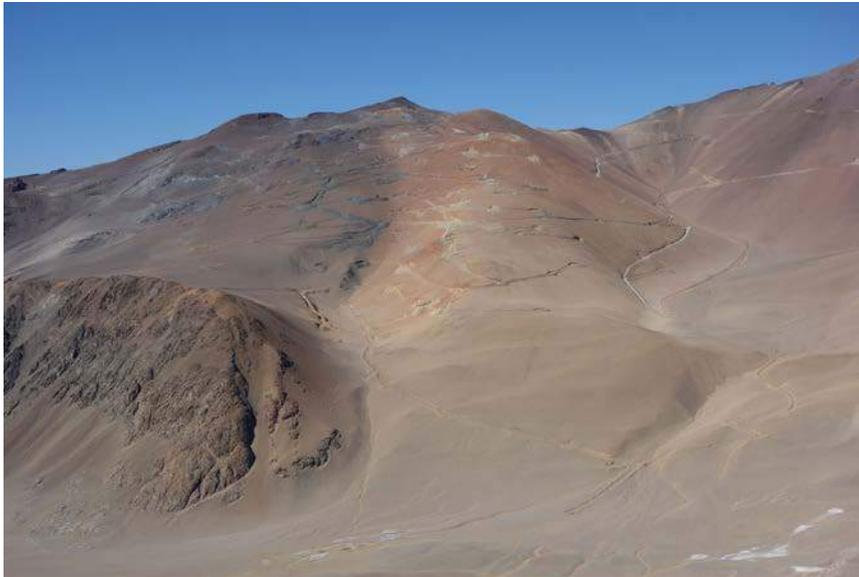




NI 43-101 Technical Report, Pre-feasibility Study for the Josemaría Copper-Gold Project, San Juan Province, Argentina

Prepared for

NGEx Resources Inc.



Prepared by



SRK Consulting (Canada) Inc.

2CN027.001

December 2018

NI 43-101 Technical Report, Pre-feasibility Study for the Josemaría Copper-Gold Project, San Juan Province, Argentina

Prepared for

NGEx Resources Inc.
2000–885 West Georgia Street
Vancouver, BC V6C 3E8
Canada

Tel: +1 604 689-7842
Web: www.ngexresources.com

Prepared by

SRK Consulting (Canada) Inc.
2200–1066 West Hastings Street
Vancouver, BC V6E 3X2
Canada

Tel: +1 604 681 4196
Web: www.srk.com

Effective Date: 20 November 2018

Report Date: 19 December 2018

Authored By:

Fionnuala Devine, P.Geo.
Merlin Geosciences Inc.

Gino Zandonai, CP (RM
CMC)
DGCS SA

Bruno Borotraeger, P.Eng.
Knight Piésold Ltd.

Robert McCarthy, P.Eng.
*SRK Consulting (Canada)
Inc.*

Neil Winkelmann, FAusIMM
*SRK Consulting (Canada)
Inc.*

Andy Thomas, P.Eng
*SRK Consulting (Canada)
Inc.*

Michael Royle, P.Geo.
*SRK Consulting (Canada)
Inc.*

Robin Kalanchey, P.Eng.
*Ausenco Engineering
Canada Inc.*

Scott Elfen, P.E.
*Ausenco Engineering
Canada Inc.*

Project No: 2CN027.001

File Name: Josemaría_Technical_Report_PFS_2CN027.001_FINAL

Copyright © SRK Consulting (Canada) Inc., 2018



Important Notice

This report was prepared as a National Instrument 43-101 Technical Report for NGEx Resources Inc. by SRK Consulting (Canada) Inc. as part of a team of consultants contracted by NGEx (“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 NGEx subject to the terms and conditions of its contracts with SRK and the Team and to the relevant securities legislation. The contracts permit NGEx 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 NGEx. 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.

© 2018 SRK Consulting (Canada) Inc.

This document, as a collective work of content and the coordination, arrangement and any enhancement of said content, is protected by copyright vested in SRK Consulting (Canada) Inc.

Outside the purposes legislated under provincial securities laws or as otherwise stipulated in SRK’s client contract, this document shall not be reproduced in full or in any edited, abridged or otherwise amended form unless expressly agreed in writing by SRK.

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
oz	troy ounce
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
CN	cyanide
NaCN	sodium cyanide

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
tpa	tonnes per annum
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	31.10348 g

Table of Contents

1	Executive Summary	1
1.1	Introduction	1
1.2	Property Description, Location and Access	1
1.3	Mineral Tenure and Surface Rights	1
1.4	History	2
1.5	Geological Setting and Mineralization	2
1.6	Exploration	3
1.7	Sample Preparation, Analyses, and Data Verification	3
1.8	Data Verification.....	4
1.9	Mineral Processing and Metallurgical Testing	4
1.10	Mineral Resource Estimates	5
1.11	Mineral Reserve Estimates	6
1.12	Mine Plan	7
1.13	Processing and Recovery Methods	8
1.14	Project Infrastructure.....	8
1.14.1	Site Wide Geotechnical.....	8
1.14.2	Tailings storage.....	8
1.14.3	Surface Water Management.....	9
1.14.4	Power	9
1.14.5	Water Supply.....	9
1.14.6	Logistics	10
1.15	Market Studies and Contracts	10
1.16	Environment, Permitting and Social.....	10
1.17	Cost Estimates.....	11
1.18	Economic Analysis.....	12
1.19	Risks and Opportunities.....	14
1.20	Conclusion and Recommendations	15
2	Introduction and Terms of Reference.....	18
2.1	Introduction	18
2.2	Responsibility.....	18
2.3	Basis of Technical Report.....	19
2.4	Effective dates	20
2.5	Qualifications of the Project Team.....	20
2.6	Site Visit	21
2.7	Declaration.....	21

3	Reliance on Other Experts	22
3.1	Ownership, Mineral Tenure and Surface Rights.....	22
3.2	Environmental, Permitting and Social.....	22
3.3	Taxation	22
4	Property Description and Location.....	23
4.1	Location	23
4.2	Mineral Tenure in Argentina	24
4.2.1	Exploration Permits (Cateo)	24
4.2.2	Exploitation Permits (Mina).....	24
4.2.3	Surface Rights	25
4.3	Environmental Regulations.....	25
4.4	Josemaría Mineral Tenure.....	26
4.4.1	Josemaría Surface Rights	27
4.5	Taxation, Royalties and Option Agreements	27
4.5.1	Corporate Income Tax.....	27
4.5.2	Provincial Mining Royalties.....	27
4.5.3	Option Agreements.....	28
4.6	Josemaría Permits.....	28
4.7	Josemaría Environmental Liabilities	29
4.8	Mining Integration and Complementation Treaty.....	29
4.9	Closure Considerations	29
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography.....	30
5.1	Accessibility	30
5.2	Climate.....	30
5.3	Local Resources and Infrastructure.....	30
5.4	Physiography	31
6	History	32
7	Geological Setting and Mineralization.....	33
7.1	Regional Geology	33
7.2	Project Geology	34
7.3	Deposit Description.....	36
7.3.1	Lithologies.....	38
7.3.2	Alteration.....	40
7.3.3	Mineral Zones	43
7.3.4	Mineralization.....	43
8	Deposit Types	46

9	Exploration	48
9.1	Grids and Surveys	48
9.2	Geologic Mapping	48
9.3	Geochemical Sampling	48
9.4	Geophysics	48
9.5	Pits and Trenches	48
9.6	Exploration Potential	49
9.6.1	Josemaría Deposit	49
9.6.2	Regional Targets	49
10	Drilling	51
10.1	Summary	51
10.2	Drill Programs	51
10.3	Geological Logging	52
10.4	Recovery	53
10.5	Collar Surveys	53
10.6	Downhole Surveys	53
10.7	Sample Length/True Thickness	53
11	Sample Preparation, Analyses, and Security	55
11.1	Surface Sampling	55
11.1.1	Talus Sampling	55
11.1.2	Chip Sampling	55
11.2	Drill Sampling	55
11.2.1	Pre-2007 Drill Sampling	55
11.2.2	NGEx Sampling	55
11.3	Density Determinations	56
11.4	Sample Preparation and Analysis	56
11.4.1	Surface and RC Samples	56
11.4.2	Core	56
11.5	Analytical and Test Laboratories	56
11.6	Quality Assurance and Quality Control	57
11.6.1	Surface and RC Sampling	57
11.6.2	Core Sampling	57
11.6.3	External Assay Checks	58
11.7	Databases	58
11.8	Sample Storage	58
11.9	Sample Security	58
12	Data Verification	59

12.1 G. Zandonai (2013).....	59
12.2 F. Devine (2014)	59
13 Mineral Processing and Metallurgical Testing.....	60
13.1 Testwork Programs.....	60
13.2 Geometallurgical Domains.....	60
13.3 Head Sample Characterization.....	62
13.4 Mineralogy	63
13.5 Physical Characterization	64
13.6 Gravity Recoverable Gold.....	66
13.7 Flotation Testing, SGS (2015)	67
13.8 Grind Size Determination.....	70
13.9 Analysis of SGS Phase II LCT Flotation Results.....	75
13.10 ALS Testwork (2018)	76
13.10.1 Feed Assay and Mineralogical Results.....	77
13.10.2 Analysis of ALS 2018 LCT Flotation Results.....	79
13.10.3 Head grade vs recovery correlation.....	81
13.11 Recovery Estimates	81
13.11.1 PFS Recovery Model Development.....	82
13.12 Metallurgical Variability	83
13.13 Deleterious Elements	83
14 Mineral Resource Estimate	84
14.1 Summary.....	84
14.2 Geological Models	84
14.3 Exploratory Data Analysis.....	84
14.4 Density Assignment	84
14.5 Grade Capping/Outlier Restrictions	85
14.6 Composites.....	85
14.7 Variography.....	85
14.8 Estimation/Interpolation Methods	86
14.9 Block Model Validation	86
14.10 Classification of Mineral Resources.....	88
14.11 Reasonable Prospects of Economic Extraction.....	88
14.12 Mineral Resource Estimate	88
14.13 Grade Sensitivity Analysis.....	90
15 Mineral Reserve Estimates.....	92
15.1 Introduction	92
15.2 Key Assumptions, Parameters and Methods	92

15.2.1	Economic Limit Definition.....	92
15.2.2	Mine Design Model	92
15.2.3	Pricing and Off-Site Costs.....	92
15.2.4	Geotechnical Slope Guidance	93
15.2.5	Metallurgical Recoveries Used for Mine Planning	96
15.2.6	Dilution and Ore Loss	97
15.2.7	NSR Calculation.....	97
15.2.8	Cost Inputs	98
15.3	Pit Optimization.....	98
15.3.1	Optimization Results	98
15.3.2	Ultimate Pit Shell Selection.....	99
15.4	Reserve Pit Design	100
15.4.1	Parameters Relevant to Mine Design	100
15.4.2	Ultimate Pit Design	102
15.5	Mineral Reserve Estimate.....	104
15.5.1	Cut-Off Grade.....	104
15.5.2	Mineral Reserve Estimate.....	104
15.6	Relevant Factors.....	104
16	Mining Methods.....	106
16.1	Open Pit Mining	106
16.1.1	Open Pit Mining.....	106
16.1.2	Equipment Selection	106
16.2	Mine Design	106
16.2.1	Pit Phase Designs.....	106
16.2.2	Waste Storage Facility Designs	112
16.3	Mine Production Schedule.....	114
16.3.1	Pre-Production Development.....	114
16.3.2	Production Mining	114
16.3.3	Equipment Requirements	115
16.4	Mining Operations.....	117
16.4.1	Drilling	117
16.4.2	Blasting	117
16.4.3	Loading	117
16.4.4	Hauling.....	118
16.4.5	Support Activities	118
16.4.6	Ancillary Equipment	118
16.4.7	Dewatering.....	118

17 Recovery Methods	119
17.1 Introduction	119
17.1.1 General Facility Description	119
17.1.2 Comminution Circuit Review	119
17.2 Processing Plant General	120
17.2.1 Crushing	120
17.2.2 Grinding	121
17.2.3 Flotation	121
17.3 Copper Concentrate Thickening and Filtration	122
17.4 Tailings Thickening and Storage	122
17.5 Fresh Water Supply	123
17.6 Process Design Criteria and Equipment Sizing	123
17.7 Major Process Requirements	125
18 Project Infrastructure.....	127
18.1 General Site Layout	127
18.2 Access Roads and Logistics.....	127
18.2.1 Road Logistics.....	127
18.2.2 Port Logistics.....	127
18.3 Power	129
18.3.1 Transmission Line Connection.....	129
18.3.2 Electrical Load.....	129
18.3.3 Main Substation	130
18.3.4 Power Distribution	130
18.4 Plant Buildings	130
18.5 Camp and Accommodations.....	131
18.6 Fuel Supply and Storage	131
18.7 Water Supply	131
18.8 Infrastructure Geotechnical Investigation	132
18.8.1 Field Program.....	132
18.8.2 Infrastructure	134
18.8.3 Groundwater	134
18.8.4 Permafrost.....	134
18.9 Tailings Storage	134
18.9.1 Introduction	134
18.9.2 Site Characteristics	135
18.9.3 Tailings Characteristics	135
18.9.4 Design Basis and Assumptions	135

18.9.5	TSF Layout, Development and Operating Strategy.....	138
18.9.6	TSF Seepage Management.....	142
18.9.7	Instrumentation	142
18.9.8	Closure and Reclamation.....	142
18.10	Site Wide Water Balance	143
18.11	Surface Water Management	145
19	Market Studies and Contracts.....	147
20	Environmental Studies, Permitting, and Social or Community Impact	150
20.1	Introduction	150
20.2	Permitting.....	150
20.3	Environmental Studies	151
20.3.1	Meteorology	151
20.3.2	Noise and Vibration.....	161
20.3.3	Glaciology	161
20.3.4	Hydrology.....	162
20.3.5	Geochemistry	162
20.3.6	Water Quality and Aquatic Biota	163
20.3.7	Soils	164
20.3.8	Flora and Fauna.....	165
20.3.9	Archaeology	167
20.4	Social Considerations	168
20.4.1	Community Identification.....	168
20.4.2	Community Relations Plan.....	168
20.4.3	Indigenous Populations.....	169
20.5	Waste and Tailings Disposal	169
20.5.1	Tailings Storage Facility.....	169
20.5.2	Site Selection	169
20.5.3	Design Parameters	169
20.6	Water Management	170
20.7	Mine Closure.....	170
21	Capital and Operating Costs	173
21.1	Capital Cost Estimate	173
21.1.1	Estimate Classification	173
21.1.2	Estimating Methodology.....	173
21.1.3	Definition of Costs.....	175
21.1.4	Exchange Rates and Foreign Content.....	175
21.1.5	Market Availability	176

21.1.6	Estimate Summary.....	177
21.2	Operating Cost Estimate.....	181
21.2.1	Common Cost Inputs	181
21.2.2	Mining Operating Cost Estimate	182
21.2.3	Processing Operating Cost Estimate	187
21.2.4	General and Administrative.....	193
22	Economic Analysis	196
22.1	Cautionary Statement	196
22.2	General	196
22.3	Production Schedule.....	201
22.4	Pricing Assumptions	205
22.5	Processing Recovery Assumptions	205
22.6	Capital Costs.....	206
22.7	Operating Costs	207
22.8	Royalties	207
22.9	Taxation	208
22.10	Off-Site Costs	208
22.11	Sensitivity Analysis.....	208
22.12	Risks and Opportunities	214
22.12.1	Risks	214
22.12.2	Opportunities.....	216
22.12.3	Note on Inferred Resources.....	217
23	Adjacent Properties	218
24	Other Relevant Data and Information	219
25	Interpretations and Conclusions	220
25.1	Mineral Resource Estimates.....	220
25.2	Mineral Reserve Estimate.....	220
25.3	Pit Geotechnical.....	221
25.4	Mining Methods.....	221
25.5	Metallurgical Testwork	222
25.6	Recovery Plan.....	222
25.7	Tailings Management Facility	223
25.8	Environmental Studies, Permitting, and Social or Community Impact	224
26	Recommendations	225
26.1	Mineral Resource Estimate.....	225
26.2	Mineral Reserve Estimate.....	225
26.3	Pit Geotechnical.....	226

26.4 Mining Method	226
26.5 Metallurgy and Processing	227
26.6 Tailings Management Facility	228
26.7 Infrastructure.....	229
26.8 Hydrology and Surface Water Management	230
26.9 Environmental Studies, Permitting, and Social or Community Impact	231
26.10 2019 Work Program	231
27 References.....	232
28 Date and Signature Page	234

List of Figures

Figure 4.1: Project location and access map	23
Figure 4.2: Mineral tenure map	27
Figure 7.1: Map showing part of the Late Oligocene to Miocene porphyry-epithermal belt	34
Figure 7.2: Regional geological map of part of the Potro area	35
Figure 7.3: Josemaría geology map	37
Figure 7.4: Josemaría vertical section 5000N lithology and alteration	39
Figure 7.5: Josemaría alteration map	41
Figure 7.6: Vertical section 5900N interpreted alteration and mineralization	42
Figure 7.7: Section 5800N mineral zones used in resource estimation	44
Figure 8.1: Porphyry copper belts and major porphyry copper deposits in the Andes	47
Figure 9.1: Exploration targets – copper values from surface talus fines geochemical sampling	50
Figure 10.1: Drill-hole collar location map, Josemaría.....	52
Figure 10.2: Example drill section 5300N, Josemaría	54
Figure 13.1: Elemental copper deportment in Phase II geometallurgical domains	64
Figure 13.2: Supzone - primary grind size effect on Cu recovery vs grade.....	71
Figure 13.3: Rhyzone - primary grind size effect on Cu recovery vs grade.....	71
Figure 13.4: Porzone - primary grind size effect on Cu recovery vs grade	72
Figure 13.5: Tonzone - primary grind size effect on Cu recovery vs grade.....	72
Figure 13.6: Supzone - regrind size effect on Cu recovery vs grade	73
Figure 13.7: Rhyzone - regrind size effect on Cu recovery vs grade	74
Figure 13.8: Tonzone - regrind size effect on Cu recovery vs grade.....	74
Figure 13.9: Porzone - regrind size effect on Cu recovery vs grade	75
Figure 14.1: Copper blocks with drill holes (Section 6855300N)	87
Figure 14.2: Gold blocks with drill holes (Section 6855300N)	87
Figure 15.1: Field program drillholes and logged sections in red (view looking south)	93
Figure 15.2: Pit shell cut-out showing geotechnical design domains and pit design sectors	95
Figure 15.3: Grade-recovery relationships by ore type for copper	96
Figure 15.4: Pit optimization results.....	99
Figure 15.5: Impact on grade of increasing mining scale for Josemaría ore types	100
Figure 15.6: Impact on waste in ore of increasing mining scale for Josemaría ore types	101
Figure 15.7: Josemaría ultimate pit design	103
Figure 16.1: Josemaría Phase 1 pit design	107
Figure 16.2: Josemaría Phase 2 pit design	108
Figure 16.3: Josemaría Phase 3 pit design	109
Figure 16.4: Josemaría Phase 4 pit design	110
Figure 16.5: Josemaría longitudinal section (A-A') of pit phase designs	111
Figure 16.6: Josemaría waste storage facilities.....	113
Figure 16.7: Mine production schedule	115
Figure 16.8: Production equipment fleet requirements	116
Figure 17.1: Simplified process flow diagram	119
Figure 18.1: General site layout - Josemaría project.....	128
Figure 18.2: Josemaría field test pit location map	133
Figure 18.3: Stage storage curve.....	137
Figure 18.4: TSF tailings deposition Year 2.....	139
Figure 18.5: Ultimate TSF tailings deposition with north and south embankments.....	140
Figure 18.6: Mine average year make-up water requirements	144

Figure 19.1: Historic copper prices	148
Figure 19.2: Historic gold prices	149
Figure 19.3: Historic silver prices	149
Figure 20.1: Water quality and aquatic biota sampling locations	164
Figure 20.2: Typical steppe habitat dominated by <i>Stipa spp</i> grasses	165
Figure 20.3: Vega in the upper Río Pirca de Bueyes dominated by <i>Oxychloe Castellanosii</i>	166
Figure 22.1: Metal production schedule graph.....	203
Figure 22.2: Mine production schedule graph.....	204
Figure 22.3: Recovered copper by lithology	206
Figure 22.4: Single factor sensitivity – net present value	211
Figure 22.5: Metals price sensitivity – net present value	212
Figure 22.6: Tornado diagram of key risk sensitivity	213
Figure 22.7: Argentine peso to US dollar historic exchange rate	214

List of Tables

Table 1.1: Mineral resource statement for the sulphide material Josemaría project, San Juan, Argentina, 7 August 2015	6
Table 1.2: Mineral resource statement for the oxide material Josemaría project, San Juan, Argentina, 7 August 2015	6
Table 1.3: Mineral reserve statement for the Josemaría project, San Juan, Argentina, 20 November 2018	7
Table 1.4: Capital cost estimate	11
Table 1.5: Operating cost estimate	12
Table 1.6: Project economics summary	13
Table 1.7: Josemaría work program cost estimate	17
Table 2.1: List of QPs and responsibilities	19
Table 2.2: List of QPs and their site visits	21
Table 4.1: Mineral tenure – Josemaría	26
Table 10.1: Drill summary table – Josemaría	51
Table 13.1: Composite description, Josemaría Phase I	61
Table 13.2: Composite description, Josemaría Phase II	61
Table 13.3: Chemical characterization of Phase I domain composites	62
Table 13.4: Chemical characterization of Phase II domain composites	62
Table 13.5: Calculated pyrite:copper sulphide ratio in Phase II composite samples	63
Table 13.6: Physical characterization of Phase I composites	65
Table 13.7: Physical characterization of Phase II composites	65
Table 13.8: Physical characterization of Phase II variability samples	66
Table 13.9: Hardness determination for Phase II composites	66
Table 13.10: Summary of LCT results for Phase II composite samples	67
Table 13.11: Final concentrate analysis from LCT testing of Phase II composite samples	68
Table 13.12: Summary of LCT results for Phase II variability samples	69
Table 13.13: Final concentrate analysis from LCT testing of Phase II variability samples	69
Table 13.14: Final concentrate analysis from LCT testing of Phase II variability samples - continued	70
Table 13.15: Master composite LCT mass and metal conservation balance for last three cycles	75
Table 13.16: Adjusted LCT results for Phase II composite samples	76
Table 13.17: Feed analysis for ALS 2018 variability and composite samples	78
Table 13.18: Mineralogical analysis for ALS composite samples	78
Table 13.19: Copper mineralogy for ALS composite samples	79
Table 13.20: Rhyzone composite sample LCT results	80
Table 13.21: Tonzone composite sample LCT results	80
Table 13.22: Porzone composite sample LCT results	81
Table 13.23: Supzone composite sample LCT results	81
Table 13.24: Summary of metallurgical performance by lithology	82
Table 13.25: Annual estimated copper, gold and silver recoveries	83
Table 14.1: Density data, Josemaría	85
Table 14.2: Mineral resource statement (sulphide material) Josemaría project, San Juan, Argentina, 7 August 2015 (assuming open pit mining methods)	89
Table 14.3: Mineral resource estimate (oxide material) for Josemaría project, San Juan, Argentina, 7 August 2015 (assuming open pit mining methods)	89
Table 14.4: Global block model quantities and grade estimates*, Josemaría project at various cut-off grades	90

Table 14.5: Global block model quantities and grade estimates*, Josemaría project at various cut-off grades	91
Table 15.1: Metal price assumptions for pit optimization	92
Table 15.2: Pit slope design criteria	95
Table 15.3: Gold and silver recoveries by ore type	97
Table 15.4: Off-site pit optimization parameters	97
Table 15.5: Ultimate pit quantities	99
Table 15.6: Geotechnical inputs used in the study	102
Table 15.7: Mineral reserve statement for the Josemaría project, San Juan province, Argentina, 20 November 2018.....	105
Table 16.1: Josemaría primary mine equipment.....	116
Table 17.1: Process plant design criteria	124
Table 17.2: Selected major process plant requirements	126
Table 19.1: Concentrate freight and TC/RC assumptions	147
Table 19.2: Commodity price assumptions.....	148
Table 20.1: Monthly mean and annual mean temperature (°C) with synthesized data set	153
Table 20.2: Estimated mean monthly potential evapotranspiration	156
Table 20.3: Long-term synthetic monthly and annual total precipitation (mm)	158
Table 20.4: Preliminary closure cost estimate	172
Table 21.1: Estimate exchange rates	176
Table 21.2: Foreign and USD content	176
Table 21.3: Capital cost estimate summary by major area (WBS Level 1)	177
Table 21.4: Initial capital cost estimate Summary Level 2.....	178
Table 21.5: Mine equipment capital cost summary with no contingency.....	180
Table 21.6: Summary of mining capital costs without contingency	181
Table 21.7: Labour rates for hourly personnel.....	183
Table 21.8: Salaries for staff (unburdened)	184
Table 21.9: Mine equipment unit costs	185
Table 21.10: Mine operating costs by activity	185
Table 21.11: Mining operating costs by year	186
Table 21.12: Processing operating cost data sources.....	188
Table 21.13: Processing operating cost summary.....	189
Table 21.14: Installed power and consumed annual power by area	190
Table 21.15: Reagent and consumable costs.....	191
Table 21.16: Process operating labour summary by function.....	192
Table 21.17: Process/operations department cost summary	194
Table 21.18: Administration cost summary.....	194
Table 21.19: Other G&A cost summary	195
Table 21.20: Contracts cost summary	195
Table 22.1: Summary of project economics.....	198
Table 22.2: LOM annual project cash flow.....	199
Table 22.3: Production schedule summary.....	202
Table 22.4: Pricing assumptions for economic analysis	205
Table 22.5: Processing recovery assumptions used for economic analysis	205
Table 22.6: Capital cost summary	207
Table 22.7: Operating costs summary	207
Table 22.8: Royalty summary	208
Table 22.9: Two-factor sensitivity – Capex and Opex	209
Table 22.10: Two-factor sensitivity – Prices and discount rate.....	209

Table 22.11: Two-factor sensitivity – Capex and metal prices	210
Table 22.12: Two-factor sensitivity – Opex and metal prices	210
Table 22.13: Sensitivity – Individual metal prices	210
Table 26.1: Josemaría work program cost estimate	231

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Technical Report, Prefeasibility Study for the Josemaria Copper-Gold Project, San Juan Province, Argentina" prepared for NGEx Resources Inc. (the "Issuer") dated December 19, 2018, with an effective date November 20, 2018 (the "Technical Report").

I, Scott Elfen, P.E., 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 Josemaria 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 1.14, 1.17, 1.19, 1.20, 18.8, 18.9, 18.10, 18.11, 20.5, 20.5.1, 20.5.2, 20.5.3, 25.7, 26.6, 26.7, 26.8, and 26.10 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 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 19 day of December 2018 in Vancouver, B.C., Canada.

"original signed"

Scott Elfen, P.E.
Ausenco Engineering Canada Inc. (Canada)

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Technical Report, Prefeasibility Study for the Josemaria Copper-Gold Project, San Juan Province, Argentina" prepared for NGEx Resources Inc. (the "Issuer") dated December 19, 2018, with an effective date November 20, 2018 (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 1.9, 1.13, 1.17, 1.19, 1.20, 13.0 (in its entirety), 17.0 (in its entirety), 18.1, 18.2, 18.3, 18.4, 18.5, 18.6, 18.7, 21.1 (not Mining component of 21.1.2 or 21.1.6), 21.2 (not including 21.2.2), 25.5, 25.6, 26.5, and 26.10 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 19 day of December, 2018 in Vancouver, B.C., Canada.

"original signed"

Robin Kalanchey, P.Eng,
Ausenco Engineering Canada Inc. (Canada)

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Technical Report, Prefeasibility Study for the Josemaría Copper-Gold Project, San Juan Province, Argentina" prepared for NGEx Resources Inc. (the "Issuer") dated December 19, 2018, with an effective date November 20, 2018 (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 1.16, 1.17, 1.19, 1.20, 20.1, 20.2, 20.3, 20.4, 20.6, 20.7, 25.8, 26.9, and 26.10 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 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 19 day of December 2018 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, Prefeasibility Study for the Josemaria Copper-Gold Project, San Juan Province, Argentina" prepared for NGEx Resources Inc. (the "Issuer") dated December 19, 2018, with an effective date November 20, 2018 (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 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 1.1, 1.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 4.0 (in its entirety), 5.0 (in its entirety), 6.0 (in its entirety), 7.0 (in its entirety), 8.0 (in its entirety), 9.0 (in its entirety), 10.0 (in its entirety), 11.0 (in its entirety), 12.2, 23.0 (in its entirety), and 24.0 (in its entirety) 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 2014 and 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 19 day of December, 2018 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, Prefeasibility Study for the Josemaria Copper-Gold Project, San Juan Province, Argentina" prepared for NGEx Resources Inc. (the "Issuer") dated December 19, 2018, with an effective date November 20, 2018 (the "Technical Report").

I, Gino Zandonai, MSc. (CSM), CP (RM CMC #0155), do hereby certify that:

- 1 I am an independent mining engineer and qualified person, residing at Camino de Los Refugios 17770, Comuna de Lo Barnechea, Santiago, Chile, tel +56 (9) 97915596, email gino.zandonai@dgcs.cl. I am employed as managing director by DGCS SA.
- 2 I am a I graduated in civil & mining engineering from the University of La Serena, Chile with degrees of Licenciado en Ciencias de la Ingenieria (B.Sc) in 1989, and from the Colorado School of Mines, Golden, Co, USA with a M.Sc. in Mining Engineering in 1999.
- 3 I am a Competent Person duly qualified in Estimation of Mineral Resources and Reserves (Record No. 0155) from the Examination Board of Competences in Mining Resources and Reserves of Chile, Law 20.235, subscribed to the Committee for Mineral Reserves International Reporting Standards (CRIRSCO #0155). I am a "qualified person" for the purposes of NI 43-101 due to my experience and current affiliation with a professional organization as defined in NI 43-101.
- 4 I have visited the property in 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 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 1.8, 1.10, 12.1, 14.0 (in its entirety), 25.1, 26.1, and 26.10 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 had prior involvement with the subject property, having completed updates to the mineral resource estimation in 2013 and 2014.
- 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 19th day of December, 2018 in Sanitago, Chile.

"original signed"

Gino Zandonai,
MSc. (CSM), CP (RM CMC #0155).

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Technical Report, Prefeasibility Study for the Josemaria Copper-Gold Project, San Juan Province, Argentina" prepared for NGEx Resources Inc. (the "Issuer") dated 19 December 2018, with an effective date 20 November 2018 (the "Technical Report").

I, Bob McCarthy, P.Eng., 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 British Columbia with a Bachelor in Applied Sciences degree in Mining and Mineral Process Engineering in 1984. 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 Canada, Brazil, Peru, Mozambique, Russia, and the United States.
3. I am a Professional Engineer registered with the Association of Professional Engineers & Geoscientists of British Columbia, license # 27309.
4. I visited the property on February 2-3, 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 1.11, 1.12, 1.17, 1.19, 1.20, 15.1, 15.2 (not including 15.2.4), 15.3, 15.4, 15.5, 15.6, 16.0 (in its entirety), 20.5, 21.1.2 (Mining component), 21.1.6 (Mining component), 21.2.2, 25.2, 25.4, 26.2, 26.4 and 26.10 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 19 day of December 2018 in Vancouver, B.C., Canada.

