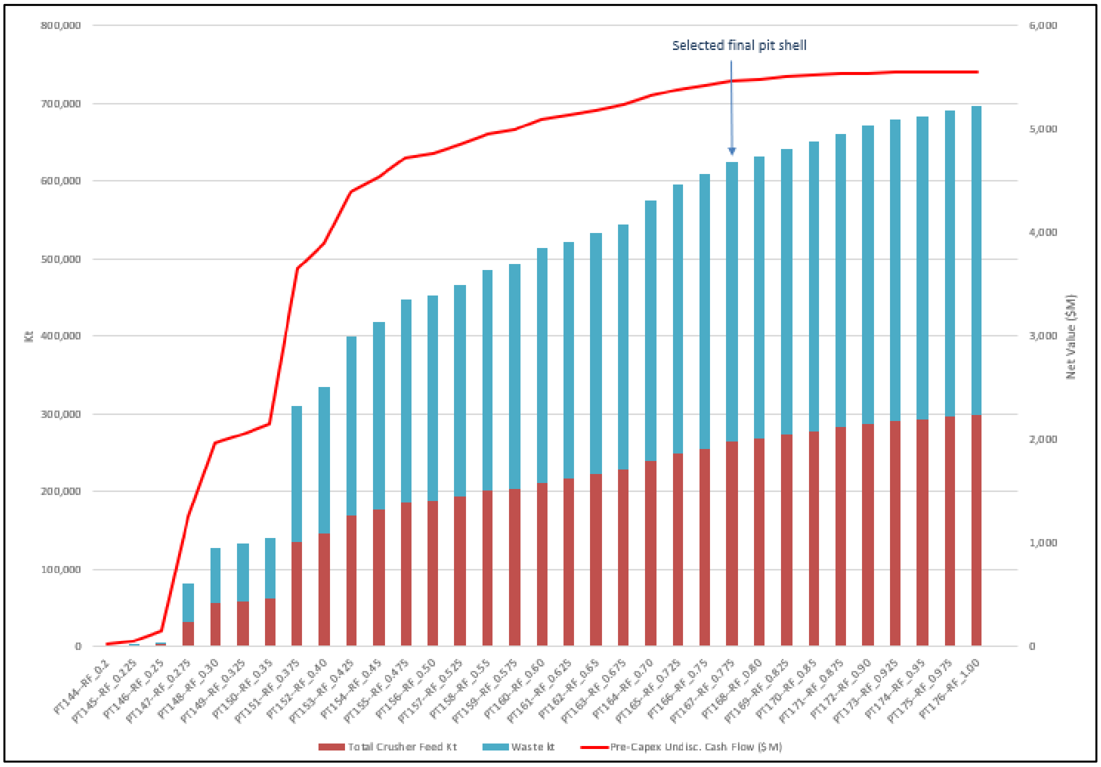


Figure 15-1: Nested LG Pit Shell 'Pit by Pit' Graph

The Revenue Factor (RF) 0.775 LG shell was selected to guide the ultimate pit design. This shell was selected based on the following considerations:

- A desire for a mine life of 12 years or more to provide a substantial tail after project payback
- The crusher feed quantity in the LG Shell matched the available capacity for a single cyanide leach pad
- Visual inspection of the Pre-Capex Undiscounted Cash Flow shown in Figure 15-1 shows that the economic gains from utilizing a higher Revenue Factor shell were small and likely less attractive when the time value of money is considered.

The outline of the RF 0.775 LG shell outline is shown with 5m topography contours is shown in below.

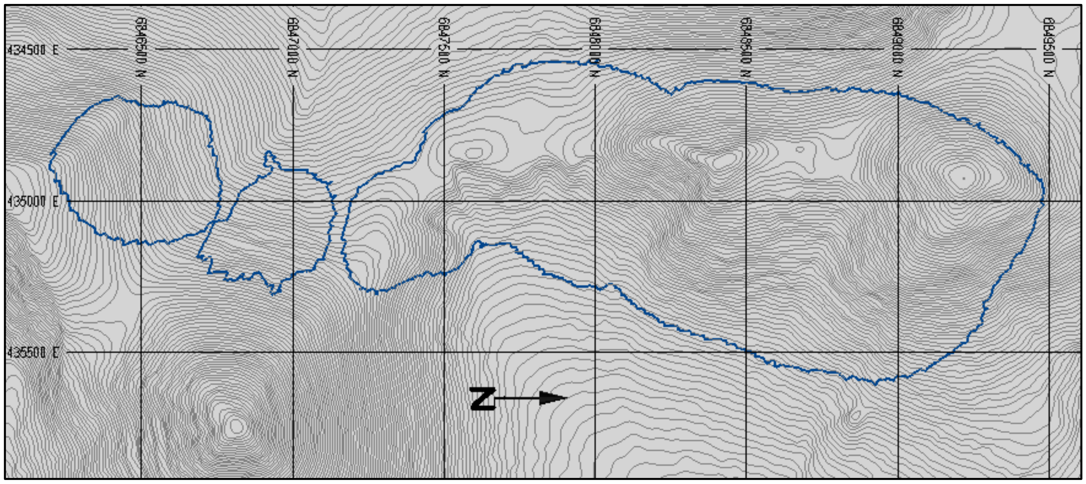


Figure 15-2: RF 0.775 LG Shell Outline

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 Tamberias 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 432 m 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 Figure 15-3, with the outline of the RF 0.775 LG shell in black.

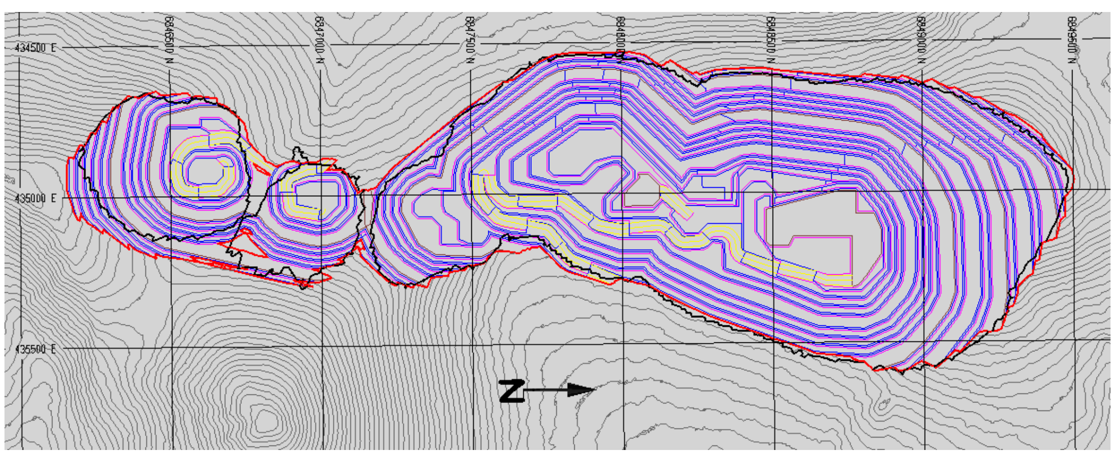


Figure 15-3: Ultimate Pit Design

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 Jay Melnyk, P.Eng. of AGP Mining Consultants Inc. who is a QP as defined under NI 43-101. The Mineral Resources are inclusive of Mineral Reserves.

Table 15-5: Mineral Reserve

Filo del Sol Mineral Reserve Statement (@ 0.01 $/t NVPT cut-off)								
	Tonnage	Grade				Contained Metal		
Category (all domains)	(Mt)	Cu (%)	Au (g/t)	Ag (g/t)	NVPT ($/t)	Cu (M lbs)	Au (k oz)	Ag (k oz)
Proven	-	-	-	-	-	-	-	-
Probable	259.1	0.39	0.33	15.1	25.30	2,226	2,764	126,028
Total Proven and Probable	**259.1**	**0.39**	**0.33**	**15.1**	**25.30**	**2,226**	**2,764**	**126,028**

Notes to accompany Filo del Sol Mineral Reserves table:

8. Mineral Reserves have an effective date of 13 January 2019. The Qualified Person for the estimate is Mr. Jay Melnyk, P.Eng. of AGP Mining Consultants, Inc.
9. The Mineral Reserves were estimated in accordance with the CIM Definition Standards for Mineral Resources and Reserves;
10. 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 $3.00/lb, Ag $20/oz, Au $1300/oz;
    - Mining cost of $2.00/t;
    - An average processing cost of $9.73/t;
    - General and administration cost of $2.02/t processed;
    - Pit slope angles varying from 29 to 45 degrees, inclusive of geotechnical berms and ramp allowances;
    - Process recoveries were based on rocktype. 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;
11. 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;
12. Ore/Waste delineation was based on a Net Value Per Tonne (NVPT) breakeven cut-off considering metal prices, recoveries, royalties, process and G&A costs as per LG shell parameters stated above;
13. The life-of-mine (LOM) stripping ratio in tonnes is 1.52:1;
14. 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-meter 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 tpd processing rate. The peak mining capacity is 65 million tonnes per annuum.

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 Tamberias 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:

- 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. Tamberias 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 of the proposed Tamberias pit are located within this domain.

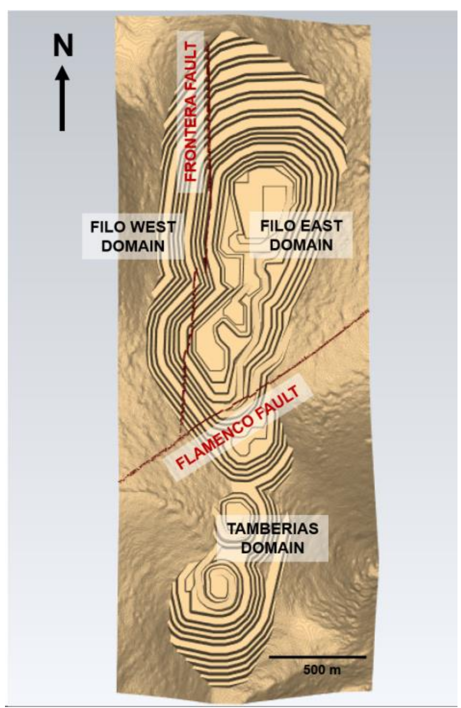


Figure 16-1: Structural Domain Boundaries Shown with Ultimate Pit

No geotechnical or exploration core drilling was completed in 2018 in either the Filo West or Tamberias 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.
- Tamberias (TAM) Unit, comprising the rock mass located south of the Flamenco fault, this unit forms the Tamberias 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 Tamberias 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[1]	Design Sector[1]	Slope Azimuth		Bench Geometry			Inter-Ramp Geometry			Slope Design Control
				Design Height	Face Angle	Width[2]	Maximum Height	Angle	Geotechnical Berm Width[3]	
		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)
	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)
Tamberias	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 pre-feasibility 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 raveling and rockfall hazard, particularly for north-facing slopes. 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.

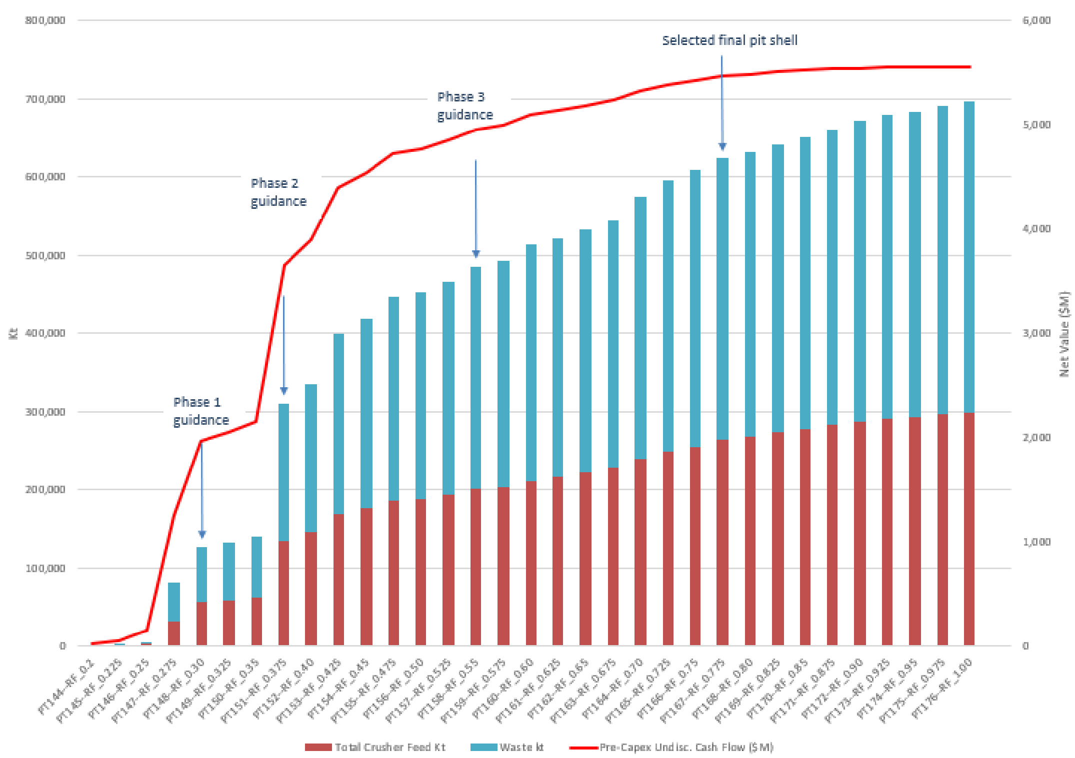


Figure 16-2: Pit by Pit Graph with Selected Filo Internal Phase Guidance

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:

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 5,309m elevation and mining to the 4,949m bottom is completed in year 5. The ramp from the bottom daylights to the east at the 5,075m 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.

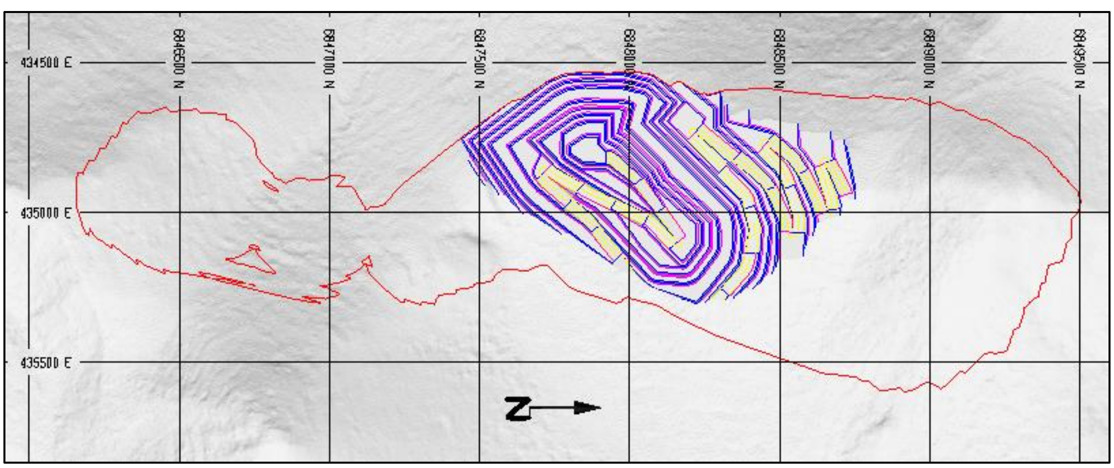


Figure 16-3: Filo Phase 1 Design with Ultimate Pit Outline in Red

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 2 at the 5,405m elevation and mining to the 4,937m bottom is completed in year 6. The ramp from the bottom daylights to the east at the 5,069m 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 16-4 below.

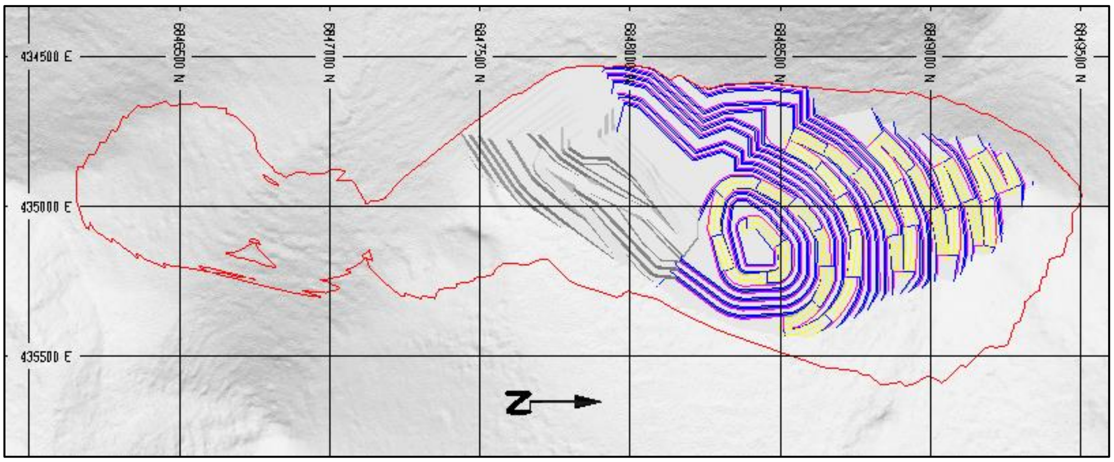


Figure 16-4: Filo Phase 2 Design

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 2 at the 5,381m crest elevation and mining to the 4,901m bottom is completed in year 11. The ramp from the bottom daylights to the east at the 5,067m 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 16-5 below.

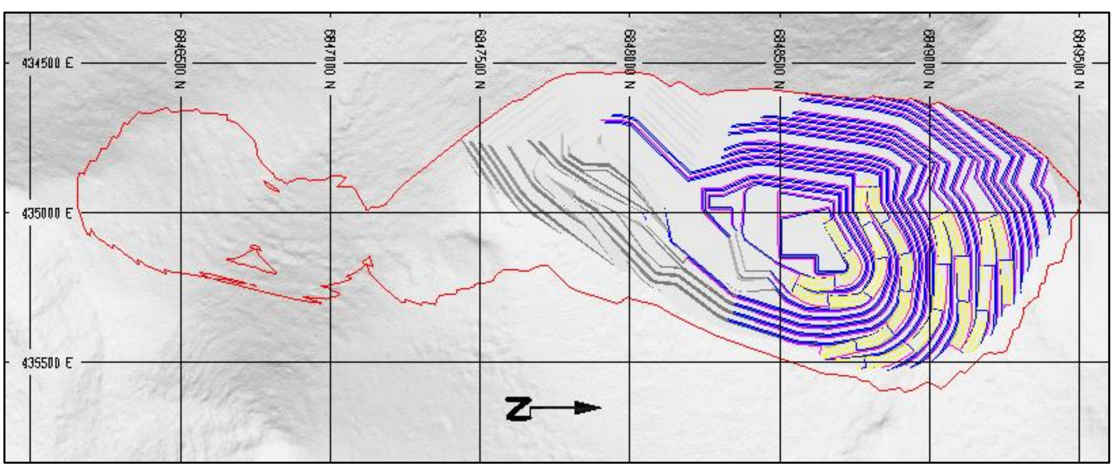


Figure 16-5: Filo Phase 3 Design

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 4 at the 5,333m elevation and mining to the 4,877m bottom is completed in year 13. The ramp from the bottom daylights to the east at the 5,055m elevation. The Filo phase 4 design is shown in Figure 16-6 below.