"original signed"

Bob McCarthy, P.Eng.
Principal Consultant (Mining)
SRK Consulting (Canada) Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Technical Report, Prefeasibility Study for the Josemaria Copper-Gold Project, San Juan Province, Argentina" prepared for NGEx Resources Inc. (the "Issuer") dated 19 December 2018, with an effective date 20 November 2018 (the "Technical Report").

I, Michael Royle, do hereby certify that:

1. I am a Principal Hydrogeologist at SRK Consulting (Canada) Inc., with an office at 2200-1066 W. Hastings St., Vancouver, BC, Canada.
2. I graduated from University of British Columbia, Vancouver Canada, in 1987 with a Bachelor of Science degree in Geology, and from University of New South Wales, Sydney Australia, in 1992 with a Masters of Applied Science in Hydrogeology. I have practiced my profession continuously as a hydrogeologist for 25 years. I have spent the balance working for different engineering consulting firms. I have been with SRK Consulting Inc since 1995, with a one year leave of absence in 1999 for volunteer work and a 2-year stint in 2013 to 2015 with Schlumberger Canada Inc.
3. I am a Professional Geoscientist registered with the EGBC, license 27830 .
4. I have not visited the property;
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 15.2.4 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 19 day of December 2018 in Vancouver, B.C., Canada.

"original signed"

Michael Royle, PGeo (BC)
SRK Consulting (Canada) Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Technical Report, Prefeasibility Study for the Josemaria Copper-Gold Project, San Juan Province, Argentina" prepared for NGEx Resources Inc. (the "Issuer") dated 19 December 2018, with an effective date 20 November 2018 (the "Technical Report").

I, Andy Thomas do hereby certify that:

1. I am a Senior Geotechnical Engineer with SRK Consulting (Canada) with an office at 22nd Floor, 1066 West Hastings Street, Vancouver, BC, V6E 3X2, Canada.
2. I am a graduate of the University of The University of Adelaide in 2004 where I obtained a Bachelor of Engineering (Civil & Environmental) and a Bachelor of Science (Geology). I am also a graduate of The University of British Columbia in 2014 where I obtained a Master of Engineering (Geological). Aside from the time spent studying at The University of British Columbia, I have practiced my profession continuously since 2005. My relevant experience includes geotechnical and hydrogeological investigations and geotechnical design of open pits in Australia, North America and South America.
3. I am a Professional Engineer registered with the Engineers and Geoscientists British Columbia, license #44961.
4. I did not visit the property before the effective date but did visit from 27 to 29 November 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 section 15.2.4, 15.4.1, 25.3, 26.3 and 26.10 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 19 day of December 2018 in Vancouver, B.C., Canada.

"original signed"

Andy Thomas, P.Eng.
SRK Consulting (Canada)

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Technical Report, Prefeasibility Study for the Josemaria Copper-Gold Project, San Juan Province, Argentina" prepared for NGEx Resources Inc. (the "Issuer") dated 19 December 2018, with an effective date 20 November (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 10 years.
3. I am registered as a Fellow of The Australasian Institute of Mining and Metallurgy (AusIMM, #323673).
4. I visited the property on 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 sections 1.15, 1.18, 1.19, 1.20, 2.0 (in its entirety), 3.0 (in its entirety), 19.0 (in its entirety) and 22.0 (in its entirety) 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 19 day of December 2018 in Vancouver, B.C., Canada.

"original signed"

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

1 Executive Summary

1.1 Introduction

Josemaría is an advanced stage copper-gold exploration project located in San Juan, Argentina. In February 2018, NGEx Resources Inc. contracted SRK Consulting (Canada) Inc., along with Ausenco Engineering Canada Inc., and Knight Piésold Ltd. to conduct a pre-feasibility study (PFS) on the project. This report, with an effective date of 20 November 2018, discloses the outcomes of the PFS and the first-time estimate of mineral reserves.

1.2 Property Description, Location and Access

Josemaría is located in the Andes Mountains, 9 km east of the Chile–Argentina border. The deposit is centred at 28.4359° S, 69.5486° W. Elevations range from approximately 4,000 metres above sea level (masl) to 4,900 masl at the ridgetop immediately south of the Josemaría deposit. Topography is mountainous, typically comprised of broad, flat-bottomed valleys with moderately steep slopes.

The best access to the project is from Copiapó, a driving distance of about 200 km, or four hours. Transit of the international boundary via this route is facilitated by the Mining Integration and Complementation Treaty, which allows for people and equipment to cross the border by either of two routes. Alternate access from Argentina is possible by major provincial highways north through San Jose de Jachal to the town of Guandacol (in La Rioja Province) and from there by approximately 150 km of regional unpaved roads and trails. Total driving time from San Juan is approximately 10 hours.

The climate in the project area is dry to arid and the temperatures are moderate to cold. Annual precipitation is about 250 mm, with snow at higher altitudes in the winter. Exploration fieldwork is generally possible from mid-October to early May. It is anticipated that mining operations will be conducted year-round.

The Josemaría project will be a greenfields development. The most important logistics centre in the region is San Juan, which has a population of about 700,000. The city has a domestic airport with scheduled flights to Buenos Aires and other Argentine cities and is home to several companies that offer services for mining and exploration.

1.3 Mineral Tenure and Surface Rights

NGEx Resources Inc. holds an indirect 100% interest in the Josemaría deposit through its Argentine subsidiary Desarrollo de Prospectos Mineros SA (Deprominsa or DPM).

For the purposes of this report, the NGEx parent and subsidiary companies are referred to interchangeably as “NGEx”.

In Argentina, NGEx holds eight exploitation licences (minas) and two exploration licences (cateos). Total holdings cover an area of approximately 16,715 ha.

NGEx has an occupancy easement for the Batidero Camp at Josemaría, and a road right-of-way, which provides access to the work area. Part of the road right-of-way is within private property. The remainder of the road, and the camp fall within the multiple usage area of the San Guillermo Provincial Reserve. Multiple usage allows mining activities.

1.4 History

Mineral rights for Josemaría were first acquired by Sr. Lirio in the early 1990s. Solitario Resources acquired these rights in 1993, with limited exploration occurring up to 2002 when Solitario (then called TNR Resource Ltd) signed an option agreement with Tenke Mining Corporation (now NGEx Resources Inc.).

The Josemaría deposit was discovered during the initial drilling campaign in the 2003/2004 field season. The first hole drilled encountered 280 metres grading 0.61% copper and 0.51 g/t gold. It was targeted on coincident talus fine copper and gold geochemical and magnetic anomalies.

Work conducted by NGEx and precursor companies has included reconnaissance prospecting; geological mapping; talus fines sampling; rock chip and trench sampling; ground-based magnetic, controlled source audio-magnetic telluric (CSAMT) and induced polarization (IP)–resistivity geophysical surveys; reverse circulation (RC) and core drilling; and metallurgical testwork.

1.5 Geological Setting and Mineralization

Based on geological features and location, the Josemaría deposit is classified as a porphyry copper-gold system.

The copper-gold mineralization at Josemaría is hosted by a Late Oligocene porphyry system developed within Permian to Triassic basement rocks. The deposit area measures ~1500 m north-south by 1000 m east-west and 600-700 m vertically from surface, within a larger alteration footprint of up to 4 km north-south by 2 km east-west. A variably-developed leached cap overlies part of the Josemaría deposit and is related to oxidation at and below the modern-day surface. The leached cap, with underlying supergene copper enrichment, ranges from 10 to 150 m in thickness, with the thicker parts preferentially developed along structures.

Mineral zones within the Josemaría deposit are defined by the relative abundance of chalcopyrite, pyrite and chalcocite, as well as the mode of occurrence of chalcocite (hypogene or supergene) and level of oxidation. Chalcopyrite and pyrite are disseminated through the potassic and overprinting chlorite-sericite zones, with minor bornite. Quartz–magnetite ± chalcopyrite veining occurs through much of the main mineralized zone, as discrete veins and locally as a more intense stockwork. Sulphide mineralization in the upper advanced argillic and sericitic domains includes a hypogene-enriched high-sulphidation assemblage of chalcocite with covellite, tennantite, and minor enargite, resulting in some of the highest hypogene grades in the deposit.

The Josemaría deposit remains open to the south, beneath a thickening cover of post-mineral volcanic rocks and also at depth.

1.6 Exploration

Work programs conducted by NGEx include geological mapping; soil, rock-chip and talus sampling; a number of geophysical surveys including IP-resistivity, magnetometer, and Mount Isa Mine's Distributed Acquisition System methodology (MIMDAS) surveys; and RC and core drilling.

Nine drilling campaigns have been carried out at the Josemaría deposit, from 2004 to 2014. Drilling at the Josemaría deposit to date totals 61,100 m in 142 drill holes, of which 48 holes (17,535 m) are RC holes, and 94 holes (43,565 m) are core holes.

Core was photographed, logged for detailed lithology, alteration and mineralization features, and (RQD) and recovery data were collected. Several of the drill holes were also logged for geotechnical information.

Core recovery data were not systematically collected on holes drilled before the 2010-2011 campaign. Core recovery from holes drilled at Josemaría between 2011 and 2014 averages 94%.

Collar locations were surveyed using a differential global positioning system (GPS) instrument.

None of the RC holes were surveyed for down-hole deflection. Diamond drill holes were surveyed for the 2009-2010 season and then systematically starting with the 2011-2012 season. Down-hole surveys were carried out at 50-m intervals on average, using a Reflex multi-shot instrument during the 2011-2012 drilling campaign. For the 2012-2013 and subsequent seasons, a SRG-gyroscope survey was completed for each drill hole by Comprobe Limitada. On average, measurements were collected at 30 m intervals down the hole.

Drill hole orientations are generally appropriate for the mineralization style. The Josemaría deposit is a porphyry system with disseminated mineralization and overlying supergene enrichment. Reported and described interval thicknesses are considered true thicknesses.

1.7 Sample Preparation, Analyses, and Data Verification

Drill holes were typically sampled on 2-m intervals.

A total of 11,754 core samples were systematically measured at Josemaría for specific gravity (SG), by NGEx technicians using the water displacement method.

Prior to 2009, ALS Chemex (ALS) in Chile was used as the primary analytical laboratory and the analytical package used was a 27-element inductively-coupled plasma atomic emission spectrometry method (ICP-AES) following a four-acid digestion, gold fire-assay atomic absorption (AA) finish and trace mercury by cold vapor/AA.

Beginning in 2009, all samples were analyzed by ACME Analytical Laboratories Ltd. (ACME) in Santiago, Chile following sample preparation at ACME's sample preparation laboratory in Mendoza, Argentina (Josemaría).

Sample preparation for core and RC chips from the Josemaría deposit included drying, crushing to better than 85% passing 10-mesh and pulverizing to 95% passing 200-mesh. Sample digestion was done by a multi-acid attack. Gold was determined by fire assay with an atomic absorption spectroscopy (AAS) finish based on a 30 g sample. A suite of 37 elements, including Cu, was determined by ICP-emission spectroscopy (ES) analyses. Samples analyzed before the 2010-2011 campaign had copper re-assayed by AAS only if the ICP result exceeded the detection upper limit of 10,000 ppm. Beginning in 2010-2011, all samples with copper grades over 5,000 ppm Cu were re-assayed by AAS. Starting in 2011-2012, copper determinations in all samples were done by both ICP and AAS. Mercury concentration was determined by cold vapour/AA in all samples up to 2010-2011.

Prior to 2009, quality control was limited to the preparation and analysis of field duplicates from the drill samples.

A rigorous quality control (QC) protocol was implemented in 2009–2010, beginning with drill hole JMDH08, and has been followed since then with minor variations. Quality assurance and quality control (QA/QC) includes insertion of standard reference materials (SRMs), coarse blank samples and duplicate samples. A set of 183 coarse rejects from the 2012 drill campaign at Josemaría were selected for re-assaying at SGS Laboratories.

1.8 Data Verification

Data verification has been conducted by independent consultants in support of technical reports on the project. This work has included field visits (drill collar monumenting; location checks for selected drill collars); witness sampling; spot checks of the assay database against assay certificates; reviews of the lithology and alteration information in drill core against drill logs; reviews of collar elevations in the database against collar elevations in the digital elevation model provided by NGEx; downhole survey deviation reviews; reviews of QA/QC data including standard, blank and duplicate sample performances; and a review of check sampling on pulps completed by a check laboratory.

1.9 Mineral Processing and Metallurgical Testing

A two-phase metallurgical testwork program for Josemaría was conducted at SGS Minerals S.A. laboratories in Santiago, Chile in 2014 and 2015. SGS Minerals S.A. laboratories is independent of NGEx. Multiple composite and variability samples were tested for mineralogy, physical characterization, gravity concentration, conventional sulphide flotation (open/locked cycle tests with different flowsheets), flotation tailings cyanidation and solids settling. Based on the testwork completed to date, life of mine metal recovery is expected to be 86% for copper, 71% for gold, and 59% for silver. Copper concentrate grades are expected to average 25% over the life of the mine. It is anticipated that the concentrate will be relatively free of impurities, endowed with precious metals and readily marketable.

Recent metallurgical testing performed at ALS, Kamloops, BC focused on confirming and improving the bulk concentrate flotation results achieved by SGS. ALS is also independent of NGEx. The work generally confirmed the results and identified several areas that could be

explored to increase the overall copper concentrate grades while maintaining similar recoveries. Additional testwork is planned during the 2018/2019 field season to optimize this work.

The Josemaría concentrates showed no major deleterious elements. However, mill feed blending strategies should be employed to generate flotation concentrates that have high copper grades whilst maintaining minimal deleterious element levels.

1.10 Mineral Resource Estimates

The Josemaría mineral resource estimate update is based on data from 116 drill holes totalling 52,725 m of drilling, of which 34 holes (13,164 m) are reverse circulation (RC) and 82 holes (39,561 m) are core holes. The total length of assayed intervals is 51,092 m and there are 27,344 assays.

A two-dimensional (2D) interpretation based on logged data was completed by NGEx geologists on east–west oriented sections spaced 100 m apart. Two-dimensional lines were then exported from GEMS and imported into the Leapfrog geological modelling software and the final three-dimensional (3D) wireframe solids were constructed.

Ordinary kriging (OK) and inverse distance squared (ID2) weighting interpolation was done in a single pass. All elements (Cu, Au, Ag, Mo, As, S and Fe) were interpolated using OK.

Model validation was carried out using visual comparison of blocks and sample grades in plan and section views; statistical comparison of the block and composite grade distributions; and swath plots to compare OK, ID2 and NN (nearest neighbour) estimates.

The classification of the mineral resources was done as a two-step process. An initial step which considered the geostatistical analysis of copper grades in the deposit was modified by a final revision to ensure consistency in the classification.

To evaluate the potential for reasonable prospects of eventual economic extraction for Josemaría, a Whittle pit shell was generated.

The analysis was done based on the copper equivalent (CuEq) grades in the block model. CuEq was calculated using metal prices of US\$3.00/lb copper, US\$1,400/oz gold and US\$23/oz Ag and were adjusted for metallurgical recoveries. Mineral resources for Josemaría are reported at a 0.2% CuEq grade for the sulphide material.

The mineral resource estimate for Josemaría, assuming open pit mining methods is reported using the 2014 CIM Definition Standards. Indicated and Inferred classifications only have been estimated; no measured mineral resources were classified.

The mineral resource estimates were prepared by Mr. Gino Zandonai, RM CMC. The Josemaría estimate has an effective date of 7 August 2015.

Mineral resource statements for Josemaría are presented in Table 1.1 and Table 1.2. Mineral Resources that are not mineral reserves do not have demonstrated economic viability.

Table 1.1: Mineral resource statement for the sulphide material Josemaría project, San Juan, Argentina, 7 August 2015

Cut-off (CuEq %)	Quantity	Grade				Contained Metal		
	(million tonnes)	Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)	Cu (billion lbs)	Au (million ozs)	Ag (million ozs)
Indicated Mineral Resources								
0.2	1,066	0.31	0.22	1.0	0.44	7.4	7.4	34.5
Inferred Mineral Resources								
0.2	404	0.24	0.15	0.8	0.33	2.0	2.0	10.8

Table 1.2: Mineral resource statement for the oxide material Josemaría project, San Juan, Argentina, 7 August 2015

Cut-off (Au g/t)	Quantity (million tonnes)	Grade			Contained Metal	
		Au (g/t)	Ag (g/t)	Cu (%)	Au (thousand ozs)	Ag (thousand ozs)
Josemaría Indicated Mineral Resources						
0.2	43	0.32	1.2	0.15	450	1,610
Josemaría Inferred Mineral Resources						
0.2	4	0.32	1.0	0	48	145

Notes to accompany Josemaría mineral resource statement:

1. Mineral resources have an effective date of 7 August 2015. The Qualified Person for the estimate is Mr. Gino Zandonai, RM CMC.
2. Mineral resources that are not Mineral Reserves do not have demonstrated economic viability.
3. Sulphide mineral resources are reported using a copper equivalent (CuEq) cut-off grade. CuEq was calculated using US\$3.00/lb copper, US\$ 1,400/oz gold and US\$23/oz Ag and was based on copper, gold and silver recoveries obtained in metallurgical testwork on four composite samples representing the rhyolite, tonalite, porphyry and supergene zones. Copper recoveries for the rhyolite, tonalite and porphyry zones were calculated as a function of copper grade, ranging from a low of 81% to a high of 97%. Copper recovery in the supergene zone was fixed at 85%. Gold recoveries were fixed between 62% and 73% and silver recoveries were fixed between 53% and 75% depending on the zone.
4. Mineral resources are reported within a conceptual Whittle™ pit that uses the following input parameters: Cu price: US\$3.00/lb, mining cost: US\$2.20/t, process cost (including G&A): US\$7.40/t processed, copper selling cost: US\$0.35/lb and Over-all slope angle of 42°.
5. Mineral resources (sulphide) have a base case estimate using a 0.2% CuEq grade; mineral resources (oxide) are reported using a 0.2 g/t Au cut-off grade.
6. Totals may not sum due to rounding as required by reporting guidelines.

1.11 Mineral Reserve Estimates

The open pit mineral reserves for Josemaría are reported within a pit design based on open pit optimization results. 3-D mine designs were completed using MineSight software.

Mineral reserves were classified using the 2014 CIM Definition standards. Indicated mineral resources were converted to probable mineral reserves by applying the appropriate modifying

factors – those being dilution (5%) and ore loss (1%). There are no measured resources and thus no proven reserves. Cut-off grades (as net smelter return) were applied to both prime mill feed (\$4.95/t) and stockpiled ore (\$5.95/t) to designate mineral reserve from waste.

The open pit mine design process resulted in open pit mining reserves of 1,008 Mt with average grades of 0.29% copper, 0.21 g/t gold, and 0.92 g/t silver, for an overall copper equivalent grade of 0.41% CuEq. The mineral reserve statement, as of 20 November 2018, for the Josemaría project is presented in Table 1.3.

Table 1.3: Mineral reserve statement for the Josemaría project, San Juan, Argentina, 20 November 2018

Category (all domains)	Tonnage (Mt)	Grade				Contained Metal		
		Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)	Cu (billion lbs)	Au (million oz)	Ag (million oz)
	Proven	-	-	-	-	-	-	-
Probable	1,008	0.29	0.21	0.92	0.41	6.5	6.5	28.8
Total	1,008	0.29	0.21	0.92	0.41	6.5	6.5	28.8

Notes to accompany Josemaría mineral reserve statement:

1. Mineral reserves have an effective date of 20 November 2018. The Qualified Person for the estimate is Mr. Robert McCarthy, P.Eng.
2. The mineral reserves were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), Definition Standards for Mineral Resources and Reserves, as prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.
3. The mineral reserves were based on a pit design which in turn aligned with an ultimate pit shell selected from a Whittle™ pit optimization exercise. Key inputs for that process are:
 - Metal prices of \$2.95/lb Cu, \$1,225/oz Au; \$19.00/oz Ag
 - Mining cost of \$1.80/t ore and waste at a reference elevation of 4180 m, plus cost adjustments of \$0.016/t/10m bench above reference and \$0.025/t/10 m bench below reference
 - Processing cost of \$3.60/t ore milled
 - General and administration cost of \$0.80/t milled
 - Sustaining capital costs of \$0.55/t
 - Pit slope angles varying from 42° to 46°
 - Process recoveries are based on grade. The average recovery is estimated to be 85% for Cu, 65% for Au and 66% for Ag
 - CuEq was calculated using total payable revenue from all metals in the mine plan, converting to payable copper, and back calculating for grade based on life of mine average copper recoveries and payables
4. Mining dilution is estimated to be 5%.
5. Ore loss is assumed to be 1%.
6. The mineral reserve has an economic cut-off, based on NSR, of \$4.95/t for direct mill feed.
7. There are 711 Mt of waste in the ultimate pit. The strip ratio is 0.71 (waste:ore).
8. All figures are rounded to reflect the relative accuracy of the estimate. Totals may not sum due to rounding as required by reporting guidelines.

1.12 Mine Plan

The study contemplates conventional open pit mining methods using autonomous haul trucks. Mine planning incorporates stockpiling strategies to focus on the early extraction of the highest-grade ore in addition to deferring waste stripping. Including pre-stripping, the open pit will be in operation for 23 years, delivering ore to the mill for 20 years, with a life of mine strip ratio of 0.71:1.

A maximum mining rate of approximately 115 Mt per year (including waste) is required to provide the nominal 150,000 tonnes per day (tpd) of ore to the mill. A total of 1,008 Mt of ore is expected to be processed over the life of the mine.

1.13 Processing and Recovery Methods

Primary crushed ore will be transported via surface conveyor to a coarse ore stockpile, and then transferred to the process plant. The process considers the use of high-pressure grinding rolls (HPGR) as part of a three-stage crushing circuit, followed by conventional ball mill grinding and sulphide flotation. Design throughput is 150 ktpd. Water obtained from the concentrate thickener, tailings thickener and concentrate filter will be recovered and sent back to the process plant to be used as make-up water. Additional water as needed to meet processing requirements will be reclaimed from the tailings storage facility (TSF).

1.14 Project Infrastructure

Key infrastructure at Josemaría will include the open pit, process plant, filter plant, ancillary administrative buildings, construction and operations camp, truck shop, electrical distribution system, water and emergency ponds, and site security.

1.14.1 Site Wide Geotechnical

As part of the design of the TSF, primary crusher, processing facilities, waste rock storage 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 rock storage facilities, stockpiles, and TSF site, 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. In addition, one geophysics line was performed along the longitudinal axis of the proposed TSF.

The Josemaría project infrastructure is situated on alluvium and colluvium typically less than 5 m thick that is underlain by weathered bedrock, except in the bottom of the TSF valley where the alluvium ranges from 66 to 172 m along the center of the valley.

1.14.2 Tailings storage

The final location of the center of the TSF will be 4 km south of the concentrator in a large open valley basin. It has been sized to contain 1,008 Mt of tailings, operational water, and runoff from the design Intensity-Duration-frequency (IDF) event equal to the “one third between 1,000 years annual exceedance probability (AEP) and the Probable Maximum Flood (PMF)” event. However, the tailings facility has the capacity to handle the PMF as tailings deposition governs the height of the TSF embankments. The impoundment is located in the Pirca de Los Bueyes basin with the starter embankment located at the southeast end of this basin. The TSF consists of a main embankment (215 m high) and two smaller embankments for protection of the regional water during operations and closure, while containing tailings within a geotechnically stable engineered

facility. The main embankment will utilize centerline construction due to its size, while the two smaller embankments will utilize phased downstream construction.

The TSF will be closed with an alluvial cap to prevent migration of fugitive tailings from the impoundment after closure, along with riprap-lined diversion channels to capture and safely pass the attenuated PMF storm runoff from the facility and watersheds above the facility. In preparation for closure, there will be an expansion of the operation spillway developed along the northern abutment of the main embankment to discharge surface flow safely out of the impoundment.

1.14.3 Surface Water Management

Components of the surface water management system include diversion channels, diversion berms, road drainage ditches, and surface drainage channels (both natural and constructed) directing runoff to ditches and channels, flow-through underdrains in the waste rock storage facilities, dumps, and sediment ponds in key locations immediately downstream of impacted areas.

1.14.4 Power

Power for the site is assumed to be supplied with electricity through a 250-km long, 220 kV, single-circuit power transmission line connected to the Guanizuil substation in San Juan Province, Argentina. Maximum demand load is estimated to be 150 MW. A price assumption of \$0.075/kWh was used for long-term power supply. Power supply alternatives from Chile were also considered as the costs were comparable, but the relative simplicity of the supply from Argentina (avoiding cross-border issues) resulted in supply from Argentina as the preferred solution. The power infrastructure will include:

- A 220-kV overhead transmission line from Guanizuil substation
- A main power substation beside the process plant

Power will be distributed at 13.8 kV via localized mine grid. A back-up generator will also be located on site to support key facilities in an emergency.

1.14.5 Water Supply

The site water balance used a deterministic model developed in GoldSim® based on a monthly timestep for the entire duration of mining operations to determine make-up water requirements. Three precipitation scenarios were considered in the model; dry year, average year and wet year.

Due to the dry conditions inherent to the arid climate of the project area, the TSF will be used as an on-site water storage pond, which will collect runoff from the TSF watershed, including extreme runoff events (i.e., rainfall and snow melt). The TSF is designed with sufficient storage capacity to avoid any discharges to the environment.

Available information such as hydrogeology, hydrology, and climate data were used in the model (PFS level). Production ramp-up, thickening improvement, accumulative storage and tailings deposition plan were considered in the model.

During wet months of the wet precipitation cycles, the TSF is able to provide the entire amount of water required by the process plant. Therefore, the minimum pump capacity of 1,400 L/s is recommended for the TSF reclaim water barge.

The make-up water requirements based on the water balance for the mean yearly precipitation cycle is 555 L/s during the dry season and 138 L/s during the wet season, while the average monthly make-up water is 353 L/s. During the dry precipitation cycles (drought) the maximum make-up water requirements are 562 L/s and the average monthly make-up water is 562 L/s.

1.14.6 Logistics

Concentrate will be transported from the Josemaría mine site to the port at Caldera, Chile using a standard combination of tandem drive tractor articulated with a tipper semi-trailer. Approximately 57 km of light vehicle road will require upgrading to connect the Josemaría mine site to the Argentina Ruta Nacional 76 highway. The route then enters Chile through the international border crossing at Pircas Negras and continues through Copiapó to Caldera. The travel distance between the site and the port is approximately 343 km. Export of concentrate was assumed to be from the Punta Padrones Terminal.

1.15 Market Studies and Contracts

The product of the mine will be a conventional copper concentrate with a concentrate grade forecast to average 25.1% Cu over the life-of-mine. No deleterious elements are forecast to be present in the concentrate and no penalties have been modelled.

No contracts in relation to concentrating, smelting, refining, transportation, handling, sales and hedging, and forward sales, nor any other marketing arrangements are currently in place. Pursuant to the terms of NGEx's acquisition of its previous partner's (Japan Oil, Gas, and Metals National Corporation or "JOGMEC") 40% interest in the Josemaría project, JOGMEC holds an option to purchase up to 40% of the material produced from any mine on the property.

The price assumptions used for this study are: \$3.00/lb copper, \$1,300/oz gold and \$20.00/oz silver.

1.16 Environment, Permitting and Social

NGEx 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 the Mining Code – Environmental Protection for Mining Activity. This is a recognized process with successful precedent in the San Juan province of Argentina. There are no known environmental issues that could materially impact the ability of NGEx to extract the mineral resources at the Josemaría project.

1.17 Cost Estimates

The capital cost estimate is summarized in Table 1.4. It is stated in United States dollars (USD) with a base date of fourth quarter 2018 and with no provision for forward escalation. 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 Class 4 PFS estimate with an accuracy range of +/-25%.

Table 1.4: Capital cost estimate

Cost Type	WBS LVL 1	LVL 1 Description	Initial (USD \$000)	Sustaining (USD \$000)	Total (USD \$000)
Direct	1000	Mine	262,990	254,917	517,907
	2000	Crushing	485,806	0	485,806
	3000	Process	456,618	0	456,618
	4000	On-site infrastructure	162,654	513,634	676,288
	5000	Off-site infrastructure	279,641	0	279,641
Direct Subtotal			1,647,709	768,551	2,416,260
Indirect	6000	Indirects	309,557	66,389	375,946
	7000	Project delivery	245,228	0	245,228
	8000	Owners costs	83,494	0	83,494
	9000	Provisions	474,657	25,492	500,149
		Indirect Total	1,112,937	91,881	1,204,818
PROJECT TOTAL			2,760,646	860,432	3,621,078

A summary of operating costs for the project is provided in Table 1.5.

Table 1.5: Operating cost estimate

Category	Life-of-Mine Total (\$M)	Unit Rates
Mining	\$2,802	\$1.63/t moved (including rehandle) \$2.78/t milled
Processing	\$3,271	\$3.23/t milled
Site G&A	\$712	\$0.71/t milled
Off-site Infrastructure (not including roads)	\$19	\$0.02/t milled
Less capitalized Opex	-\$49	n/a
Total Operating Costs	\$6,754	\$6.74/t milled

1.18 Economic Analysis

The Economic Analysis of the Josemaría project indicates that the project as conceived has the potential for economic execution.

The base-case after-tax net present value (NPV) evaluated at a discount rate of 8% is \$2.03B. The after-tax internal rate of return (IRR) is 18.7%. A summary of KPIs and economic analysis inputs is shown in Table 1.6.

A positive valuation is maintained across a wide range of sensitivities on key assumptions such as prices, costs, metallurgical recoveries and schedule. The operating margin averages 65% over the life-of-mine for the base case.

Table 1.6: Project economics summary

Project Metric	Units	Value
Pre-Tax NPV @ 8%	\$ B	2.91
Pre-tax IRR	%	21.4
After-Tax NPV @ 8%	\$ B	2.03
After Tax IRR	%	18.7
Undiscounted After-Tax Cash Flow (LOM)	\$ B	6.58
Payback Period from start of processing (undiscounted, after-tax cash flow)	years	3.4
Initial Capital Expenditure	\$ M	2,761
LOM Sustaining Capital Expenditure (excluding closure)	\$ M	860
LOM C-1 Cash Costs (Co-Product exc. royalty)	\$/lb CuEq.	1.26
Nominal Process Capacity	ktpd	150
Mine Life	years	20
LOM Mill Feed	kt	1,008,078
LOM Grades		
Copper	%	0.29
Gold	grams per tonne	0.208
Silver	grams per tonne	0.920
LOM Waste Volume	kt	711,556
LOM Strip Ratio (Waste:Ore)	ratio	0.71
First Three Years Average Annual Metal Production		
Copper	tonnes	170,000
Gold	ounces	352,000
Silver	ounces	1,026,000
LOM Average Annual Metal Production		
Copper	tonnes	123,000
Gold	ounces	232,000
Silver	ounces	791,000
LOM Average Process Recovery		
Copper	% contained metal	86
Gold	% contained metal	71
Silver	% contained metal	59

1.19 Risks and Opportunities

The main opportunities identified for the project include:

- Higher metals pricing: the project has significant leverage to copper prices
- Delineation of additional mineralization or potential conversion of inferred resources into the mine plan (particularly higher-grade material) through further exploration drilling
- Potential to heap-leach oxide gold mineralization
- Optimization of the mine plan, including dynamic operating strategies and cut-off grade policies
- Improvements in process plant throughput, concentrate grades, and metallurgical recoveries through additional testwork
- Determining a more cost-effective power supply option
- Potential for regional synergies with other mining operations
- Optimization of the tailings deposition plan could reduce embankment volumetrics and consequently reduce both capital and sustaining capital costs for the TSF

Risks noted with the PFS assumptions include:

- Long-term depressed metals prices and fluctuations with metals pricing
- Political risks and uncertainties affecting legislation, regulatory requirements or general business climate in Argentina
- Inflation and increased prices for infrastructure, equipment and consumables, resulting in changes to operating and capital cost estimates
- Unforeseen geotechnical issues in the foundation of the TSF embankments that might increase construction costs
- Limited geochemistry of the tailings suggests it is potentially acid generating (PAG) and this has been taken into account, but any significant changes or unforeseen geochemistry issues may affect TSF design
- Implementation of additional monetary controls or restrictions on imports by the Argentine government
- Obtaining the appropriate permits to support project construction and operation
- Timely completion of the environmental permitting process
- Environmental concerns that may be raised due to proximity concerns: the proximity of the El Potro glacial area, rock glaciers in the broader periglacial environment, and cultural heritage sites

- Uncertainties in long term management of acid rock drainage and metal leaching from mine, waste and tailings
- Continuity and effectiveness of community relations programs
- Large-scale structures (that were not able to be considered in the PFS design) dictating stability of portions of the pit walls requiring shallower design angles.

1.20 Conclusion and Recommendations

Mineral resource estimates presented in this report represent the global mineral resources for the Josemaría deposit as of 7 August 2015. 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.

SRK confirms that the Josemaría mineral resource can be converted to a sizeable mineral reserve through application of open pit mining methods. SRK perceives little risk in the mineral reserves as stated. There are several opportunities to optimize the project and increase project value and it is recommended that trade-off studies related to the location/capacity of the WSF versus haulage cost and low-grade ore stockpile capacity versus cut-off grade strategy be completed.

Historical large-scale structures data was not included in the pit geotechnical assessment and SRK recognises that these input data limitations mean that the recommended design pit walls in this study may be steeper than achievable. SRK recommends completion of a comprehensive pit geotechnical program, including photologging of existing drill core, completion of dedicated geotechnical drillholes in all design domains, pit sectors and lithology units, and instrumentation and testing of all geotechnical drillholes.

The Josemaría project is amenable to open pit mining methods. However, as the project is in a high elevation operating environment, where labour productivity and equipment utilization can be impacted, SRK has evaluated 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, it is SRK's opinion that autonomous haulage has sufficiently matured to be considered for Josemaría. SRK recommends that additional autonomous technologies, such as drilling, be considered for the project and that the cut-off grade strategy be further optimized to sustain higher grade feed to the mill by maintaining higher mining rates when the mine plan currently calls for these to decrease.

The Josemaría metallurgical testwork programs (Phases I and II) were conducted at SGS Minerals S.A. (SGS) in Santiago, Chile (SGS, 2015). Further testwork was undertaken as part of this PFS to confirm the work that had been completed. The work was undertaken at the ALS Laboratory in Kamloops, British Columbia and focussed on the grade-recovery relationship for the different zones of the deposit. The ALS testwork showed a potential to further improve recovery with some grade adjustments in the next phase of testwork. A thorough testwork campaign is strongly recommended before the commencement of a feasibility study and should address primary grind size, regrind size, and optimum reagent types and flotation conditions.

Ausenco developed a phased TSF design that has a storage capacity of 1,008 Mt of tailings, corresponding to approximately 20 years at a rate of 54 Mt/y and was designed in accordance to international and national (Argentina) best available technology (BAT). It is recommended that comprehensive infrastructure studies be completed, including: geotechnical studies of the plant, TSF and WSF sites, and other project related locations; hydrological/hydrogeological studies; and tailings and TSF characterization studies (geotechnical investigations and laboratory testwork).

NGEx has conducted environmental studies in the project area for a number of years, which provides a defensible baseline. Mine development will require an Environmental Impact Assessment and permitting under the Mining Code. This is a recognized process with successful precedent in the San Juan province of Argentina. There are no known environmental issues that could materially impact the project. It is recommended that the environmental and social programs be continued and are calibrated to the project as its design evolves.

The Josemaría project has robust economics with an NPV of \$2.03B, an IRR of 18.7% and a payback period of 3.4 years. A positive valuation is maintained across a wide range of sensitivities on key assumptions such as prices, costs, metallurgical recoveries and schedule.

The work program for the upcoming year consists 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
- Pit geotechnical field work and data analysis
- Metallurgical testing
- Infrastructure studies, including:
 - Geotechnical studies of the plant site, TSF and WSF sites, and other project related locations
 - Hydrogeological studies
 - Tailings and TSF characterization studies (geotechnical investigations and laboratory testwork)
- Continuation of existing environmental baseline studies and social programs
- Initiation of a Feasibility Study

The estimated cost for completing this work is summarized in Table 1.7.

Table 1.7: Josemaría work program cost estimate

Program Component	Cost Estimate (\$000)
Resource drilling	5,200
Pit geotechnical - drilling and engineering	4,100
Metallurgical testing	1,200
Infrastructure studies - field program and laboratory work	3,000
Environmental baseline studies and social programs	1,000
Feasibility study	3,000
Total Cost	17,500

2 Introduction and Terms of Reference

2.1 Introduction

Josemaría is an advanced stage copper-gold exploration project located in San Juan Province, Argentina. In February 2018, SRK Consulting (Canada) Inc., along with Ausenco Engineering Canada Inc., and Knight Piésold Ltd., were contracted to conduct a PFS on the project. Work commenced with site visits in February 2018, followed by strategic mine planning, and geotechnical and metallurgical testing – progressing through to core PFS activities from July to November 2018.

This technical report discloses the outcomes of the PFS, including the first-time reporting of mineral reserves for the Josemaría project.

2.2 Responsibility

The PFS Qualified Persons (QP), as defined by CIM, and their areas of responsibility are summarized in Table 2.1.

Table 2.1: List of QPs and responsibilities

Qualified Person	Company	Area(s) of Responsibility
Scott Elfen	Ausenco	1.14, 1.17, 1.19, 1.20, 18.8, 18.9, 18.10, 18.11, 20.5, 20.5.1, 20.5.2, 20.5.3, 25.7, 26.6, 26.7, 26.8, 26.10
Robin Kalanchev	Ausenco	1.9, 1.13, 1.17, 1.19, 1.20, 13.0 (in its entirety), 17.0 (in its entirety), 18.1, 18.2, 18.3, 18.4, 18.5, 18.6, 18.7, 21.1 (not Mining component of 21.1.2 or 21.1.6), 21.2 (not including 21.2.2), 25.5, 25.6, 26.5, 26.10
Bruno Borntraeger	Knight Piésold	1.16, 1.17, 1.19, 1.20, 20.1, 20.2, 20.3, 20.4, 20.6, 20.7, 25.8, 26.9, 26.10
Fionnuala Devine	Merlin Geoscience	1.1, 1.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 4.0 (in its entirety), 5.0 (in its entirety), 6.0 (in its entirety), 7.0 (in its entirety), 8.0 (in its entirety), 9.0 (in its entirety), 10.0 (in its entirety), 11.0 (in its entirety), 12.2, 23.0 (in its entirety), 24.0 (in its entirety)
Gino Zandonai	DGCS SA	1.8, 1.10, 12.1, 14.0 (in its entirety), 25.1, 26.1, 26.10
Bob McCarthy	SRK	1.11, 1.12, 1.17, 1.19, 1.20, 15.1, 15.2 (not including 15.2.4), 15.3, 15.4, 15.5, 15.6, 16.0 (in its entirety), 20.5, 21.1.2 (Mining component), 21.1.6 (Mining component), 21.2.2, 25.2, 25.4, 26.2, 26.4, 26.10
Michael Royle	SRK	15.2.4
Andy Thomas	SRK	15.2.4, 15.4.1, 25.3, 26.3, 26.10
Neil Winkelmann	SRK	1.15, 1.18, 1.19, 1.20, 2.0 (in its entirety), 3.0 (in its entirety), 19.0 (in its entirety), 22.0 (in its entirety)

2.3 Basis of Technical Report

The following technical reports have been filed on the Josemaría project by NGEx:

- Zandonai, G.A., Carmichael, R.G. and Charchaflié D., 2013: Updated Mineral Resource Estimate for the Josemaría Property, San Juan Province, Argentina: technical report prepared by Behre Dolbear and NGEx Resources Inc., effective date 27 September 2013.
- Zandonai, G., 2013: Second Updated Mineral Resource Estimate for the Josemaría Property, San Juan Province, Argentina: technical report prepared by Behre Dolbear for NGEx Resources Inc., effective date 27 September 2013, amended 24 March 2014.
- Ovalle, O., 2016, et al. 2016: Constellation Project incorporating the Los Helados Deposit, Chile and the Josemaría Deposit, Argentina NI 43-101 Technical Report on Preliminary Economic Assessment, prepared by Alfonso Ovalle, RM CMC; Cristian Quiñones, RM CMC; Cristian Quezada, RM CMC; David Frost, FAusIMM; and Vikram Khera, P.Eng., all of whom are with Amec Foster Wheeler International Ingeniería y Construcción Limitada; and by Gino

Zandonai, RM CMC, of DGCS SA, filed under the Corporation's profile on SEDAR on 11 April 2016.

Technical reports prepared prior to NGEx's involvement in the Josemaría deposit include:

- Chapman, J., and Harrop, J., 2004: Summary Report for the Batidero Project, San Juan Province, Argentina: report prepared by Tamri Geological Ltd and Cyberquest Geoscience Ltd. for TNR Gold Corp, 24 August 2004.
- Harrop, J., 2005: Summary Report for the Josemaría-Batidero Project, San Juan Province, Argentina: technical report prepared by Cyberquest Geoscience Ltd. for Tenke Mining Corporation, effective date 20 April 2005.
- Nilsson, J., and Rossi, M., 2006: Preliminary Resource Estimate for the Josemaría Project, San Juan Province, Argentina: technical report prepared by Nilsson Mine Services Ltd and Geosystems International for Tenke Mining Corporation, effective date 12 January 2006.
- Nilsson, J., and Rossi, M., 2007: Exploration Update for the Josemaría Project, San Juan Province, Argentina: technical report prepared by Nilsson Mine Services Ltd and Geosystems International for Suramina Resources Inc., effective date 15 June 2007.

2.4 Effective dates

The following effective dates are applicable to this report:

- Date of mineral resource estimate for Josemaría: 7 August 2015
- The overall effective date of this PFS report is taken to be the date of the financial analysis and is 20 November 2018

2.5 Qualifications of the Project Team

The SRK Group comprises over 1,400 professionals, offering expertise in a wide range of resource engineering disciplines. SRK has a demonstrated track record in undertaking independent assessments of Mineral Resources and Mineral Reserves, project evaluations and audits, technical reports and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies and financial institutions worldwide. The SRK Group has worked with a large number of major international mining companies and their projects, providing mining industry consultancy service inputs. SRK has extensive PFS experience for open pit projects, considerable cold weather/extreme environment project experience and significant South American experience.

Ausenco is a global diversified engineering, construction and project management company providing consulting, project delivery and asset management solutions to the resources, energy and infrastructure sectors. Ausenco's experience in copper-gold projects ranges from conceptual, pre-feasibility and feasibility studies for new project developments to project execution with EPCM and EPC delivery. Ausenco is currently engaged on a number of global projects with similar characteristics and opportunities to the Josemaría project.

Knight Piésold is an international consulting company providing engineering and environmental services for the mining, power, water, transportation and construction sectors. Knight Piésold has significant experience with design, environmental assessment and permitting of mining projects in Argentina and throughout South America.

2.6 Site Visit

The list of QPs and dates of their site visits are summarized in Table 2.2.

Table 2.2: List of QPs and their site visits

Qualified Person	Company	Date(s) of Site Visit
Scott Eifen	Ausenco	February 2018
Robin Kalanchey	Ausenco	February 2018
Bruno Borntraeger	Knight Piésold	March 2018
Fionnuala Devine	Merlin Geoscience	January/February 2014, May 2014, March 2018
Gino Zandonai	DGCS SA	2014
Bob McCarthy	SRK	February 2018
Michael Royle	SRK	Did not visit site
Andy Thomas	SRK	Did not visit site as of effective date (subsequently visited 27-29 November 2018)
Neil Winkelmann	SRK	February 2017

2.7 Declaration

The opinions of SRK, Ausenco and Knight Piésold contained herein and effective 20 November 2018, are based on information collected throughout the course of our investigations, which in turn reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be more or less favourable.

This report may include technical information that requires subsequent calculations to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK, Ausenco and Knight Piésold do not consider them to be material.

SRK, Ausenco and Knight Piésold are not insiders, associates or affiliates of NGEx, and none of us nor any affiliate has acted as advisor to NGEx, its subsidiaries or its affiliates in connection with this project. The results of the technical review by SRK, Ausenco and Knight Piésold are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

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 of this Report 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 NGEx staff and legal experts retained by NGEx for this information, including, but not limited to the following document:

- Nicholson y Cano Abogados – Title Opinion Letter to W. Wodzicki, 21 November 2018

This information is used in Section 4 of the Report and in support of the mineral reserve estimate in Section 15 and the financial analysis in Section 22.

3.2 Environmental, Permitting and Social

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

- BGC Ingeniería Ltda, 2015: 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.
- Social and community engagement as conducted by and communicated from the Lundin Foundation

This information is used in Section 20 of the Report.

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 NGEx for this information.

This information is used in Section 22 of the Report.

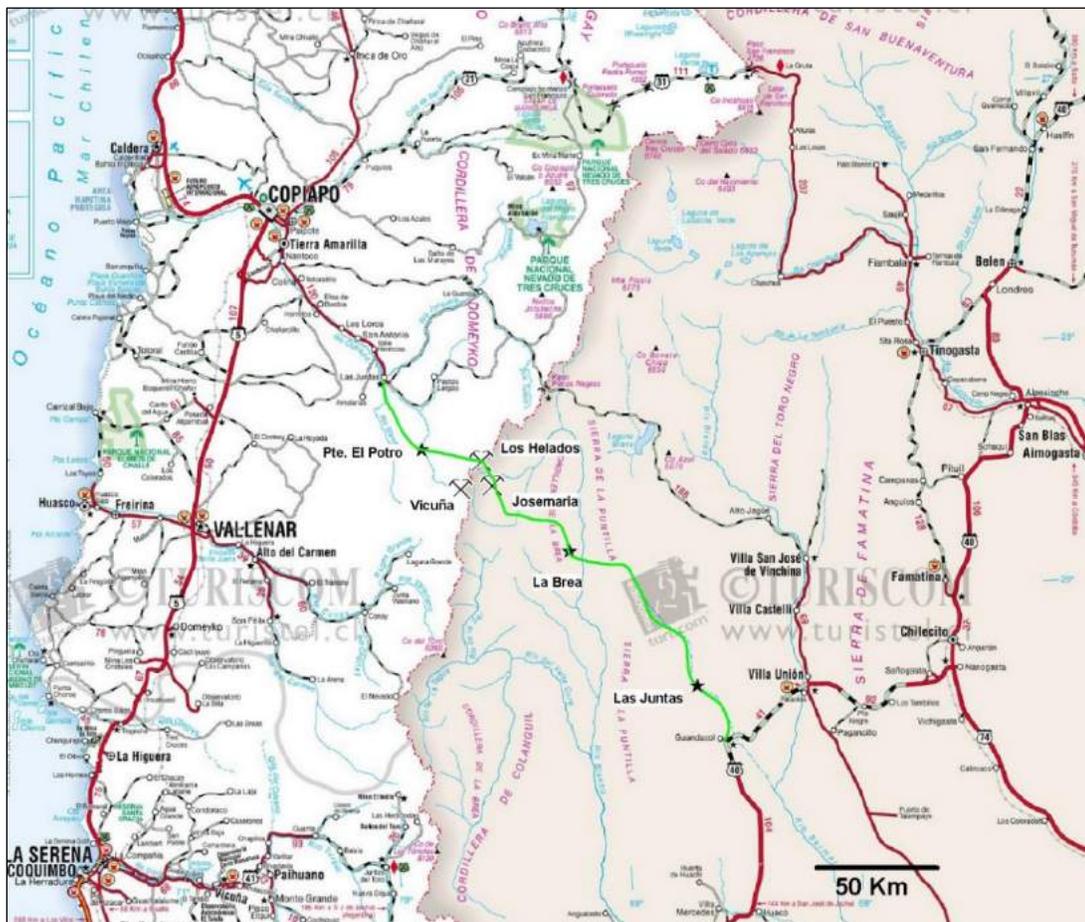
4 Property Description and Location

4.1 Location

The Josemaría deposit is located approximately 145 km southeast of the city of Copiapó, Chile, across the international boundary in San Juan Province, Argentina. The deposit is centred at 28.4359° S, 69.5486° W (Figure 4.1). The total area of the property is approximately 16,717 ha. There is some uncertainty as to the exact area due to the northern boundary, which is along the currently undefined border between La Rioja and San Juan provinces.

The best access to the project currently is from Copiapó, a driving distance of about 200 km, or four hours. Alternate access from Argentina is possible by major provincial highways north through San Jose de Jachal to the town of Guandacol and from there by approximately 150 km of regional unpaved roads and trails. Total driving time from San Juan is approximately 10 hours.

A new 57-km long, two-lane dirt access road is planned to branch off from highway RN 76 to access the proposed Josemaría mine and the process plant. From that intersection, the Pircas Negras border pass is about 22 km away on the existing road.



Source: NGEx, 2018

Figure 4.1: Project location and access map

4.2 Mineral Tenure in Argentina

Under the Argentine Mining Code, two types of permits can be granted: exploration permits (cateos) and exploitation permits (concesiones de explotación or mina).

4.2.1 Exploration Permits (Cateo)

Exploration permits typically are awarded in units of 500 ha, termed the measurement unit. Holders may acquire a maximum of 20 measurement units (10,000 ha), but may not hold, in aggregate, any more than 400 measurement units (200,000 ha) in any one Province.

Grant of an exploration permit gives the holder the right to explore and prospect within the measurement unit boundary, for a 150-day period. The term is extended by 50 days for each additional measurement unit that has been granted, with the largest possible term being 1,100 days. However, once 300 days have been reached, where the holding is over four measurement units the holder must relinquish half of the land. At the 700-day point, the holder must relinquish half of the remaining measurement units.

Prior to grant of an exploration permit, holders must pay a one-off fee of ARS\$400 for each measurement unit requested and provide a work plan and commit to starting that work program within 30 days of permit grant. Compensation must be paid to landowners inconvenienced by any exploration activities. An activities report must also be provided to the appropriate regulatory authorities within 90 days after expiry of the measurement unit.

4.2.2 Exploitation Permits (Mina)

Exploitation permits allow for mining activity. Holders must initially apply for a discovery claim (manifestación de descubrimiento) and the application is advertised for public comment.

The measurement unit area for such claims, the pertenencia, will vary depending on the mineralization to be exploited. Claims over gold, silver, and copper, and, generally, hard rock minerals deposits (e.g. vein-style and discrete deposits) are typically 6 ha in extent; however, disseminated mineralization style deposits may see claim sizes reach a maximum of 100 ha. Exploitation permits can consist of one or more pertenencias.

Exploitation permit grant is contingent on a number of factors, including:

- Provision of official cartographic coordinates for the deposit and the area required for operating facilities
- Provision of a sample of the mineral discovered
- Approval of an Environmental Impact Assessment (EIA).

Approval and registration of the legal survey request by the relevant Provincial mining authority constitutes formal title to the exploitation permit. Assuming mining is active, and all other requirements are met, exploitation permits can have an indefinite grant period.

After three years from the date the discovery claim was registered, an annual fee (canon) becomes payable. The amount of the annual canon depends on the pertenencia size, and ranges from ARS\$80 for the 6 ha pertenencias, to ARS\$800 for the 100 ha pertenencias.

A further condition is required of a holder, which is to invest, at a minimum, 300 times the value of the annual canon in fixed assets on the exploitation permit over a five-year period. Twenty percent of the required investment must be made each year for the first two years of the designated investment period. For the final three years, the remaining 60% of the investment requirement is at the holder's discretion as to how it is expended. The exploitation permit can be cancelled if the minimum expenditures are not met in the manner stipulated.

Permits may also be cancelled if mining activity ceases for more than four years and the holder has no plans to reactivate mining within a five-year period.

4.2.3 Surface Rights

The Argentine Mining Code (AMC) sets out rules under which surface rights and easements can be granted for a mining operation, and covers aspects including land occupation, rights-of-way, access routes, transport routes, rail lines, water usage and any other infrastructure needed for operations.

In general, compensation must be paid to the affected landowner in proportion to the amount of damage or inconvenience incurred. However, no provisions or regulations have been enacted as to the nature or amount of the compensation payment.

In instances where no agreement can be reached with the landowner, the AMC provides the mining right holder with the right to expropriate the required property.

4.3 Environmental Regulations

Minimum environmental standards are enacted federally, with Provincial governments able to enact supplementary legislation to these minimum standards. The AMC incorporates National Law No. 24.585, key features of which include:

- An environmental impact statement (EIS) must be filed with the relevant regulatory authority
- The AMC has adopted a sectorial approach, in that each mining stage, including prospecting, exploration, exploitation, development, extraction, storage and beneficiation phases, as well as mine closure, require separate environmental impact reports (EIRs), each of which are reviewed separately prior to any approval
- If the EIS meets the relevant requirements under National Law No. 24.585, an environmental impact declaration (EID, or DIA [Declaración de Impacto Ambiental] in Spanish) will be granted; this allows work to commence
- EIDs have a two-year duration, and a set of conditions and requirements that must be met to keep the EID current.

Provinces may also have their own, additional, requirements relating to EIS preparation.

Provinces also regulate the generation of hazardous waste, water extraction for mining purposes, liquid effluent discharges, and soil protection. Some Provinces (e.g. Chubut and Mendoza) have banned open pit mining and/or the utilisation of cyanide and other chemicals in the mining process. Open pit mining and the use of cyanide are both permitted in San Juan Province.

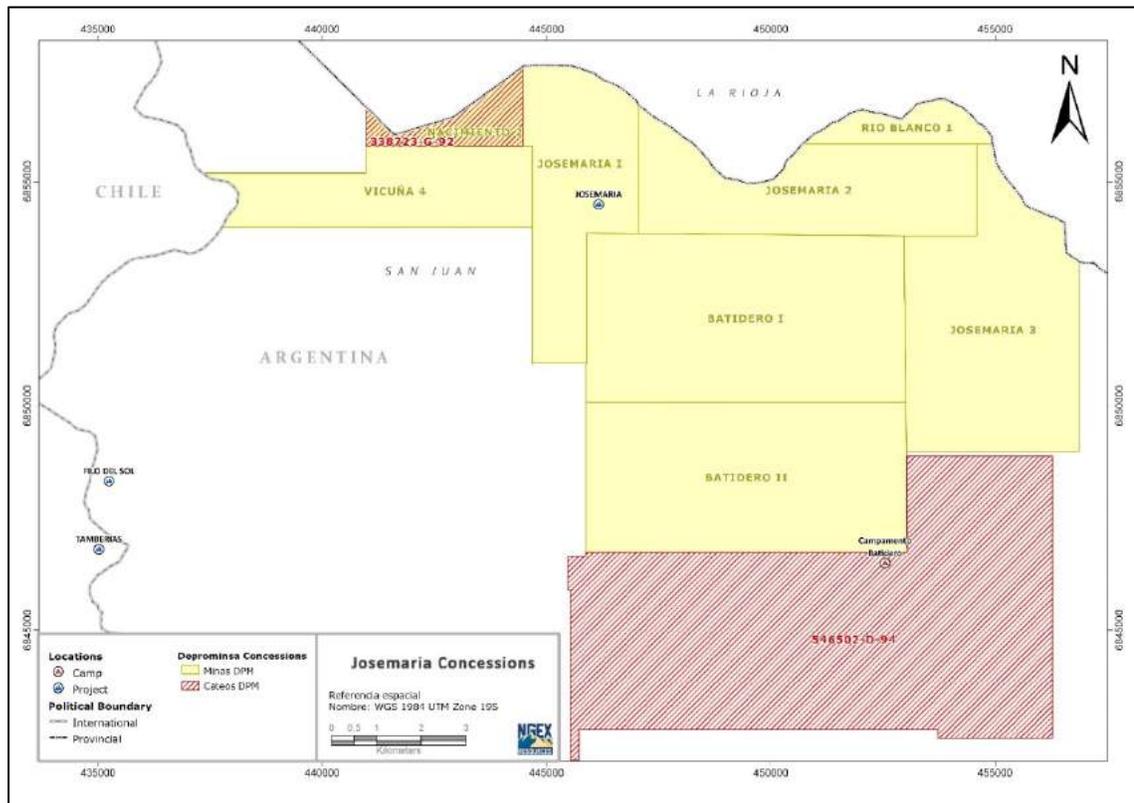
4.4 Josemaría Mineral Tenure

Legal opinion was provided that supported that DPM holds eight exploitation licences (minas) and two exploration licences (cateos). These are summarized in Table 4.1. Total holdings cover an area of approximately 16,716 ha. The Josemaría deposit is located within the “Josemaría 1” exploitation licence (mina), as shown in Figure 4.2.

Table 4.1: Mineral tenure – Josemaría

Concession	Type	Agreement	File Number	Area (Ha)	Mining Units	Annual Fee (ARS)	5 Year Investment (ARS)
	Cateo	Filo	338.723-G-92	291			
	Cateo	Lirio	546.502-D-94	5,011			
Rio Blanco 1	Mina	Lirio	520-0347-D-99	271	3	9,600	2,880,000
Josemaría 1	Mina	Lirio	414280-L-04	1,222	13	41,600	12,480,000
Josemaría 2	Mina	Lirio	414281-L-04	1,500	15	48,000	14,400,000
Josemaría 3	Mina	Lirio	1124.284-D-14	2,054	21	67,200	20,160,000
Vicuña 4	Mina	Filo	520-0447-B-99	1,033	11	35,200	10,560,000
Nacimiento 2	Mina	Filo	1124-285-F-14	291	3	9,600	2,880,000
Batidero I	Mina	Batidero	425066-C-01	2,656	27	86,400	25,920,000
Batidero II	Mina	Batidero	425065-C-01	2,387	24	76,800	23,040,000

Note: ARS = Argentine peso



Source: NGEx, 2018

Figure 4.2: Mineral tenure map

4.4.1 Josemaría Surface Rights

NGEx currently has an occupancy easement for the Batidero camp and a road right-of-way, which provides access to the work area. Part of the road right-of-way is within private property. The remainder of the road and the camp fall within the multiple usage area that has been designated by the San Guillermo Provincial Reserve. Multiple usage includes allowances for mining activities.

4.5 Taxation, Royalties and Option Agreements

4.5.1 Corporate Income Tax

A corporate tax rate in Argentina of 25% was presumed to be in place when the project is in production.

4.5.2 Provincial Mining Royalties

There is a 3% mine head value royalty payable to the Province of San Juan, which can be reduced in certain circumstances.

4.5.3 Option Agreements

The Josemaría project is subject to three underlying agreements: the Lirio agreement, the Batidero agreement and the Filo agreement. Table 4.1 indicates which concessions are incorporated in each agreement.

Lirio Property Agreement

The Lirio property was acquired from the Lirio family through an exploration agreement with an option to purchase, dated 15 July 2003. This option was exercised on 25 June 2009 for US\$813,000.

NGEx holds a 100% interest in the property, subject to a 0.5% net smelter return (NSR) royalty (for a period of 10 years), and an additional US\$2 million payment within six months of the completion of the second full year of mine operations.

The Lirio property agreement covers the area of the mineral reserve estimate for the Josemaría deposit.

Batidero Property Agreement

The Batidero property was acquired through an agreement with Compania Minera Solitario S.A. dated 1 July 2002 and transferred to DPM through public deed No. 01 dated 4 January 2013. NGEx holds a 100% participating interest in the Batidero property, subject to a 7% net profit interest.

The currently-estimated mineral reserves for Josemaría do not fall within the Batidero property agreement.

Filo Property Agreement

The Filo property was acquired from Filo del Sol Exploración S.A. through an agreement dated 11 January 2018. NGEx holds a 100% interest in the Filo property subject to a 3.0% NSR royalty in favour of Filo del Sol. NGEx has the right to buy back 2% of the NSR for \$2 million.

The currently-estimated mineral reserves for Josemaría do not fall within the Filo property agreement.

4.6 Josemaría Permits

Surface exploration work in the Josemaría area is permitted under a DIA. The original DIA application was submitted on 10 November 2006 for the Josemaría 1 and 2 exploitation concessions (minas) and was granted on 16 November 2010 under Resolution 287-SEM-2010.

On 20 November 2012, an amendment request was filed to include the Rio Blanco 1 exploitation concession (mina) in the DIA.

The Environmental Impact Report for the Batidero exploitation concessions was filed on 30 April 2007, and the DIA was granted on 5 August 2008.

4.7 Josemaría Environmental Liabilities

Existing environmental liabilities are limited to those associated with exploration-stage properties and would involve removal of the exploration camps and rehabilitation of drill sites and drill site access roads.

4.8 Mining Integration and Complementation Treaty

On 29 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 20 August 1999, Chile and Argentina subscribed to the Complementary Protocol and on 18 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 NGEx's "Proyecto de Prospección Minera Vicuña" (Vicuña Mining Prospecting Project), dated 6 January 2006. This Protocol allows for prospecting and exploration activities in the Josemaría area. The main benefit of the Vicuña Protocol is the authorization that 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".

In September 2012, the "Proyecto de Prospección Minera Vicuña" was amended by the "Protocol of Amendment to Article 8". With this amendment, the defined "operational area" was expanded, enabling a new border crossing area to be demarcated.

4.9 Closure Considerations

Closure must be covered by submission of a new EIR, or an update/amendment to an existing approved EIR. The document must include details of the proposed environmental rehabilitation, reclamation or adjustment activities, and discuss how post-closure environmental impacts will be avoided. The EIR must include data on post-closure monitoring, but current regulatory requirements do not entail submission of formal closure plans.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

The Josemaría project area is accessed from San Juan by major provincial highways north through San Jose de Jachal to the town of Guandacol (in La Rioja Province) followed by approximately 150 km of regional unpaved roads and trails (Figure 4.1). Josemaría is approximately 10 hours drive from the city of San Juan.