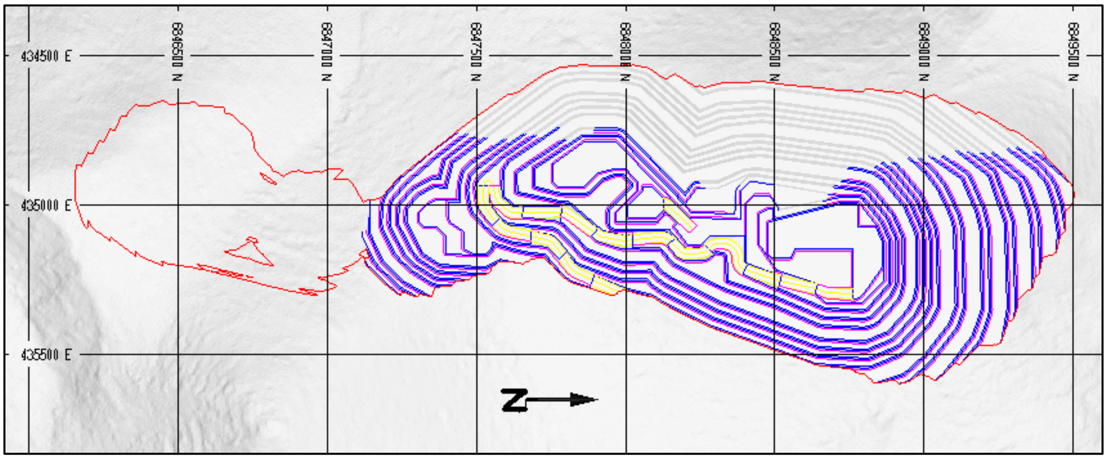


Figure 16-6: Filo Phase 4 Design

Tamberias Pit

The Tamberias pit is a small single phased pit to the south of the Filo pit. Stripping starts in year 1 at the 5,393m elevation and mining to the twin 5,105m bottoms is completed in year 3. The ramps from the southern lobe pit bottom daylights to the west at the 5,153m elevation, and ramp from the bottom of the northern lobe daylights at the 5,133m elevation. The Tamberias pit design is shown in Figure 16-7 below.

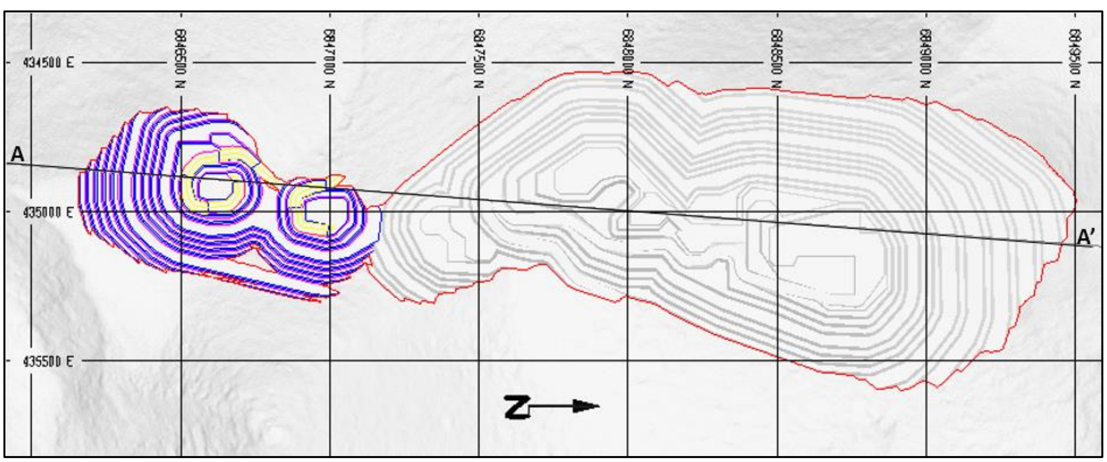


Figure 16-7: Tamberias Pit Design

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 of the phase pit bottoms.

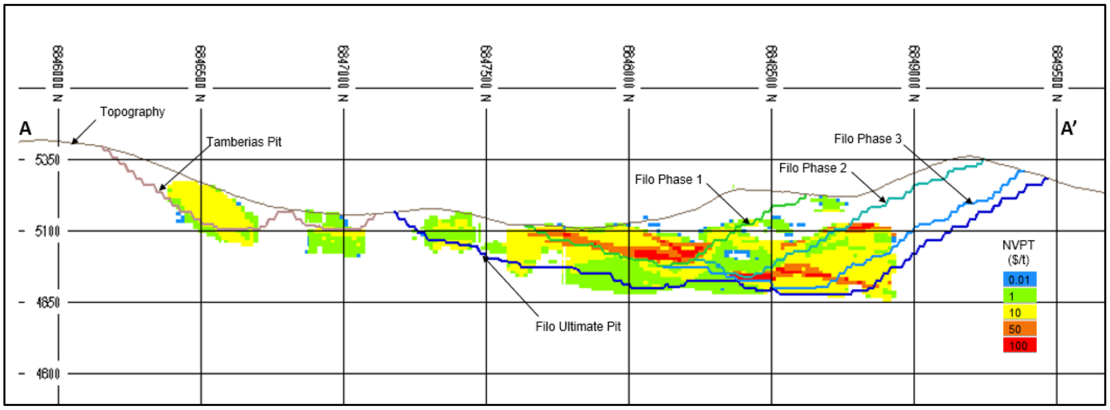


Figure 16-8: ~ West Looking Vertical Section Showing Pit Phases and Above Cut-off Mineralization

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	50,715	0.46	0.35	24.7	37.38	84,391	135,105	1.7
Filo Ph2	46,829	0.33	0.33	7.4	18.67	76,920	123,749	1.6
Filo Ph3	59,594	0.43	0.30	18.5	28.38	123,832	183,426	2.1
Filo Ph4	75,714	0.34	0.34	15.6	21.76	82,651	158,364	1.1
Tamberias	26,226	0.41	0.33	1.3	16.99	26,734	52,961	1.0
Total	259,078	0.39	0.33	15.1	25.30	394,527	653,605	1.5

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 toe area (key) to provide good contact and stability are required. In order 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.

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%

Plan view images of the Phase 1 and 2 dump designs are shown in Figure 16-9 and Figure 16-10 below.

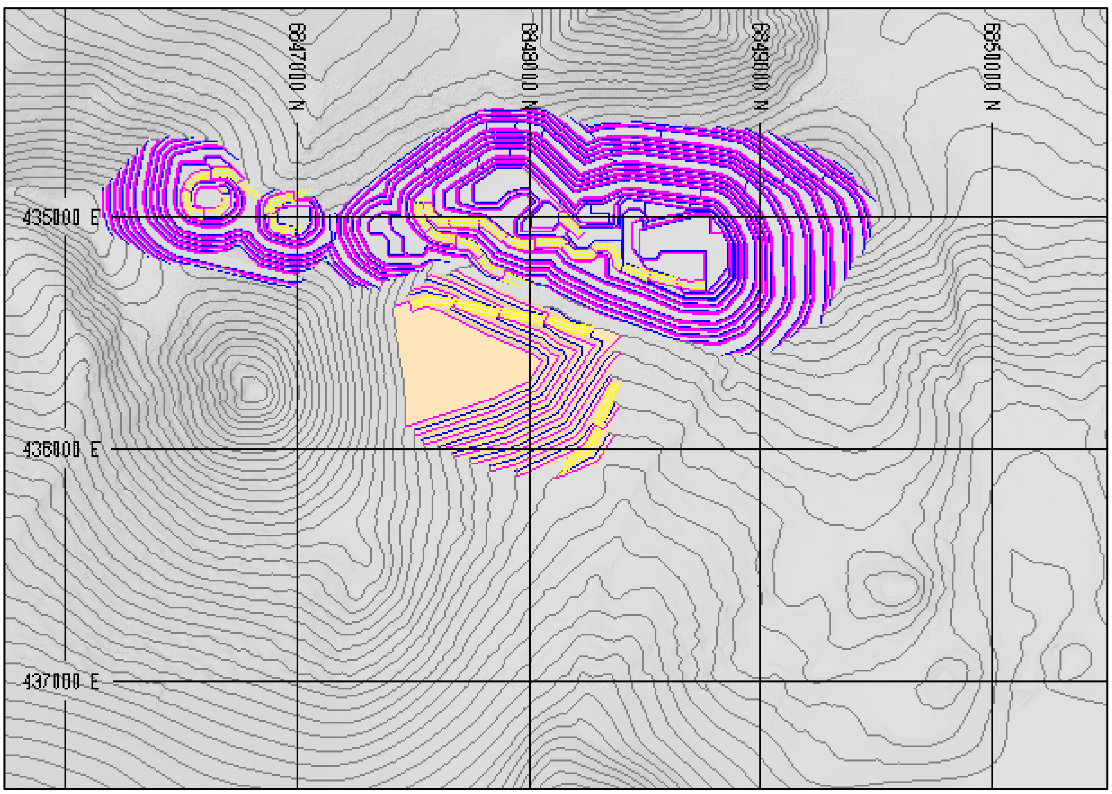


Figure 16-9: Phase 1 Waste Dump

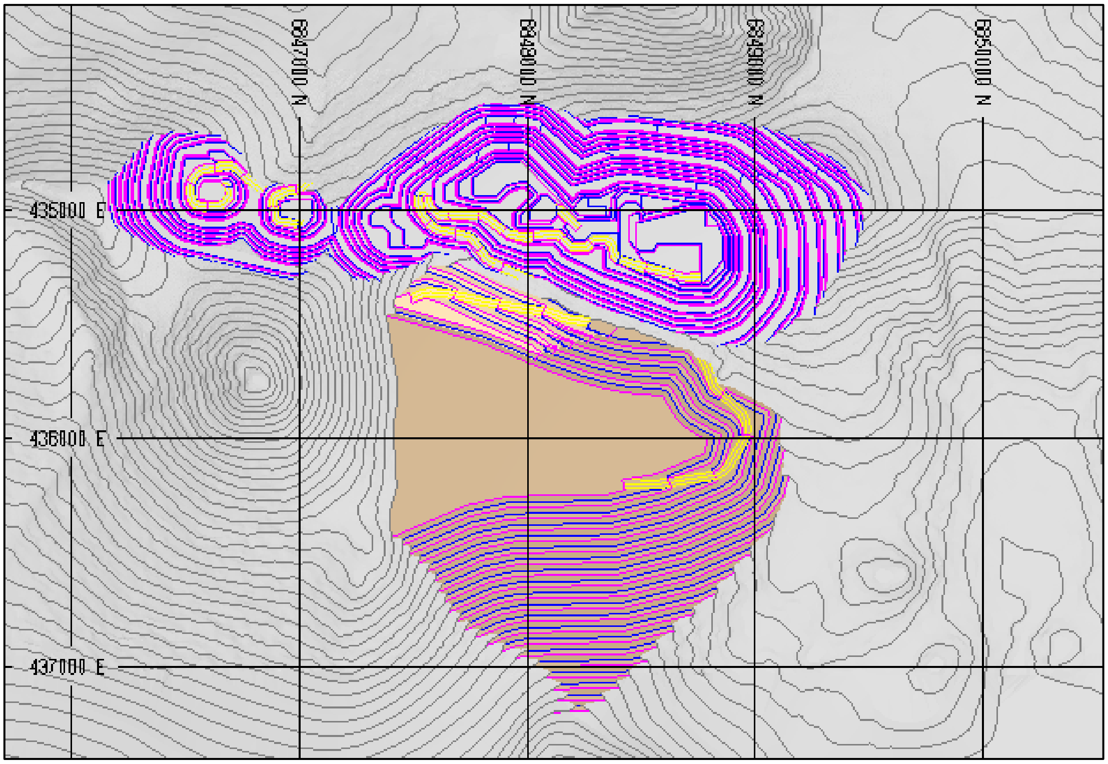


Figure 16-10: Phase 2 Waste Dump

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.

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 Production Schedule

The mine plan presented in this report was developed using MineSight'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 120,000m^3 of key excavation at the toe of the phase 1 dump, and dump underdrain installation
- Strip 30.8 Mt of waste rock from Filo phase 1, exposing sufficient ore to allow continuous ore delivery during production

- Stockpile 231 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 16,240 kt, which is inclusive of the pre-production stockpiled ore reclaim. In production year 2 through 12, the full 21,900 kt (60,000 tpd) are delivered to the crusher area. The last year of production, year 13, is a partial year with 1,928 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 tramming as required. The peak mining capacity of 65 million tonnes per annum or 178.1 kt/d is reached in years 2, 3 and 4. The material moved from the mine is shown in Figure 16-11 below, and material delivered to final destinations is shown in Table 16-4 below.

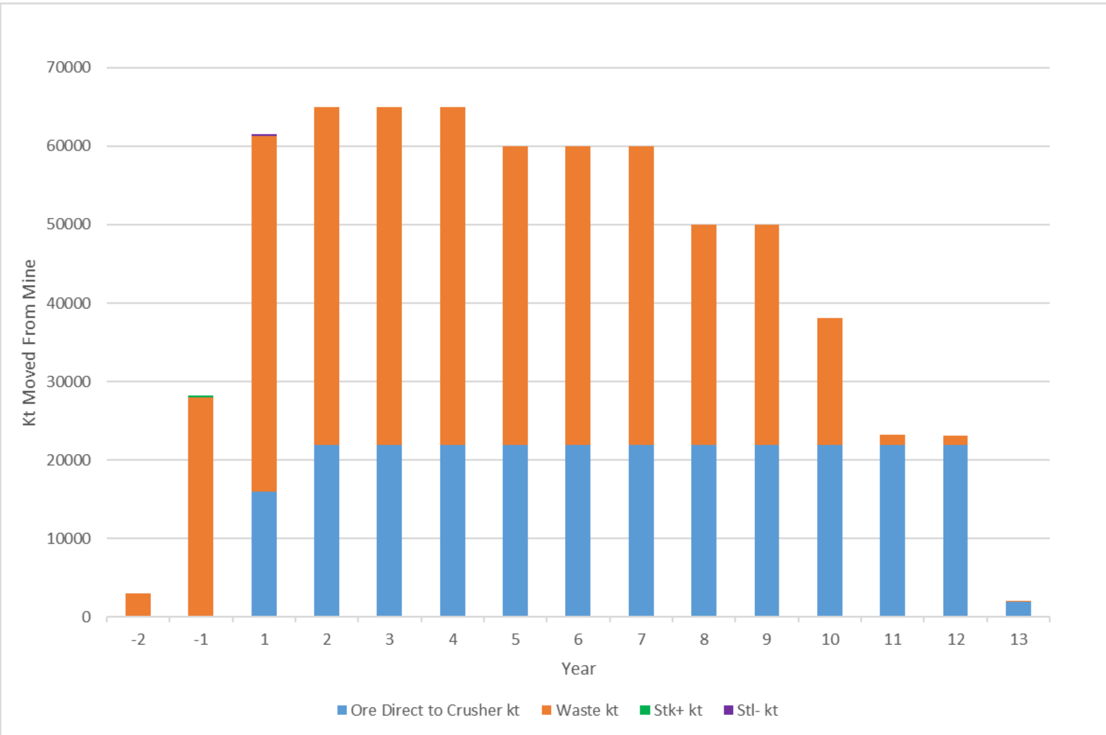


Figure 16-11: Material Moved from Mine

Table 16-4: Material Delivered to Final Destination

	Ore Delivered to Crusher					Waste	Total Material	SR
Years	**kt**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**	**NVPT ($/t)**	**kt**	**kt**	**W/O**
-2						3,000	3,000	

	Ore Delivered to Crusher					Waste	Total Material	SR
-1						27,769	27,769	
1	16,250	0.29	0.39	6.2	19.37	45,243	61,493	2.78
2	21,900	0.40	0.34	5.6	21.35	43,100	65,000	1.97
3	21,900	0.44	0.34	5.3	21.98	43,100	65,000	1.97
4	21,900	0.41	0.38	17.4	30.89	43,100	65,000	1.97
5	21,900	0.43	0.33	29.1	35.86	38,100	60,000	1.74
6	21,900	0.40	0.28	12.7	23.54	38,100	60,000	1.74
7	21,900	0.41	0.23	2.1	17.09	38,100	60,000	1.74
8	21,900	0.36	0.29	8.0	17.75	28,100	50,000	1.28
9	21,900	0.40	0.34	25.8	31.08	28,100	50,000	1.28
10	21,900	0.34	0.30	11.7	20.08	16,205	38,105	0.74
11	21,900	0.42	0.37	30.4	34.84	1,290	23,190	0.06
12	21,900	0.36	0.40	25.8	29.43	1,219	23,119	0.06
13	1,928	0.29	0.31	4.9	11.53	1	1,929	0.00
Total	259,078	0.39	0.33	15.1	25.30	394,527	653,605	1.52

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.

Table 16-5: Primary Production Equipment

Equipment Type	Equipment Class	Maximum Fleet Size
Haul Truck	220 t	27
Hydraulic Shovel	26 m^3	3
Front End Loader	18 m^3	2
Track Dozer	4.7 m blade	2
Grader	4.9 m blade	2
Rubber Tired Dozer	5.2 m blade	1

Equipment Type	Equipment Class	Maximum Fleet Size
Support Backhoe	3.0 m^3	1
Water Truck	136 t	2
Blast Hole Drill	34,000 kg pulldown, 200 mm bit	7
Small Drill	22 t operating wt., 140 mm bit	1

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.

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 mt 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, ad-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. Operation 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 50% improvement in tire life (from 4,500 hrs to 6,750 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 $19.8 million to the mine operations department. Further savings are recognised 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 27 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.

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, in order 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 P_{80} 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 7 drills are required. A secondary drill capable of drilling a 140mm 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.

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. 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 in order to provide a full blasting service (priming, loading, stemming, sequencing, firing and magazine management). The blasting supplier will provide two 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

Equipment	Maximum Fleet Size
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 'digable' 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 km2. 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 58 staff. Additionally, there will be 5 blasting contractors and 6 dispatch/autonomous system operators on site at all times. The peak total mine operations workforce in camp is 171 people.

17 Recovery Methods

17.1 Process Description

The process plant for the Filo del Sol project is designed to treat 60,000 tpd of ore through a sequential heap leach process, to produce copper cathodes and gold/silver doré.

Flowsheet selection was based upon results of laboratory test work developed and managed by HydroProc Consultants.

Figure 17-1 shows a simplified schematic of the process flowsheet.

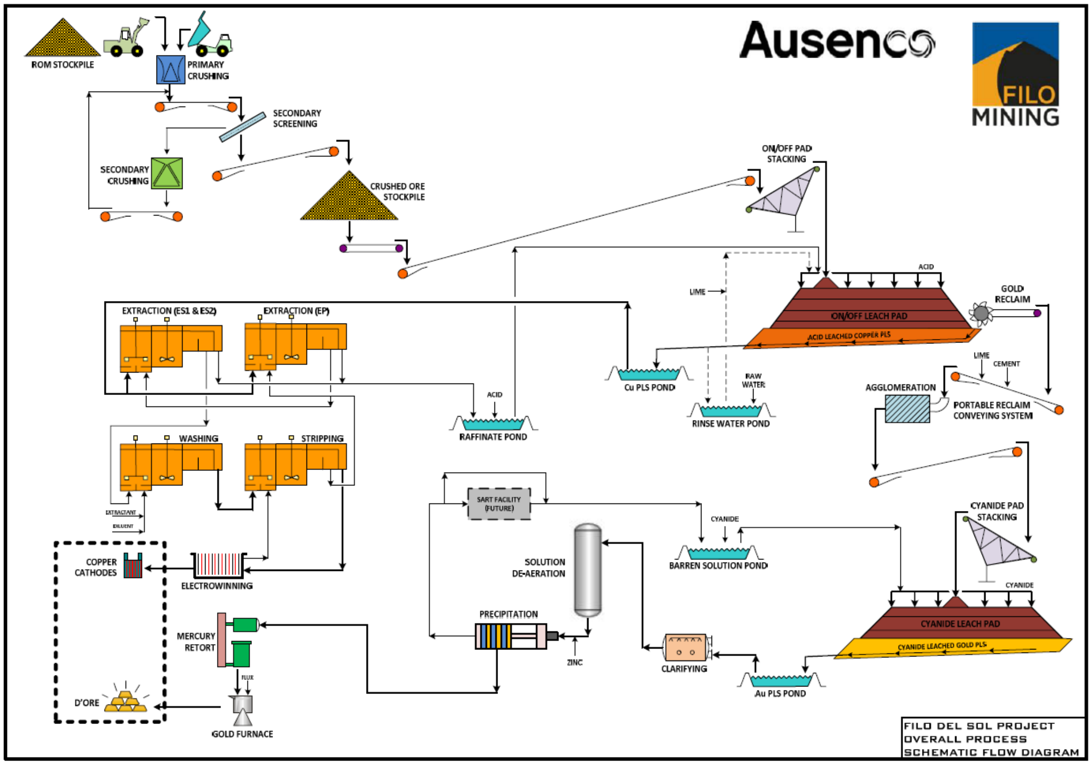


Figure 17-1: Overall Process Schematic Flow Diagram

17.2 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.