Alternate access from Chile is provided through the Mining Integration and Complementation Treaty between Chile and Argentina. This treaty allows personnel and equipment to access the Josemaría area from Chile, providing that they also return to Chile and do not cross out of the Treaty area into Argentina. Josemaría is approximately four hours drive from the city of Copiapó.

The C-35 paved road from Copiapó passes in a southeasterly direction through the town of Tierra Amarilla and Punta del Cobre, along the Copiapó River valley, through the small villages of Pabellon, Los Loros, La Guardia, and Iglesia Colorada. After these small villages, the road continues towards the El Potro bridge. At about kilometre 130, the paved road ends, and the remaining road to the project area is gravel. Access is generally possible during the summer months from September to May but may be curtailed if there is inclement weather.

5.2 Climate

The climate in the Josemaría area is dry to arid and the temperatures are moderate to cold. Annual precipitation is about 250 mm, with snow at higher altitudes in the winter.

Exploration fieldwork is generally possible from mid-October to early May.

It is expected that any future mining operation will be able to be conducted on a year-round basis, based on the success of several other mines operating in similar climates and altitudes in Chile and Argentina.

5.3 Local Resources and Infrastructure

The Josemaría project will be a greenfields development. The most important logistics centre in the region is San Juan, which has a population of about 700,000. San Juan has a domestic airport with scheduled flights to Buenos Aires and other Argentine cities. The city of Mendoza, south of San Juan city, has an international airport with flights to Santiago and elsewhere. Copiapó has a modern airport, with several daily flights to Santiago.

There is no infrastructure in the area except for NGEx's Batidero exploration camp located near the Josemaría deposit.

The Batidero camp is located 7 km south of the Josemaría deposit at an elevation of approximately 4,000 masl. The camp consists of portable structures with infrastructure for septic, water distribution and electricity generation. It is currently configured to house about 210 people.

5.4 Physiography

The Josemaría project is located in the Andes Mountains in Argentina, 9 km east of the Chile–Argentina border. Terrain in the Josemaría area and near the proposed processing plant site varies from broad flat alluvial plains one kilometre or wider, to rounded ridges and peaks with varying steepness. Colluvial cover thickens on lower slopes and in places fresh outcrop is difficult to locate.

The Josemaría deposit itself underlies a north–south-trending ridge that lies along the southern side of the broad Rio Blanco river valley. Relief along the ridge ranges from 4,000 masl at the valley bottom at the northern edge to 4,900 masl at the top of the hill in the south.

6 History

There is no record of significant exploration activity at Josemaría prior to NGEx's interest. The deposit was discovered by NGEx in 2004.

There is no reported production from the project area.

Prior to 2001 there is no known history of mineral exploration fieldwork or mining on the Josemaría property other than several regional prospecting programs conducted during the 1990s that probably collected talus or drainage samples, and a program of LANDSAT imagery interpretation, which identified a large area that had spectral response characteristics of hydrothermal alteration.

This activity prompted Sr. Lirio, a local landholder, to acquire the mineral rights for various areas, including Josemaría.

Rights to the Lirio holdings were acquired by Solitario Resources in 1993, and a small amount of prospecting work was completed in the claims area. At the time, the area was referred to as Cateo 17 or the Arroyo Batidero project.

During 1998, Toscana Resources Ltd, (later TNR Resources Ltd, and now TNR Gold Corp) took over Solitario. Exploration work recommenced in 2000, when Solitario had concluded a joint venture exploration agreement with Barrick Exploraciones de Argentina S.A. (BEASA). The agreement created a joint venture, Compania Minera San Juan S.A. (CMSJ). However, when the joint venture was dissolved in 2001, CMSJ was deregistered and the mineral tenure returned to Solitario's ownership.

In June 2002, the parent company of Solitario (then called TNR Resource Ltd) signed an option agreement with Tenke Mining Corporation (now NGEx Resources Inc).

The Josemaría deposit was discovered during the initial drilling campaign in the 2003-2004 field season. The first hole drilled encountered 280 metres grading 0.61% copper and 0.51 g/t gold. It was targeted on coincident talus fine copper and gold geochemical and magnetic anomalies.

7 Geological Setting and Mineralization

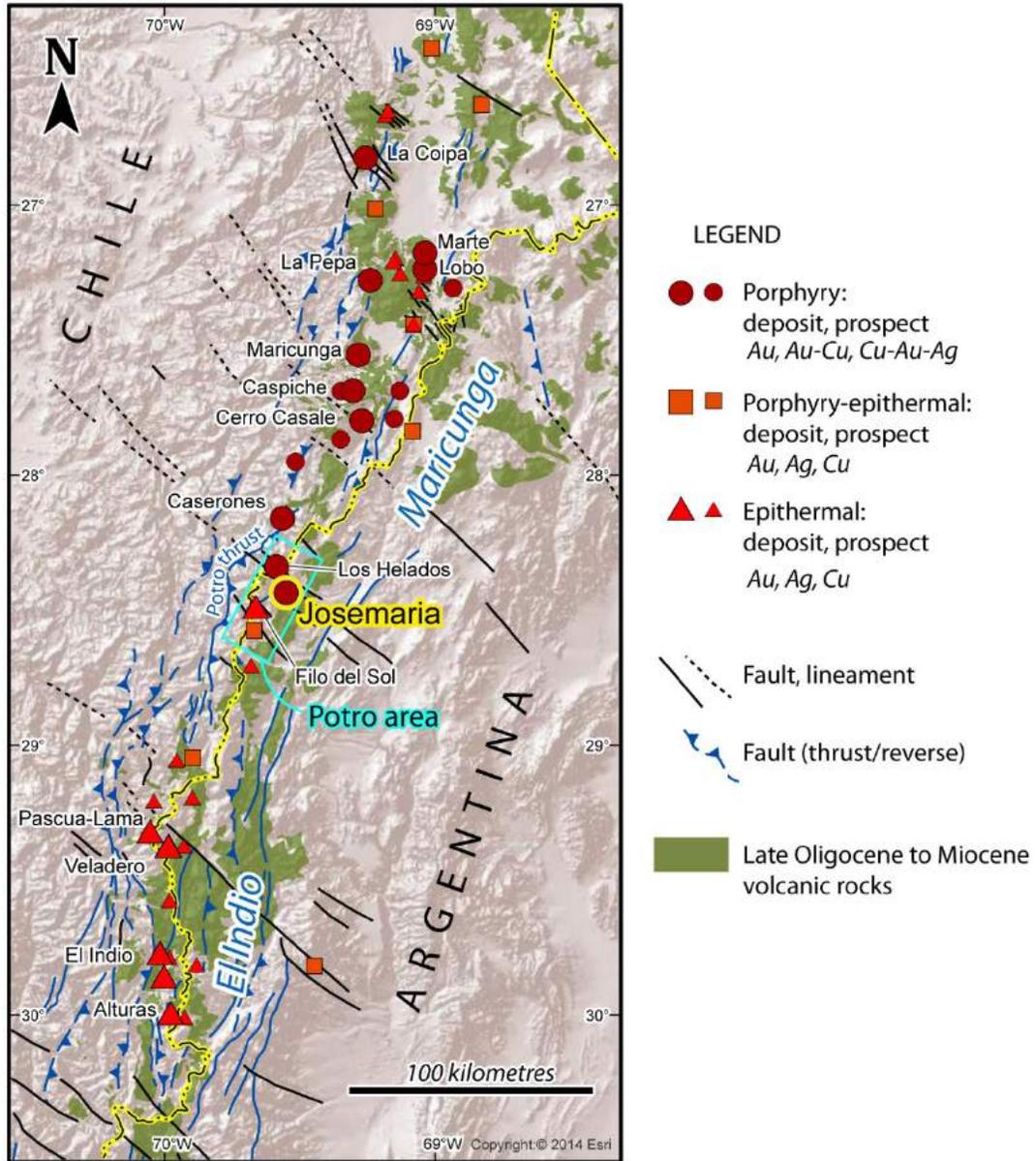
7.1 Regional Geology

The Potro area in the central Andes encompasses the crest of the range along the Chile-Argentina border and the area westward into Argentina at approximately 28.5° N (Figure 7.1). It lies within the present-day non-volcanic segment of the Andes, correlative with the flat-slab portion of the subducted Nazca plate.

Basement rock in the area includes Permian-Triassic granitic and rhyolitic volcanic rocks, intruded by Triassic tonalite-diorite intrusive complexes. The Triassic extensional rift basin deposits and the Jurassic – Early Cretaceous backarc basin sedimentary rocks that are found farther north are not present in the area, and Eocene volcanic and intrusive rocks are preserved only to the east and north. Latest Oligocene to Miocene porphyry intrusions and associated porphyry copper-gold and epithermal mineralization occur primarily within the Permo-Triassic basement rocks, but also locally within relatively small remnants of Late Oligocene to Miocene sedimentary and volcanic rocks where they have escaped erosion.

A high degree of tectonic inversion in the area has led to the predominant exposure of basement rocks, and the lack of preservation of overlying sedimentary and volcanic sequences. Faults related to extension during the pre-Andean and early Andean arc development were reactivated as early as Late Oligocene, followed by a main pulse of compression and inversion as high-angle reverse faults in the Miocene. The Potro fault is a significant reverse structural feature in the region, responsible for a large degree of upthrow of the Paleozoic basement rocks to the west and their juxtaposition with younger sedimentary units.

Mineral exploration is focused on copper and gold mineralization related to porphyry and epithermal systems developed during the main Late Oligocene to Miocene compressive stage of Andean arc development. The Maricunga belt to the north is notable for its porphyry Au-Cu systems and the El Indio belt to the south, including Pascua Lama, hosts world-class high-sulphidation epithermal deposits. The Potro area has historically been overshadowed by these two high-profile metallogenic belts, partly due to the lack of preservation of extensive Miocene volcanic rocks, which was incorrectly interpreted to reflect a paucity of Miocene mineral deposits. Mpodozis and Kay (2003) proposed that the Potro area is in fact prospective for porphyry copper-gold and epithermal systems, and subsequent work by NGEx has shown this to be the case, with the discovery of Josemaría, Filo del Sol and Los Helados deposits with Late Oligocene to Late Miocene ages. While the contemporaneous volcanic rocks have been largely removed through erosion, the porphyry and local epithermal systems remain, although they are developed within the basement and older sedimentary rocks, rather than within Late Oligocene to Miocene volcanic sequences.

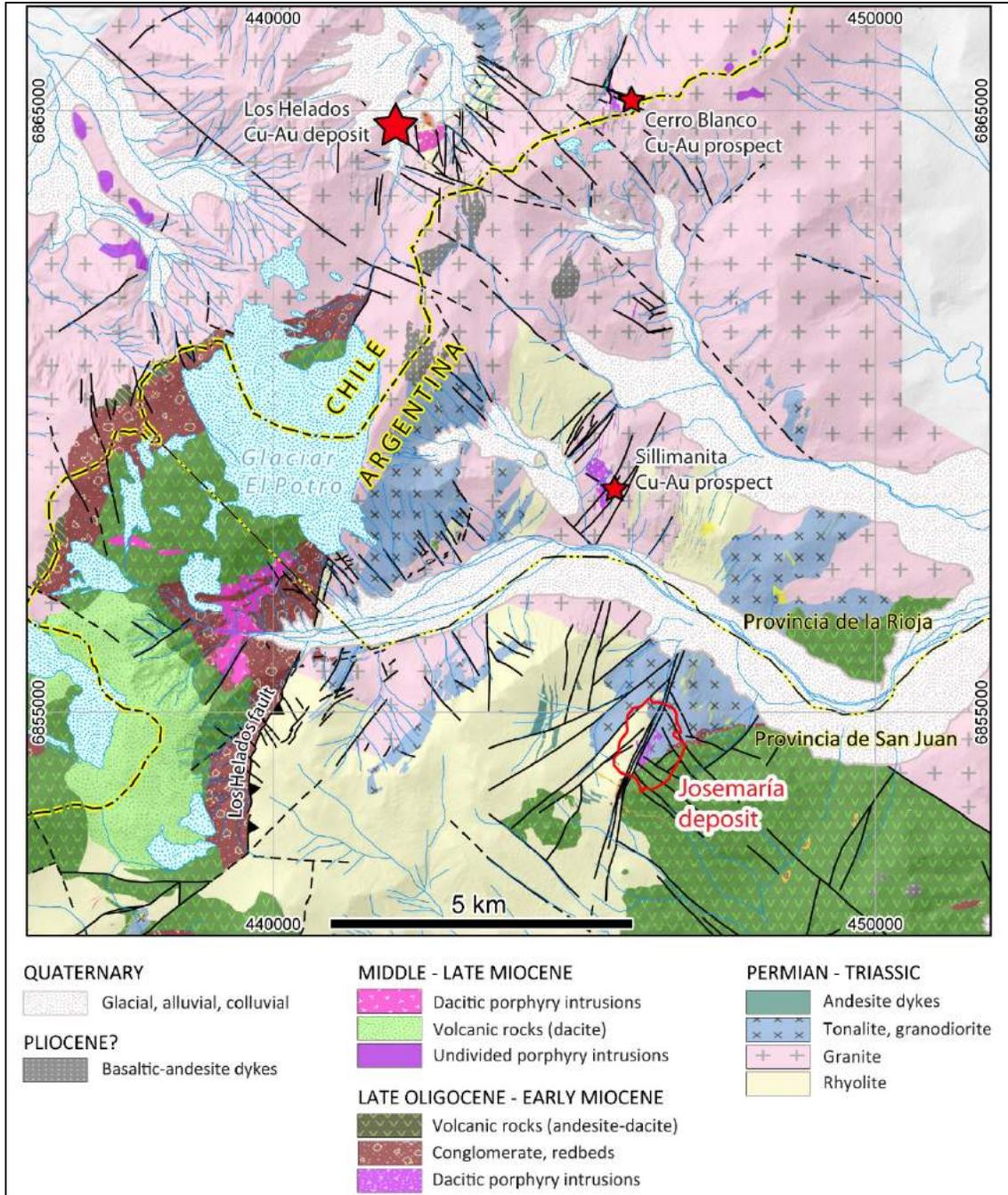


Source: NGEx, 2018

Figure 7.1: Map showing part of the Late Oligocene to Miocene porphyry-epithermal belt

7.2 Project Geology

The Josemaría project area is underlain most extensively by Permo-Triassic rocks assigned to the Choiyoi Group, which forms the Andean basement in the region (Figure 7.2). They include volcanoclastic and ignimbritic rhyolites as well as broadly equivalent granites. Triassic intrusive complexes of tonalite, diorite and granodioritic composition intrude the rhyolites and granite. Swarms of andesite dykes, which are typical of the Permo-Triassic in this region, cut the older Permo-Triassic units.



Source: NGEx, 2018

Figure 7.2: Regional geological map of part of the Potro area

Inferred Late Oligocene to Miocene sedimentary and volcanic rocks are located in the western part of the area to the west of Josemaría, in the footwall of the Los Helados fault at the crest of the range. Late Oligocene sedimentary and volcanic rocks also occur over top, and to the east of Josemaría, preserved in the relatively low-lying Macho Muerto basin.

Regional faults, such as the Los Helados fault, were active as early as Late Oligocene, but particularly post-20 Ma (million years ago) when the most significant compressive stage of Andean mountain building began. The uplift due to this compressive event is responsible for the more deeply eroded nature of the area, exposing the basement rocks most extensively and eroding the Late Oligocene to Miocene sequences.

Porphyry intrusive rocks and associated porphyry and epithermal mineralized systems are largely hosted within basement rocks. Late Oligocene systems form a north-south trend extending from Josemaría to the Sillimanita and Cerro Blanco copper-gold porphyry prospects formed around similar dacitic intrusions. Middle Miocene systems such as the Filo del Sol porphyry-epithermal copper-gold-silver deposit and the Los Helados porphyry deposit occur several kilometres westward.

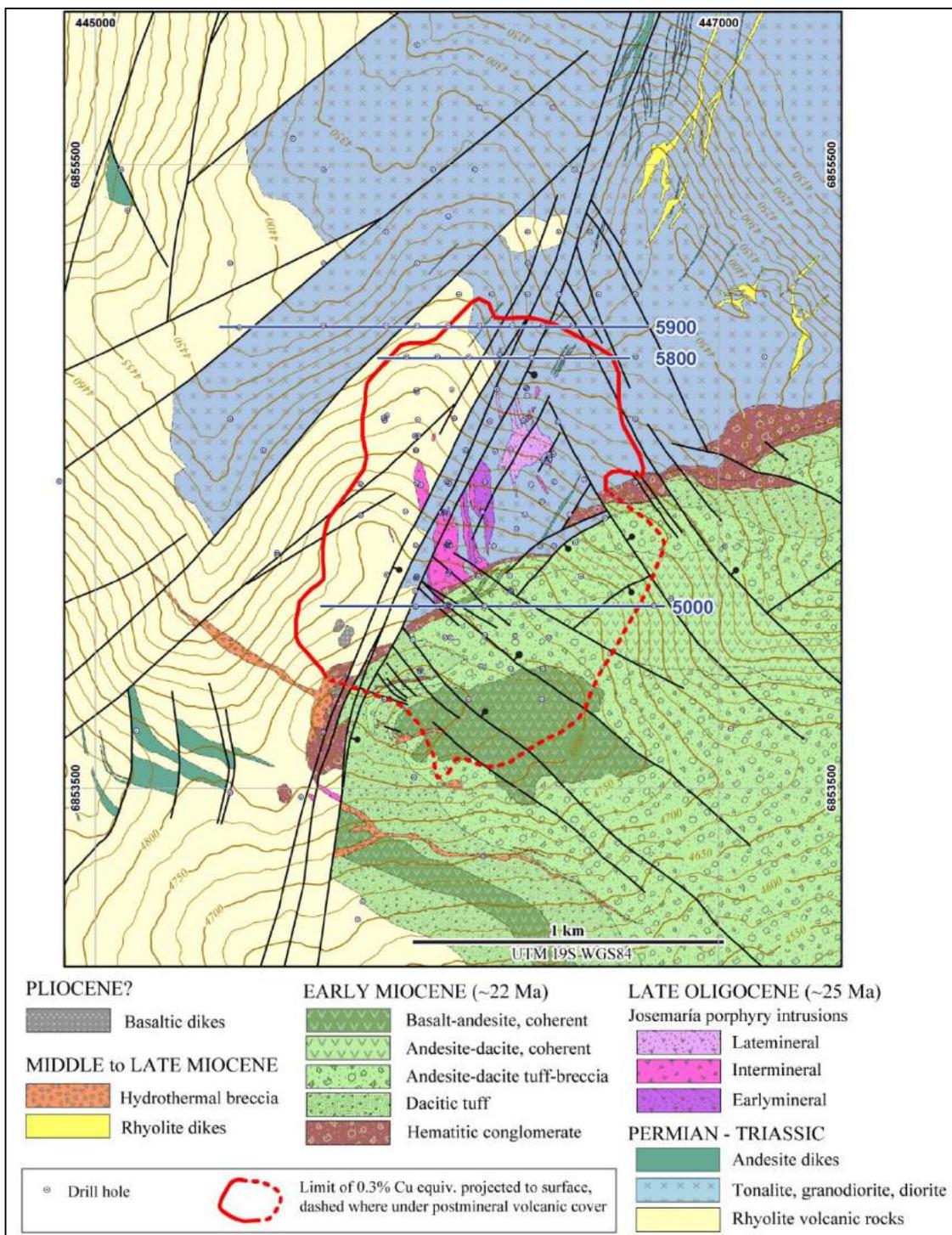
7.3 Deposit Description

The Josemaría porphyry copper-gold deposit is centred on a Late Oligocene dacitic porphyry intrusive complex emplaced into Permo-Triassic rhyolite and tonalite (Figure 7.3). Porphyry ascent and localization appears to have been guided by a pre-existing north-south structural zone.

The deposit was developed within and around the upper parts of the porphyry intrusions at ~24.5 Ma (Sillitoe, et al., in prep). Disseminated and vein-related chalcopyrite with minor bornite occurs within the domain of sericite-chlorite-clay alteration overprinting earlier potassic, centred on a multiphase porphyry intrusive complex. The upper part of the deposit is enhanced by overprinting high-sulphidation mineralization including hypogene chalcocite and covellite, which upgrades the copper values and correlates with an increase in gold values.

Although the deposit formed at ~24.5 Ma, it was rapidly unroofed and overlain by a redbed conglomerate unit and post-mineral volcanic rocks by 22 Ma (earliest Miocene). This rapid unroofing is evident in the high degree of telescoping of alteration and mineralization within the system, as well as the lack of a well-developed leached cap below the Late Oligocene erosional surface. The conglomerate at the erosional contact contains sulphide mineralized clasts that were not oxidized, an indication of lack of time for penetrative leaching. Deposition of the earliest Miocene post-mineral volcanic rocks effectively halted the rapid erosion into the porphyry system. It remained buried beneath Miocene post-mineral volcanic cover until it was once again exposed during the recent development of the modern erosional surface.

A significant, post-mineral north-northeast fault system trends through the centre of the deposit. It forms a structural zone with early (pre-22 Ma) high-angle reverse motion, but most recent down-to-the-east normal displacement on the order of 100 to 200 metres. A series of northwest-trending faults also cut the overlying post-mineral volcanic rocks, with similar normal displacement to the northeast. These structures, while responsible for relatively minor recent offsets of the mineralized domains within the deposit, played an important role in the development of the supergene copper enrichment blanket.



Source: After Sillitoe et al., in preparation; mapping by F. Devine, 2014

Figure 7.3: Josemaría geology map

Supergene copper enrichment has developed since the most recent erosion into the deposit through the post-mineral volcanic rocks. The supergene profile varies greatly in thickness and is most strongly developed within the main NNE structural zone through the central part of the deposit. Fracturing along faults allowed for downward flow of the copper-charged surface waters, whereas the areas away from faults remained relatively impermeable due to retention of sulphate veins or low fracture density. The supergene mineralization in the deposit developed immediately over top of the hypogene zones with little evidence for lateral transport or exotic copper mineralization.

7.3.1 Lithologies

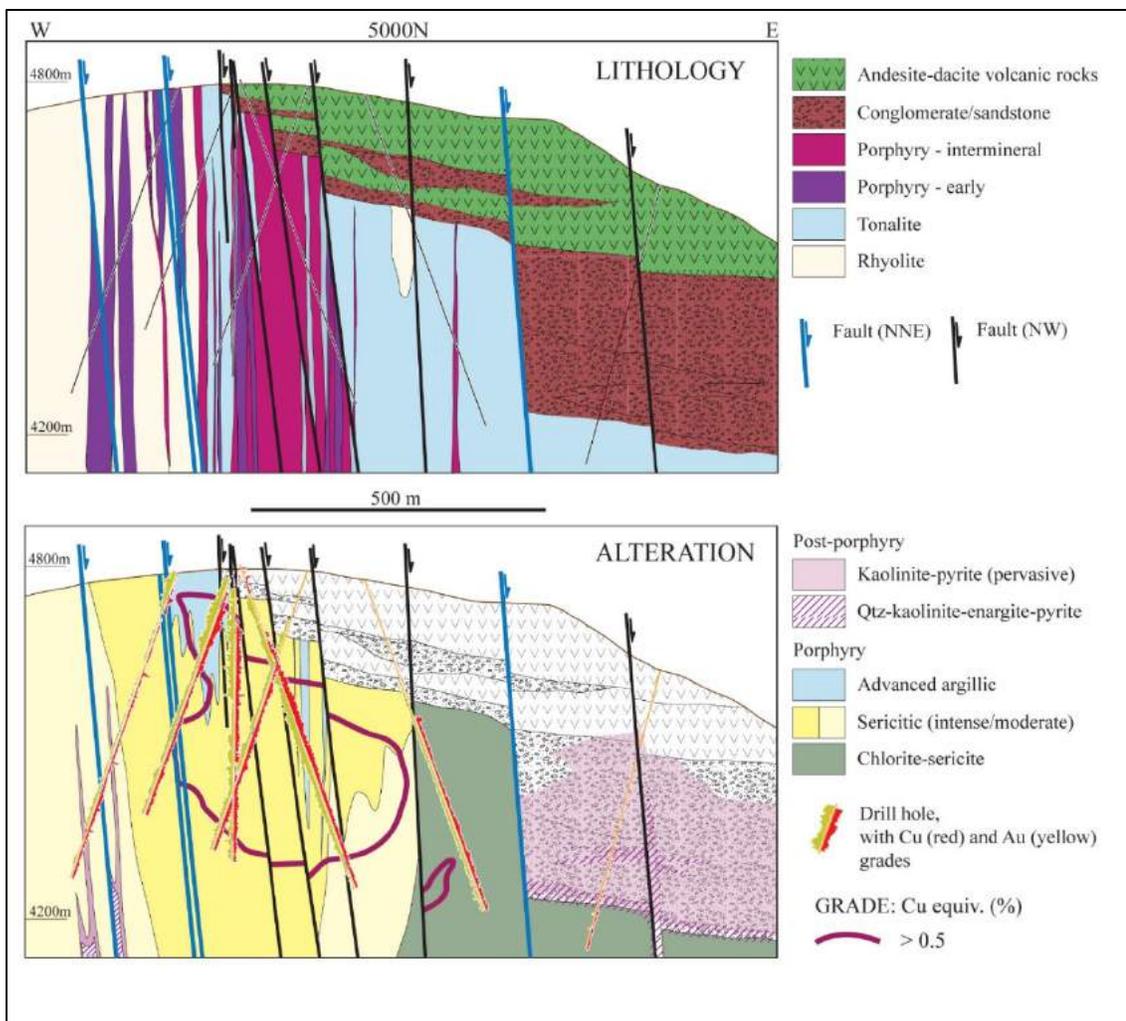
The host rock units in the Josemaría area are assigned to the Permo–Triassic Choiyoi Group. To the west of the main Josemaría NNE structure, rhyolite ignimbrite and tuff-breccia form the predominant unit at surface (Figure 7.3). Bedded volcanoclastic textures are mapped locally and welded, black to cream coloured, quartz and feldspar-phyric rhyolite with an aphanitic groundmass is common where primary textures are preserved. These volcanoclastic rhyolites overlie, and are interpreted to be intruded by, the tonalite-granodiorite unit.

Tonalite, granodiorite, and diorite intrusive rocks are exposed on the northern and eastern sides of Josemaría. They are medium- to coarse-grained and equigranular with varying quartz content.

Andesite dykes ranging from sub-metre to 10 m wide cut both the tonalite and rhyolite, locally as dyke swarms with a northerly trend. These are similar to andesite dykes common in the Permo-Triassic basement rocks throughout the region.

The Josemaría Late Oligocene porphyry intrusions are centred on the upper part of the north-facing slope immediately below the height of land at Josemaría (Figure 7.4). They occur over an approximate 1000 m x 400 m area, on both sides of the main structural corridor, although predominantly to the east. They include a series of feldspar-quartz- hornblende-biotite-phyric dacitic intrusions that have been divided into three main phases based on their compositions as well as the presence of vein fragments and relative vein density and intensity of mineralization. The early-mineral porphyry intrusions are fine-grained and feldspar phyric with feldspar phenocrysts lath-shaped and <5mm and occasional small quartz phenocrysts. They can be difficult to distinguish from the host tonalite where strongly overprinted by porphyry-related alteration. The inter-mineral phase includes strongly quartz and feldspar-phyric variants, with up to 50% feldspar phenocrysts, and round clear to grey quartz phenocrysts up to 1cm. The late mineral phases are quartz and feldspar porphyritic with an aphanitic groundmass.

The Late Oligocene erosional surface that cuts down into the porphyry system is overlain by a distinct, hematitic redbed conglomeratic unit. It includes cobble conglomerate and wacke with a variety of clasts types, including a predominance of rhyolite clasts, but also mineralized porphyry intrusive clasts. The conglomerate is overlain by an andesitic to dacitic volcanoclastic and coherent volcanic package of earliest Miocene age (together referred to as the PMV, ~22 Ma).



Source: NGEx, after Sillitoe et al., in preparation.

Figure 7.4: Josemaría vertical section 5000N lithology and alteration

Hydrothermal breccias (HBX), younger than the porphyry system and also younger than the post-mineral volcanic rocks, cut all units in the southern part of Josemaría. They are narrow, dominantly northwest-trending, quartz–alunite-cemented, polymictic breccias that expand in size where their trend intersects the Josemaría structural corridor but taper out into quartz–kaolinite-cemented dyke-like bodies laterally. Associated chalcedony-alunite and kaolinite alteration with pyrite-enargite mineralization is mapped more broadly around these bodies and extends along post-mineral volcanic layering; similar alteration and arsenic values are found within narrow, structurally-controlled domains along the Josemaría structural corridor.

Fine-grained, northerly-trending rhyolite dykes are found on the northern slope of Josemaría, and locally within the deposit area. They are generally less than 10 m wide where intersected in drilling, and on the northern slopes form interconnecting dykes and intrusive bodies with domains up to 30 m wide. While relative age relationships particularly with the younger sedimentary and volcanic units at Josemaría are not conclusive, to the north of Rio Blanco similar rhyolite dykes are relatively young (Miocene?) and cut Late Oligocene mineralization.

Local, small basaltic plugs occur at the top of Josemaría. They are vesicular, black, and inferred to post-date all other local units.

7.3.2 Alteration

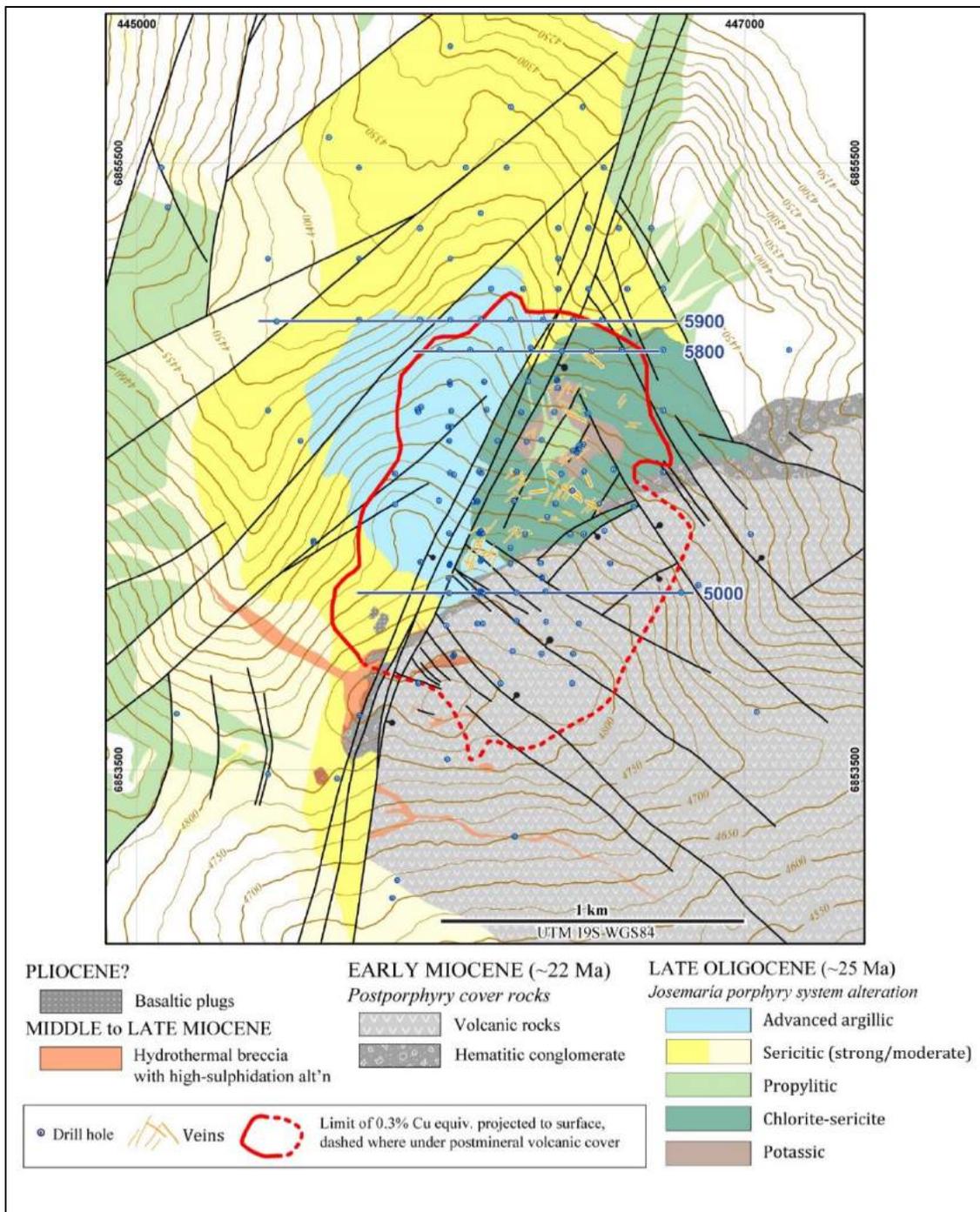
Alteration zonation within the Josemaría porphyry system is centred on the porphyry intrusions that underlie the top and uppermost northern slope at Josemaría (Figure 7.5). The alteration footprint of the system extends for ~2 km east-west, and ~4 km north-south, where it is covered by alluvium in the Rio Blanco valley to the north.

Deepest alteration is potassic, occurring in all holes in the central part of the system within the tonalite host rock below 400 to 600 m (Figure 7.6). A steeply inclined column of potassic alteration is also preserved around the late mineral porphyry intrusions in the northern part of the system. Fine-grained biotite with disseminated and vein magnetite define the mineralogy of the potassic zone, with some bleached feldspars indicating replacement by albite or K-feldspar alteration.

Multidirectional quartz veinlets, mainly A-type chalcopyrite-magnetite veins, were introduced with the potassic event, with slightly more distal molybdenite B-type quartz veins.

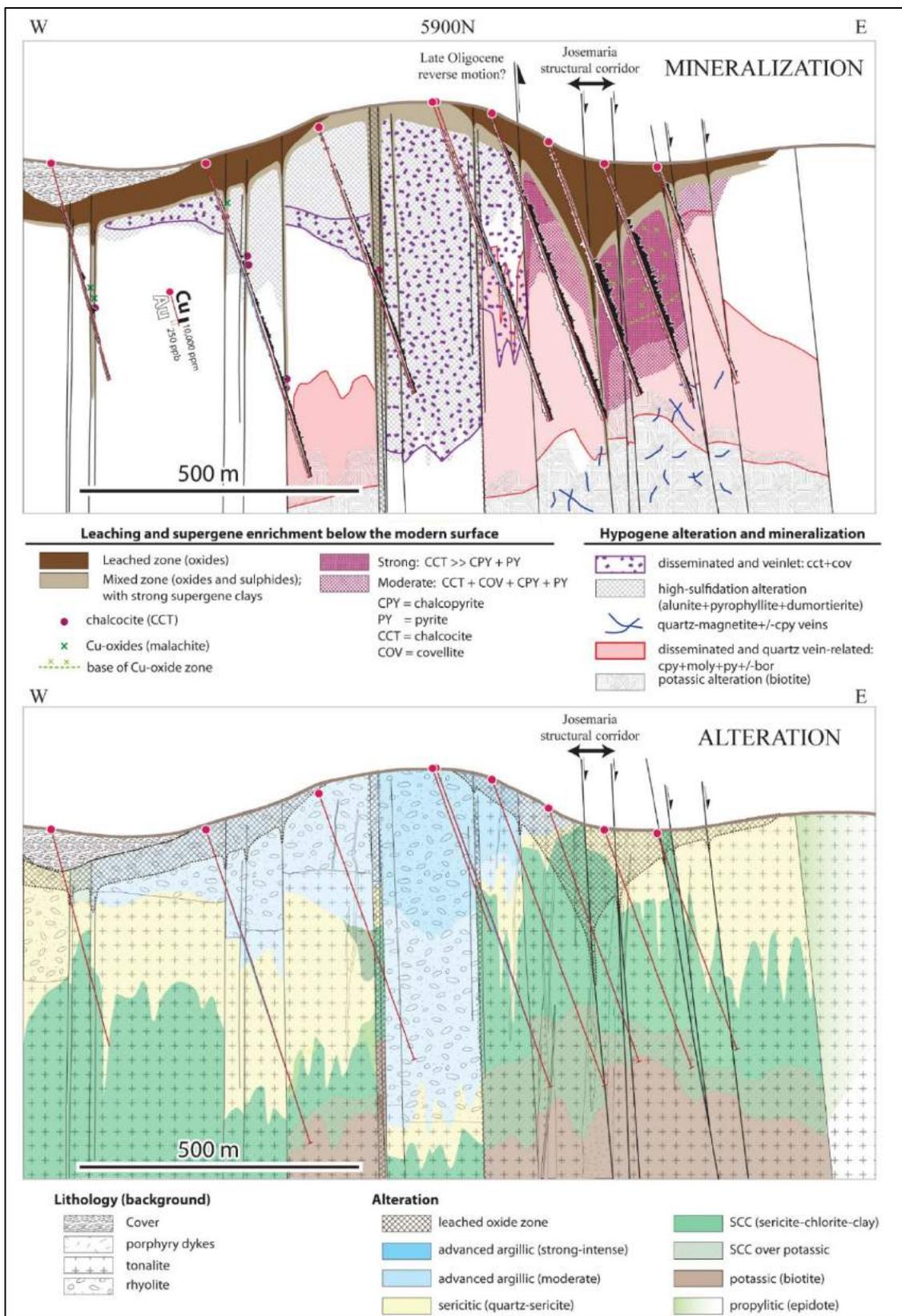
Sericite-chlorite-clay alteration formed at the expense of potassic, with the intensity of the overprint decreasing with depth. Surface exposures in the central part of Josemaría, to the east of the NNE structural zone, are SCC altered (sericite-chlorite-clay), with distinctive mineralogy and hematitic overprint of magnetite. Potassic alteration is also preferentially preserved within the host rock andesite dykes, even where they are enclosed by SCC-altered tonalite.

High-sulphidation alteration and underlying sericitic are best preserved to the west of the NNE structural zone, within the rhyolite host rock. However, the system displays significant telescoping of alteration within its central most part. Advanced argillic alteration has overprinted potassic alteration, most evident in the tonalite, early-, and inter-mineral porphyry phases that underlie the topographically highest part of the deposit. Advanced argillic alteration in the rhyolite includes quartz-pyrophyllite and local quartz alunite alteration, while related alteration within the more reactive tonalite is represented by sericitic alteration below the higher-level advanced argillic assemblage.



Source: After Sillitoe et al., in preparation; mapping by F. Devine, 2014

Figure 7.5: Josemaría alteration map



Source: NGEx, 2018

Figure 7.6: Vertical section 5900N interpreted alteration and mineralization

The porphyry intrusions also record progressive development of alteration within the system with the early mineral phase displaying strong potassic and SCC alteration, while the late mineral phase is only propylitic. A relatively weak sericitic and propylitic halo surround the deposit.

A second high-sulphidation alteration event post-dates the Josemaría system as well as the post-mineral volcanic rocks. It is associated with the quartz-alunite and quartz-kaolinite cemented breccia dykes, predominantly in the southern part of the deposit area. The breccias and related chalcedony-alunite-kaolinite-pyrite-energite alteration invade the NW structures and locally the NNE structures, and also the Late Oligocene unconformity.

7.3.3 Mineral Zones

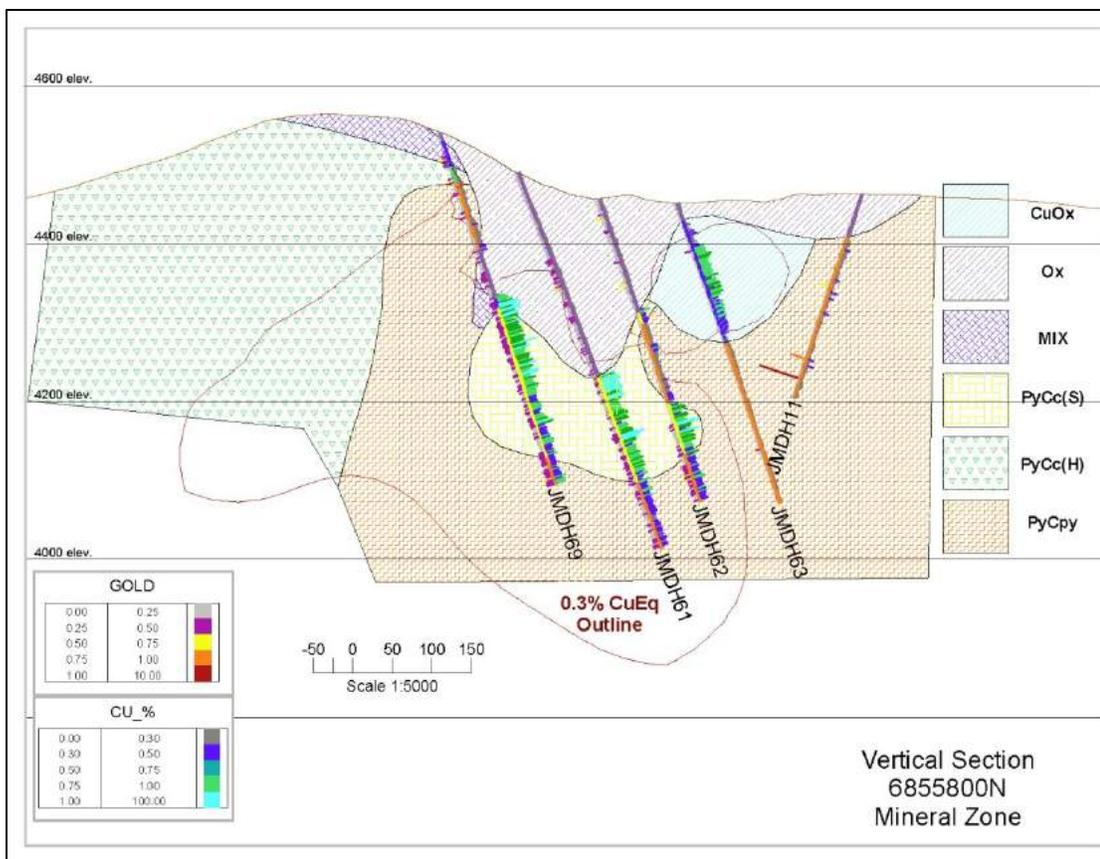
Mineral zones within the Josemaría deposit were defined by the relative abundance of chalcopyrite, pyrite and chalcocite, as well as the mode of occurrence of chalcocite (hypogene or supergene), and the level of oxidation. Six main zones were modeled and used to develop the resource estimate as follows (Figure 7.7):

- Copper oxide (CuOx)
- Barren oxide (leached cap) (Ox)
- Pyrite + hypogene chalcocite (PyCc(H))
- Pyrite + supergene chalcocite (PyCc(S))
- Mixed sulphide and oxide (MIX)
- Pyrite + chalcopyrite (PyCpy)

7.3.4 Mineralization

The copper-gold mineralization at Josemaría is hosted by a porphyry system that includes two main types of hypogene mineralization. These two types occur in proximity to one another due to a high degree of telescoping of high-sulphidation alteration and mineralization over deeper mineralization related to potassic alteration. Late supergene enrichment within the northern part of the deposit has upgraded copper values over part of the system. Deposit dimensions, defined by the current resource, are ~1000 m east-west, ~1500 m north-south, and 600 to 700 m vertically.

The most widespread hypogene copper and gold mineralization is associated with the upper parts of the potassic alteration zone (Min zone PyCpy). Disseminated and vein-style chalcopyrite mineralization is associated with an A-type quartz-magnetite veinlet stockwork in the area above and around the porphyry intrusions. Minor bornite is present, but in an approximate ratio of 30:1 (chalcopyrite:bornite) within the potassic zone.



Source: NGEx, 2013

Figure 7.7: Section 5800N mineral zones used in resource estimation

Sericite-chlorite-clay alteration overprints potassic but was not grade-destructive and some of the best copper grades are found in the SCC domain. Where overprinted, which is through much of the deposit, the sulphide assemblage has been variably reconstituted to pyrite-chalcopyrite with pyrite:chalcopyrite ratios of approximately 3-10:1. Copper and gold values are in the range of ~0.35% Cu and 0.2 g/t Au.

This copper-gold mineralization is overlapped by a molybdenite-bearing annulus best developed on the northern and eastern sides, with grades averaging > 50 ppm Mo. It is related to molybdenite-bearing B-veins surrounding the central part of the system.

The second type of hypogene sulphide mineralization is located along the western and central parts of the system, associated with the advanced argillic domain and the underlying sericitic alteration (Min zone PyCc(H)). This high-sulphidation assemblage includes disseminated grains of pyrite rimmed by hypogene chalcocite, bornite and/or covellite with trace amounts of tennantite and enargite. Arsenic values are relatively low, in the range of ~10–100 ppm. Pyrite:copper-bearing sulphide ratios are roughly 10:1.

In the central part of the system, where the highest degrees of alteration telescoping are mapped, the high-sulphidation alteration extends downward over the potassic- and SCC-related chalcopyrite mineralization. In this area, the early potassic-related sulphide mineralization is reconstituted and upgraded by the high-sulphidation sulphide assemblage, reflected in higher gold and hypogene copper grades in the central part of the system. In places values of ~0.6% Cu and ~0.7 g/t Au are attained in the south-central, highest grade part of the system.

Supergene copper enrichment (PyCc(S)) is focused along the NNE structural zone through the northern part of the deposit. The Late Oligocene erosional event removed the upper parts of the mineralized system, but erosion took place at a rapid rate that did not allow for development of an extensive leached cap or supergene enrichment at that time. Only more recently, likely during most recent glacially-aided erosion into the system, has a leached cap been developed over the system (Ox and Mix) with an underlying supergene enrichment zone. The leached zone ranges from 10–20 m in thickness over the relatively impermeable felsic volcanic rocks in the west to a maximum of 230 m within the Josemaría NNE structural corridor and the tonalite farther east where facilitated by damage zones along faults and increased permeability through groundwater removal of sulphate veins. The underlying supergene enrichment domain attains grades in the range of 0.8-1.5% Cu.

Appreciable oxide copper (malachite and neotocite) mineralization (CuOx) is restricted to a small zone of fractures within the leached cap in the northern part of the deposit. This interpreted to be the result of leaching of the pyrite-poor potassic domain. Also, a significant gold-rich portion of the leached cap occurs along the centre of the deposit, between section lines 5000N and 5500N, with values of up to 0.35 g/t Au within the oxide zone. This area corresponds to the central, and perhaps deepest, parts of the advanced argillic alteration zone within the system.

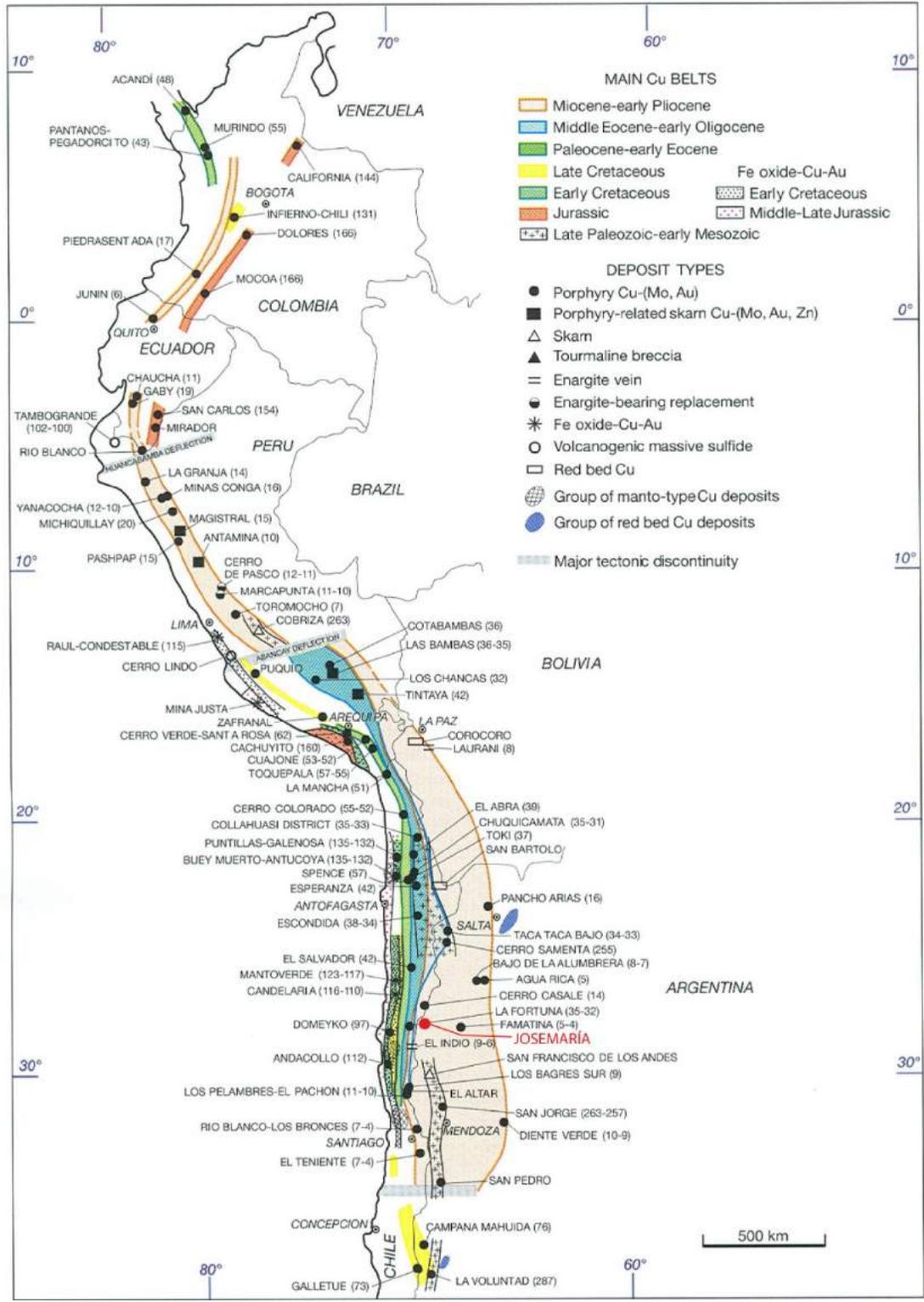
8 Deposit Types

Based on geological features and location, the Josemaría deposit is classified as a copper-gold porphyry system. Porphyries are well documented along the Andes and represent a widespread type of deposit in Chile and Argentina (Figure 8.1).

Porphyry deposits in general are large, low- to medium-grade magmatic-hydrothermal deposits in which primary (hypogene) sulfide minerals occur as veinlets and disseminations within large volumes of altered rock that are spatially and genetically related to felsic to intermediate porphyritic intrusions (Seedorf et al., 2005). The large size and styles of mineralization (e.g., veins, vein sets, stockworks, fractures and breccia pipes), and association with intrusions distinguish porphyry deposits from a variety of other deposit types that may be peripherally associated, including skarns, high-temperature mantos, breccia pipes, and epithermal precious metal deposits. Secondary minerals may be developed in supergene-enriched zones in porphyry copper deposits by weathering of primary sulphides. Such zones typically have significantly higher copper grades, thereby enhancing the potential for economic exploitation (Sinclair, 2007).

Porphyry deposits occur throughout the world in a series of extensive, relatively narrow, linear metallogenic provinces. They are predominantly associated with Mesozoic to Cenozoic orogenic belts in western North and South America and around the western margin of the Pacific Basin, particularly within the South East Asian Archipelago. However, major deposits also occur within Paleozoic orogens in Central Asia and eastern North America, and to a lesser extent, within Precambrian terranes (Sinclair, 2007).

Porphyry deposits are large and typically contain hundreds of millions of tonnes of mineralization, although they range in size from tens of millions to billions of tonnes. Grades for the different metals vary considerably but generally average less than 1%. In typical porphyry copper deposits, copper grades range from 0.2% to more than 1% Cu; Mo content ranges from approximately 0.005% to about 0.03% Mo; gold contents range from 0.004 to 0.35 g/t Au; and silver content ranges from 0.2 to 5 g/t Ag (Sinclair, 2007).



Source: NGEx, October 2013; modified from Sillitoe and Perelló (2005)

Figure 8.1: Porphyry copper belts and major porphyry copper deposits in the Andes

9 Exploration

9.1 Grids and Surveys

Josemaría drill collar coordinates are reported using Gauss Krüger (Campo Inchauspe, Zone 2) coordinates.

The base topography used for mineral resource estimation was obtained from PhotoSat Information Ltd. in Vancouver who provided a 5-m digital elevation model (DEM) produced from stereo 2.5-m resolution satellite images.

9.2 Geologic Mapping

Several phases of geological mapping have been completed at Josemaría, with each phase building on and refining the previous phase. The most recent mapping update was performed by Fionnuala Devine in November 2018.

9.3 Geochemical Sampling

During the period 2003–2005, 315 rock chip and 459 talus fines samples were collected. A central feature of approximately 2.5 km in diameter was delineated by coincident gold, copper and molybdenum anomalies and encouraged further exploration studies.

9.4 Geophysics

IP surveys were completed at Josemaría during the 2003–2004, 2006–2007 and 2009–2010 field seasons. Magnetic surveys were done during the 2003–2004, 2004–2005 and 2006–2007 seasons. Other types of geophysical surveys completed include a CSAMT survey conducted in 2003 and a MIMDAS survey undertaken in 2008–2009.

The porphyry intrusive rocks closely correspond to magnetic (high) anomalies, and the main structural features are also outlined by magnetics.

IP chargeability shows a partial pyrite “ring” around the western and northern parts of the main deposit. The response to the south and east appears to be masked by the post-mineral volcanic cover and chargeability is generally low in this area.

9.5 Pits and Trenches

Trenches were completed primarily following road cuts. Samples were taken over a 3-m interval whenever possible. However, since the trenches followed roads with curves, sampled lengths are not true lengths.

9.6 Exploration Potential

9.6.1 Josemaría Deposit

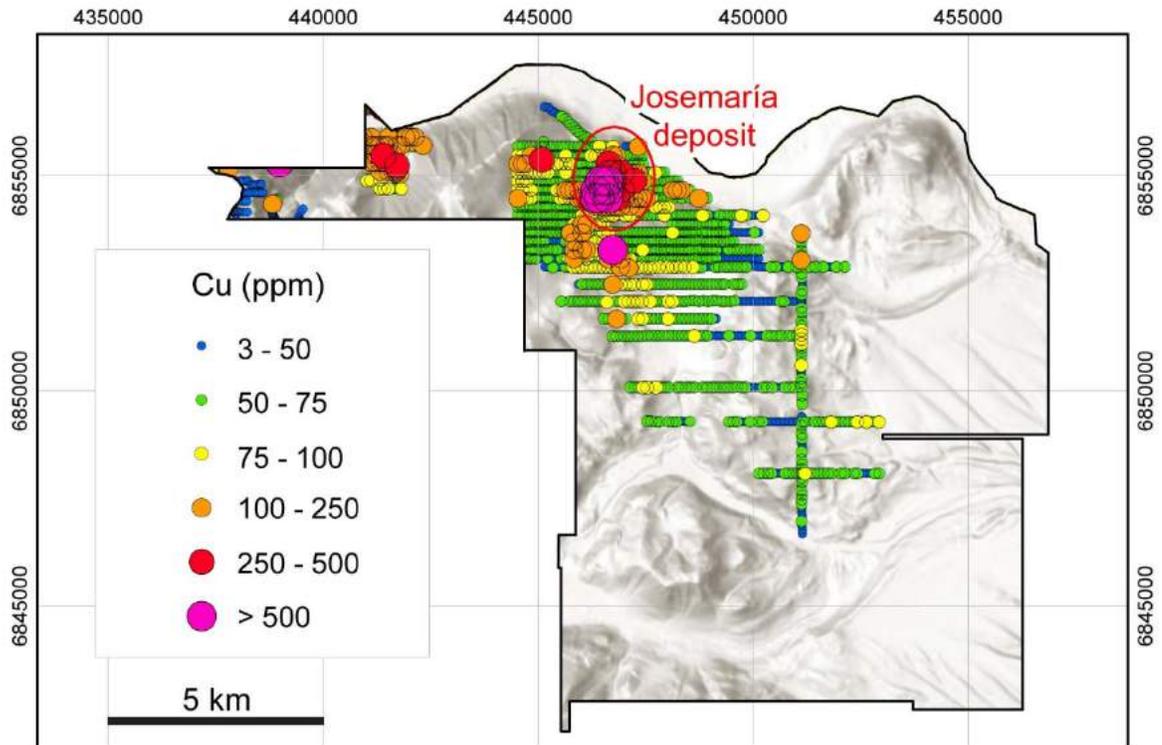
The Josemaría deposit remains open to the south, beneath a thickening cover of post-mineral volcanic rocks, and also at depth. Drilling was planned with a conceptual open-pit configuration in mind, and only two drill holes were extended beyond depths of about 600 m (JMDH06 and 07). Both drill holes encountered lower-grade mineralization; however, they intersected the late mineral porphyry unit, which tends to be lower grade, and potential remains to extend the mineralization at depth within the tonalite unit.

9.6.2 Regional Targets

Several exploration targets were developed in the area during the surface exploration programs that led to the discovery of the Josemaría. At that time, prior to the discovery of Josemaría, several targets were being advanced in parallel, ultimately resulting in the initial drill program. Once the main deposit was discovered, all the exploration effort shifted to deposit definition drilling, and exploration on the other exploration targets was suspended.

These additional targets include the southward extension of the Josemaría deposit, as well as a second major geochemical anomaly on the western side of the property, similar in size and tenor to the Josemaría deposit, that has alteration features consistent with porphyry-style mineralization. These targets, as defined by copper in talus fine samples, are shown in Figure 9.1.

Given that porphyry deposits occur in clusters, and the exploration targets are in the vicinity of the Josemaría deposit and other deposits in the region, there is excellent exploration potential to identify additional porphyry-hosted mineralization. Additional exploration work is recommended to continue to advance them.



Source: NGEx, 2018

Figure 9.1: Exploration targets – copper values from surface talus fines geochemical sampling

10 Drilling

10.1 Summary

Nine drilling campaigns have been carried out at the Josemaría deposit, from 2004 to 2014.

Core was photographed, logged for detailed lithology, alteration and mineralization features, and (RQD) and recovery data were collected. Several of the drill holes were also logged for geotechnical information and two dedicated geotechnical holes were drilled.

Core recovery data were not systematically collected on holes drilled before the 2010-2011 campaign. Core recovery from holes drilled at Josemaría between 2011 and 2014 averages 94%.

Collar locations were surveyed using a differential global positioning system (GPS) instrument.

None of the RC holes were surveyed for down-hole deflection. Diamond drill holes were surveyed for the 2009-2010 season and then systematically starting with the 2011-2012 season. Down-hole surveys were carried out at 50 m intervals on average, using a Reflex multi-shot instrument during the 2011–2012 drilling campaign. For the 2012–2013 and subsequent seasons, a SRG-gyroscope survey was completed for each drill hole by Comprobe Limitada. On average, measurements were collected at 30-m intervals down the hole.

Drill hole orientations are generally appropriate for the mineralization style. The Josemaría deposit is a porphyry system with disseminated mineralization. Reported and described interval thicknesses are considered true thicknesses.

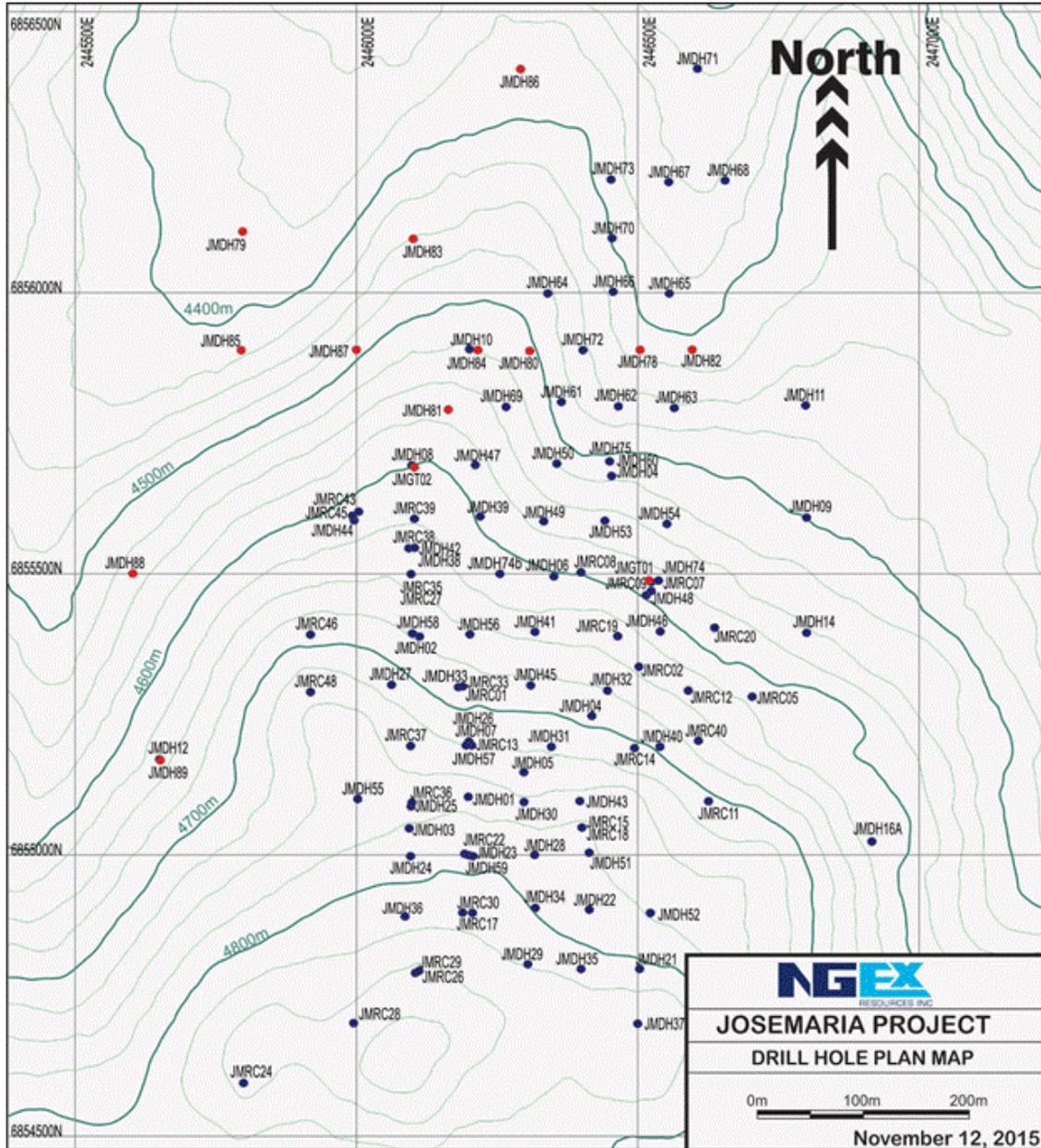
10.2 Drill Programs

Nine drilling campaigns have been carried out at the Josemaría deposit, from 2004 to 2014. Drilling at the Josemaría deposit to date totals 61,100 m in 142 drill holes (Table 10.1), of which 48 holes (17,535 m) are RC holes, and 94 holes (43,565 m) are core holes. More than 90% of the metres drilled were HQ (63.5-mm diameter core).