17.2.1 Crushing

The crushing plant is designed to operate at a nominal rate of 60,000 tpd. 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.

The two stage crushing circuit will consist of primary gyratory crushing, and twin secondary cone crushing, operating in a closed circuit with vibrating screens for product size control.

17.2.2 Copper On/off Leaching

Crushed ore will be reclaimed from the stockpile, transferred to the on/off leach pad, stacked to a height of 7.5 m, and 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.

For the copper on/off pad, irrigation will be provided by a series of drippers which distribute the acidic leach solution. Pregnant leach solution (PLS) from the heap will flow to the copper on/off PLS pond and will then 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.

17.2.3 Solvent Extraction and Electrowinning

Copper will be extracted from PLS in the solvent extraction (SX) plant, which consists of three stages of extraction, a wash stage and a single strip stage.

Copper will be transferred from the PLS into the organic and will then be removed from the organic using acidic spent electrolyte. This generates 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.

17.2.4 Permanent Cyanide Leaching

Leached ore from the copper on/off pad will be 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.

17.2.5 Gold Recovery

PLS will be pumped through the Merrill-Crowe plant where gold and silver will be recovered by precipitation with zinc powder. The precipitate will be smelted to Doré bars using a smelting furnace and casting machine and the bars will be stored in the gold room safe.

17.2.6 SART

A Sulfidization, 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.

17.2.7 Reagents

Package plants will be provided to supply the following reagents required for the process:

- Lime
- Sulfuric acid
- Salt
- Cobalt sulphate
- Diluent
- Extractant
- Smoothing agent

17.2.8 Services

Air

Instrument and process air will be provided by a packaged air compressor system and a packaged drier. Only instrument air will be dried.

Water

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.

A fire water system will be included with its own tank, electric pump, diesel pump, and jockey pump.

Potable water will be generated as required, from raw water.

A treatment plant will collect raffinate bleed, storm water run-off, 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.

17.3 Design Criteria

The principal criteria used for the design of the processing circuit are summarized in Table 17-1.

Table 17-1: Processing Design Criteria Summary

Item	Design Criteria
Annual tonnage processed	21,900,000 t
Crushing production rate	60,000 t/d (nominal)

Item	Design Criteria
Crusdhing operation	12 h/shift, 2 shifts /day, 7 days /week
Crusher availability	72%
Crushing circuit P80 product size	38 mm
Copper leaching cycle	52 days
Gold and silver leaching cycle	60 days
Average copper recovery	80%
Average gold recovery	70%
Average silver recovery	82%

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 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.

18 Project Infrastructure

18.1 General Site Layout

The overall site plan, included as Figure 18-1 overleaf, shows the general arrangement of the plant, the mine and major infrastructure.

Ausenco

FILO MINING

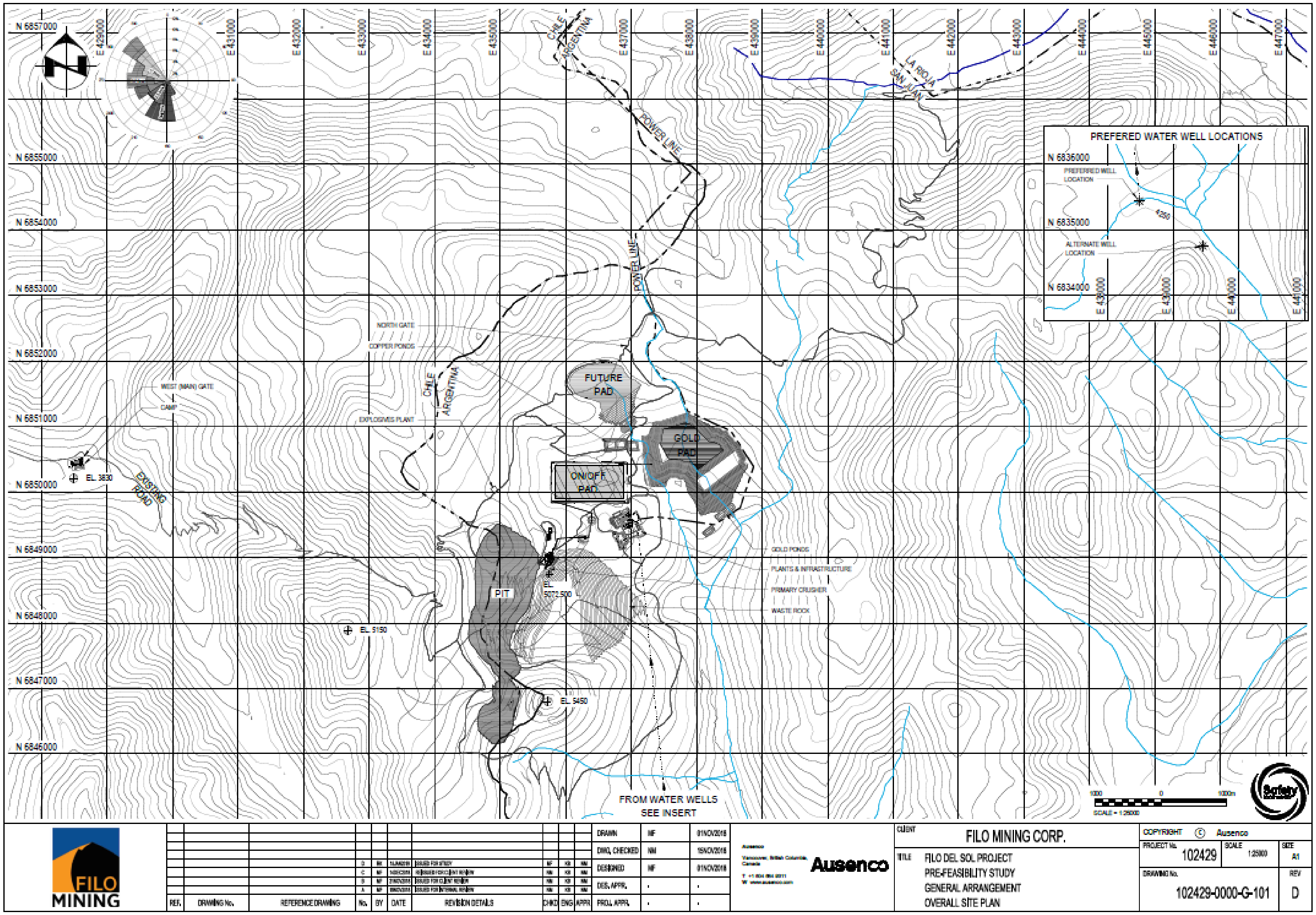


Figure 18-1: Overall Site Plan

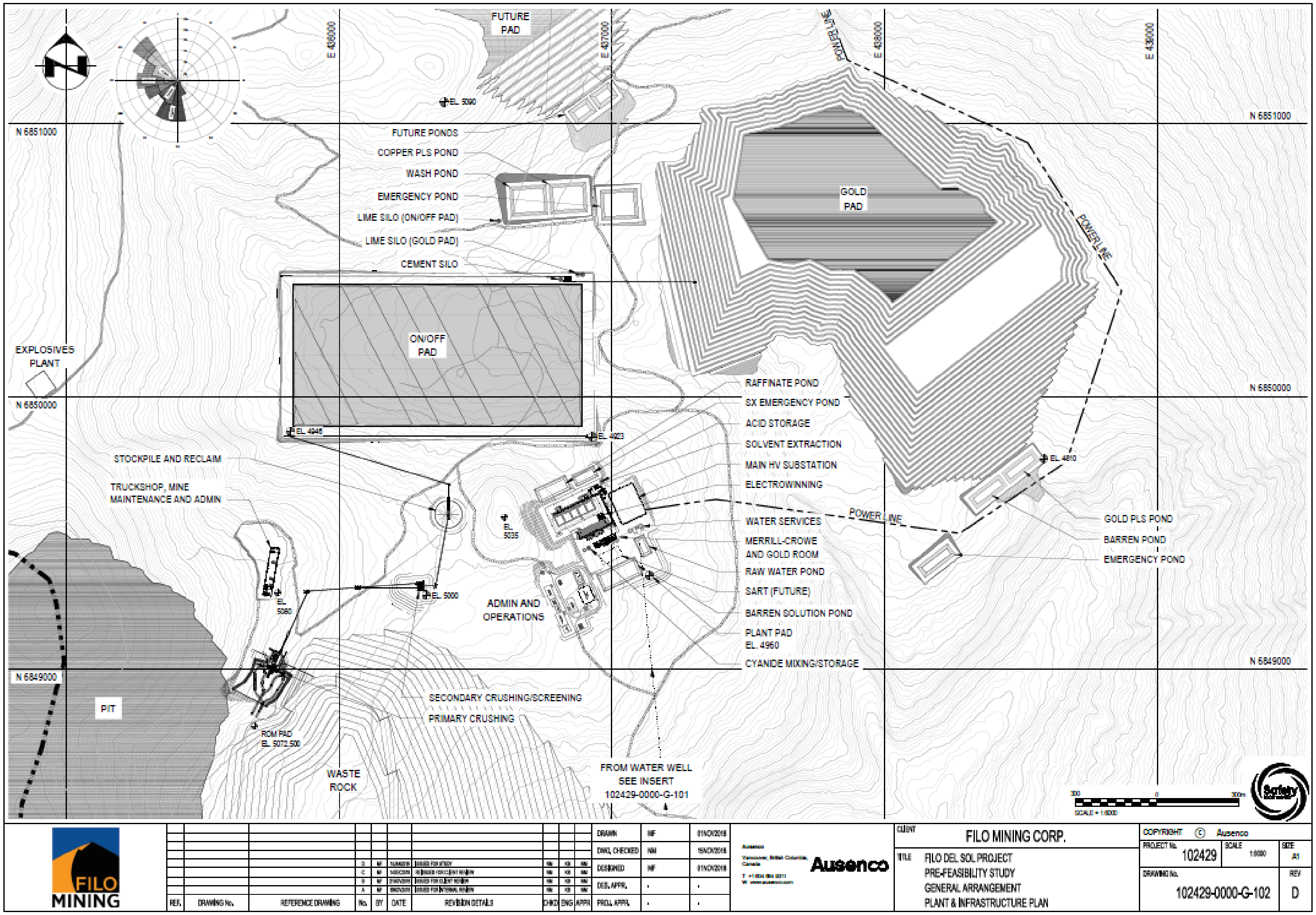


Figure 18-2: General Arrangement

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 carried out at the aforementioned sites.

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

18.3 Road and Logistics

18.3.1 Road

The travel distance between the site and the port is approximately 245 km. Approximately 48 km of light vehicle road will require upgrading to a nine meter-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 Nte) in Copiapó. Ruta 5 passes the city of Caldera, which is located approximately 77 km from Copiapó, and accesses the port of Caldera.

Copper cathode will be transported by flat bed truck to Puerto Caldera, a port near the city of Caldera- and doré will be transported approximately 175 km to Aeropuerto Desierto de Atacama for ongoing airfreight.

Operating consumables required by the mine that have foreign supply will be imported to the port of Caldera. The route to access the mine will be the same as 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

The Port of 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 3,800m 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 also 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 2,500 man 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 about 375 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 52 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 of these substations to the project site and are shown in the Figure 18-3 below.

Figure 18-3: Transmission Line Route Options

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.

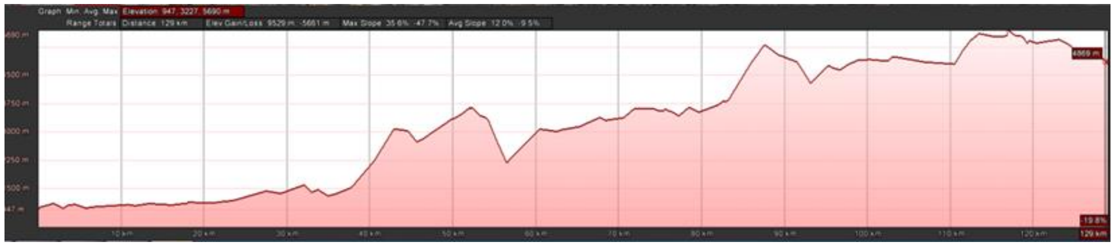


Figure 18-4: Transmission Profile

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 a 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 centralised 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 located in the admin 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 located. 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 centers (MCC), with incoming breakers. The MCC's will be located 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 uninterruptable 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 make-up requirement is estimated to be 75 L/s based on 60,000 tpd 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 the majority 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 make-up 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.

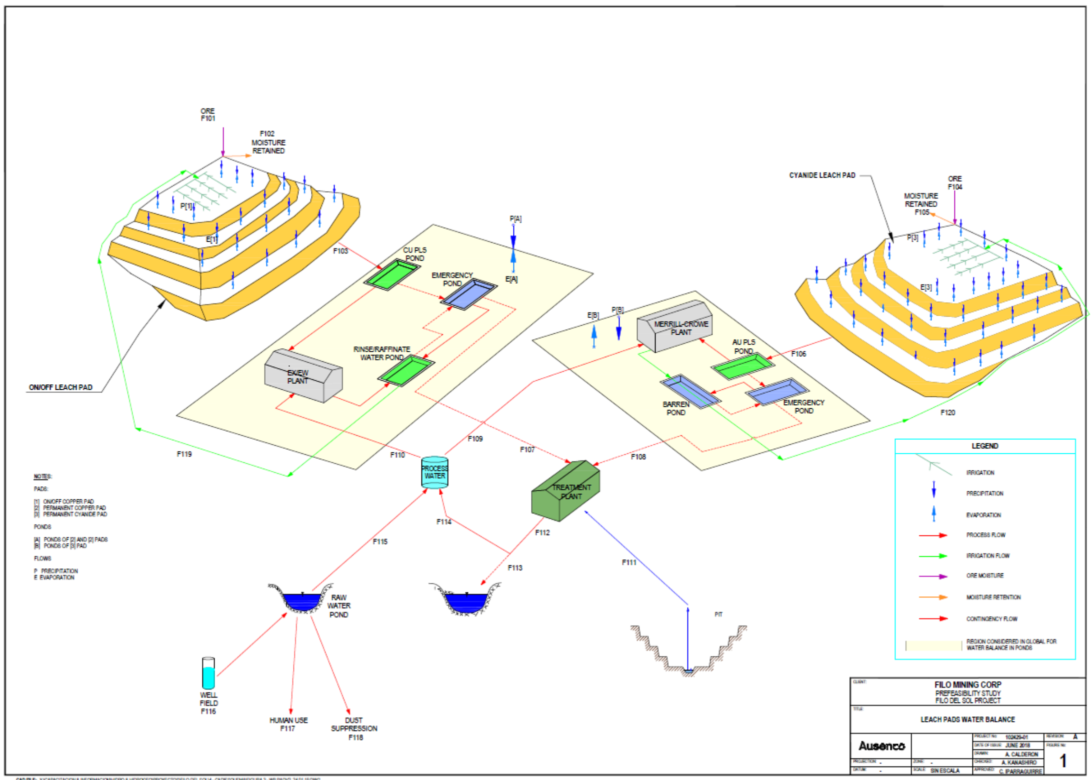


Figure 18-5: Water Balance

The average make-up water requirement for the project is estimated to be 75 L/s based on 60,000 tpd nominal feed rate.

18.9 Surface Water Management

The climate at the project site is classified as sub-tropical semi-arid, typical of the high altitude central Andean Cordillera. This climate is characterized by low precipitation, low relative humidity and high solar radiation, and is notably influenced by cyclical climate 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 Cordillera is highly influenced by the ENSO climate cycles. ENSO cycles are characterized by a tendency for more 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 up to 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 Andeans, 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 the management of surface water 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 run-off 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 storm-water 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 make-up 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 run-off volume as possible in water retaining ponds to diversion structuresfor 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 run-off from disturbed sites.

Surface run-off which can be intercepted and directed by the diversion works will be considered to be non-contact water. Any water stream run-off 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 run-off 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 fresh water 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 run-off 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 are provided, one for the ono/ff 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 pre-feasibility 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 type 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 Argenitnean 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 permeanant 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 Merril-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 permenant 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. Pre-feasibility 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.
- In order 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 feasbility 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 off loading (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 maximising 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 andFigure 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 in order 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.