Table 10.1: Drill summary table – Josemaría

Year	RC Holes	RC Metres	Core Holes	Core Metres
2003–2004	10	3,475	—	—
2004–2005	21	7,822	5	2,406
2005–2006	—	—	2	1,700
2006–2007	17	6,238	0	—
2007–2008	—	—	—	—
2008–2009	—	—	—	—
2009–2010	—	—	7	2,253
2010–2011	—	—	8	2,419
2011–2012	—	—	39	19,236
2012–2013	—	—	19	8,241
2013–2014	—	—	14	7,310
Totals	48	17,535	94	43,565

Drill hole collar locations are shown in Figure 10.1.



Source: NGEx, 2015

Figure 10.1: Drill-hole collar location map, Josemaría

10.3 Geological Logging

Drill core was transported by pick-up truck from the drill sites to the Josemaría camp. At the camp core logging facility, the core was photographed, logged for rock quality designation (RQD) and recovery, and a quick log of the key geological features was prepared. The core was then prepared for cutting and sampling. Prior to the 2011–2012 season, core was cut at the field camp, but during

the 2011-2012 and 2013-2014 campaigns the core was cut at the NGEx sampling facility located in San Juan. Detailed geological logging was also completed in San Juan.

10.4 Recovery

Core recovery data was not systematically collected on holes drilled before the 2011-2012 campaign but was systematically collected for all holes drilled between 2011 and 2014, and averaged 94%.

Recovery was measured with a metric tape between drill core marks, annotated and the percentage recovery calculated. RQD was calculated as the total length of recovered core (measured from pieces) that exceeded or equaled 10 cm.

10.5 Collar Surveys

Drill sites were initially located in the field by hand-held global positioning system (GPS) instrument and marked with stakes for the collar location and a front and back site indicating the azimuth. The drill was moved on to the site and then lined up with the stakes by the supervising geologist. Following completion of the drill hole, final collar locations were surveyed using a differential GPS instrument.

10.6 Downhole Surveys

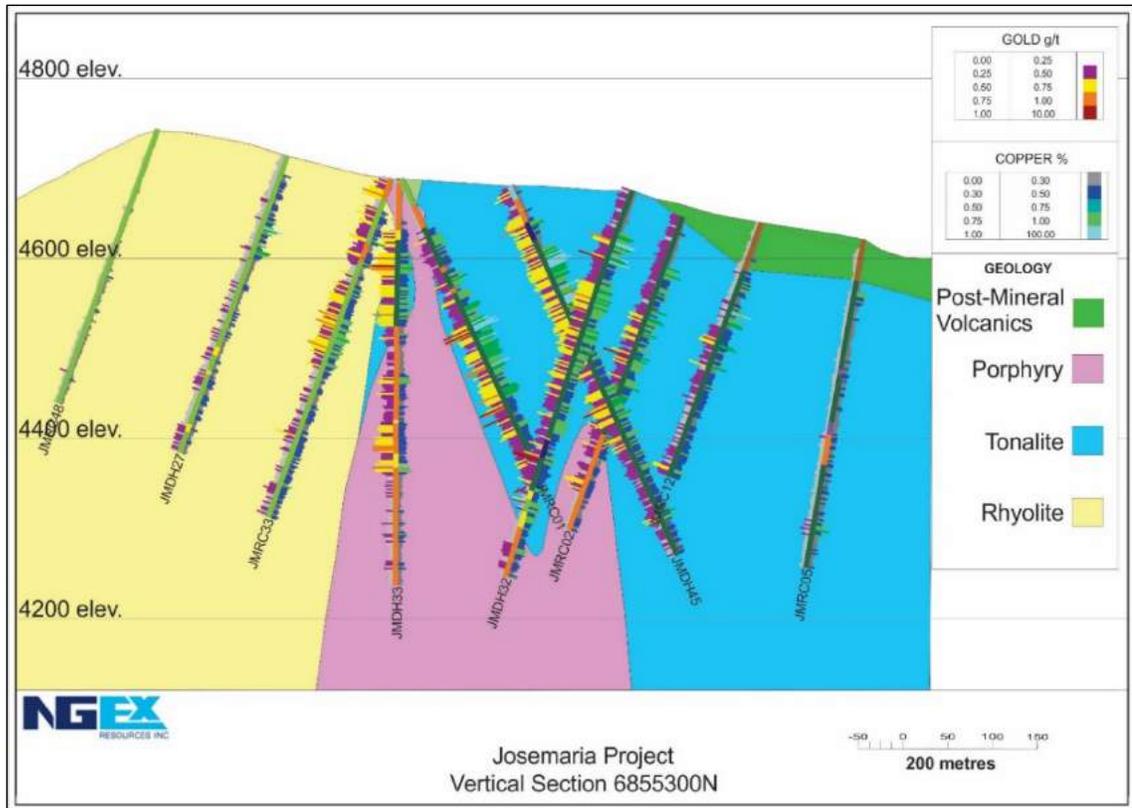
Beginning in 2009, downhole surveys were carried out using a Reflex multi-shot instrument at, on average, 50-m intervals within the hole.

For the 2012-2014 seasons (JMDH78) a SRG-gyroscope survey was completed for each drill hole by Comprobe Limitada, with measurements collected at 30-m intervals down the hole.

Earlier core and RC holes were not surveyed for down-hole deflection. Hole deflection is typically less than 0.001° per metre in dip and 0.01° per metre in azimuth. Given the low deflection of the holes and the continuous, disseminated nature of the mineralization, the lack of survey data from the RC holes is not considered to be a significant issue.

10.7 Sample Length/True Thickness

Josemaría is a porphyry deposit that contains disseminated mineralization. Reported and described interval thicknesses are considered true thicknesses. A drill section through the deposit illustrating the typical drill orientations in relation to the mineralization is illustrated in Figure 10.2.



Source: NGEx, 2015

Figure 10.2: Example drill section 5300N, Josemaría

11 Sample Preparation, Analyses, and Security

11.1 Surface Sampling

11.1.1 Talus Sampling

Sampling of talus fines was carried out using a compositing method that results in samples representative of 100 m along the sampling line. Talus fines were collected as composites of 10 sites located at 10-m intervals, centred if possible, on a 100-m line station.

11.1.2 Chip Sampling

Chip sampling followed conventional methods of following as close to the centre line of the sample as practical. Samples were chipped not cut. The majority of chip samples were taken along road-cut type trenches. Sample width was kept constant within each trench as much as possible.

11.2 Drill Sampling

11.2.1 Pre-2007 Drill Sampling

The entire length of the holes was logged, on a systematic 2-m interval in the case of RC, and on a systematic 1-m core length in the case of DDH holes. RC chips were collected at the drill in large sacks weighing about 40 kg. These were taken to the camp where they were weighed and run through a quartering and homogenizing process using riffle splitters that results in a 5-kg split for shipment to the lab. Representative samples are retained as a geological record of the hole and for re-assay.

The core intervals were split in half by saw with one half then being submitted for assay and the balance being store in San Juan for reference. Also, from the saved one-half core, samples were taken for density measurements.

No geologic breaks dictated breaks in the uniform 2-m (RC) or 1-m (DDH) sampling, which is appropriate for a bulk tonnage, low-grade deposit. HQ diameter core was drilled to provide adequate sample weights. The average weight of a half core sample for a 2-m interval is 8.0 kg, and therefore a significant weight that provides for sample preparation and assaying.

The rock is generally very competent, and overall recoveries are in the order of 95% or better, with only very occasional fracture zones having recoveries of less than 70%.

11.2.2 NGEx Sampling

All drilling since 2009 has been core drilling. Core was sampled continuously from the beginning of recovery to the end of the hole. Samples are generally 2 m long (except for JMDH01 to 07 that were sampled on 1-m intervals). Drill core was cut in half using a circular, water-cooled rock saw. Half-cores are randomly weighed and compared to verify that 50% of the material was sampled.

One half of the core was used as a geochemical sample and the other stored in boxes or trays for reference and future revisions.

11.3 Density Determinations

A total of 11,752 core samples have been systematically analyzed for specific gravity (SG) since the 2011–2012 drilling program. Specific gravity was measured by NGEx technicians using the water immersion method at the NGEx core logging and sampling facility in San Juan.

11.4 Sample Preparation and Analysis

11.4.1 Surface and RC Samples

Sample preparation included; drying the sample, crushing to >70% passing -2 mm mesh, and pulverizing to >85% passing -75 µm screen.

Gold was determined using an AAS finish on a 50-g sample. The detection limit and the upper range of this method was 0.005 ppm Au and 10 ppm Au respectively.

The sample was also digested using a HF–HNO₃–HClO₄ acid digestion, HCl leach and finished using ICP-AES for 27 elements. In addition, Hg was determined using an aqua regia digestion and cold vapour AAS.

11.4.2 Core

Sample preparation included drying the sample, crushing to better than 85% passing 10-mesh and pulverizing to 95% passing 200-mesh.

Sample digestion was done by a multi-acid attack with the exception of one submission during the 2009-2010 campaign.

Gold was determined by fire assay with an atomic absorption spectroscopy (AAS) finish based on a 30 g sample. A suite of 37 elements, including copper, was determined by ICP-emission spectroscopy (ES) analyses.

Samples analyzed before the 2010-2011 campaign had copper re-assayed by AAS only if the ICP result exceeded the upper detection limit of 10,000 ppm. Beginning in 2010, all samples with copper grades over 5,000 ppm Cu were re-assayed by AAS. Starting in 2012, copper determinations in all samples were done by both ICP and AAS.

Mercury concentration was determined by cold vapour/AA in all samples up to 2010.

11.5 Analytical and Test Laboratories

Surface and RC samples were analysed by ALS Chemex (ALS) in Chile. At the time of analysis, ALS held ISO9001 accreditations for selected procedures.

From 2009, all core samples have been analyzed by ACME Laboratories in Chile (ACME). ACME's accreditations have included ISO9001:2000 and ISO/IEC17025. Sample preparation was undertaken at ACME's sample preparation laboratory in Mendoza, Argentina, which holds ISO 9000:2001 accreditation.

SGS Laboratories (SGS) in Chile was used as an umpire laboratory during 2012-2013. At the time the analyses were performed, SGS held ISO/IEC17025 accreditations.

ACME and ALS were also used for surface sample analyses.

11.6 Quality Assurance and Quality Control

11.6.1 Surface and RC Sampling

There is only limited information on the overall precision of the assay data for surface and RC sampling, and no information regarding its accuracy.

Duplicate samples were collected in the field and routinely examined using regression methods. A total of 447 duplicate samples were collected from drilling, including RC drilling, up to 2007. Statistical analyses made on these duplicates indicate that the overall precision of the samples was good or very good.

11.6.2 Core Sampling

A quality control protocol was implemented in the 2009–2010 season, beginning with JMDH08; the program, with some minor variations, has been followed since that date. The programs include blanks, duplicates and standard reference materials inserted in the sampling sequence.

The programs included a total of seven quality control samples inserted for every 77 samples submitted to the laboratory to provide sufficient controls for the 78 and 36 element trays used in the laboratory. These control samples consist of:

- Standard #1 (medium-grade, approximately deposit average grades)
- Standard #2 (low-grade, approximately equates to the cut-off grade used in estimation), implemented during the 2011–2012 campaign
- Blank (coarse material)
- Field duplicate (second half core)
- Preparation duplicate (second pulp)
- Assay duplicate (second assay)

Standard Reference Materials

Certified reference materials (CRMs) utilized in the 2009–2010 and 2010–2011 campaigns were acquired from SGS in Argentina.

In September 2011, five standard reference materials (SRMs) were prepared by NGEx using selected coarse rejects from the previous drill season at Los Helados and used during the 2011-2012 campaign. The samples were prepared by Vigalab SA (Vigalab; now part of the Intertek Group). At the time, Vigalab held ISO9001:2009 accreditation.

Five analytical laboratories located within the region were used to perform a round robin test of results: ACME, Activation Laboratories Ltd (Actlabs; at the time, Actlabs was ISO 17025 accredited and/or certified to 9001: 2008), SGS, ALS and Vigalab. Based on the round robin results, the SRMs were assigned an averaged best value.

Coarse Blanks

Blank material was obtained from an andesite outcrop located a few kilometres away from the deposit.

Duplicates

Field duplicates were obtained by cutting a half-core into quarter core to be analyzed independently.

11.6.3 External Assay Checks

A set of 183 coarse rejects from the 2012 drill campaign were selected for re-assaying at SGS Laboratories. Grades reported by ACME on the coarse rejects ranged from 0.093 to 11.10% Cu and 0.05 to 0.751 g/t Au.

Samples were submitted for preparation at the SGS facilities in San Juan, Argentina and assayed in Callao, Peru.

11.7 Databases

Drill hole data are stored in a GEOVIA GEMS database, which is a Microsoft Access database platform created and manipulated using GEMS.

Data stored for each drill hole includes collar information, downhole surveys, codes and comments for lithology, alteration and mineralization, assays, specific gravity, magnetic susceptibility, recovery, RQD and metallurgical sample information.

11.8 Sample Storage

Drill core is stored in a core storage warehouse in San Juan. Core is well organized and stored in racks, easily available for review. The laboratory returns the pulps and coarse reject for each sample that has been sent for analysis. These are stored at the San Juan facility.

11.9 Sample Security

The logging facility is fenced, locked when not occupied, and is secure. Samples are handled only by company employees or their designates (i.e. laboratory personnel).

Samples are in the control of an NGEx employee or contractor to NGEx from the time they leave the site until they arrive at the San Juan lab.

12 Data Verification

12.1 G. Zandonai (2013)

Checks completed included spot checks of database assays against the laboratory certificates; and review of lithology and alteration information in drill logs against the drill core.

In the central part of the deposit there is reasonably good outcrop or subcrop. Surface trench samples from this area confirm the presence of copper mineralization over a broad area. Verification carried out included an overview of the general nature of the site, checking locations of the drill collars, and a brief examination of core samples selected to observe the nature of the copper mineralization and a range of alteration types logged by NGEx geologists.

In Zandonai's opinion, verification was adequate to confirm that, as represented by NGEx, Josemaría was a copper-gold mineralized deposit associated with high level siliceous porphyry intrusives and associated breccias, that it contained significant disseminated and vein copper mineralization, both oxide and sulphide, and displayed alteration types characteristic of a porphyry environment.

12.2 F. Devine (2014)

F. Devine was responsible for the 2014 surface mapping program, which included extensive traverses over, and peripheral to, the deposit area. Many drill sites were located and correlated with the surface maps and 3-D database. Mapping of surface outcrop included lithology, alteration and mineralization features which were correlated with the drill database and sections previously developed by NGEx geologists. Updates to the geological model were completed by F. Devine following 7 days of core review in San Juan in May 2014, which included extensive review of assay data as well as 10 witness samples taken of quartered drill core. The values of the witness samples correlate well with the drill database. A 3-D model of the deposit was developed following this work. An additional trip the core facility in San Juan was undertaken in March 2018 to review drill core across several sections to work toward a co-authored peer-reviewed publication of the geology of the Josemaría deposit. F. Devine has also made several trips to other deposits and prospects in the immediate district. In Devine's opinion, the geological and geochemical data presented in this report is an adequate and accurate reflection of the geology of the Josemaría deposit.

13 Mineral Processing and Metallurgical Testing

13.1 Testwork Programs

Two Josemaría metallurgical testwork programs (Phases I and II) were conducted at SGS Minerals S.A. (SGS) in Santiago, Chile (SGS, 2015). The main activities completed during the metallurgical test programs were:

- Sample selection for the metallurgical test programs
- Chemical characterization including mineralogical analysis
- Physical characterization and parameter determination (including Bond work index (BW_i), Bond rod work index (RW_i), abrasion index (A_i), SAG mill competency (SMC) and SAG power index (SPI) testing, and specific gravity)
- Gold recovery using gravity processing techniques
- Leaching of the copper and gold oxide ore types
- Copper, gold and silver recovery using conventional sulphide flotation
- Liquid-solid separation testwork

A subsequent testwork program (ALS, 2018) was undertaken as part of this PFS to confirm results that had been generated in SGS 2015 testing, and to provide information for planning a future testwork program in support of a feasibility study. The ALS 2018 testing was completed by ALS Laboratory in Kamloops, British Columbia and was focussed on the grade-recovery relationship for different lithological zones of the deposit.

13.2 Geometallurgical Domains

Sample selection for the SGS 2015, Phase I metallurgical test program was based on the then current geological model for the Josemaría deposit, which divided the deposit into five geometallurgical domains approximately by depth and mineralogy, as follows:

- Copper oxide
- Gold oxide
- Transitional
- Upper Fresh Sulphide
- Lower Fresh Sulphide

Composite samples were selected to represent each of these domains (Table 13.1).

Table 13.1: Composite description, Josemaría Phase I

Geometallurgical Domain	Depth from Surface	Fraction of Total Cu (%)	Fraction of Total Ore Mass (%)	Sample ID
Oxide-rich Au	At surface	1	2	Oxide Au
Oxide-rich Cu	At surface	4	3	Oxide Cu
Transitional	40 m from the base of the oxide zone	12	8	Transition
Sulphide Upper Fresh	100 to 300m	34	32	Upper Fresh
Sulphide Lower Fresh	> 300 m depth	50	55	Lower Fresh
Totals		100	100	

Following completion of the SGS Phase I program, an updated geological model was developed for the Josemaría deposit that divided the deposit into four principal lithologies. Accordingly, during the Phase II testwork program, sample selection was modified to reflect this revised interpretation of the geometallurgical domains, and these lithology-based geometallurgical domains were used for the balance of the testwork. The four lithology-based domains included:

- Supergene
- Rhyolite
- Tonalite
- Porphyry

Composite samples representing each of these four zones were created (Table 13.2).

Table 13.2: Composite description, Josemaría Phase II

Geometallurgical Domain	Main Characteristic	Fraction of Total Cu* (%)	Fraction of Total Ore Mass* (%)	Sample ID
Supergene	Mineralogical enrichment zone	8	4	Supzone
Rhyolite	Lithology	21	24	Rhyzone
Tonalite	Lithology	55	54	Tonzone
Porphyry	Lithology	16	18	Porzone

*As percentage of total quantity in deposit. Note that these fractions differ from those in the final mine plan and are based on the resource, rather than the reserve.

In addition to composite samples for each of the four lithology-based geometallurgical domains, the Phase II program included selection of 25 variability samples for comminution and flotation testing to test the variability within the orebody. Results generated in the Phase II program provided a basis for the metallurgical projections used in the process design and financial modelling.

13.3 Head Sample Characterization

Representative splits from each of the composite samples prepared to represent the geometallurgical domains in Phases I and II of the SGS 2015 program were assayed by a combination of inductively coupled plasma (ICP) mass spectrometry and fire assay, and the average analyses are summarized in Table 13.3 and Table 13.4.

Table 13.3: Chemical characterization of Phase I domain composites

Phase I Geometallurgical Domain	Analysis (%)				Analysis (g/t)				
	Cu	Cu _{Sol} *	Fe	S	Au	Ag	Mo	As	Hg
Upper Fresh Sulphide	0.402	<0.005	3.32	3.34	0.335	1.5	74	69	<2
Lower Fresh Sulphide	0.346	<0.005	3.27	3.21	0.233	2.7	50	156	<2
Transition	0.527	0.054	3.53	2.39	0.273	2.3	71	41	<2
Oxide Au	0.029	na**	4.01	0.43	0.393	2.4	64	13	na**
Oxide Cu	0.298	na**	3.88	0.46	0.226	2.9	70	<1	na**

* Cu_{Sol} = weak acid soluble copper, ** na = not assayed

Table 13.4: Chemical characterization of Phase II domain composites

Phase II Geometallurgical Domain	Analysis (%)				Analysis (g/t)				
	Cu	Cu _{Sol} *	Fe	S	Au	Ag	Mo	As	Hg
Supzone	0.617	0.047	2.95	1.34	0.32	1.6	61	11	<2
Rhyzone	0.338	0.013	3.40	3.48	0.22	0.6	99	50	<2
Tonzone	0.331	0.003	3.01	1.16	0.25	1.3	87	13	3
Porzone	0.306	0.005	2.45	2.39	0.33	1.6	50	29	<2

* Cu_{Sol} = weak acid soluble copper

The analyses highlighted variation between the geometallurgical domains with respect to copper grade, soluble copper, sulphur and arsenic contents. Given the compositional variance of the geometallurgical composite samples, it is expected that the physical characteristics and mineralogy between domains will also be distinct.

13.4 Mineralogy

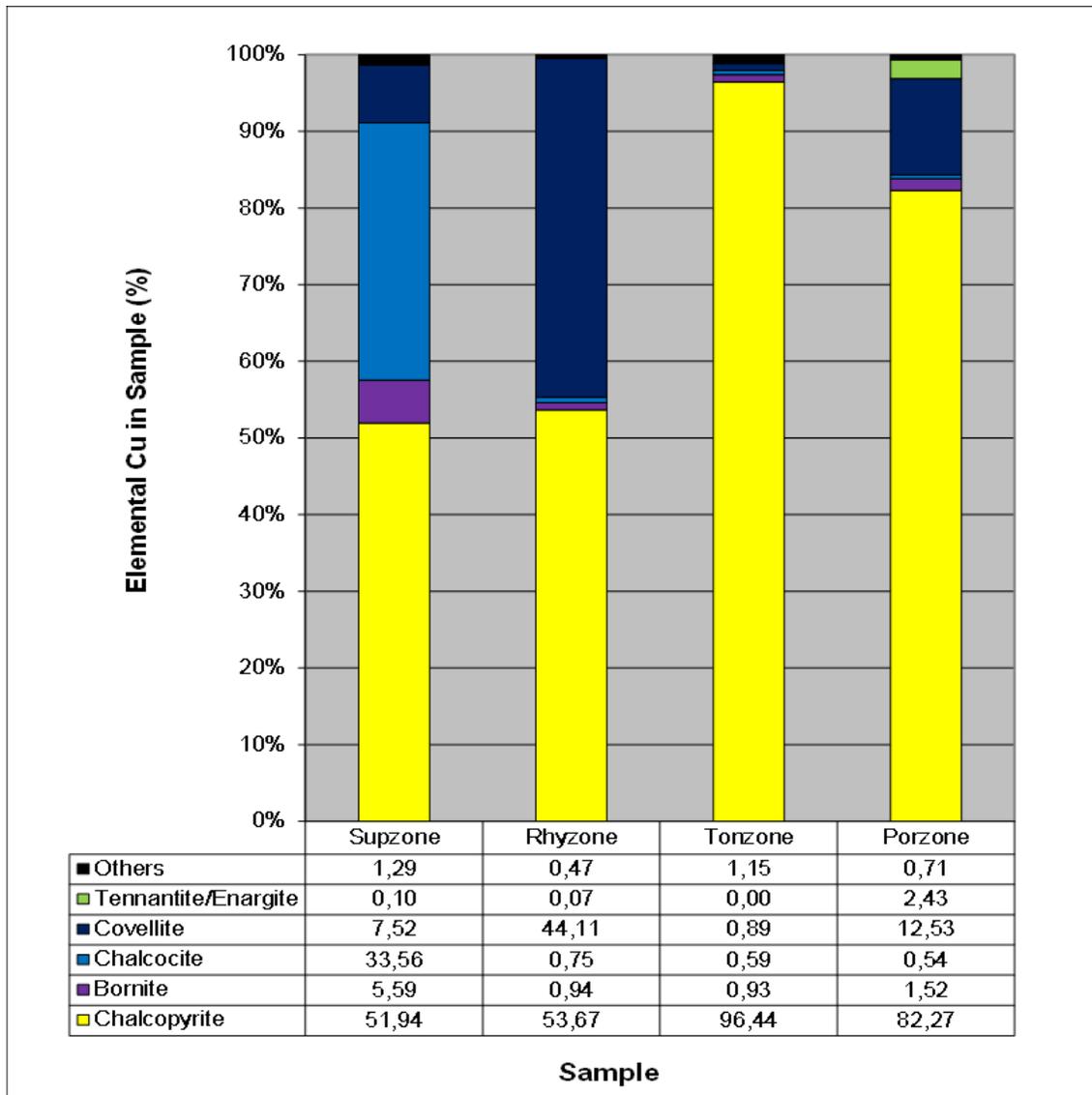
Mineralogical analysis of composites samples prepared to represent the geometallurgical domains in the SGS 2015 Phase I and II testwork was completed by quantitative scanning electron microscopy (QEMSCAN) to facilitate development of recovery models and identification of the preferred process flowsheet for the Josemaría deposit.

The mineralogical analysis highlighted variation in pyrite:copper sulphide ratios present in the different geometallurgical zones, with the highest ratio observed in the Rhyolite zone. High pyrite:copper sulphide ratios can cause difficulty in the separation of copper minerals from pyrite by conventional sulphide flotation techniques. A summary of the calculated pyrite:copper sulphide ratio for each geometallurgical domain is provided in Table 13.5.

Table 13.5: Calculated pyrite:copper sulphide ratio in Phase II composite samples

Geometallurgical Domain	Pyrite:Copper Sulphide Ratio
Rhyzone (Rhyolite zone) composite	10.7 : 1
Supzone (Supergene zone) composite	1.5 : 1
Tonzone (Tonalite zone) composite	2.3 : 1
Porzone (Porphyry zone) composite	4.4 : 1

Chalcopyrite was the principal copper mineralization in each of the four geometallurgical domains as illustrated in Figure 13.1. While each of the samples contained secondary copper sulphide minerals, the abundance of secondary copper and the mineralization were different for each geometallurgical domain composite. These differences in secondary copper have the potential to influence both copper recoveries and final copper concentrate grades.



Note: the table below the figure uses the Spanish protocol of using commas for decimal points, so 1,29 = 1.29.

Figure 13.1: Elemental copper department in Phase II geometallurgical domains

13.5 Physical Characterization

As part of the SGS Phase I work, physical characterization was completed for each of the five composite domain samples, including Bond work index (BWi), Bond rod work index (RWi), abrasion index (Ai), physical parameter tests including SAG mill competency (SMC) and SAG power index (SPI) testing, and specific gravity determination. Average results for each domain are summarized in Table 13.6.

Table 13.6: Physical characterization of Phase I composites

Geometallurgical Domain	Specific Gravity	Bond Ball BWi (kWh/t)	Bond Rod RWi (kWh/t)	Bond Abrasion index (g)	SMC (A x b)	SPI (min)
Upper Fresh Sulphide	2.80	10.9	11.9	0.111	45.1	98
Lower Fresh Sulphide	2.75	12.3	12.3	0.120	50.3	91
Transition	2.79	11.9	11.3	0.119	58.4	69
Oxide Au	2.78	11.8	11.0	0.107	92.0	59
Oxide Cu	2.75	12.2	11.4	0.064	50.0	79

Based on the SMC test results, Upper Fresh Sulphide, Lower Fresh Sulphide, and Oxide Cu samples can be classified as medium–hard material; the Transition sample can be classified as moderately soft material; and the Oxide Au sample can be classified as soft material. All of the samples are classified as medium-hard using the SPI test procedure. The descriptive classifications noted above are based on a comparison of the individual test results against a database of historical data for various materials tested by these methods.

The BWi and RWi results indicate that all of the samples tested are moderately hard, although each of the samples tested reported a low Ai classification. The results indicate that ores from these domains, when processed in a conventional SAG-ball mill comminution circuit, could be expected to generate low wear and grinding media consumption rates.

As part of the SGS Phase II testwork program, physical characterization testwork including BWi, RWi, SMC, Ai and specific gravity determination, was completed on composite samples representing the four Phase II geometallurgical domains and on the 25 variability samples. Results are summarized in Table 13.7 for the composite domain samples and Table 13.8 for the variability samples.

Table 13.7: Physical characterization of Phase II composites

Geometallurgical Domain	Specific Gravity	Bond Abrasion Index (g)
Supzone	2.785	0.161
Rhyzone	2.849	0.169
Tonzone	2.892	0.206
Porzone	3.114	0.161

Table 13.8: Physical characterization of Phase II variability samples

Geometallurgical Domain	Sample ID/Test	Specific Gravity	Bond Ball BWi (kWh/t)	Bond Rod RWi (kWh/t)	SMC (A x b)
Supzone	VAR 1 - VAR 7	2.722 - 3.131	10.44 - 18.73	9.85 - 16.29	29.3 - 74.6
Rhyzone	VAR 8 - VAR 13	2.792 - 3.111	8.81 - 11.52	9.11 - 13.47	35.2 - 61.9
Tonzone	VAR 14 - VAR 18	2.695 - 2.808	13.94 - 17.71	13.32 - 14.45	27.0 - 37.2
Porzone	VAR 19 - VAR 25	2.742 - 2.911	11.24 - 14.76	10.51 - 12.49	36.7 - 50.8

Abrasion index results summarized in Table 13.9 highlighted differences in the hardness of the material between zones. The Supergene zone includes materials classified as very hard to soft while the Rhyolite zone included materials classified as hard to moderately soft. The Tonalite and the Porphyry zones reported lower variability in terms of A x b values and were generally harder materials. The Tonalite zone included material classified as very hard to hard, while the Porphyry zone reported material classified as hard to medium.

Table 13.9: Hardness determination for Phase II composites

Geometallurgical Domain	A x b Range		
	Min	Max	Average
Supzone	29.3	74.6	46.7
Rhyzone	35.2	61.9	46.1
Tonzone	27.0	37.2	31.9
Porzone	36.7	50.8	42.4

The testwork program to support a feasibility study should include further evaluation of physical characteristics of the various ore types, including characteristics specifically required to design high pressure grinding roll (HPGR) crushing and HPGR testwork on bulk samples.