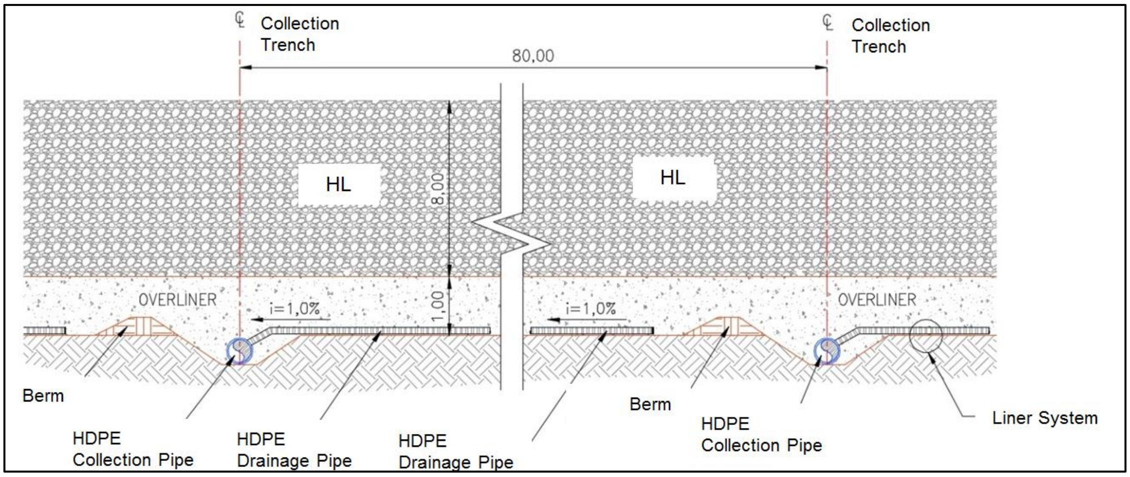


Figure 18-6: HLF Drainage System Detail

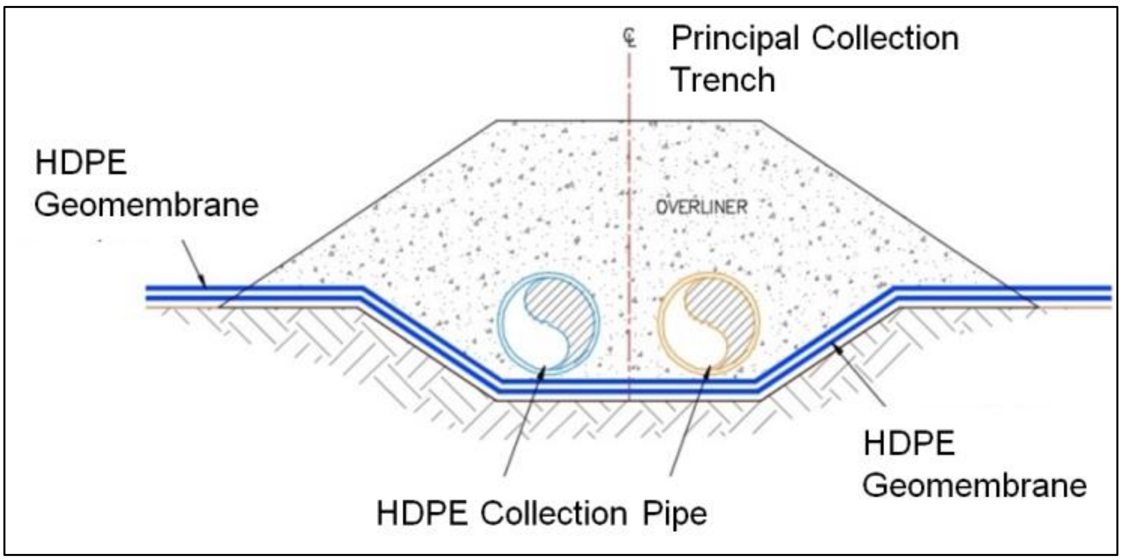


Figure 18-7: HLF Collection System Detail

The HLF external collection pipes will be covered with over-liner 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 liftoff 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 storm water 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 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,

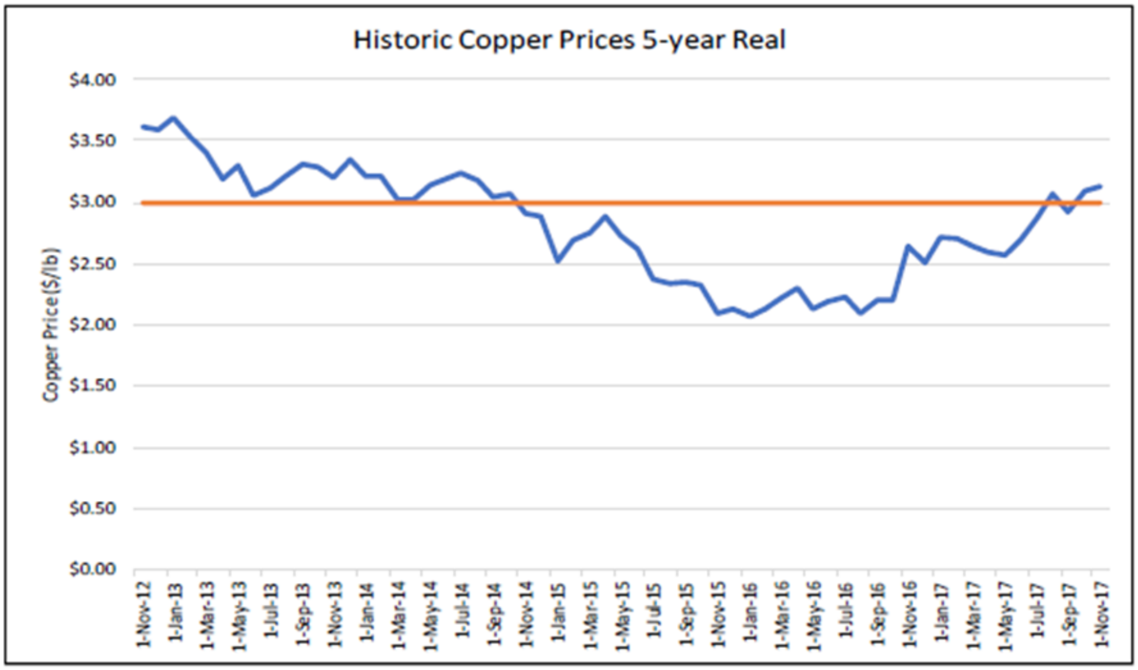
Figure 19-2 and Figure 19-3). Ausenco also considers the prices used in this study to be consistent with the range of prices being used for other project studies.

Note that the copper price excludes a 1.5% 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.

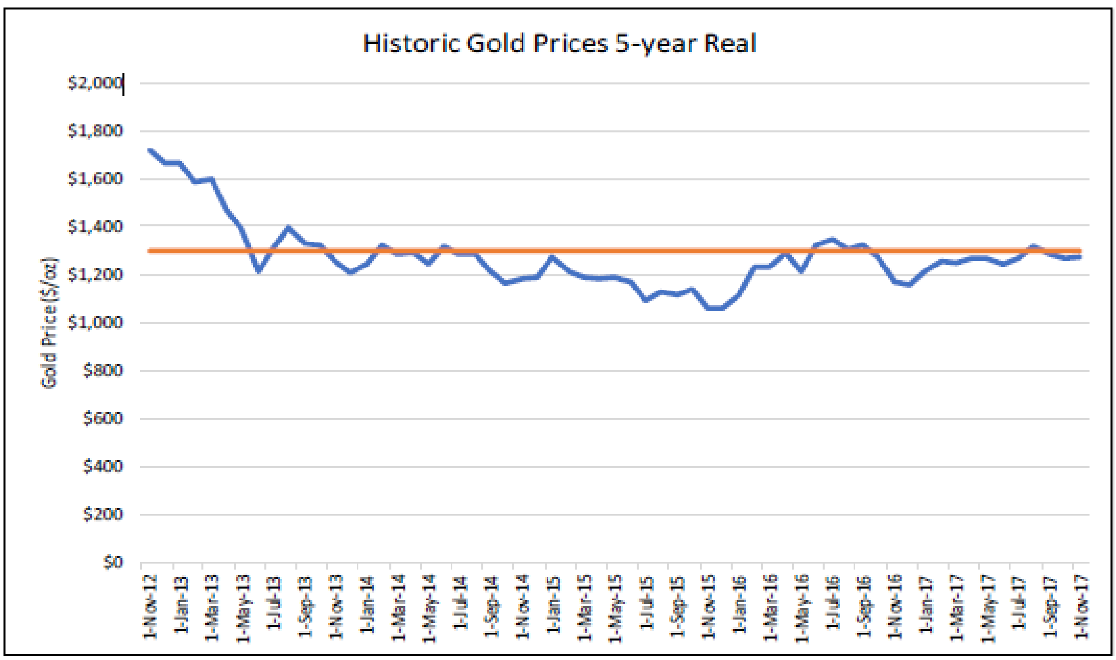
Table 19-1 : Pricing Assumptions for Economic Analysis

Commodity	Price
Copper (Cu)	$3.00 per pound (lb)
Gold (Au)	$1300 per ounce (oz)
Silver (Ag)	$20.00 per ounce (oz)



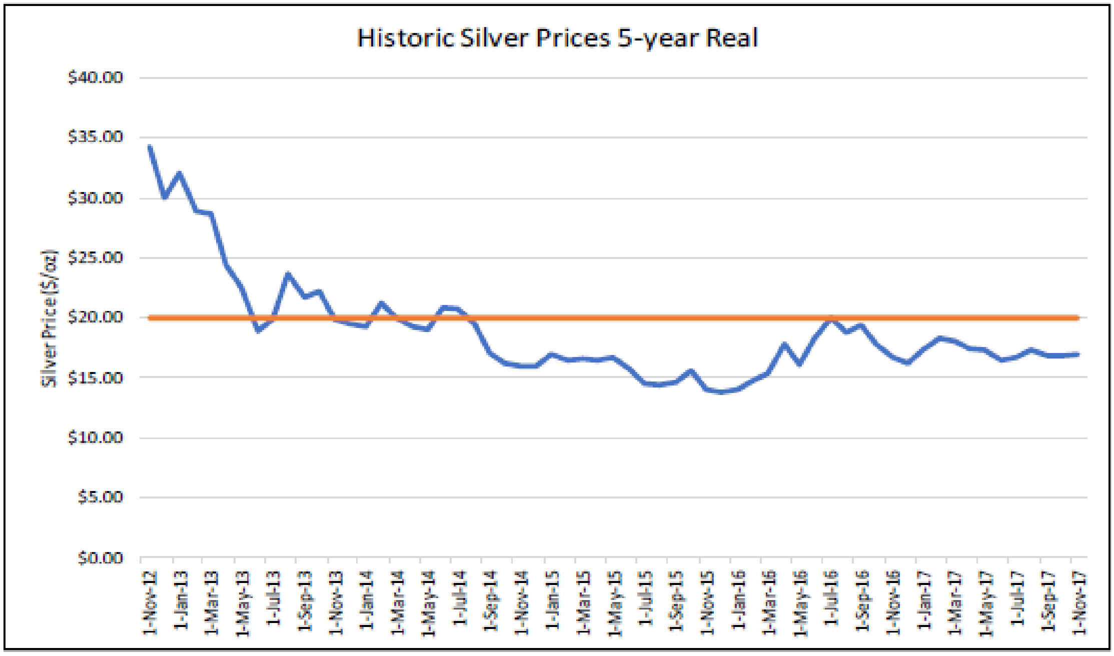
Source: SRK, 2018

Figure 19-1: Historic Copper Prices



Source: SRK, 2018

Figure 19-2: Historic Gold Prices



Source: SRK, 2018

Figure 19-3: Historic Silver Prices

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 summarised 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 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 in order to facilitate the advancement of 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 29 March 2017 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 modeling, 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 an Informe Consolidado de Solicitud de Aclaraciones, Rectificaciones y Ampliaciones, or 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 of 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.

20.3 Environmental Studies

A summary of the results of the environmental studies conducted to date is provided below.

20.3.1 Meteorology

Recent 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 neighboring 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 of 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 December 2017. The mean annual temperature for the Project area was -5.4 °C for the period of 2015 to 2017. For the same period, the maximum and minimum hourly air temperatures were 12.0 °C and -25.6 °C, 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 (ºc) 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

Year	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
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

Year	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
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.

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**

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.

20.3.1.4 Precipitation

Precipitation at Filo del Sol is an infrequent occurrence, with very 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-Term 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

Year	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
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

Year	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
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.

20.3.3 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.

20.3.4 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.

20.3.5 Geochemistry

20.3.5.1 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.

20.3.5.2 Analytical Program

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), sulfate 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.

Kinetic Tests

Kinetic tests are long-term leach tests that provide insight into the weathering characteristics of materials over time including NP depletion, sulfide 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, sulfate, chloride, fluoride), nutrients (ammonia, nitrate, nitrite, phosphorus) and dissolved metals by ICP-MS.

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.

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, micas etc. 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 suphide 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 CaCO3/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 CaCO3/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 sulfide 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%).

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.

20.3.6 Water Quality and Aquatic Biota

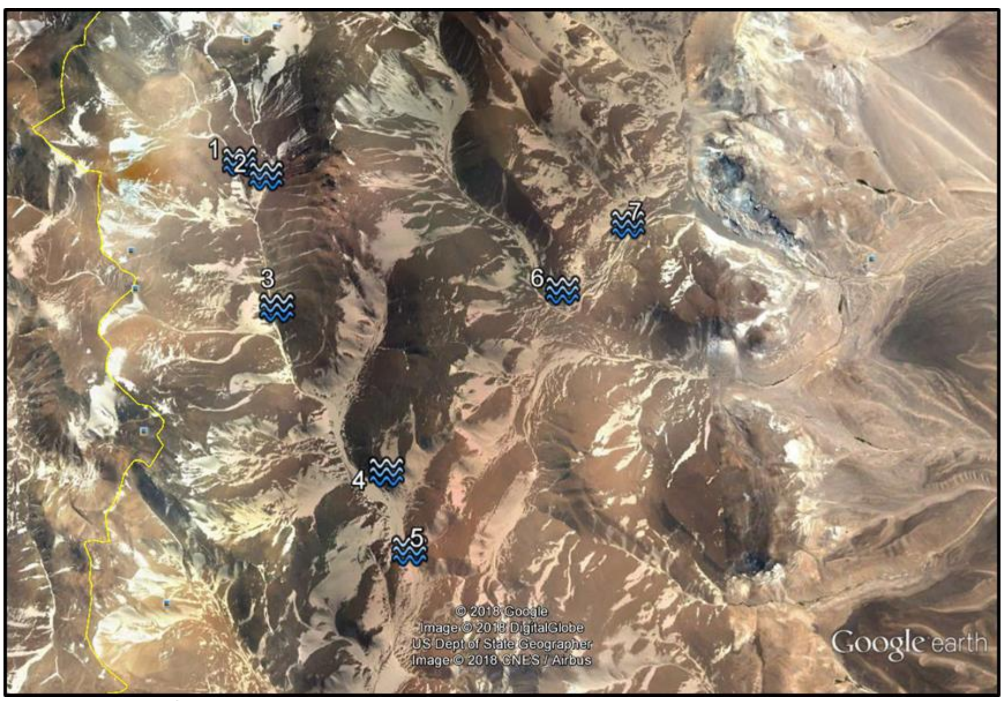
A focused study program for the Filo del Sol project was carried out by Knight Piésold (2018b 2018c, and 2018d), which followed several previous regional studies. Sites throughout the area and in downstream catchments were sampled for water quality and for invertebrates and phytoplankton. Sample locations 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.



Source: Knight Piésold, 2018

Figure 20-1: Water Quality and Aquatic Biota Sampling Locations

20.3.7 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.

20.3.8 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 4,700m 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 diguse Stipa spp grasses

Figure 20-3: Vega in the riparian zones of the lower Mogotes River dominated by *Oxychloe castellanosii*

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.

20.3.9 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 in excess of 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

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 150km 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 they 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 $51M 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	500,000	7.5
Access control / safety berm around pit	m	5,000	200	1.0
Retraining waste dump diversion ditches to the pit	km	2	350,000	0.7
Rinsing of Heap Leach Pads	year	4	200,000	0.8
Re-profiling and placement of evaporative earthen cover	ha	282	30,000	8.5
Scarification and contouring of the footprint	ha	27	200,000	5.4
Scarification and contouring of the internal access roads	km	80	6,000	0.5
Dismantling of electrical transmission line	km	140	2,500	0.4
Detailed closure engineering and planning	Study	1	800,000	0.8
Active closure monitoring	year	2	190,000	0.4
Post closure monitoring	year	5	165,000	0.8

Closure Aspect	Unit	Quantity	Unit Cost ($)	Total ($M)
Misc. (Waste management and disposal, specialist contracts, etc.)	Lump Sum	1	1,200,000	1.2
			Subtotal of Direct Costs	**28.0**
Indirect Costs				
Contractor Fees (25% of Direct Costs)				6.9
Administration (15% of Direct Costs)				4.2
			Subtotal of Indirect Costs	**11.1**
Subtotal				**39.1**
Contingency at 30%				11.7
Total Closure Costs (USD)				**50.8**

21 Capital and Operating Costs

21.1 Summary of Capital Cost Estimates

Capital costs were estimated from a variety of sources including derivation from first principles, equipment quotes and factoring from actual costs incurred in the construction of other similar facilities. Costs are estimated to an accuracy of +/- 25% which is equivalent to an AACE International, Class 4 Estimate. Capital costs are summarised in Table 21-1 and show initial costs of $1,266 million, with sustaining costs of $217 million and closure costs of $51 million, for a life-of-mine total capital cost of $1,534 million.

Table 21-1: Capital Cost Summary

Estimated Capital Costs	($M)
Mine Pre-strip	59
Mining	121
Crushing	67
Processing	325
On-Site Infrastructure	94
Off-Site Infrastructure	124
Total Direct Costs	**789**

Estimated Capital Costs	($M)
Indirect Costs	132
Project Delivery	101
Owner's Costs	50
Contingency	194
TOTAL INITIAL CAPEX	**1,266**
LOM Sustaining Capital	217
Closure	51
Total Life of Mine Capital	**1,534**

21.1.1 Initial Capital Cost

The Initial capital cost estimate has been summarized at the levels indicated by the following tables and stated in United States Dollars (USD) with a base date of 4th Quarter 2018 and with no provision for forward escalation.

Initial capital costs of $1,266 million are shown in various formats in the following tables:

- By major areas (Level 1): Table 21-2
- By major disciplines: Table 21-3
- Summary (Level 2): Table 21-4

Table 21-2: Initial Capital Estimate Summary Level 1 Major Area

Cost Type	WBS LVL 1	LVL 1 Description	Total ($M)
Direct	1000	MINE	**180**
	3000	PROCESS PLANT	**392**
	4000	ON-SITE INFRASTRUCTURE	**94**
	5000	OFF-SITE INFRASTRUCTURE	**123**
		Direct Subtotal	**789**
Indirect	6000	INDIRECTS	**132**
	7000	EPCM SERVICES (PROJECT DELIVERY)	**101**
	8000	OWNERS COSTS	**50**
	9000	PROVISIONS (Contingency)	**194**
		Indirect Total	**477**
PROJECT TOTAL			**1,266**

Table 21-3: Initial Capital Estimate by Major Discipline

Disc.	WBS Description	Total Cost ($M)
A	SITE DEVELOPMENT	88
B	EARTHWORKS	69
C	CONCRETE	32
D	STRUCTURAL STEEL	21
E	ARCHITECTURAL	16
F	PLATEWORK	7
G	MECHANICAL EQUIPMENT	188
H	MOBILE EQUIPMENT	4
I	PAINTING AND COATINGS	1
J	PIPING	54
K	ELECTRICAL EQUIPMENT	106
L	ELECTRICAL BULKS	22
M	INSTRUMENTATION	7
R	THIRD PARTY ESTIMATES (mining)	172
	Subtotal Direct Costs	**788**
S	FIELD INDIRECTS	103
T	SPARES & FIRST FILLS	20
U	VENDORS	10
V	EPCM SERVICES (PROJECT DELIVERY)	101
W	OWNER's COSTS	50
y	PROVISIONS	194
	Subtotal Indirect Costs	**478**
	PROJECT TOTAL	**1,266**

Table 21-4: Initial Capital Estimate Summary (Level 2)

Cost Type	WBS LVL 2	LVL 2 Description	Total ($M)
Direct	1100	MINE DEVELOPMENT SURFACE	59
	1200	DEWATERING	0
	1300	MINING EQUIPMENT	114
	1400	ANCILLARY SERVICES	7
	1500	MINE EXPLOSIVES MAGAZINE	0
	3100	CRUSHING	68
	3200	COPPER ON/OFF CIRCUIT	113
	3400	COPPER PROCESSING (SX-EW)	105
	3500	GOLD CIRCUIT	75
	3600	GOLD PROCESSING (MERRILL-CROWE)	31
	3700	SART (future)	0
	4100	SITE DEVELOPMENT	27
	4200	POWER SUPPLY AND DISTRIBUTION	23
	4300	UTILITIES	24
	4400	GENERAL BUILDINGS	5
	4500	PLANT BUILDINGS	10
	4600	MOBILE EQUIPMENT	4
	5100	OFF-SITE ROADS	43
	5200	POWER SUPPLY	43
	5300	WATER SUPPLY	33
	5400	PERMANENT ACCOMMODATION	4.
		Subtotal Direct Costs	**789**
Indirect	6100	FIELD INDIRECTS	26
	6200	HEAVY LIFT CRANES	3
	6300	ACCOMMODATION & MESSING	73
	6400	VENDOR REPRESENTATIVES	10
	6600	SPARES AND FIRST FILLS	20
	7100	PROJECT DELIVERY (EPCM SERVICES)	97

Cost Type	WBS LVL 2	LVL 2 Description	Total ($M)
	7300	PROJECT DELIVERY (EPCM EXPENSES)	5
	8100	OWNER'S COSTS	2
	8200	PERMITTING, SOCIAL AND ENVIRONMENTAL	6
	8500	LAND	2
	8600	PRE-PRODUCTION COSTS	37
	8700	FINANCING	3
	9100	CONTINGENCY	194
	9200	CLIENT CONTINGENCY (RISK)	Excluded
	9300	FOREX	Excluded
	9400	ESCALATION	Excluded
		Subtotal Indirect Costs	**477**
PROJECT TOTAL			**1,266**

21.1.2 Sustaining Capital Costs

The Sustaining capital cost estimate has been summarized at the levels indicated by the following table and stated in United States Dollars (USD) with a base date of 4th Quarter 2018 and with no provision for forward escalation.

Table 21-5: Sustaining Capital by Major Area

Cost Type	WBS LVL 1	LVL 1 Description	Total ($M)
Direct	1000	MINE	120
	3000	PROCESS	97
	4000	ON-SITE INFRASTRUCTURE	0
	5000	OFF-SITE INFRASTRUCTURE	0
TOTAL SUSTAINING			**217**

21.2 Definition of Capital Costs

21.2.1 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.

21.2.2 Methodology General

The estimate is developed based on a mix of 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.

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
- finalized the estimate, estimate basis and obtained sign off by the Study Manager

21.2.3 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

21.2.4 Basic Information

The following basic information pertains to the estimate:

- the estimate base date is Q4, 2018
- 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

21.2.5 Estimate Classification

The estimate has been prepared in accordance with the recommended practices of the American Association of Cost Engineers (AACE) and is classified as an AACE and Ausenco Class 4 Pre-Feasibility Study estimate with an accuracy range of +/-25%. The typical purpose of the estimate will be for budgetary, viability purposes, to determine validity of a business case or option validation and assessment.

21.2.6 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 September 14th, 2018 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 = 1.30 CAD
EURO	Euro	1 USD = 0.88 EUR
GPB	Great Britain Pound	1 USD = 0.77 GPB
ARS	Argentine Peso	1 USD = 39.90 ARS

Code	Currency	Exchange Rate
AUD	Australian Dollar	1 USD = 1.40 AUD
CLP	Chilean Peso	1 USD = 684.93 CLP

The following Table 21-7 identifies the foreign priced content and USD priced content.

Table 21-7: Foreign and USD Priced Content

Country	Initial CAPEX ($M) (excl contingency)	% of Costs (excl contingency)
United States priced content	1,024.1	96%
Canadian priced content	0.2	0.02%
European priced content	27.3	25%
Great Britain priced content	0	0%
Argentine priced content	20.4	1.9%
Australian priced content	0.6	0.06%
Chilean priced content	0	0%
Total – Directs and Indirects (less contingency)	**1,072.8**	**100%**

21.2.7 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 Operating Cost Estimates

21.3.1 Summary

The operating costs are estimated, C1 cash costs (co-product basis), over the life of mine at an average of $1.23/lb CuEq. C1 cash costs include at-mine cash operating costs, treatment and refining charges, royalties, selling costs, and transportation costs and are reported on a $/equivalent payable unit of the primary metal.

All costs are presented in US dollars.

Table 21-8 below summarises the operating costs, including mining, processing and G&A. Average cost is $ 14.19/t ore processed.

Table 21-8: Operating Cost Estimate Summary

Operating Cost	$/t Processed
Mining	3.86
Processing	8.90
Site G&A	1.44
TOTAL	**14.19**

Costs are presented across the life-of-mine (LOM), including costs for the SART plant from Year 2.

Operating cost estimates are accurate to within ±25%.

Contingency was not included in the operating cost estimate.

Note that in some tables the totals presented may differ from totals on individual table values because of rounding.

21.3.2 Mining

Mining costs were estimated by building up the cost estimate over the LOM and presenting an average annual cost of $ 84 million, which is equivalent to $3.86/t processed.

The build-up is summarised in Table 21-9, which shows the cost components relevant to each cost centre. Costs were developed for each year of operation.

Mining 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 57 to 58, and mine labour varies by the year and ranges between 100 and 250, 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, 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.

Costing is based on the following inputs:

- Diesel price at $0.62 /L based on Filo Mining's long range diesel price forecast
- Electricity price at $0.075 based on Filo Mining's long range power price forecast

Table 21-9: Mining Costs (LOM)

Centre	Salaries and Wages ($M)	Fuel and Power ($M)	Consumables ($M)	Totals ($M)
General	56	0	6	62
Drilling	15	1919	77	111
Blasting	0	0	196	196
Loading	16	45	72	132
Hauling	29	129	202	361
Support	16	22	36	74
Sundry costs				63
TOTAL	**132**	**215**	**589**	**999**

Total mining cost for the LOM is $999 million. Total ore crushed is 259 million tonnes. Mining costs are calculated as $3.86/ t processed.

21.3.3 Processing

Processing costs consist of costs for power, consumables maintenance and labour, as summarised in Table 21-10.

Table 21-10: Processing Costs

Processing Cost Item	Annual Cost ($M)	Annual Cost ($/t processed)
Power	37	1.68
Consumables	134	6.13
Maintenance	17	0.77
Labour	7	0.32
TOTAL	**195**	**8.90**

Basis of Estimate

Processing operating costs were estimated based on:

- Filo Mining's recommendations for fuel costs, electricity, miscellaneous expenses, reviewed against Ausenco's database for reasonableness.
- Labour rates and rotation schedule were established based on Ausenco Argentina and Chile offices information.
- Fixed 60,000 tpd mine production schedule was used for the OPEX estimation.

- Operating consumables are based on benchmarks from similar operations and from vendor information, and reagent usages are calculated based on SGS 2018 test results.

All costs are presented in US dollars, unless stated otherwise.

Inclusions

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
- 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 $0.075/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.

21.3.3.1 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 utilisation and load factors.

Unit cost of power was supplied by Filo Mining at $0.075/kWh.

Annual power consumption is shown in Table 21-12 at a total annual consumption of 490 815 MWh which costs $36.8 million. This is equivalent to $1.68/t processed.

Average on-line power demand is 52 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 that the costs for the future installation of the SART plant have been included and averaged over the life-of-mine.

21.3.3.2 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 summarised in Table 21-13 below, which shows individual costs for consumables, reagents and SART operating costs. The table also show $14.9 million for transport of consumables and reagents, taken as 12% of supply cost. Total annual cost is $134 million, which is equivalent to $6.13/t processed.

Table 21-13: Operating Costs - Consumables and Reagents

	Annual Costs ($M)
Consumables	5.7
Reagents	105.8
SART	7.8
Transport	14.9
TOTAL	**134.2**

Table 21-14: Consumables

LOM Consumables	Annual Consumption	Annual Cost ($M)
Primary Crusher Bowl/Mantles/etc.	1 set	0.6
Coarse Screen Top deck	8 decks	0.3
Coarse Screen Bottom deck	8 decks	0.2
Secondary Crusher Bowl/Mantles/etc.	2 sets	0.9
Lime Slaker - Consumable Parts	1 set	0.02
Lime Slaker - Balls	34 t/y	0.03
Anodes	361 units	0.2
Cathodes (incl. edge strips)	505 units	0.2
Diesel (Hot Water Cathode Wash)	639 kL	0.4
Mist Suppression (Beads)	53 m^3	0.7
Mobile Equipment fuel	3249 kL	2.0
Lab consumables	(allowance)	0.15
TOTAL		**5.7**

Table 21-15: Reagents

Reagent	Annual Consumption (t/y)	Annual Cost ($M)
Sulfuric acid	5,595	0.5
Cement	87,600	17.5
Sodium cyanide	30,240	69.8
Lime	83,111	5.7
Extractant	134	1.0
Diluent	1,411	2.7
Clay	43	0.03
Anthracite	23	0.01
Garnet Sand	59	0.03
Filter Sand	59	0.01
Smoothing Agent	13	0.3
Cobalt Sulphate	1	0.01
Salt	5	0.0
Antiscalant	110	0.6
Zinc Dust	1,643	6.7
Lead Nitrate	164	0.4
Gold room reagents		0.6
TOTAL		**105.7**

Table 21-16: SART Operating Costs

Reagent	Reagent Use (t/y)	Annual Cost ($M)
Sodium hydrosulphide	2,915	3.6
Lime (hydrated)	6,200	1.3
Sulfuric Acid	12,770	1.2
Other Consumables		1.7
TOTAL		**7.8**

Costs for the SART plant were derived from reported values.

21.3.3.3 Maintenance

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 concentrator 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, e.g. steel plate and general liners, miscellaneous structural steel, etc.

The plant maintenance spares and consumables exclude:

- Maintenance labour, which is included under labour costs.
- Special wear parts 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.

Note that 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 ($)	Factor of Equipment Cost	Annual Maintenance ($M)
3100	Crushing	28,203,872	8%	2.3
3200	Copper On/Off Circuit	35,674,647	8%	2.8
3400	Copper Processing (SX-EW)	40,540,607	8%	3.2
3500	Gold Circuit	45,200,446	8%	3.6
3600	Gold Processing (Merrill-Crowe)	8,091,970	8%	0.6
4300	Reagents & Services	4,828,172	8%	0.4
4000	On-Site Infrastructure	599,031	4%	0.02
5000	Off-Site Infrastructure	746,820	4%	0.03
1400	Ancillary Services	164,902	4%	0.0

WBS	Area Description	Total Equipment Cost ($)	Factor of Equipment Cost	Annual Maintenance ($M)
future	SART	47,000,000	8% (ave)	3.2
	Mobile equipment maintenance	719,712		0.7
TOTAL				**16.9**

21.3.3.4 Labour

Labour costs include all processing and maintenance costs.

Labour costs were based on salaries and labour rosters provided by Ausenco Argentina office and Filo Mining.

- On-costs: (payroll burdens) 32.5% was selected as advised by Ausenco Argentina office
- All personnel are on shift schedule working 12 hours per day, 2 weeks on and 2 weeks off.
- Regular pay for the first 2080 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 summarised in Table 21-18 below.

Table 21-18: Operating Costs - Labour

Cost Centre	Number of People	Annual Cost ($M)
Plant management	28	1.9
Shift crew	96	2.7
Laboratory and refinery	38	1.1
Maintenance	36	1.1
Mobile equipment operators	4	0.2
TOTAL	**198**	**7.0**

21.3.4 Site G&A

Operating cost estimates for G&A were prepared by Ausenco and were confirmed by Filo MIning.

The G&A costs include camp operations, G&A personnel, off-site offices as well as miscellaneous project costs. An annual G&A operating cost of $31.1M was estimated.

Table 21-19: G&A Cost Summary

Cost Centre	Annual Cost ($M)	Annual Cost ($ /t processed)
Labour	3.8	0.18
Processing and operations	0.6	0.03
Administration and other costs	10.0	0.46
Contracts	16.7	0.76
Mobile equipment maintenance	0.2	0.01
TOTAL	**31.2**	**1.44**

The majority of G&A costs are based on benchmarked data from similar projects in South America. The following sections describe the build-up of department's cost in the G&A area.

21.3.4.1 Labour

Annual labour costs for G&A were estimated at $3.9 million for 86 personnel.

Costs include 32.5% on-costs.

Table 21-20: G&A Costs - Labour

Cost Centre	Number of People	Annual Cost ($M)
Admin including General Manager	5	0.7
Supply and clerks	9	1.0
IT support	4	0.1
Emergency and first aid	5	0.2
Camp manager	1	0.1
Transportation	18	0.8
HR	3	0.1
Environmental management	5	0.1
Environmental and Sustainability	20	0.7
TOTAL	**86**	**3.9**

21.3.4.2 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 $200/person/year is allowed for safety equipment.

The cost items are summarised in Table 21-21.

Table 21-21: G&A Costs - Processing and Operations

Cost Centre	Annual Cost ($M)
Metallurgical test work	0.1
Training	0.3
Safety equipment	0.06
Environmental test work	0.05
Laboratory equipment maintenance	0.05
TOTAL	**0.6**

21.3.4.3 Administration and Other Costs

An allowance of $750k per year was made for corporate travel. This allowance includes all travel and conferences for senior personnel.

Recruitment allowance of $2,500/person was made for 10% estimated turnover rate.

Costs for camp lodging and catering were $40 per person per day which was benchmarked against other projects.

Table 21-22: G&A Costs - Administration and Other

Cost Centre	Annual Cost ($M)
Travel expenses	0.8
Recruitment	0.07
Camp	6.5
Insurance	1.0
Allowances	1.7
TOTAL	**10.6**

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.

21.3.4.4 Contracts

Road maintenance costs were calculated based on an allowance of 25% 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 $ 0.70/t based on annual tonnage placed on the heap.

Employee transport cost is estimated at $30 per person per week over 1/3 year each, based on a total of 448 people.

Table 21-23: G&A Costs - Contracts

Cost Centre	Annual Cost ($M)
Road maintenance	0.9
Security	0.1
Cleaning service	0.1
Effluent handling/ garbage removal	0.1
Heap leach development	15.3
Employee transport	0.2
TOTAL	**16.7**

21.3.4.5 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.

Table 21-24: G&A Costs - Mobile Vehicle Maintenance

Cost Centre	Annual Cost ($k)
Warehouse forklift	72.6
Fire truck	12.7
Crew bus	50.7
Crew van	17.5
Hazardous response vehicle	0.7
Mobile gen-sets	8.7

Cost Centre	Annual Cost ($k)
TOTAL	166.4

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) 2018 USD and modelled the project cashflows in annual periods.

The model assumes a 24-month physical construction period, and production period 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 feasibility study period of the project are summarized.

Table 1-5 shows a summary of key project parameters and project economics. LOM project annual cash flow is shown in Table 22-4.

22.3 Financial Model Parameters

22.3.1 Mineral Resource, 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 summarise the Mineral Reserves.

Table 22-1,Table 22-2, and Table 22-3 summarise the mine production and Net Revenue calculations by year.