13.6 Gravity Recoverable Gold

Standard Knelson three-stage gravity recoverable gold tests were completed as part of the SGS 2015 testwork. Gravity testwork highlighted that ore samples from the Josemaría deposit as tested did not contain appreciable free gold, indicating that most of the gold in the deposit was contained in sulphide minerals. This assertion was further supported by results of the sulphide flotation testwork, which resulted in high recoveries of metal values and sulphide minerals. The testwork results do not support incorporation of a gravity-recoverable gold circuit into the proposed processing flowsheet.

13.7 Flotation Testing, SGS (2015)

The SGS Phase II testwork included sulphide flotation testing on the composite domain samples prepared to represent the four lithology-based geometallurgical domains including Supzone, Rhyzone, Tonzone and Porzone samples. These results have been used as the basis for the PFS metallurgical recoveries used in the financial model.

Flotation testing included both open circuit tests (OCT) and locked cycle tests (LCT). Generally, OCTs were completed to test amenability of the ore and the influence of adjustments to specific process parameters. The OCTs generally defined preferred conditions for flotation, including reagent scheme and residence time. Multiple-cycle LCTs were subsequently completed to confirm OCT results and demonstrate recoveries with conditions approximating operation of the commercial plant under preferred conditions. LCT testing comprised both rougher-scavenger and cleaner flotation tests, over several cycles with appropriate recycling of intermediate streams.

One LCT was completed for each of the Supzone, Rhyzone, Tonzone and Porzone composite samples. Table 13.10 and Table 13.11 show average results from the final three cycles of LCT testing.

Table 13.10: Summary of LCT results for Phase II composite samples

Geometallurgical Domain	Calculated Feed Cu Grade (%)	Mass to Concentrate (%)	3 rd Cleaner Concentrate Grade			Overall Recovery to Final Concentrate (%)		
			Cu (%)	Au (g/t)	Ag (g/t)	Cu	Au	Ag
Supzone	0.61	1.74	30.0	13.7	52.4	85.3	72.2	78.3
Rhyzone	0.32	0.98	28.9	11.0	49.0	89.0	59.9	68.1
Tonzone	0.33	1.24	24.3	18.0	48.1	90.6	77.0	51.1
Porzone	0.29	1.28	20.3	13.7	54.8	90.0	62.6	60.3

Table 13.11: Final concentrate analysis from LCT testing of Phase II composite samples

Analysis	Geometallurgical Domain			
	Supzone	Rhyzone	Tonzone	Porzone
Calculated feed Cu grade (%)	0.61	0.32	0.33	0.29
Sb (%)	0.008	0.020	0.033	0.026
As	0.070	0.308	0.095	0.200
Cd	0.003	0.003	0.004	0.002
Cl	0.054	0.018	0.009	0.072
Cu	30.0	28.9	24.3	20.3
Cu _{Sol}	0.353	0.105	0.093	0.021
Fe	25.4	25.9	23.7	31.6
S	36.2	36.2	28.3	37.1
Zn	0.544	0.111	0.187	0.161
Au (g/t)	13.7	11.0	18.0	13.7
Hg	0.8	3.1	1.0	0.5
Ag	52.4	49.0	48.1	54.8
Insoluble (%)	7.47	8.50	15.7	11.9

Following identification of preferred conditions, each of the 25 variability samples was tested in OCTs, and ten of those variability samples were further tested in LCTs. Average results of the last three cycles from the LCTs are summarized in Table 13.12 to Table 13.14.

Table 13.12: Summary of LCT results for Phase II variability samples

Sample ID	Calculated Feed Cu Grade (%)	Mass to Concentrate (%)	Overall Recovery to Final Concentrate (%)				
			Cu	Au	Ag	Fe	S
VAR 3	0.65	1.71	72.5	66.3	71.6	10.7	54.4
VAR 7	0.39	1.07	80.2	43.5	69.0	7.2	45.1
VAR 9	0.37	1.06	88.1	62.2	78.7	13.6	17.0
VAR 10	0.35	1.54	90.5	69.1	75.4	30.5	33.9
VAR 12	0.33	2.27	89.8	75.0	77.4	48.3	55.7
VAR 14	0.40	1.46	92.2	75.8	67.2	13.2	23.9
VAR 16	0.36	1.47	90.7	64.1	76.6	9.4	60.5
VAR 18	0.25	1.07	82.6	57.5	75.0	10.8	46.7
VAR 20	0.24	1.22	82.4	57.7	43.0	8.6	13.0
VAR 25	0.30	1.38	85.4	53.1	55.5	9.0	18.6

Table 13.13: Final concentrate analysis from LCT testing of Phase II variability samples

Analysis	Variability Samples				
	VAR 3	VAR 7	VAR 9	VAR 10	VAR 12
Calculated feed Cu grade (%)	0.65	0.39	0.37	0.35	0.33
Sb (%)	<0.005	<0.005	0.04	<0.005	0.02
As	0.086	0.064	0.273	0.464	0.3
Cd	<0.001	0.002	0.004	0.004	0.002
Cl	0.018	0.045	insufficient sample	0.018	0.018
Cu	27.7	29.5	30.7	20.8	13.1
Cu _{sol}	0.76	0.435	0.285	0.175	0.174
Fe	21.6	21.3	26.5	27.0	40.8
S	31.8	30.2	38.1	35.2	46.4
Zn	0.066	0.093	0.035	0.196	0.122
Au (g/t)	7.1	7.1	9.8	7.0	5.5
Hg	1.7	1.1	3.2	1.5	1.7
Ag	33.1	54.7	35.9	45.2	48.7
Insoluble (%)	17.4	21.8	6.38	13.3	5.29

Table 13.14: Final concentrate analysis from LCT testing of Phase II variability samples - continued

Analysis	Variability Samples				
	VAR 14	VAR 16	VAR 18	VAR 20	VAR 25
Calculated feed Cu grade (%)	0.40	0.36	0.25	0.24	0.30
Sb (%)	<0.005	0.078	0.009	<0.005	0.410
As	0.033	0.160	0.360	0.007	0.240
Cd	0.002	0.004	0.010	0.002	0.008
Cl	0.018	0.027	0.018	0.018	0.036
Cu	25.4	22.5	19.7	16.2	18.7
Cu _{sol}	0.143	0.070	0.102	0.089	0.108
Fe	26.9	24.7	23.6	18.7	18.5
S	31.4	28.2	29.2	23.9	22.2
Zn	0.369	0.219	0.327	0.041	1.01
Au (g/t)	14.9	11.0	8.20	7.80	8.80
Hg	<0.1	0.2	<0.1	0.7	0.6
Ag	43.3	78.7	76.3	21.8	52.3
Insoluble (%)	29.2	33.0	23.7	21.0	28.2

13.8 Grind Size Determination

As part of the SGS 2015 Phase II testwork, flotation testing over a range of primary grind sizes between 100 and 160 µm was completed to determine the influence of particle size on flotation recovery/grade results. Figure 13.2 to Figure 13.5 illustrate the influence of feed particle size on the grade-recovery relationship for Supzone, Rhyzone, Porzone and Tonzone composite domain samples, respectively.

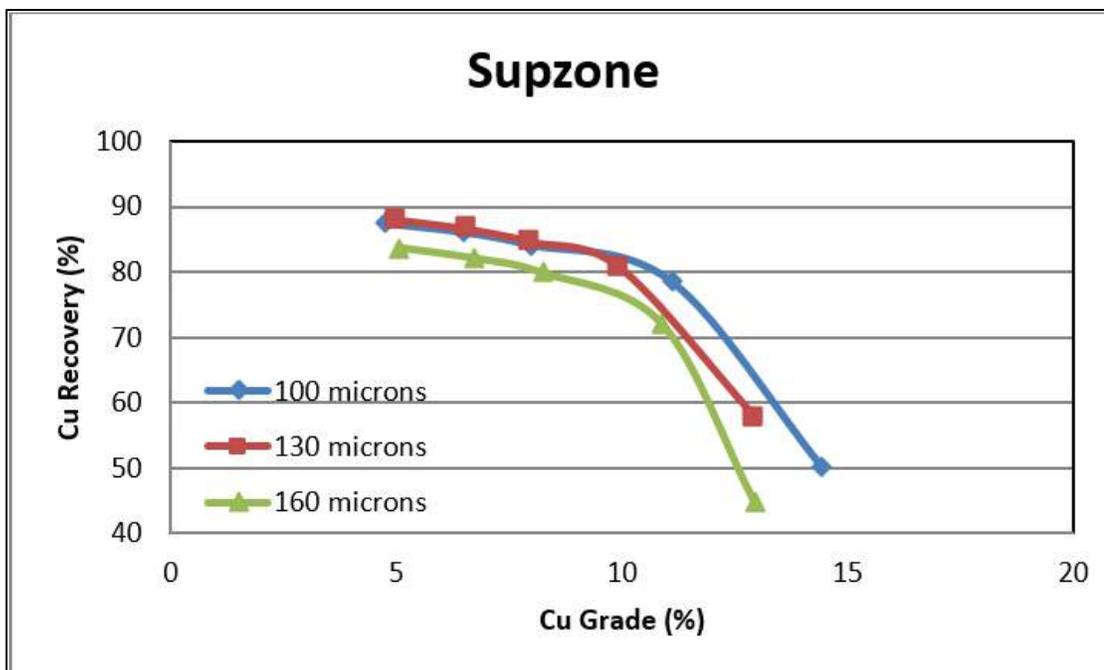


Figure 13.2: Supzone - primary grind size effect on Cu recovery vs grade

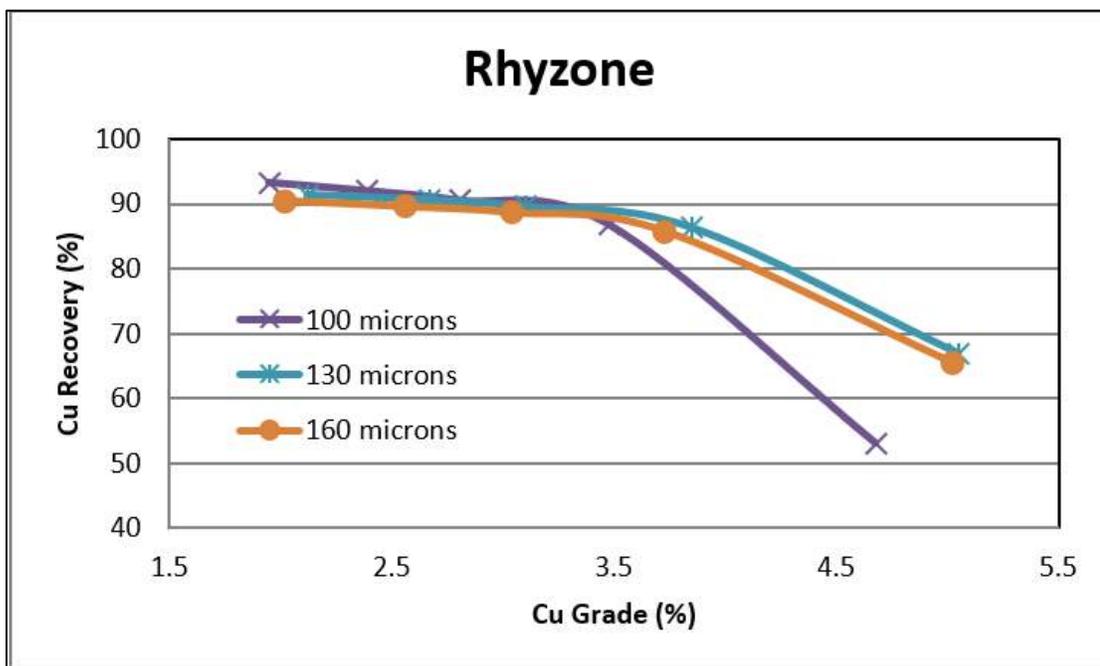


Figure 13.3: Rhyzone - primary grind size effect on Cu recovery vs grade

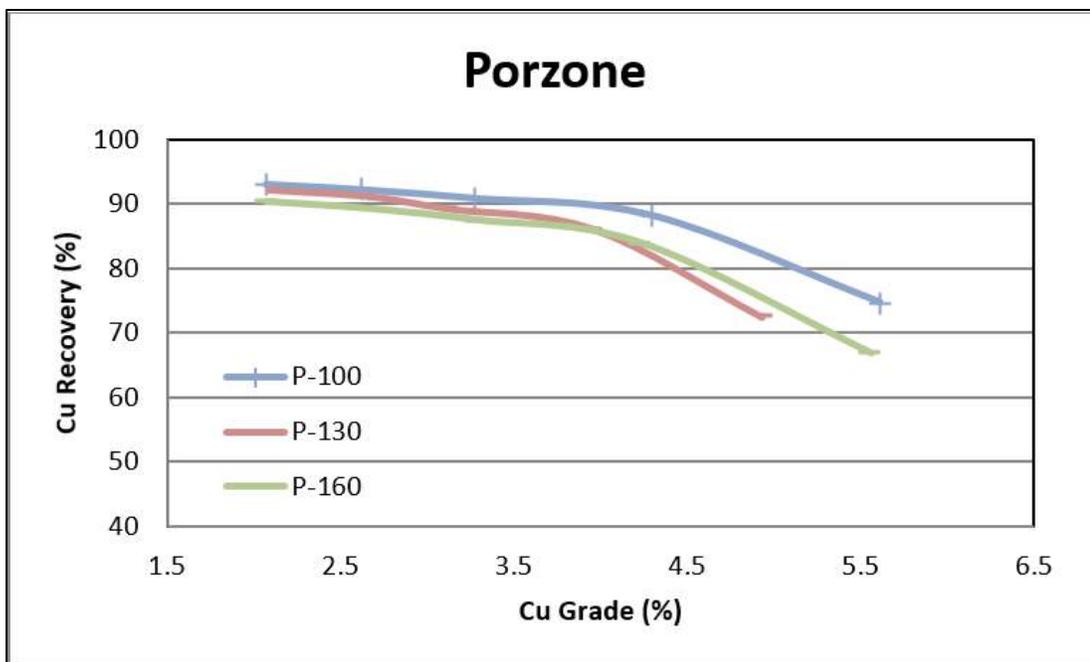


Figure 13.4: Porzone - primary grind size effect on Cu recovery vs grade

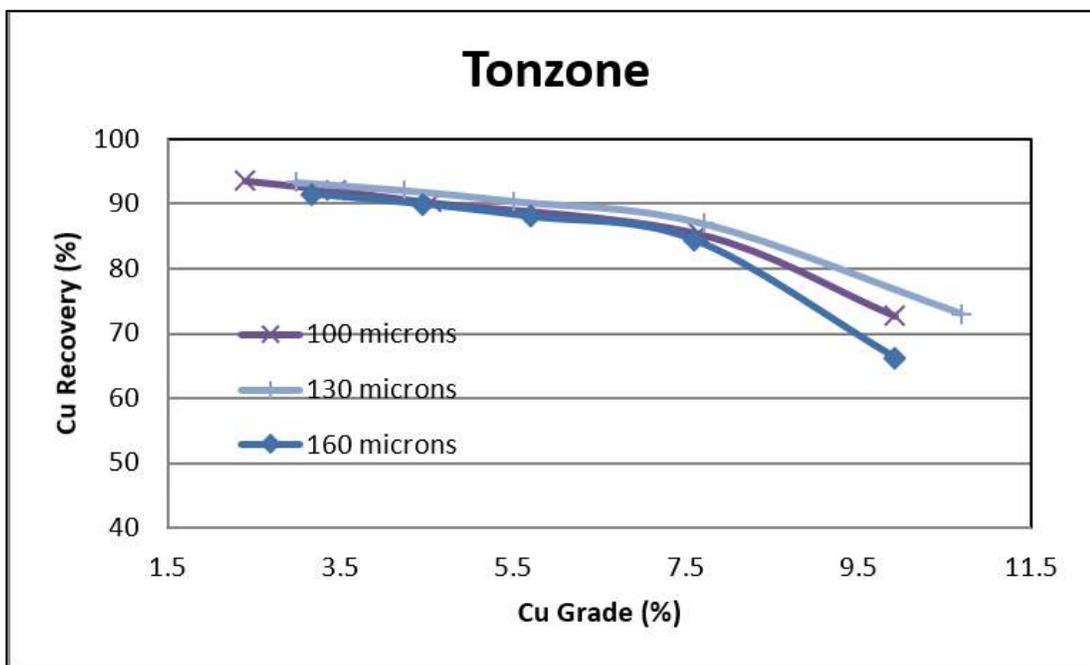


Figure 13.5: Tonzone - primary grind size effect on Cu recovery vs grade

Based on these results, a primary grind size P_{80} of 130 μm was selected for the PFS design, as there was negligible improvement in flotation results at finer feed sizes.

Similarly, the requirement for a concentrate regrind between the rougher and cleaner flotation stages, as well as the influence of target regrind particle size distribution on flotation results including grade and recovery of copper concentrates, were also evaluated as part of the SGS Phase II testwork. The results for the individual geometallurgical domain composite samples are illustrated in Figure 13.6 to Figure 13.9.

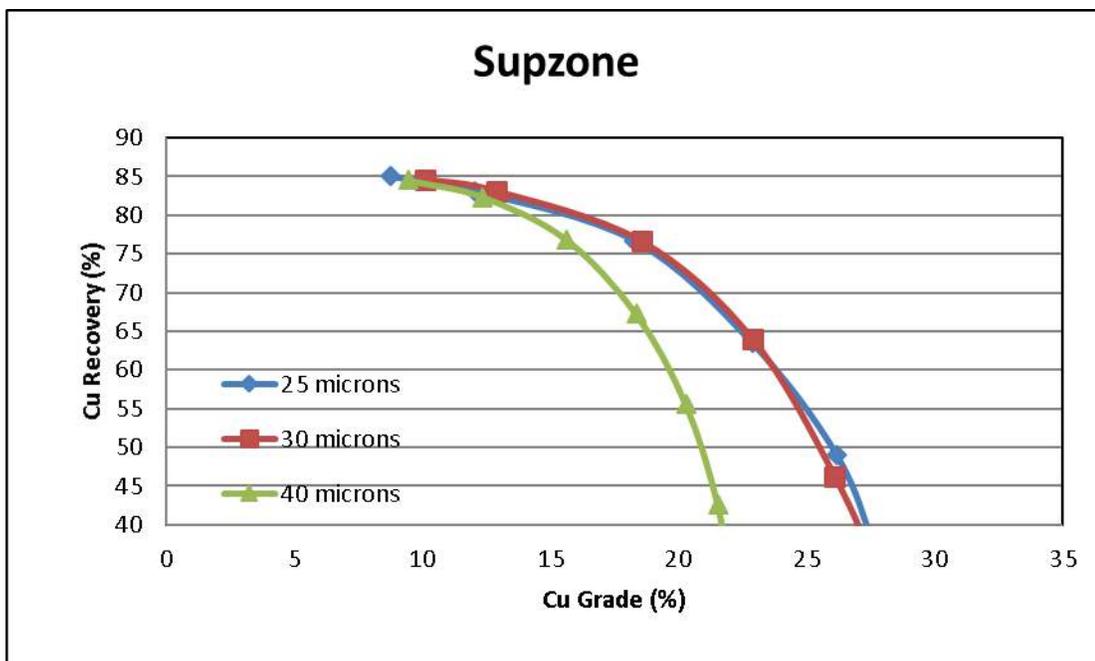


Figure 13.6: Supzone - regrind size effect on Cu recovery vs grade

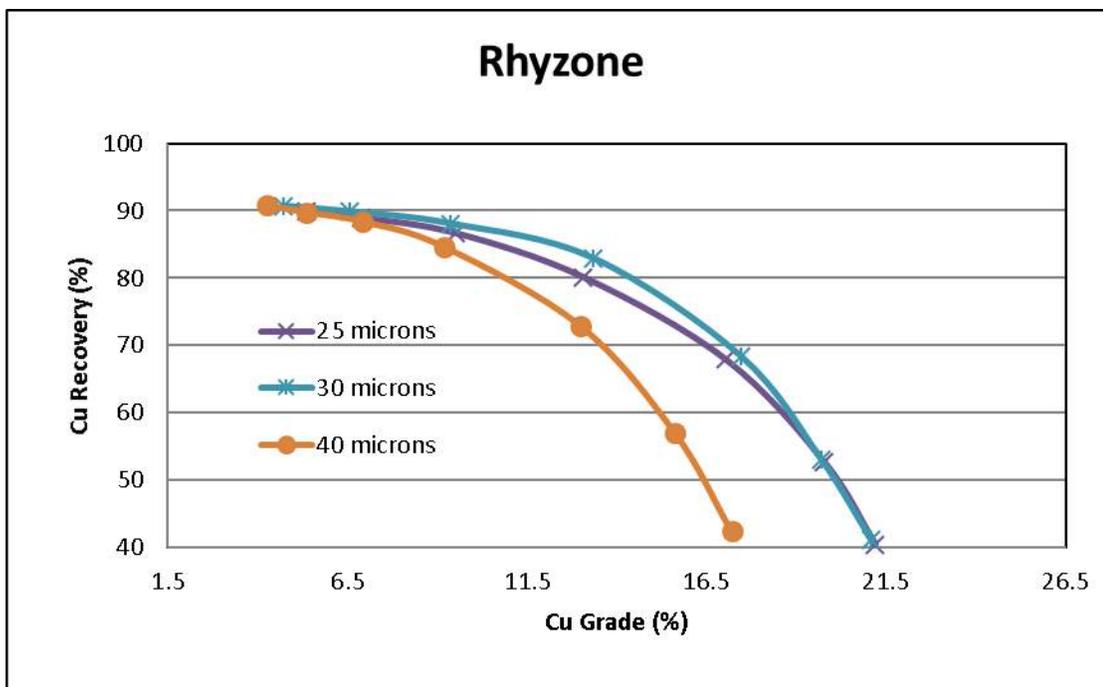


Figure 13.7: Rhyzone - regrind size effect on Cu recovery vs grade

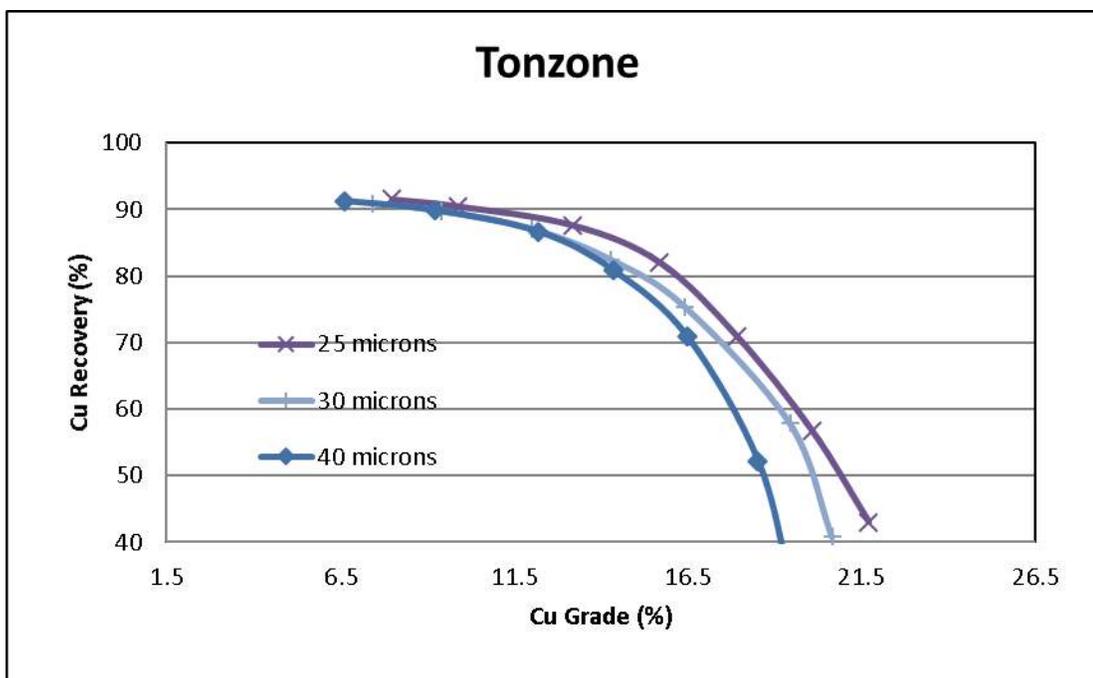


Figure 13.8: Tonzone - regrind size effect on Cu recovery vs grade

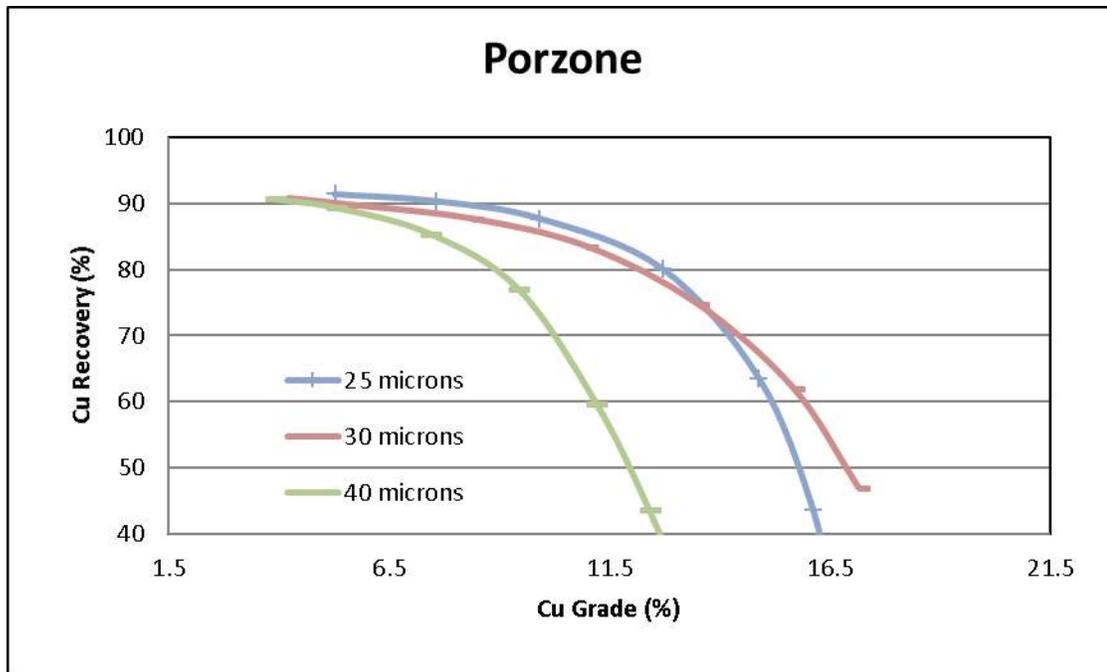


Figure 13.9: Porzone - regrind size effect on Cu recovery vs grade

Based on the results of the SGS Phase II testwork, a target regrind particle size P_{80} of 25 μm was chosen for the PFS as this size distribution showed preferred cleaner flotation results for copper recovery and grade.

Further evaluation and confirmation of the target particle size distributions for both the primary grind and regrind should be completed in support of the feasibility study design.

13.9 Analysis of SGS Phase II LCT Flotation Results

The SGS Phase II LCT data as summarized below in Table 13.15 indicated relatively poor metal accountabilities for all tests, indicating that metal was being retained in the middlings streams, and potentially affecting the reported metal recoveries and/or concentrate grades.

Table 13.15: Master composite LCT mass and metal conservation balance for last three cycles

Geometallurgical Domain	Accountability (%)			
	Mass	Cu	Fe	S
Supergene	99	88	88	74
Rhyolite	99	97	100	99
Tonalite	99	79	96	91
Porphyry	99	97	93	93

To accommodate the influence of poor material balances in the flotation testing, an adjustment was made to the calculated recoveries, as summarized in Table 13.16.

Table 13.16: Adjusted LCT results for Phase II composite samples

Geometallurgical Domain	Cu Grade in Con (%)	Cu Recovery (as tested) (%)	Cu Recovery (adjusted) (%)
Supergene	29.9	74.4	84.4
Rhyolite	28.9	86.4	88.8
Tonalite	24.3	72.3	91.4
Porphyry	20.3	86.6	89.5

Results indicate that SGS Phase II LCTs did not complete enough cycles to reach steady state and generally accumulated copper and sulphur in the middlings streams, usually in the cleaner scavenger concentrate. Additional testing should be considered to validate LCT results from Phase II and provide confidence in design assumptions.

13.10 ALS Testwork (2018)

Additional testing was completed at ALS, Kamloops, Canada (ALS, 2018), to verify SGS Phase II test results and identify opportunities for improvements.

The ALS testwork samples were selected from the 2012 drilling program cores and as such the age and extent of oxidation of the core samples used to prepare the geometallurgical composites could have affected the flotation results. Testwork undertaken in support of a future feasibility study should use freshly recovered drill core for which the sample history is well known and for which potential oxidation is limited.

Based on a cut-off grade of 0.3% Cu, head grade variability samples were identified as low (below 0.3%), medium (between 0.3% and 0.4%) and high grade (above 0.4%), with the exception of Supzone, and 20 m intervals of corresponding drill core samples, averaging approximately 40 kg for each sample, were collected and shipped to ALS, Kamloops, Canada. Low grade, medium grade, and high-grade samples from the Supzone lithology were 0.59, 0.60, and 0.78%, respectively. Three grade variability samples (high, medium and low) for each lithological domain (Rhyzone, Supzone, Tonzone and Porzone) totalling 12 variability samples were tested as part of the ALS (2018) testwork.

The ALS test program included:

- QEMSCAN on geometallurgical zone composite samples
- Kinetic flotation tests on zone composite samples to determine optimum flotation conditions
- OCT (including cleaning) tests on geometallurgical zone grade variability samples
- LCT on each of the four zone composite samples

For the ALS testing, target particle size distributions of P_{80} of 130 μm in the primary grind and P_{80} of 25 μm in the regrind were chosen, to be consistent with the conditions identified in the SGS Phase II test program.

The ALS flotation test program was focused only on copper metallurgy. Gold and silver metallurgy improvements should be further evaluated as part of future testwork.

13.10.1 Feed Assay and Mineralogical Results

Each zone variability and composite sample was analysed for copper, copper oxide, iron, sulphur, gold and the results are summarized in Table 13.17. Copper oxides analysis was determined by applying weak acid to samples to dissolve any acid soluble copper in the samples. Copper oxides, as determined, may indicate the presence of copper sulphate forms in Table 13.18 or the QEMSCAN is mis-reporting shallow sulphide borders as sulphates and the copper oxide is acid soluble digenite or covellite. QEMSCAN BMA results (Table 13.19) indicate that there are no oxide copper mineral species present in the four lithological zones.