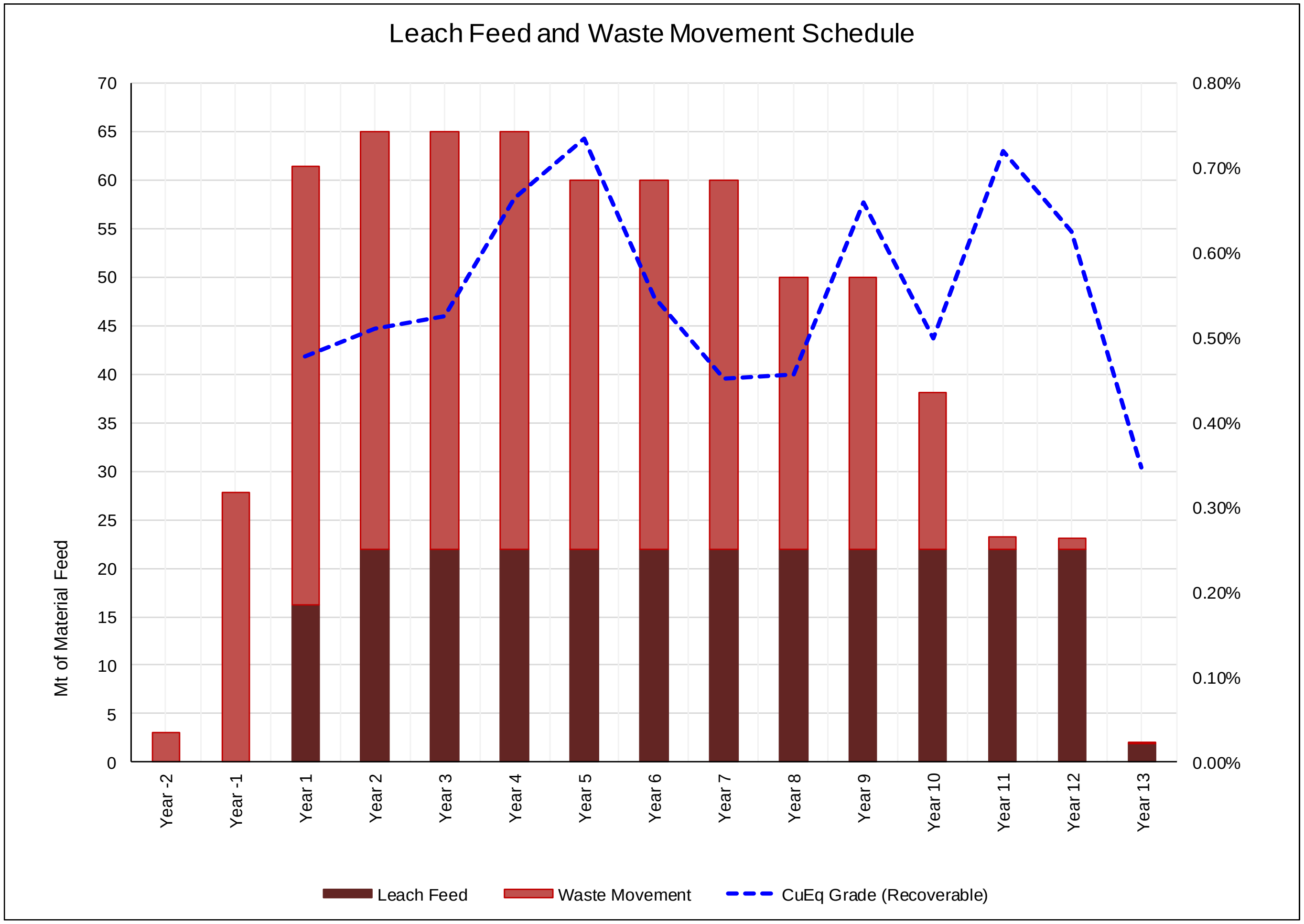


Figure 22-1: Leach Feed and Waste Movement

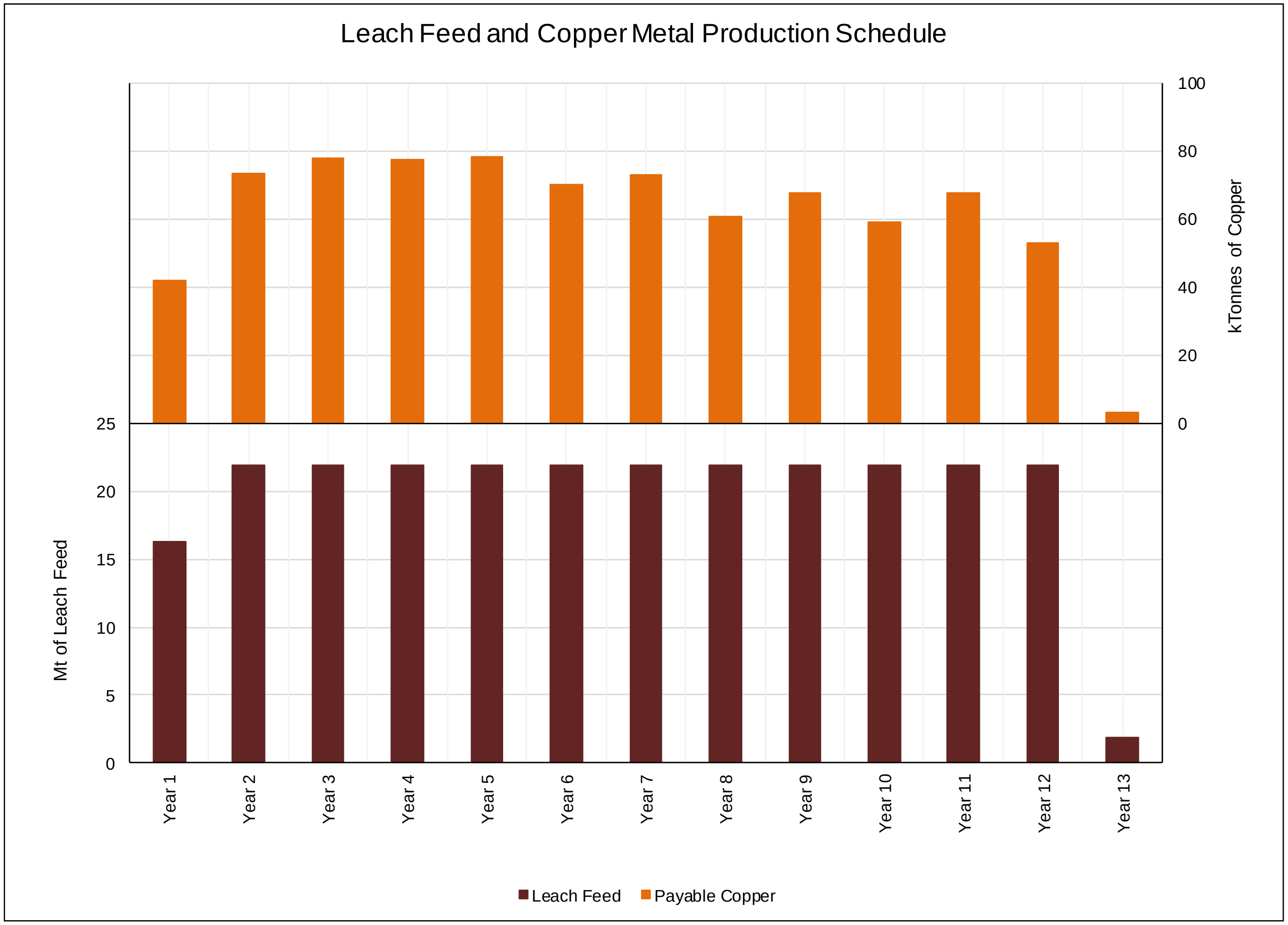


Figure 22-2: Leach Feed and Payable Copper

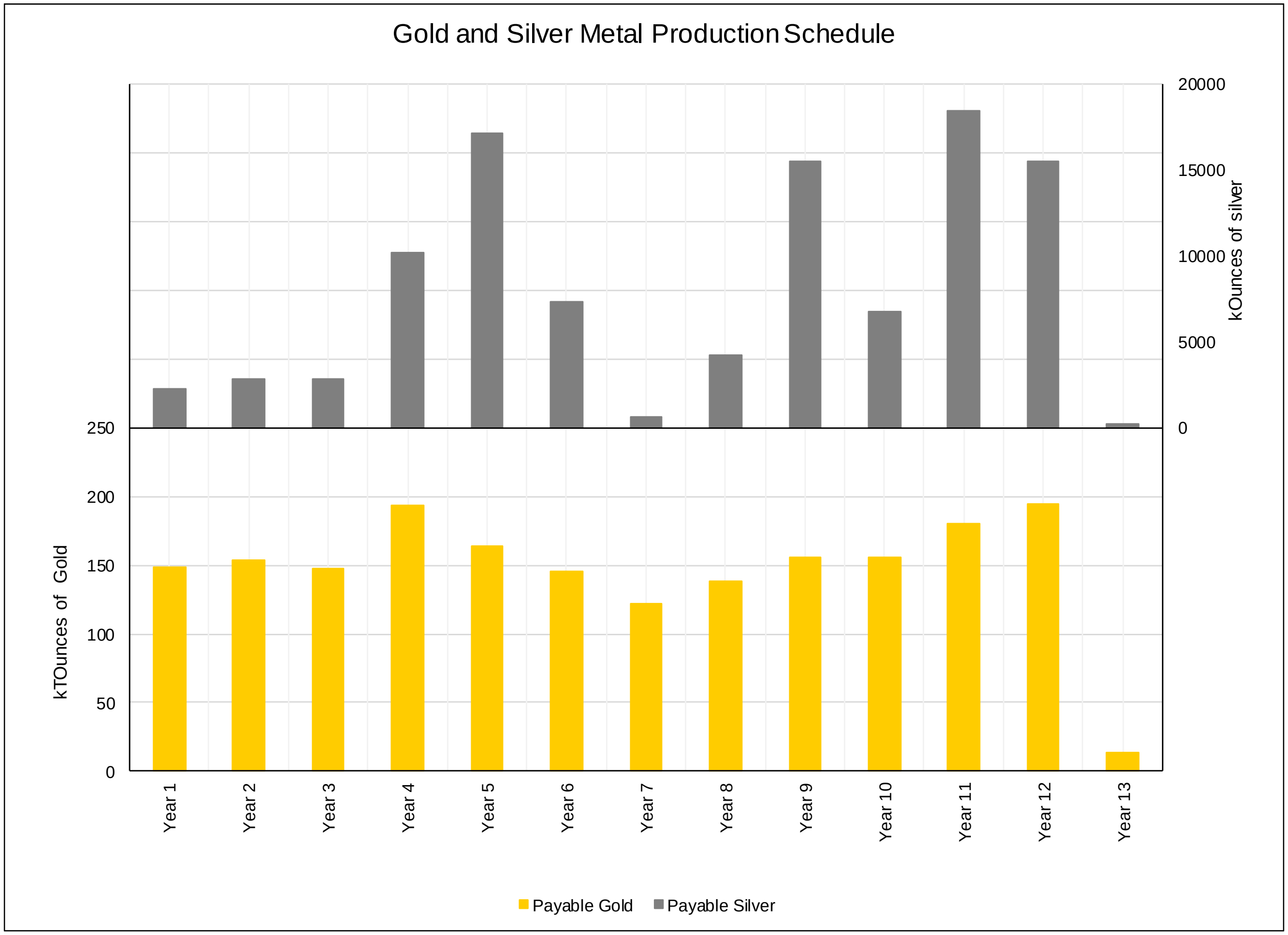


Figure 22-3: Payable Gold and Silver

Table 22-1: Production Summary

Production Summary	Units	LOM Total	Period	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Leach Feed	Mt	259.1		0.0	0.0	16.3	21.9	21.9	21.9	21.9	21.9	21.9	21.9	21.9	21.9	21.9	21.9	1.9	0.0	0.0	0.0	0.0
Waste Movement	Mt	394.5		3.0	27.8	45.2	43.1	43.1	43.1	38.1	38.1	38.1	28.1	28.1	16.2	1.3	1.2	0.0	0.0	0.0	0.0	0.0
Payable Gold	000 Oz	1,913.8		0.0	0.0	148.9	154.1	147.2	193.8	163.6	145.6	121.9	138.4	156.0	155.2	179.8	195.0	14.2	0.0	0.0	0.0	0.0
Payable Silver	000 Oz	103,841.9		0.0	0.0	2,303.4	2,825.3	2,848.4	10,176.2	17,132.1	7,300.6	671.0	4,204.0	15,491.6	6,736.0	18,451.0	15,479.9	222.4	0.0	0.0	0.0	0.0
Payable Copper	Kt	804.3		0.0	0.0	42.1	73.6	78.0	77.6	78.2	70.0	73.2	60.6	67.9	59.1	67.8	53.0	3.3	0.0	0.0	0.0	0.0

Table 22-2: Recovered Metals by Country

Metal Production by country			Period 1	2	3	4	5	6	7	8	9	10	11	12	13
Commodity Prices															
Gold Price	$/oz		$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,30 0	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300
Silver Price	$/oz		$20	$20	$20	$20	$20	$20	$20	$20	$20	$20	$20	$20	$20
Copper Price (incl. 1.5% Cathode premium)	$/lb		$3.05	$3.05	$3.05	$3.05	$3.05	$3.05	$3.05	$3.05	$3.05	$3.05	$3.05	$3.05	$3.05
Recovered Metals		**Total**													
Argentina															
Gold	oz	1,632.2	123.1	71.2	56.8	181.1	136.6	142.8	120.9	125.3	146.3	150.7	175.1	190.0	12.3
Silver	oz	93,115.6	2,110.0	2,141.1	2,442.2	9,209.8	13,239.3	7,282.3	635.1	3,273.6	14,579.1	6,101.3	17,586.0	14,476.9	38.9
Copper	tonnes	666.0	33.8	26.0	32.5	72.6	67.1	67.7	71.3	56.0	63.8	56.7	65.1	50.6	2.7
Chile															
Gold	oz	281.5	25.8	82.9	90.4	12.7	27.0	2.8	1.0	13.1	9.7	4.5	4.6	5.0	1.9
Silver	oz	10,726.3	193.4	684.2	406.3	966.4	3,892.7	18.4	35.9	930.4	912.4	634.7	864.9	1,003.0	183.6
Copper	tonnes	128.6	7.8	46.7	44.5	4.2	10.2	1.4	1.0	3.9	3.3	1.7	1.8	1.7	0.5

Table 22-3: Net Revenue detail

Net Revenues				Period 1	2	3	4	5	6	7	8	9	10	11	12	13
Leach Revenue Argentina		**Total**	**NPV**													
Total Gross Revenue	**$M**	**8,455.1**	**2,880.2**	**429.3**	**310.2**	**341.1**	**906.9**	**892.7**	**786.0**	**648.6**	**604.1**	**910.1**	**698.5**	**1,016.4**	**876.3**	**34.9**
Payable Revenue																
Gold	$M	2,119.8	738.8	159.9	92.5	73.8	235.2	177.5	185.5	157.0	162.7	190.0	195.7	227.4	246.8	15.9
Silver	$M	1,843.7	577.0	41.8	42.4	48.4	182.4	262.1	144.2	12.6	64.8	288.7	120.8	348.2	286.6	0.8
Copper	$M	4,470.9	1,557.8	227.1	174.8	218.4	487.2	450.2	454.7	478.7	375.8	428.4	380.6	437.1	339.8	18.2
Total Payable Revenue	**$M**	**8,434.4**	**2,873.6**	**428.7**	**309.7**	**340.6**	**904.8**	**889.8**	**784.4**	**648.3**	**603.3**	**907.0**	**697.1**	**1,012.7**	**873.2**	**34.9**
Total Refining Charges	$M	33.2	0.0	0.8	0.8	0.9	3.3	4.7	2.6	0.3	1.2	5.2	2.2	6.2	5.1	0.0
Total Freight and Insurance	$M	74.5	39.3	3.4	2.8	3.4	7.9	8.0	7.2	6.5	5.6	8.0	6.1	8.5	6.8	0.4
Argentina Mine Head Royalty	$M	182.7	96.2	9.1	6.8	8.2	20.4	20.4	16.4	12.4	11.9	20.6	13.9	23.2	19.1	0.5
Total Argentina Leach Net Revenue	**$M**	**8,143.9**	**4,278.3**	**415.4**	**299.3**	**328.1**	**873.2**	**856.7**	**758.2**	**629.1**	**584.6**	**873.3**	**674.9**	**974.8**	**842.2**	**34.0**

Net Revenues				Period												
				1	2	3	4	5	6	7	8	9	10	11	12	13
Leach Revenue Chile																
Total Gross Revenue	**$M**	**1,443.8**	**1,194.6**	**89.6**	**435.2**	**424.4**	**63.8**	**181.6**	**13.4**	**8.5**	**61.7**	**52.9**	**30.1**	**35.2**	**37.7**	**9.7**
Payable Revenue																
Gold	$M	365.6	310.1	33.6	107.7	117.4	16.5	35.0	3.6	1.3	17.1	12.6	5.8	6.0	6.5	2.5
Silver	$M	212.4	121.6	3.8	13.5	8.0	19.1	77.1	0.4	0.7	18.4	18.1	12.6	17.1	19.9	3.6
Copper	$M	863.3	761.3	52.1	313.8	298.8	27.9	68.7	9.5	6.5	26.0	22.0	11.5	11.9	11.1	3.5
Total Payable Revenue	**$M**	**1,441.3**	**1,193.0**	**89.5**	**435.0**	**424.2**	**63.6**	**180.8**	**13.4**	**8.5**	**61.5**	**52.7**	**29.9**	**35.1**	**37.5**	**9.6**
Total Refining Charges	$M	3.9	0.0	0.1	0.3	0.2	0.3	1.4	0.0	0.0	0.3	0.3	0.2	0.3	0.4	0.1
Total Freight and Insurance	$M	13.3	8.9	0.7	4.3	4.0	0.5	1.5	0.1	0.1	0.5	0.4	0.3	0.3	0.3	0.1
Chilean Private NSR Royalty	$M	7.3	0.0	0.0	0.0	0.0	0.9	2.7	0.2	0.1	0.9	0.8	0.4	0.5	0.6	0.1
Total Chile Leach Net Revenue	**$M**	**1,416.9**	**949.7**	**88.7**	**430.4**	**420.0**	**61.7**	**175.2**	**13.1**	**8.3**	**59.7**	**51.1**	**29.0**	**33.9**	**36.3**	**9.3**
SART Revenue																
Payable Revenue - SART Argentina	**$M**	**53.9**	**28.4**	**2.5**	**1.9**	**2.3**	**5.4**	**5.2**	**5.4**	**5.7**	**4.6**	**5.3**	**4.7**	**5.7**	**4.9**	**0.3**
Total TCRC Freight	**$M**	**4.2**	**2.2**	**0.2**	**0.1**	**0.2**	**0.4**	**0.4**	**0.4**	**0.4**	**0.4**	**0.4**	**0.4**	**0.4**	**0.4**	**0.0**
Net Revenue - SART Argentina	**$M**	**49.7**	**26.1**	**2.3**	**1.7**	**2.2**	**5.0**	**4.8**	**5.0**	**5.3**	**4.3**	**4.9**	**4.3**	**5.2**	**4.5**	**0.3**
Payable Revenue - SART Chile	**$M**	**10.5**	**7.3**	**0.6**	**3.7**	**3.8**	**0.3**	**0.8**	**0.1**	**0.1**	**0.4**	**0.3**	**0.1**	**0.1**	**0.1**	**0.1**
Total TCRC Freight	**$M**	**0.8**	**0.6**	**0.0**	**0.3**	**0.3**	**0.0**	**0.1**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**
Net Revenue - SART Chile	**$M**	**9.7**	**6.7**	**0.5**	**3.5**	**3.5**	**0.3**	**0.7**	**0.1**	**0.1**	**0.3**	**0.3**	**0.1**	**0.1**	**0.1**	**0.0**
Total SART Net Revenue	**$M**	**59.4**	**32.8**	**2.8**	**5.2**	**5.6**	**5.3**	**5.5**	**5.1**	**5.3**	**4.6**	**5.2**	**4.4**	**5.4**	**4.6**	**0.3**

Table 22-4: 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	**$M**	**8,143.9**	**4,278.3**	415.4	299.3	328.1	873.2	856.7	758.2	629.1	584.6	873.3	674.9	974.8	842.2	34.0
Total Chile Leach Net Revenue	**$M**	**1,416.9**	**949.7**	88.7	430.4	420.0	61.7	175.2	13.1	8.3	59.7	51.1	29.0	33.9	36.3	9.3
Total SART Net Revenue	**$M**	**59.4**	**32.8**	2.8	5.2	5.6	5.3	5.5	5.1	5.3	4.6	5.2	4.4	5.4	4.6	0.3
Grand Total Minesite Net Revenue	**$M**	**9,620.2**	**5,260.9**	**506.9**	**735.0**	**753.8**	**940.2**	**1,037.5**	**776.4**	**642.7**	**649.0**	**929.6**	**708.4**	**1,014.1**	**883.1**	**43.6**

22.3.2 Metallurgical Recoveries

Metallurgical recoveries 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 overall achieved LOM recoveries are shown in Table 22-5.

Table 22-5: LOM Average Process Recoveries

Metal	Value
Copper (including 1% from SART)	80%
Gold	70%
Silver	82%

22.3.3 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 floatation products) at 65% contained copper. SRK understands that this is typical for the precipitate produced by SART circuits, which consists primarily of precipitated chalcocite.

22.3.4 Metal Prices

Flat real prices were assumed for the life of the project. Table 22-6 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 22-6: Pricing Assumptions for Economic Analysis

Commodity Market Prices	Units	Price
Copper Price excl. 1.5% Cathode Premium	$/lb	$3.00
Gold Price	$/oz	$1,300
Silver Price	$/oz	$20.00

A 1.5% premium associated with selling cathode copper from the SX-EW plant was assumed, increasing the received price to $3.05 per pound.

22.3.5 Operating Costs

The operating costs modelled are detailed in Section 22.

Table 22-7 summarises the overall unit costs resulting from the incorporation into the economic model.

Table 22-7: Operating Cost Summary

Operating & Sustaining Costs	LOM ($000)	Unit Opex ($/t)	Unit Opex ($/lb Payable Cu.Eq)
Mine Operating Cost	**$999,013**	**$3.86**	***$0.31***
Process Operating Costs			
Power	$435,088	$1.68	*$0.13*
Consumables	$1,564,397	$6.04	*$0.48*
Maintenance	$204,500	$0.79	*$0.06*
Labour	$101,040	$0.39	*$0.03*
Total Processing Costs	**$2,305,025**	**$8.90**	***$0.70***
General and Administrative	**$373,072**	**$1.44**	***$0.11***
Total Operating Costs	**$3,677,110**	**$14.19**	***$1.12***
Offsite Unit Costs			
Total Payable Deduction	$25,594	$0.10	*$0.01*
Total TCRC Freight and Insurance	$129,881	$0.50	*$0.04*
Total Royalties	$190,009	$0.73	*$0.06*
Total Offsite Unit Costs	**$345,483**	**$1.33**	***$0.11***
Sustaining Capex	**$217,119**	**$0.84**	***$0.07***
Closure	**$50,765**	**$0.20**	***$0.02***
All-in Sustaining Costs	**$4,290,477**	**$16.56**	***$1.31***

The mine operating costs were $1.53 per tonne of total material moved.

22.3.6 Capital Costs

Capital costs used for the economic evaluation are shown by period in Table 22-14 and summarised in Table 22-8. Initial capital costs were scheduled to be spent equally across the two-year timeline for project construction, with the exception of mining costs which were scheduled to match the expected spend in the two years of project construction.

Table 22-8: Capital Cost Summary

Capital Expenditure	Initial ($M)	Sustaining ($M)	LOM ($M)
Mine	$180	$120	$300
Process Plant	$392	$97	$489
On-Site Infrastructure	$94	$0	$94
Off-Site Infrastructure	$123	$0	$123
Sub-Total Direct Costs	**$789**	**$217**	**$1,006**
Indirects	$132	$0	$132
EPCM Services	$101	$0	$101
Owner's Costs	$50	$0	$50
Provisions	$194	$0	$194
Sub-Total Indirect Costs	**$477**	**$0**	**$477**
Project Total	**$1,266**	**$217**	**$1,483**
Closure Capex	**N/A**	**$51**	**$51**

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 productions and waste movement profile. In the case of plant, 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.

22.3.7 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 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.

22.3.7.1 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.

22.3.7.2 Chilean Royalties

Chilean royalties were estimated on the basis of 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.

22.3.8 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 22-9 summarises the assumption made.

Table 22-9: Pricing Assumptions for Economic Analysis

Working Capital	Units	Value
Receivables outstanding	days	15
Payables outstanding	days	30
Annual operating costs in stores	% of annual opex	12.00%
Contraction discount stores value lost per year	% of stores balance	5.00%

22.3.9 Taxes

Two tax models were created, splitting the revenues and costs by country, and estimating taxable income for each. The rates used for corporate taxes were 25% for Argentina and 27% for Chile.

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 applied only to years of positive taxable income was 7.4%, resulting in a payment of $49.8M over the life of the project. SRK considers it possible that more detailed analysis, and consideration of how periods of loss are utilised, 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.

22.3.10 Closure Costs and Salvage Value

An allowance of $51M 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 salvage value was assumed for any items. For the plant and infrastructure, salvage value was assumed to be netted-off the closure cost estimate. In the case of the mining fleet, optimisation of sustaining capital expenditure was assumed, rendering salvage value to be effectively zero.

22.3.11 Financing

No consideration of financing was made. The model considers the cashflow only at an asset level and assumes 100% equity ownership.

22.3.12 Inflation

The modelling was primarily undertaken in real 2019 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.

22.4 Financial Results

Analysis of the project demonstrates that the mine plan has positive economics under the assumptions used. The project post-tax NPV at and 8% discount rate is estimated to be $1.28Bn, with an IRR of 23%. The project financial summary is shown in Table 22-10. The year-by-year cashflows are summarised in Table 22-11.

Table 22-10: Project Economic Summary

Project Metric	Units	Value
Pre-Tax NPV (8%)	$B	$1.86
Pre-tax IRR	%	27%
After-Tax NPV (8%)	$B	$1.28
After-Tax IRR	%	23%
Undiscounted After-Tax Cash Flow (LOM)	$B	$3.23
Average Operating Margin[1]n*	%	62%
Payback Period from start of processing (undiscounted, after-tax cash flow)	years	3.4
Initial Capital Expenditures (rounded)	$B	$1.27
LOM Sustaining Capital Expenditure (excluding closure)	$M	$0.22
LOM C-1 Cash Costs (Co-Product)	$ per lb Cu.Eq.	$1.23
Nominal Process Capacity	tonnes per day	60,000
Mine Life (including pre-stripping)	years	14
Average Annual Copper Production[2]	Tonnes Cu	67,000
Average Annual Gold Production[1]	Oz Au	159,000
Average Annual Silver Production[1]	Oz ag	8,653,000

Project Metric	Units	Value
LOM Average Process Recovery - Copper[3]	%	80%
LOM Average Process Recovery - Gold	%	70%
LOM Average Process Recovery - Silver	%	82%

1 Operating Margin = Operating Cashflow/Net Revenue;
2 Rounded and excluding final year of minimal leach operation;
3 Including 1% Cu recovery to concentrate for SART process.

Table 22-11: Capital Expenditure by Period

Capital Cost Summary		Total	NPV	Period -2	-1	1	2	3	4	5	6	7
Mine	$M	$180	$168	$107.6	$72.3	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Process Plant	$M	$392	$363	$196.1	$196.1	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
On-Site Infrastructure	$M	$94	$87	$46.9	$46.9	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Off-Site Infrastructure	$M	$124	$114	$61.8	$61.8	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Sub-Total Direct Costs	**$M**	**$789**	**$733**	**$412.3**	**$377.1**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**
Indirects	$M	$132	$122	$66.0	$66.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Epcm Services	$M	$101	$94	$50.7	$50.7	$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
Provisions	$M	$194	$179	$96.8	$96.8	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Sub-Total Indirect Costs	**$M**	**$477**	**$442**	**$238.5**	**$238.5**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**
Project Total Initial Capex	**$M**	**$1,266**	**$1,175**	**$650.8**	**$615.6**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**
Project Total Sustaining Capex	$M	$217	$120	$0.0	$0.0	$13.6	$18.4	$18.4	$18.4	$18.4	$18.4	$18.4
Closure Capex	$M	$51	$14	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Grand Total Project Capex	**$M**	**$1,534**	**$1,309**	**$650.8**	**$615.6**	**$13.6**	**$18.4**	**$18.4**	**$18.4**	**$18.4**	**$18.4**	**$18.4**

Capital Cost Summary (cont.)		Period 8	9	10	11	12	13	14	15	16	17
Mine	$M	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Process Plant	$M	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
On-Site Infrastructure	$M	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Off-Site Infrastructure	$M	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Sub-Total Direct Costs	**$M**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**
Indirects	$M	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Epcm Services	$M	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Owner's Costs	$M	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Provisions	$M	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Sub-Total Indirect Costs	**$M**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**
Project Total Initial Capex	**$M**	**$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	$18.4	$18.4	$18.4	$18.4	$18.4	$1.6	$0.0	$0.0	$0.0	$0.0
Closure Capex	$M	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$16.9	$16.9	$16.9	$0.0
Grand Total Project Capex	**$M**	**$18.4**	**$18.4**	**$18.4**	**$18.4**	**$18.4**	**$1.6**	**$16.9**	**$16.9**	**$16.9**	**$0.0**

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Table 22-12: Summary Cashflow

				Period								
Summary Cashflow		**Total**	**NPV**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**
Grand Total Minesite Net Revenue	**$M**	**9,620.2**	**5,260.9**	**0.0**	**0.0**	**506.9**	**735.0**	**753.8**	**940.2**	**1,037.5**	**776.4**	**642.7**
Operating Costs												
Mine	$M	$999	$590	$0.0	$0.0	$95.4	$102.0	$103.3	$102.6	$94.3	$102.2	$100.4
Processing and Infrastructure	$M	$2,305	$1,264	$0.0	$0.0	$138.1	$183.2	$179.5	$200.6	$200.6	$200.6	$200.5
General and Administrative	$M	$373	$206	$0.0	$0.0	$23.4	$31.5	$31.5	$31.5	$31.5	$31.5	$31.5
Total Operating Costs	**$M**	**$3,677**	**$2,061**	**$0.0**	**$0.0**	**$256.9**	**$316.7**	**$314.3**	**$334.7**	**$326.4**	**$334.4**	**$332.5**
Operating Cashflow	**$M**	**$5,943**	**$3,200**	**$0.0**	**$0.0**	**$250.0**	**$418.3**	**$439.4**	**$605.4**	**$711.0**	**$442.0**	**$310.2**
Total Project Capex	**$M**	**$1,534**	**$1,309**	**$650.8**	**$615.6**	**$13.6**	**$18.4**	**$18.4**	**$18.4**	**$18.4**	**$18.4**	**$18.4**
Working Capital	$M	$24	$35	$0.0	$0.0	$32.1	$13.5	$2.6	$10.5	$5.8	-$8.1	-$3.5
PRE-TAX CASHFLOW	**$M**	**$4,385**	**$1,857**	**-$650.8**	**-$615.6**	**$204.3**	**$386.4**	**$418.4**	**$576.6**	**$686.8**	**$431.8**	**$295.4**
Total Tax	$M	$1,158	$574	$0.0	$0.0	$0.0	$17.2	$70.5	$46.6	$182.8	$110.9	$77.9
After-tax Net Cash Flow (Real)	**$M**	**$3,227**	**$1,283**	**-$650.8**	**-$615.6**	**$204.3**	**$369.2**	**$348.0**	**$529.9**	**$504.0**	**$320.9**	**$217.5**

		Period									
Summary Cashflow (cont.)		**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**
Grand Total Minesite Net Revenue	**$M**	**649.0**	**929.6**	**708.4**	**1,014.1**	**883.1**	**43.6**	**0.0**	**0.0**	**0.0**	**0.0**
Operating Costs											
Mine	$M	$73.6	$73.9	$65.3	$41.7	$40.3	$4.1	$0.0	$0.0	$0.0	$0.0
Processing and Infrastructure	$M	$191.1	$191.9	$200.0	$200.6	$200.6	$17.7	$0.0	$0.0	$0.0	$0.0
General and Administrative	$M	$31.5	$31.5	$31.5	$31.5	$31.5	$2.8	$0.0	$0.0	$0.0	$0.0
Total Operating Costs	**$M**	**$296.2**	**$297.3**	**$296.8**	**$273.8**	**$272.4**	**$24.5**	**$0.0**	**$0.0**	**$0.0**	**$0.0**
Operating Cashflow	**$M**	**$352.7**	**$632.3**	**$411.6**	**$740.3**	**$610.7**	**$19.1**	**$0.0**	**$0.0**	**$0.0**	**$0.0**
Total Project Capex	**$M**	**$18.4**	**$18.4**	**$18.4**	**$18.4**	**$18.4**	**$1.6**	**$16.9**	**$16.9**	**$16.9**	**$0.0**
Working Capital	$M	$0.6	$13.4	-$7.1	$13.4	-$3.5	-$43.5	-$2.7	$0.0	$0.0	$0.0
Pre-tax Cashflow	**$M**	**$333.7**	**$600.5**	**$400.3**	**$708.5**	**$595.8**	**$61.0**	**-$14.2**	**-$16.9**	**-$16.9**	**$0.0**
Total Tax	$M	$79.2	$151.2	$99.8	$182.7	$150.6	$2.7	-$3.4	-$3.3	-$4.2	-$2.0
After-tax Net Cash Flow (Real)	**$M**	**$254.5**	**$449.4**	**$300.5**	**$525.8**	**$445.2**	**$58.3**	**-$10.8**	**-$13.6**	**-$12.7**	**$2.0**

22.5 Sensitivity Analysis

Sensitivity analysis was conducted to estimate the response of the project net present value to changes in assumptions on key inputs of metals prices, capital costs and operating costs. The results across a range of +/-20% 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 could lead to better outcomes than what the sensitivity analysis suggests.

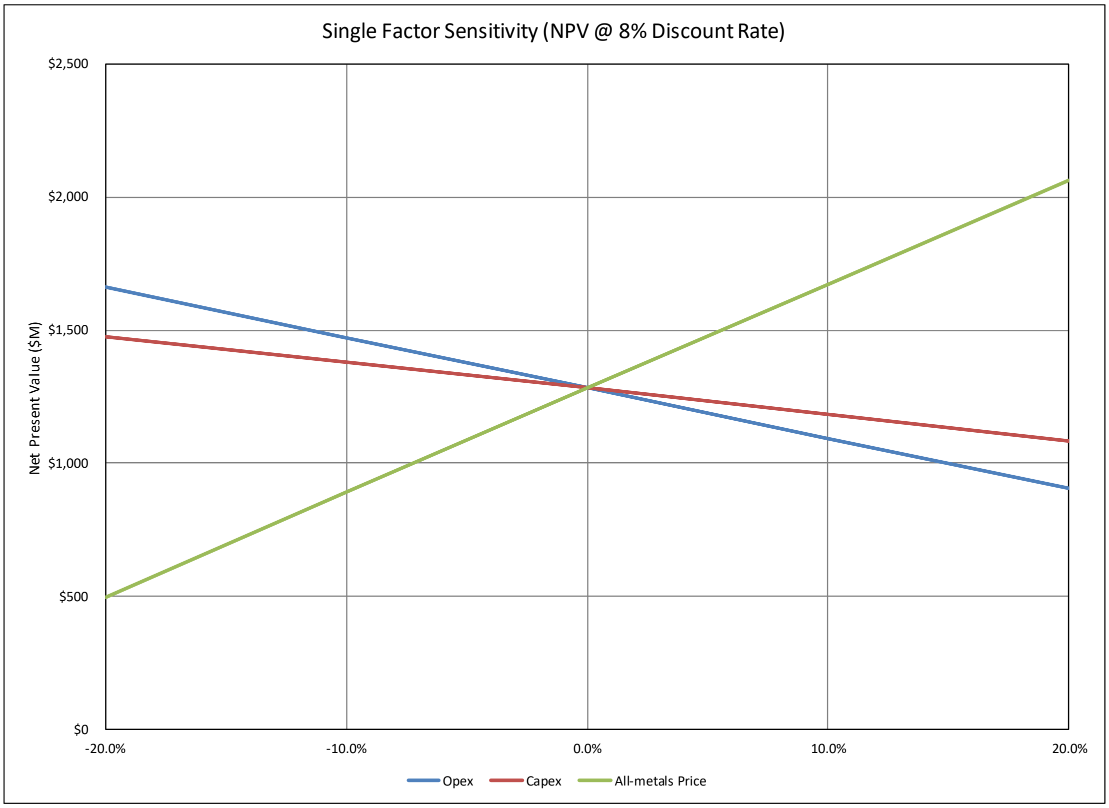


Figure 22-4: Single-Factor Sensitivity Chart

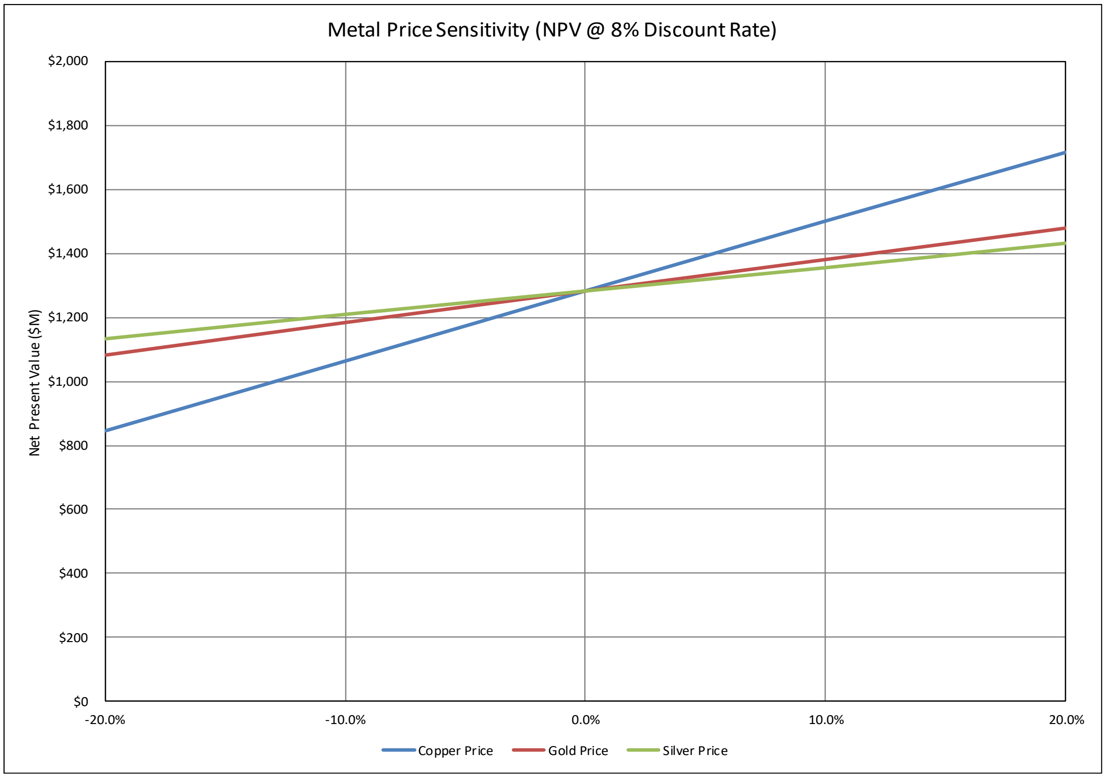


Figure 22-5: Metal Price Sensitivity Chart

Further analysis of key risks was undertaken by varying certain parameters and is summarised in Figure 22-6.

22.5.1.1 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 for the Feasibility Study.

22.5.1.2 Sensitivity Tables

Table 22-13 to Table 22-17 show the sensitivity of project NPV to variations in various key factors. They are generally presented as two-factor tables so that the effect of combinations of assumptions can be seen.

Table 22-13: Opex and Capex Sensitivity – NPV @ 8% ($B)

		Operating Costs				
			-10.0%	0.0%	10.0%	20.0%
Capital Costs	-20.0%	$1.85	$1.67	$1.48	$1.29	$1.10
	-10.0%	$1.76	$1.57	$1.38	$1.19	$1.00
	0.0%	$1.66	$1.47	**$1.28**	$1.09	$0.90
	10.0%	$1.56	$1.37	$1.18	$0.99	$0.81
	20.0%	$1.46	$1.27	$1.09	$0.90	$0.71

Table 22-14: Metal Price and Discount Rate Sensitivity – NPV @ 8% ($B)

		Discount Rate				
		6.0%	7.0%	8.0%	9.0%	10.0%
Metal Prices	-20.0%	$0.73	$0.61	$0.50	$0.40	$0.31
	-10.0%	$1.18	$1.03	$0.89	$0.77	$0.65
	0.0%	$1.63	$1.45	**$1.28**	$1.13	$1.00
	10.0%	$2.08	$1.87	$1.67	$1.50	$1.34
	20.0%	$2.53	$2.28	$2.06	$1.86	$1.68

Table 22-15: Metal Price and Capex Sensitivity – NPV @ 8% ($B)

		Metal Prices				
		-20.0%	-10.0%	0.0%	10.0%	20.0%
Capex	-20.0%	$0.69	$1.09	$1.48	$1.87	$2.26
	-10.0%	$0.59	$0.99	$1.38	$1.77	$2.16
	0.0%	$0.50	$0.89	**$1.28**	$1.67	$2.06
	10.0%	$0.40	$0.79	$1.18	$1.57	$1.96
	20.0%	$0.30	$0.69	$1.09	$1.48	$1.87

Table 22-16: Metal Price and Opex Sensitivity – NPV @ 8% ($B)

		Metal Prices				
		-20.0%	-10.0%	0.0%	10.0%	20.0%
Opex	-20.0%	$0.88	$1.27	$1.66	$2.05	$2.44
	-10.0%	$0.69	$1.08	$1.47	$1.86	$2.25
	0.0%	$0.50	$0.89	**$1.28**	$1.67	$2.06
	10.0%	$0.30	$0.70	$1.09	$1.48	$1.87
	20.0%	$0.11	$0.51	$0.90	$1.29	$1.68

Table 22-17: Individual Metal Price Sensitivity – NPV @ 8% ($B)

	Metal Prices				
	-20.0%	-10.0%	0.0%	10.0%	20.0%
Copper Price	$0.85	$1.06	**$1.28**	$1.50	$1.72
Gold Price	$1.08	$1.18	**$1.28**	$1.38	$1.48
Silver Price	$1.13	$1.21	**$1.28**	$1.36	$1.43

22.5.1.3 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.

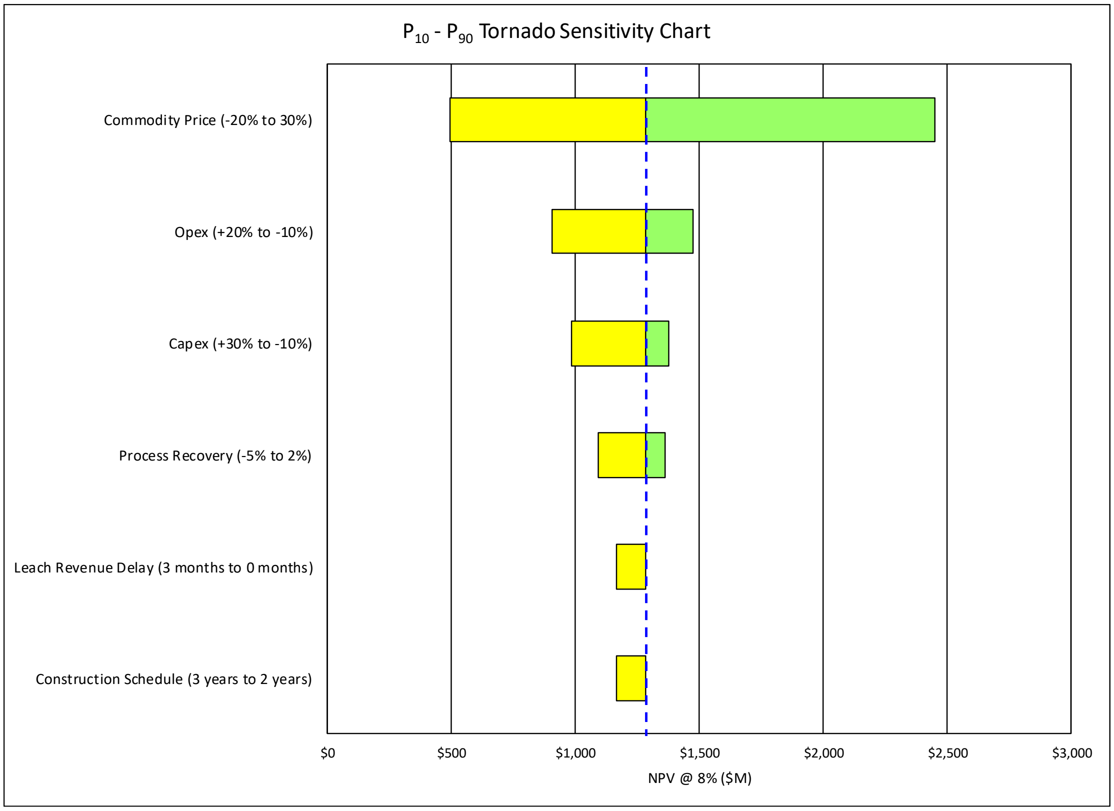


Figure 22-6: Key Risk Tornado Diagram

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

The Filo del Sol deposit remains open in most directions with several additional exploration targets outside of the immediate deposit area that have seen only preliminary exploration. Work at this high elevation is logistically challenging and some areas are not easily accessed without significant effort. Because of this, there remain several areas near the deposit that have high exploration potential. In addition, there are areas of cover near the deposit that are prospective for discovery that have not been drill tested. The Ventanas, Maranceles, and south Tamberias areas are more peripheral to the Filo deposit, but occur at distances of well over a kilometre and may host similar, broadly related systems. They are considered ongoing exploration targets that may, or may not, have direct ties to the Filo -age mineralization.

Within the Filo del Sol system, geological understanding has advanced significantly over the past couple of years, with new drilling proving early exploration concepts. The extension of the gold oxide zone into the Tamberias area is an important development not only because it demonstrates the potential for expansion of mineralization, but also because it has advanced understanding of the deposit geology. At present, the recognition that progressively younger porphyry intrusions and epithermal alteration within the highly telescoped system host the most significant mineralization discovered to date is an important concept that will guide ongoing exploration. It encourages exploration for relatively late, blind porphyry intrusions and breccia bodies even within the partly eroded porphyry domains. This also applies to exploration beneath the Filo section of the deposit itself where recent modelling of porphyry intrusions below the Filo zone shows a spatial correlation of relatively young porphyry intrusions with grades of up to 1 g/t Au and 0.5% Cu to 1% Cu. The hypogene copper-gold zone remains open at depth and targeting the upper parts of these late intrusions is an important direction for expanding the sulphide portion of the deposit at Filo del Sol.

The oxide zones at Filo form the most metal-rich part of the deposit. These domains are complex and patterns indicate some pre-mineral structural control on supergene enrichment. These high-grade zones within the deposit have room for expansion, particularly to the west up to the Frontera Fault, which will be tested with further drilling. The high-grade silver zone is similarly open for expansion in several directions within the immediate deposit area.

The presence of hydrothermal breccias as gold exploration targets, particularly at the north end of the system, has seen drilling effort over the past two campaigns, however more work is necessary to fully evaluate the large and geologically-complex area at North Filo. This area remains highly prospective.

To the immediate north of the deposit, the high-grade silver zone plunges northward and geological relationships become more complex with post-mineral faulting. Drilling to date has not successfully reached the projected deeper extension of this zone due to challenging drilling conditions. Several lines of evidence have indicated that this area at the immediate north of the Filo deposit may be the hydrothermal up-flow source zone for the Filo system; these include: the projection of the high-grade silver domain, increasing molybdenum values in this direction, and well as the intersection of northwest and north-south pre-mineral faults in this area. Given the presence of numerous high-level phreatic and hydrothermal breccias in this domain it is also considered to be prospective for gold-mineralized breccias at depth. It is an intriguing exploration target that has not been fully explored.

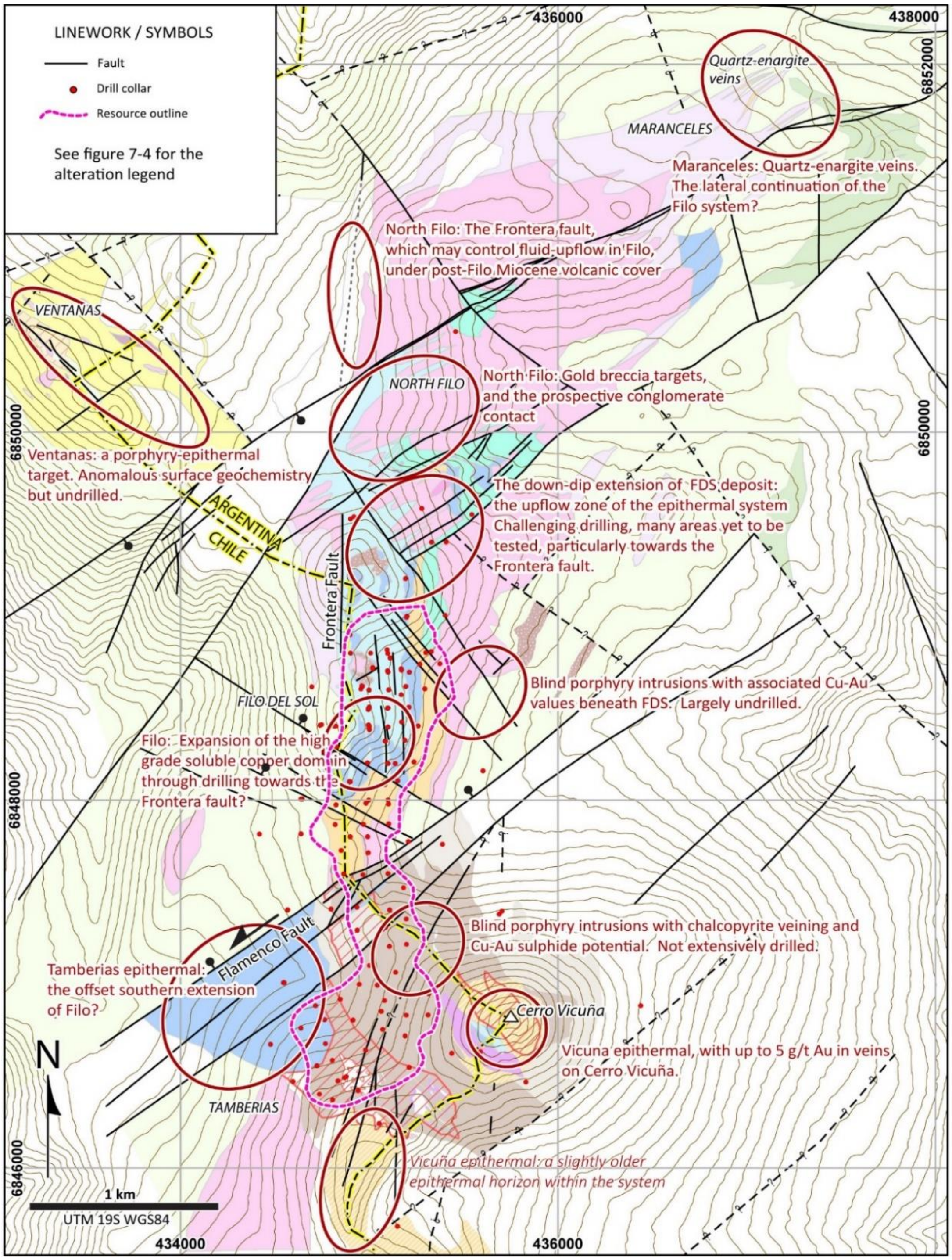
Farther north into North Filo several holes have been drilled to target hydrothermal breccias as potential gold targets in this area, with some success. However, the area is largely covered with surface talus and gravels. There are, however, at least two potential exploration concepts here that warrant further evaluation: 1) The base of the conglomerate unit that is the host for the high-grade silver zone at Filo occurs through this area, perhaps as the northern mirror-image of Filo across the hydrothermal upflow zone, and 2) further evaluation of the hydrothermal breccia bodies as gold targets. The hydrothermal breccia bodies have been difficult to distinguish from quartz-alunite-altered conglomeratic host rocks, particularly in RC drill holes. This is a challenge, however a thorough evaluation of all surface geochemistry data and correlation with available mapping may provide some new insight.

In the Ventanas area, new work in the 2016 and 2017 seasons discovered intrusive-cemented and hydrothermal breccias associated with the margin of a dacitic feldspar-biotite porphyry intrusion. This coincides with anomalous copper and gold surface geochemistry. This area has not yet been further explored and presents an exciting new porphyry Cu-Au target that may be the northern extension of the Filo porphyry trend, offset across the north-east-trending Maranceles fault.

The discovery of the Tamberias gold oxide zone with limited drilling in 2017 shows the potential for the extension of the Filo epithermal system into this area, as well as expansion of the porphyry-related gold mineralization. Understanding of the Filo -Tamberias system as a highly telescoped porphyry-epithermal system with relatively minor fault displacement following the development of the youngest Filo epithermal domain allows for more extensive continuation of the Filo epithermal system to the south of the Flamenco fault. Also within Tamberias, recent 3D modelling has highlighted the presence of a blind porphyry intrusion that is spatially associated with the best surface copper grades in talus and rock chip sampling. This intrusion was not previously identified and is a new porphyry target that adds to the two other gold-mineralized porphyry centres at Tamberias that have not yet been extensively drilled.

Preserved at the top of Cerro Vicuña and to the south of Tamberias are the remnants of an older epithermal system that pre-dates the later telescoped Filo Mining system. While gold values in quartz veins have been the focus of some drilling, the extension of this epithermal horizon remains largely unexplored to the south of Tamberias.

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.



Devine et al. 2017

Figure 24-1: Exploration Potential and Targets

25 Interpretation and Conclusions

25.1 Conclusions

Mineral Tenure, Surface Rights, and Royalties

Filo Mining, through its Argentinian and Chilean subsidiaries, Filo del Sol Exploracion 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.

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. Several target areas have been identified and are described in Section 24.

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.

Drilling

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.

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.

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.

Mineral Resources Estimation

Mineral resource estimates presented in this report represent the global mineral resources located at Filo del Sol as of 11 June 2018. 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 Whittle shell that constrains mineral resources amenable to open pit mining methods
- 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.

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,

Metallurgical Testing and Process Design

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 and 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.

The plant design has been significantly optimised from that of the PEA, by eliminating one heap leach pad and simplifying processing so that all ore is first leached for copper then transferred to the gold-silver heap leach pad. There is further opportunity to adjust the recovery process in the interests of reducing technological risks given the project location and climate, while potentially increasing the operability of the process and the options available to manage a broad range of ore variability.

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 favorable 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.

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.

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.

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 Ausenco Class 4 prefeasibility Study estimate with an accuracy range of +/-25%, 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 to 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.

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 $1.28 billion, with an IRR of 23%. 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

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.

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.

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.

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.

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.

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.

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

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.

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.

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 cut-off 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.

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 more aggressive metal recovery estimates and improve project revenues.

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.

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.

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 in order to further reduce costs or increase cut-off grades.

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

26.1 Resource Estimate

Additional infill diamond drilling should be carried out to convert a portion of the Indicated mineral resource to the Measured category, which will allow for the declaration of a portion of the mineral reserve as a Proven mineral reserve. This infill drilling should target areas scheduled for production within the first five years of the mine life, with special emphasis on drilling areas with relatively high geological complexity. This drill program should be carried out in conjunction with the pit geotechnical drill program, as it can share many of the same infrastructure, support and personnel costs.

All drill core from this work should be logged, photographed and sampled in accordance with established protocols for the project. This new data should then form the basis of a resource update, which will incorporate any revisions to the geological model resulting from the new drilling as well as the new assay data from the drill core sampling. This updated resource will then allow for an optimization of the mine plan, which will result in the declaration of a Proven mineral reserve.

This drill program should also take into account the requirements for sample collection for the recommended metallurgical testwork program. Consideration should be given to ensuring that the program results in collection of a sufficient quantity of the representative samples needed to provide the material for the metallurgical program. Metallurgical samples should be collected following logging and assaying of the drill core, by cutting the remaining half-core sample into quarter-core for the intervals selected to be sampled.

This drill program should be carried out with diamond drills, beginning as soon as access is established to the site and camp and other infrastructure is available. It is anticipated to take approximately two months to complete the drilling, and an additional two months to complete the sampling, assaying and resource estimation.

26.2 Mining

Engineering work related to open pit slope design for a Feasibility Study should be focused on improving the confidence level of the design criteria, designing interim pit slopes, and developing an optimized final pit design. Additional investigations will be required at the feasibility level, including: Drilling of 6 to 10 diamond drill holes, sited specifically for geotechnical purposes and ranging in length from 200 to 500 m, and totaling approximately 3000 m; with each of the major faults crossed by 1 to 2 holes. Advanced sampling methods, such as Lexan tubes or a Mazier core barrel, are recommended to reduce disturbance to the core in the LIX geotechnical unit. Installation of 4 to 6 vibrating wire piezometer and thermistor nests is recommended to evaluate groundwater pore pressures and permafrost distribution in the proposed open pit footprint. Further characterization of intact strengths of the geotechnical units through laboratory testing of intact drill core. This includes additional UCS and BTS testing, triaxial compressive strength testing, additional direct shear testing of discontinuities. A separate testing program is recommended for fault gouge and weak rock, such as the leached rock of the LIX unit. Thaw consolidation and strength testing (uniaxial and triaxial) of frozen rock mass samples is also recommended. Advanced numerical modeling to assess potential deformations and to validate assumptions related to depth of disturbance. Stability analyses should incorporate an evaluation of seismic loads.

Modeling of surface and groundwater flows that will report to the open pits is recommended for future studies. These flows should be predicted throughout the proposed life of the pit. A pit dewatering and depressurization plan should be developed and incorporated into the overall water management plan. The infrastructure and power requirements associated with this plan will need to be estimated.

A geohazard assessment of major infrastructure, including the open pits, plant site, camp and access and/or powerline routes, is recommended. This should include an assessment of avalanche potential within the project area and including the open pits. Large accumulations of drifting snow have been observed in the project area.

The PFS pit phase designs were developed by closely following guidance from nested 'revenue factor' LG shells. The opportunity exists to mine some of the phases to final depth and create backfill opportunities at the cost of reducing initial head grades somewhat. Given the long downhill waste hauls required by the 'bottom up' waste dump construction geotechnical requirement, a trade-off between the current design approach and one that generates backfill opportunities should be investigated.

The mine equipment selection focused only on diesel equipment. There may be benefits from utilizing electric powered shovels and drills. During the next planning stage, trade-off study should be performed comparing diesel vs electric shovels and drills.

The PFS study incorporated autonomous haulage. Autonomous blasthole drilling should be investigated as well during the next iteration of planning.

26.3 Marketing and Logistics

As part of FS level investigations, an independent, formal marketing study on the planned products from Filo del Sol (copper cathode, copper precipitate, and gold-silver doré) should be commissioned. Similarly, logistics studies in support of the FS should include a formal review of costs to get these final products to market.

26.4 Site Wide Geotechnical Program

A geotechnical field and laboratory program to a feasibility level needs to be performed in the next phase to understand subsurface conditions and limits of permafrost below infrastructure, faults in the area and limit and depth of permafrost in the area of infrastructure. This will be performed via:

- Drilling, test pit and trenching program
- Conduct in-situ permeability testing
- Conduct borrow source investigation
- Geophysical surveys
- Laboratory testing program, and
- Seismic hazard study.

26.5 Environmental and Social Programs

It is recommended that the environmental and social programs be continued and are calibrated to the project as its design evolves. Data collection should be suitable to be used for baseline in an Environmental Impact Assessment, and to provide a reference against future monitoring programs.

26.6 Hydrological Testing

No hydrogeological testing data were collected for this study - additional testing is recommended.

26.7 Solvent Extraction Modelling

Sufficient test work has been done at SGS during the testwork program to validate the choice of solvent extraction for recovery of copper from leach solutions. It is recommended that further testing be performed to determine design parameters for the solvent extraction plant. In addition, it is normal practice to have the proposed solvent extraction design modelled by the supplier of the organics; this is recommended.

26.8 Further Studies

The Filo del Sol PFS has provided a technical and economic solution that is an excellent basis on which to further advance the project.

The next phases of the project are to complete optimization studies and proceed to a Feasibility Study as follows:

Phase 1 – Optimization Studies

Optimisation studies should evaluate additional aspects of the project which require or would benefit from further investigation to substantiate the design and options considered for the project, prior to or as part of a Feasibility Study (FS).

- Continued resource definition through additional exploration drilling to:
  - determine the extent of the orebody;
  - determine further confidence in the resource and reserve estimates;
  - sterilize areas where process and waste facilities have been sited; and
  - determine the opportunity to exploit an underlying sulphide mineralization, if any.
- Additional engineering and cost estimating to develop the alternative processing option (grinding and tank leaching or similar configuration) to a similar level of confidence to support a definitive determination and decision on the preferred ore processing route, including semi-quantitative evaluation of perceived operational and maintenance risks at site.
- Additional metallurgical testwork to substantiate the preferred process route and further validate the recovery assumptions and correlations carried in the PFS, including testwork to improve copper and silver recovery, particularly from high cyanide soluble copper-bearing ores, as well as static and kinetic geochemical characterization testwork for the waste dump and permanent leach pad closure facilities;
- Mine plan optimization based on the preferred process route, with dynamic production modelling to provide insight into the process inventory growth as part of the financial modelling;
- Development of updated project economics based on the results of these optimization studies.

Phase 2 – Feasibility Study (FS)

Once the Optimization Studies are completed, the project can proceed to a FS, contingent on the results of the studies maintaining a net positive financial outcome for the project.

The FS work program would consist of:

- Infill diamond drilling to convert a portion of the Indicated mineral resource into the Measured category, which will allow for the declaration of a portion of the mineral reserve as a Proven mineral reserve;
- Field investigations including geotechnical and hydrology studies to support final selection and design of the process plant, waste rock and raw water facilities;
- Pit geotechnical field work and data analysis;
- Further metallurgical testwork (if required);
- Completion of a detailed logistics study to further support the transportation costs applied to goods and materials to site, and products to port, as well as definitive costing of necessary port upgrades to manage additional seaborne and land traffic;
- Completion of a dedicated marketing study to support the product payability assumptions used as the basis for revenue calculations made in the PFS;
- Design of infrastructure including site access road, raw water supply and electrical power supplies to allow rapid mobilization upon a project construction decision;
- Equipment procurement strategy including alternate commercial options for supply of equipment and materials to site;
- Completion of a trade off study, with potential field investigation as required, to determine the economic benefits of exploiting a proximate limestone quarry, if available, for preparation of lime at site; and,
- Continuation of existing environmental baseline studies and social programs.

The estimated cost for completing this work is summarized in Table 26-1.

Table 26-1: Filo del Sol Work Program Cost Estimate

Program Component	Cost Estimate ($M)
Optimization Studies	2.0
Resource drilling	6.0
Pit geotechnical - drilling and engineering	3.0
Metallurgical testing	1.5
Infrastructure studies - field program and laboratory work	2.0
Environmental baseline studies and social programs	1.5
Engineering Studies	2.0
Total Cost	**18.0**

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