Table 13.17: Feed analysis for ALS 2018 variability and composite samples

Geometallurgical Domain	Assay (%)						S:Cu
	Cu	CuOx	CuO in Cu	Fe	S	Au (g/t)	ratio
Rhyzone-Low	0.22	0.008	3.6	2.25	2.98	0.16	13.5:1
Rhyzone-Med	0.32	0.022	6.9	2.27	2.45	0.20	7.7:1
Rhyzone-High	0.43	0.027	6.3	4.30	0.61	0.35	1.4:1
Rhyzone Average	0.32	0.019	5.9	2.94	2.01	0.24	6.2:1
Composite Rhyzone	0.30	0.017	5.7	2.95	2.08	0.24	6.9:1
Tonzone-Low	0.23	0.008	3.5	4.70	1.21	0.17	5.3:1
Tonzone-Med	0.28	0.004	1.4	3.55	0.38	0.24	1.4:1
Tonzone-Hight	0.45	0.013	2.9	3.85	0.80	0.32	1.8:1
Tonzone Average	0.32	0.008	2.6	4.03	0.80	0.24	2.5:1
Composite Tonzone	0.31	0.006	1.9	3.71	0.82	0.23	2.6:1
Porzone-Low	0.26	0.007	2.7	2.66	1.77	0.17	6.8:1
Porzone-Med	0.38	0.013	3.4	3.41	2.37	0.24	6.2:1
Porzone-High	0.39	0.013	3.3	2.58	1.72	0.36	4.4:1
Porzone Average	0.34	0.011	3.2	2.88	1.95	0.26	5.7:1
Composite Porzone	0.35	0.012	3.4	2.85	1.88	0.33	5.4:1
Supzone-Low	0.54	0.025	4.6	5.10	0.70	0.54	1.3:1
Supzone-Med	0.55	0.069	12.5	2.40	0.34	0.22	0.6:1
Supzone-High	0.69	0.086	12.5	2.72	0.58	0.31	0.8:1
Supzone Average	0.59	0.060	10.1	3.41	0.54	0.36	0.9:1
Composite Supzone	0.58	0.058	10.0	2.89	0.52	0.33	0.9:1

Note: the individual domain composite samples were generated by combining 10 kg of each corresponding grade variability samples.

Table 13.18: Mineralogical analysis for ALS composite samples

Mineral	Porzone	Rhyzone	Supzone	Tonzone
Copper sulphide (%)	16.5	9.50	76.4	61.1
Pyrite (%)	74.9	87.8	19.5	34.9
Other sulphide minerals (%)	0.7	0.4	3.2	0.1
Sulphate minerals (%)	8.0	2.3	0.9	3.9
Total (%)	100.0	100.0	100.0	100.0

Table 13.19: Copper mineralogy for ALS composite samples

Mineral	Porzone	Rhyzone	Supzone	Tonzone
Chalcopyrite (%)	89.4	58.5	35.9	83.2
Bornite (%)	2.4	4.0	4.9	16.3
Chalcocite/Digenite (%)	0.5	0.8	18.6	0.2
Covellite (%)	6.9	36	40.3	0.3
Tennantite/Enargite (%)	0.7	0.7	0.3	<0.1
Total (%)	100.0	100.0	100.0	100.0

The Rhyolite zone had a relatively high 5.7% copper oxide component, the highest pyrite to copper sulphide ratio of 9.2:1, and 36% of copper mineralization as covellite. The mildly soluble copper species present in the Rhyolite zone will tend to activate the pyrite for flotation and may result in slower flotation kinetics and generally poorer selectivities.

The Tonalite zones appeared to have the least complicated metallurgy of the four zones tested given that only 1.9% of the total copper was present as copper oxide, it had a low pyrite to copper sulphide ratio and there was relatively little secondary copper mineralization.

The Supergene zone included 10% of the copper as copper oxides and the lowest pyrite to copper sulphide ratio of all four lithological samples. The composition of copper mineral species was 35.9% chalcopyrite, 18.6% chalcocite and 40.3% covellite (Table 13.19), reflecting the supergene nature of the ore. Higher copper concentrate grade is expected with the high proportion of chalcocite and covellite.

In general, samples with high oxidized copper, or secondary copper mineralization will respond less favourably to the flotation process. Similarly, samples with higher pyrite:copper sulphide ratios will be harder to selectively float and clean, resulting in poorer quality concentrates.

13.10.2 Analysis of ALS 2018 LCT Flotation Results

In general, results from the ALS 2018 testwork were consistent with the mineralogy, though as noted previously, generated poorer flotation results than anticipated, likely attributable to the age and extent of oxidation of the core samples used in preparing the metallurgical samples.

The LCT flotation tests completed as part of the ALS 2018 testwork achieved steady-state for at least the final two cycles in each test, and generally showed good material accountabilities. The ALS 2018 testwork considered a different reagent scheme than had been considered previously without any marked adverse effect on overall recoveries.

As was the case with previous flotation testwork, the Rhyolite zone materials proved to have the poorest response to the flotation conditions selected. Flotation results from tests completed on the Rhyzone sample showed significant copper reporting to the tails, in agreement with previous

results, and pyrite suppression was less successful with an appreciable amount reporting to the concentrate. A summary of LCT results, based on an average of results for the final two cycles, for the Rhyzone samples is provided in Table 13.20.

Table 13.20: Rhyzone composite sample LCT results

Flotation Stream	Analysis (%)				Distribution (%)				
Product	Cu	Au (g/t)	Fe	S	Mass	Cu	Au	Fe	S
Calculated Feed	0.32	0.24	3.1	2.24	100	100	100	100	100
Cleaner Concentrate (3 rd Stage)	27.6	14.2	25.9	39.6	1.0	82.7	55.8	8.0	17.0
Tailings	0.06	0.11	2.9	1.88	99.0	17.3	44.2	92.0	83.0

Testing with the Rhyolite materials generated concentrate with a 27.6% copper grade at a relatively low copper recovery of 82.7%. Copper (and gold) recoveries could be increased by limiting pyrite suppression, though likely at the expense of concentrate grade. Future phases of testwork could be expected to highlight preferred conditions for optimized grade and recovery.

The Tonzone sample responded most favourably to the flotation conditions, as was observed in previous testing. Final concentrate recovery and grade of 91.4% and 30.8%, respectively, were reported. A summary of the LCT results, based on an average of results for the final two cycles, for the Tonzone samples is provided in Table 13.21. Further refinement of flotation conditions could likely result in higher recoveries while maintaining a marketable concentrate grade and should be included in future testing.

Table 13.21: Tonzone composite sample LCT results

Flotation Stream	Assay (%)				Distribution (%)				
Product	Cu	Au (g/t)	Fe	S	Mass	Cu	Au	Fe	S
Rougher Feed	0.34	0.26	4.00	0.63	100.0	100.0	100.0	100.0	100.0
3 rd Cleaner Concentrate	30.80	17.50	26.40	32.20	1.0	91.4	67.5	6.7	52.4
Final Tail	0.03	0.09	3.80	0.30	99.0	8.6	32.5	93.3	47.6

Results of LCT flotation testing on the Porzone sample showed 87.8% copper recovery at 25.2% copper grade, as summarized for the final two cycles in Table 13.22. Results indicated that there were no significant losses of copper and good rejection of pyrite throughout.

Table 13.22: Porzone composite sample LCT results

Flotation Stream	Assay (%)				Distribution (%)				
Product	Cu	Au (g/t)	Fe	S	Mass	Cu	Au	Fe	S
Rougher Feed	0.35	0.29	3.10	1.84	100.0	100.0	100.0	100.0	100.0
3 rd Cleaner Concentrate	25.20	14.70	29.90	38.30	1.2	87.8	62.9	11.9	25.4
Final Tail	0.04	0.11	2.80	1.39	98.8	12.2	37.1	88.1	74.6

LCT testing of the Supzone samples showed significant losses of copper to the tailings, consistent with the abundance of fine secondary sulphide minerals in that material. Pyrite rejection in the cleaning stages was without difficulty. The final recovery and grade were 83.0% and 35.8%, respectively, as summarized for the final two cycles in Table 13.23. Testing with the Supzone sample indicated that there is very little opportunity for adjustments in flotation conditions to significantly improve results.

Table 13.23: Supzone composite sample LCT results

Flotation Stream	Assay (%)				Distribution (%)				
Product	Cu	Au (g/t)	Fe	S	Mass	Cu	Au	Fe	S
Rougher Feed	0.60	0.31	3.20	0.52	100.0	100.0	100.0	100.0	100.0
2 nd Cleaner Concentrate	35.80	15.30	17.90	27.30	1.4	83	69.5	7.8	73.4
Final Tail	0.10	0.09	3.00	0.14	98.6	17	30.5	92.2	26.6

13.10.3 Head grade vs recovery correlation

There was insufficient data to confirm the correlation between head grade and copper recovery based on the ALS testwork.

13.11 Recovery Estimates

The PFS recovery model was developed on the basis of LCT testing of the four geometallurgical zone composites, as completed in the SGS Phase II testwork without adjustment for head grade. Average results from the last three cycles were used, with some adjustment to accommodate the gold accountabilities in the testwork. A summary of the metal recoveries and corresponding concentrate grades per the model is provided in Table 13.24.

Table 13.24: Summary of metallurgical performance by lithology

Zone	Ore Mass Distribution (%)	Recovery to Concentrate (%)			Concentrate Grade		
		Cu	Au	Ag	Cu (%)	Au (g/t)	Ag (g/t)
Supergene	6.6	85.3	72.2	78.3	30.0	13.7	52.4
Rhyolite	29.4	89.0	64.1	68.1	28.9	12.9	49.0
Tonalite	47.9	90.6	77.0	51.1	24.3	18.0	48.7
Porphyry	16.1	90.0	62.6	60.3	20.3	13.7	54.8
Weighted average	100.0	89.7	70.6	59.4	25.4	15.5	50.0

At ore feed grades below 0.31% copper, the following model was used to estimate recoveries:

- Rhyolite: $\text{Cu Recovery (\%)} = 78.571 \times (\text{Cu head grade \%}) + 63.667$
- Tonalite: $\text{Cu Recovery (\%)} = 80.435 \times (\text{Cu head grade \%}) + 64.809$
- Porphyry: $\text{Cu Recovery (\%)} = 74.229 \times (\text{Cu head grade \%}) + 67.731$
- Supergene: 85.3% of feed content – Fixed

The algorithms were bounded by the lower 10th percentile (0.22% Cu) and upper 90th percentile (0.31% Cu) with respect to the Cu feed grade.

13.11.1 PFS Recovery Model Development

It is reasonable to assume that future flotation tests could improve recovery with some grade adjustments in the next phase of testwork.

Gold and silver recovery prediction has a greater level of uncertainty than copper, given the relatively large degree of variability in the test results.

Table 13.25 summarizes the annual weighted average copper, gold, and silver recoveries estimated for the life of the mine.

Table 13.25: Annual estimated copper, gold and silver recoveries

Operating Year	Cu Grade in Ore (%)	Cu Recovery (%)	Au Recovery (%)	Ag Recovery (%)
1	0.37	87.9	71.6	62.0
2	0.39	88.8	72.4	58.9
3	0.36	88.1	71.3	60.7
4	0.35	86.7	70.5	63.6
5	0.28	86.6	69.7	58.7
6	0.25	84.6	69.7	58.5
7	0.26	86.0	70.7	57.4
8	0.30	87.0	71.3	58.4
9	0.29	85.4	70.6	60.9
10	0.28	85.1	70.6	60.1
11	0.29	86.0	71.1	59.1
12	0.29	86.0	70.2	59.6
13	0.28	85.8	70.4	58.7
14	0.27	85.7	70.6	58.4
15	0.25	84.7	70.8	57.4
16	0.26	84.3	71.0	57.8
17	0.30	85.4	70.7	60.0
18	0.30	88.6	72.1	55.8
19	0.28	88.4	71.3	54.7
20	0.23	85.0	63.2	59.9
Overall	0.29	86.4	70.9	59.0

13.12 Metallurgical Variability

The metallurgical testwork to date is based on samples that adequately represent the variability for this stage of study. However, additional variability testwork will be required to support more detailed studies and this work will be undertaken as part of a feasibility study program. Physical characterization was conducted on variability samples along with some preliminary flotation testwork. The results indicate that additional tests will be required to confirm the physical characteristics of the different lithological zones and to confirm the copper recovery algorithms that were developed and possibly aid in the development of gold and silver algorithms.

13.13 Deleterious Elements

No major issues with deleterious elements were noted from the testwork that has been completed. However, there are some zones within the Josemaría deposit that have elevated arsenic levels. Mill feed blending strategies should be employed to generate flotation concentrates that have high copper grades, whilst minimizing arsenic and other deleterious element contributions to levels below penalty impositions in the final concentrate.

14 Mineral Resource Estimate

14.1 Summary

The mineral resource estimate discussed in this section for Josemaría is unchanged from that described in the Project Constellation technical report from 2016 (Ovalle et.al., 2016).

Mr. Gino Zandonai prepared the updated Josemaría estimate. The mineral resource estimate is based on 116 drill holes totalling 52,725 m of drilling, of which 34 holes (13,164 m) are RC and 82 holes (39,561 m) are core holes. Some early exploration holes which were drilled on the property but distal to the deposit were not included in the resource estimate. The total length of assayed intervals is 51,092 m and there are 27,344 assays.

14.2 Geological Models

A two-dimensional (2D) interpretation based on the logged data was completed by NGEx geologists on east–west oriented sections spaced 100 m apart. Two-dimensional lines were then exported from GEMS and imported into Leapfrog geological modelling software and the final three-dimensional (3D) wireframe solids were constructed.

Three separate geological models were constructed to guide resource estimation: lithology; alteration and mineral zones. These models were used with the assay data to develop an understanding of the main controls on mineralization, to provide input into the block model and to control the interpolation. The primary controls on mineralization at Josemaría are a combination of the mineral zones and lithologies, and the mineral zone wireframes were the primary controls for the interpolation, using hard boundaries between different zones. Three lithology solids (PMV, HBX and PORL) were also used in the interpolation.

14.3 Exploratory Data Analysis

Statistical analyses were performed for Cu, Au, Ag, S, Fe and As by lithological domain. Reviews included the number of samples, total length, minimum, maximum mean value, standard deviation, and co-efficient of variation (CV).

Copper and gold assay grades exhibit a moderate correlation. The coefficient of correlation between copper and gold obtained from the least square linear regression is around 0.636, while the coefficient of correlation between gold and silver is 0.475.

Hard boundaries were used between the different units during estimation.

14.4 Density Assignment

SG values used in estimation are outlined in Section 11. Average specific gravities for each mineral zone were calculated as summarized in Table 14.1.

Table 14.1: Density data, Josemaría

Rock Type / Mineral Zone	SG
Post Mineral Volcanics (PMV)	2.58
Hydrothermal Breccia (HBX)	2.69
Late Porphyry (PORL)	2.65
Pyrite – Chalcocite (hypogene) (PyCC(h))	2.70
Pyrite - Chalcopyrite	2.65
Oxide (OX)	2.58
Supergene (PyCC(s))	2.72

14.5 Grade Capping/Outlier Restrictions

No capping was applied.

14.6 Composites

The drill hole assays were composited to 2 m to maintain the majority sampling interval (86% of assayed intervals at 2 m) and to avoid spreading composites across geological domains in case of bigger composite size. Geological codes for lithology, mineral zone and alteration were assigned to each composite. There is a total of 23,622 composites in the database, all of which have assay values assigned.

14.7 Variography

Experimental variogram analysis for Cu, Au, Ag, As, Fe and S was performed using the composites based on the mineral zone and post-mineral volcanic unit.

The experimental variography was performed using the Super VISOR 3-D variogram modeller software (Supervisor).

Directional variograms were explored within each domain. The best geospatial correlation of samples is described by omnidirectional variograms over any specific preferential orientation, demonstrating the widespread disseminated nature of the mineralization in the deposit.

Omnidirectional experimental variograms were fitted with single and multiple nested spherical models where applicable, these variograms describe the spatial continuity, in all directions, of the mineralization in the deposit. Down-hole variograms at smaller lags were used to determine the nugget effect which was then manually entered when calculating a 3D auto-fit model based on the omnidirectional variograms.

14.8 Estimation/Interpolation Methods

A 3D block model of the deposit was built with 25 x 25 x 15 m blocks for mineral resource estimation purposes. The block model covered an area of 1.5 x 2.1 km on plan and had a 1.5-km vertical extent.

The interpolation plan and the search distances for ordinary kriging (OK) and inverse distance weighting to the second power (ID2) methods were based on the geostatistical analysis and variogram parameters. According to this plan, total Cu, Au, Ag, As, S and Fe values were interpolated within the mineral zones in the model. All elements were interpolated using OK. The ID2 weighting method and nearest-neighbour (NN) method was performed only for copper and gold for validation and checking purposes of the global bias.

OK and ID2 interpolation was done in a single pass. A minimum of two and a maximum of 50 composites, with maximum 15 composites from the same hole, were used for the interpolation to allow maximum spread of the data used to estimate blocks. For estimation of the kriging and block variance, a 3 x 3 x 3 discretization of the block was selected. The major, semi-major and minor axis of the search ellipse was set to the corresponding radius defined in by the omnidirectional variograms.

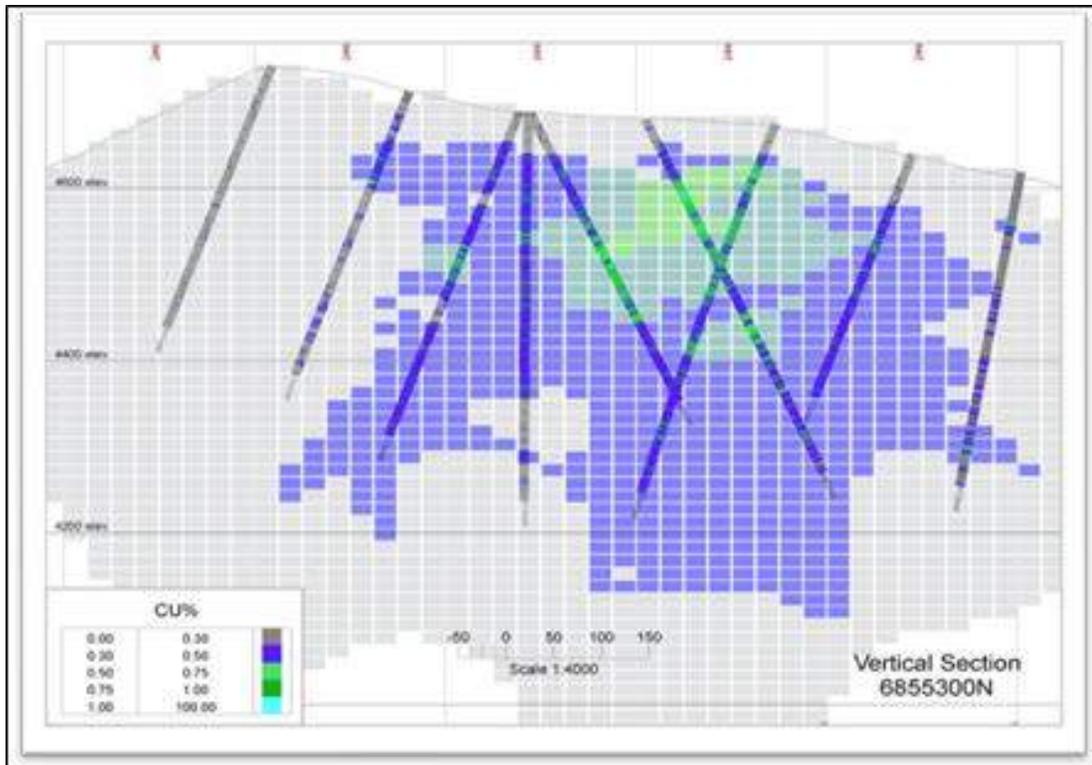
14.9 Block Model Validation

Validation steps included:

- Statistic validation comparing average grade of blocks and composites
- Swath plots comparing kriged block with NN and ID2 models
- Visual validation in plans and sections (Figure 14.1 and Figure 14.2)

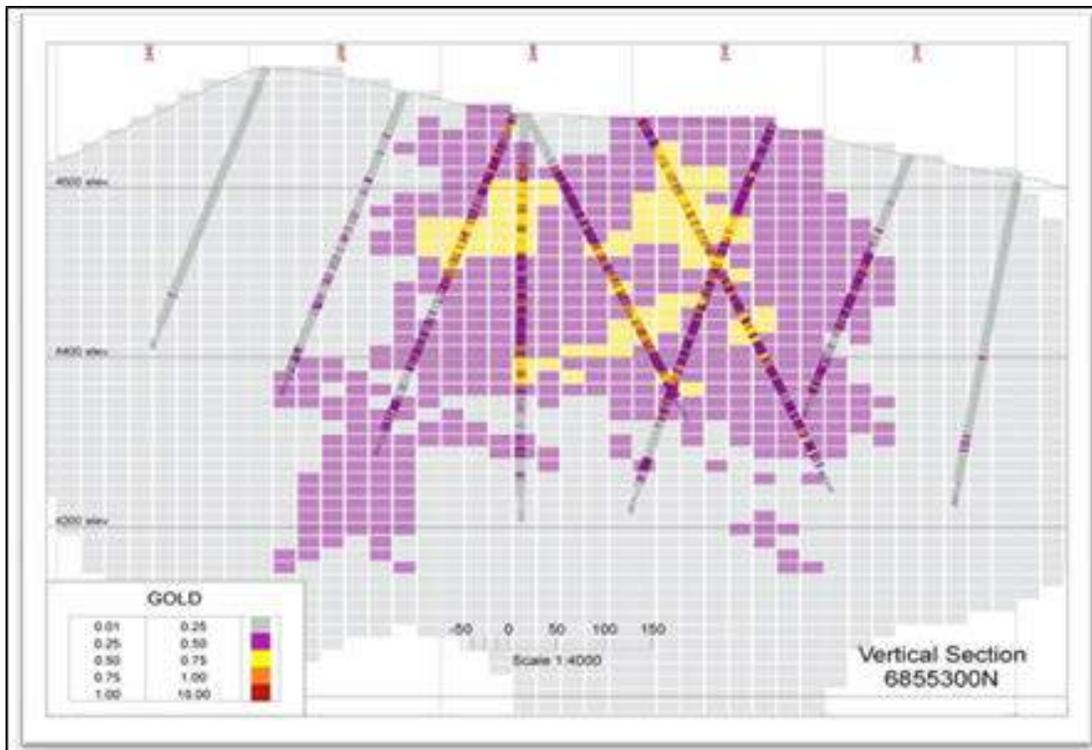
The estimated block grades show good correlation with adjacent composite grades; however, due to the large volume of some of the mineralogical domains, there is trend to smooth the kriged block mean when compared to the composites mean.

Swath plot results show a good comparison between the ID2 and OK estimate and, as expected, OK has smoothed the data against the NN estimates.



Source: NGEx, 2015

Figure 14.1: Copper blocks with drill holes (Section 6855300N)



Source: NGEx, 2015

Figure 14.2: Gold blocks with drill holes (Section 6855300N)

14.10 Classification of Mineral Resources

Mineral resource classification uses the 2014 CIM Definition Standards. Classifications were based on a two-step process, as follows:

- Indicated: the distance to the nearest drill hole from the centre of the block was less than or equal to 75 m and there were at least three drill holes used for the grade interpolation and the kriging efficiency estimation was more than 0.5
- Inferred: the distance to the nearest drill hole from the block was 75 to 150 m and there were at least two drill holes used for the grade interpolation and the kriging efficiency estimation was less than 0.5

Two smoothed buffer wireframes were created in Leapfrog as the final step; one at 75 m and one at 150 m. Inferred blocks inside the 75-m wireframe were re-classified as Indicated, while any Indicated blocks outside of the 75-m buffer, but within the 150-m buffer, were re-classified as Inferred. A final phase of visual inspection of the resulting classification was performed for validation purposes.

14.11 Reasonable Prospects of Economic Extraction

In order to evaluate for reasonable prospects of eventual economic extraction, a Whittle™ pit shell was generated using the following parameters:

- Copper price: US\$3.00/lb
- Mining cost: US\$2.20/t
- Process cost (including general and administrative (G&A) costs): US\$7.40/t processed
- Copper selling cost: US\$0.35/lb
- Over-all pit slope angle: 42°

The analysis was done based on the copper equivalent (CuEq) grades in the block model. CuEq was calculated using US\$3.00/lb copper, US\$1,400/oz gold and US\$23/oz Ag and was based on copper, gold, and silver recoveries obtained in metallurgical testwork on four composite samples representing the rhyolite, tonalite, porphyry and supergene zones. Copper recoveries for the rhyolite, tonalite and porphyry zones were calculated as a function of copper grade, ranging from a low of 81% to a high of 97%. Copper recovery in the supergene zone was fixed at 85%. Gold recoveries were fixed between 62% and 73% and silver recoveries were fixed between 53% and 75% depending on the zone.

14.12 Mineral Resource Estimate

The mineral resource estimate assuming open pit mining methods is reported using the 2014 CIM Definition Standards. Indicated and inferred classifications only have been estimated; no measured mineral resources were classified.

The mineral resource estimate was prepared by Mr. Gino Zandonai, RM CMC.

Mineral resources are summarized in Table 14.2 and Table 14.3, and 0.2 g/t Au for oxide was chosen based on comparison with other similar, nearby deposits. These projects provide useful benchmarks and have been used to select the base-case cut-off grade for oxide Josemaría mineral resources.

Table 14.2: Mineral resource statement (sulphide material) Josemaría project, San Juan, Argentina, 7 August 2015 (assuming open pit mining methods)

Cut-off (CuEq %)	Quantity	Grade				Contained Metal		
	(million tonnes)	Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)	Cu (billion lbs)	Au (million ozs)	Ag (million ozs)
Indicated Mineral Resources								
0.2	1,066	0.31	0.22	1.0	0.44	7.4	7.4	34.5
Inferred Mineral Resources								
0.2	404	0.24	0.15	0.8	0.33	2.0	2.0	10.8

Table 14.3: Mineral resource estimate (oxide material) for Josemaría project, San Juan, Argentina, 7 August 2015 (assuming open pit mining methods)

Cut-off	Quantity	Grade			Contained Metal	
(Au g/t)	(million tonnes)	Au (g/t)	Ag (g/t)	Cu (%)	Au (thousand ozs)	Ag (thousand ozs)
Josemaría Indicated Mineral Resources						
0.2	43	0.32	1.2	0.15	450	1,610
Josemaría Inferred Mineral Resources						
0.2	4	0.34	1.0	0	48	145

Notes to accompany Josemaría mineral resource tables:

1. Mineral resources have an effective date of 7 August 2015. The Qualified Person for the estimate is Mr. Gino Zandonai, RM CMC.
2. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
3. Sulphide mineral resources are reported using a copper equivalent (CuEq) cut-off grade. CuEq was calculated using US\$3.00/lb copper, US\$ 1,400/oz gold and US\$23/oz Ag and was based on copper, gold and silver recoveries obtained in metallurgical testwork on four composite samples representing the rhyolite, tonalite, porphyry and supergene zones. Copper recoveries for the rhyolite, tonalite and porphyry zones were calculated as a function of copper grade, ranging from a low of 81% to a high of 97%. Copper recovery in the supergene zone was fixed at 85%. Gold recoveries were fixed between 62% and 73% and silver recoveries were fixed between 53% and 75% depending on the zone.
4. Mineral resources are reported within a conceptual Whittle™ pit that uses the following input parameters: Cu price: US\$3.00/lb, mining cost: US\$2.20/t, process cost (including G&A): US\$7.40/t processed, copper selling cost: US\$0.35/lb and over-all slope angle of 42°.
5. Mineral resources (sulphide) have a base case estimate using a 0.2% CuEq grade; mineral resources (oxide) are reported using a 0.2 g/t Au cut-off grade.
6. Totals may not sum due to rounding as required by reporting guidelines.

14.13 Grade Sensitivity Analysis

The mineral resources of the Josemaría project are sensitive to the selection of the reporting cut-off grade. To illustrate this sensitivity, the block model quantities and grade estimates within the conceptual pit used to constrain the mineral resources are presented in Table 14.4 and Table 14.5 at different cut-off grades for copper and gold, respectively. The reader is cautioned that the figures presented in this table should not be misconstrued with a mineral resource statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade.

Table 14.4: Global block model quantities and grade estimates*, Josemaría project at various cut-off grades

Cut-off (CuEq %)	Quantity (million tonnes)	Grade				Contained Metal		
		Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)	Cu	Au	Ag
						(billion lbs)	(million ozs)	(million ozs)
Indicated Mineral Resources (sulphide)								
0.6	148	0.56	0.38	1.5	0.76	1.8	1.8	6.9
0.5	295	0.47	0.34	1.3	0.65	3	3.2	12.6
0.4	559	0.4	0.29	1.2	0.55	4.9	5.2	21.8
0.3	835	0.35	0.25	1.1	0.49	6.5	6.6	29.7
0.2	1,066	0.31	0.22	1.0	0.44	7.4	7.4	34.5
Inferred Mineral Resources (sulphide)								
0.5	9	0.37	0.28	1.1	0.52	0.1	0.1	0.3
0.4	85	0.31	0.23	1.0	0.45	0.6	0.6	2.7
0.3	236	0.28	0.19	0.9	0.38	1.4	1.4	6.8
0.2	404	0.24	0.15	0.8	0.33	2.0	2.0	10.8

Table 14.5: Global block model quantities and grade estimates*, Josemaría project at various cut-off grades

Cut-off (Au g/t)	Quantity	Grade			Contained Metal	
	(million tonnes)	Au (g/t)	Ag (g/t)	Cu (%)	Au	Ag
					(thousand ozs)	(thousand ozs)
Josemaría Indicated Mineral Resources (oxide)						
0.4	10	0.46	1.4	0.18	150	460
0.3	23	0.4	1.3	0.16	290	950
0.2	43	0.32	1.2	0.15	450	1,610
0.1	77	0.25	1.0	0.13	610	2,520
Josemaría Inferred Mineral Resources (oxide)						
0.4	2	0.43	1.2	0	27	73
0.3	3	0.4	1.1	0	37	102
0.2	4	0.34	1.0	0	48	145
0.1	7	0.26	0.9	0.02	62	214

* The reader is cautioned that the figures in this table should not be misconstrued with a mineral resource statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade.

15 Mineral Reserve Estimates

15.1 Introduction

SRK was contracted by NGEx to conduct the mine engineering and mineral reserve estimation for the Josemaría project. SRK adopted standard mine planning processes to determine the mineral reserve estimate for this surface mineable project. The following describes those processes, their inputs and the mineral reserve outcome.

15.2 Key Assumptions, Parameters and Methods

15.2.1 Economic Limit Definition

To determine the economic limit of surface mining for Josemaría, and thus the basis for the mineral reserve, SRK utilized GEOVIA's Whittle™ software, which is based on the industry standard Lerchs Grossmann pit optimization algorithm. SRK collaborated with NGEx and other consulting team members to derive the key inputs for the pit optimization, including metal pricing, metal recoveries, and operating costs.

The remainder of this section describes the inputs to the pit optimization process.

15.2.2 Mine Design Model

The resource model upon which the pit optimization was based was that reported on previously in March 2016 (Ovalle et.al., 2016) and continues to be featured in this PFS technical report (see Section 14). In addition to assay grades and densities for mineralized zones, the block model included metallurgical recoveries from grade-recovery relationships (Section 15.3.5). SRK reviewed the resource model and the data on which it was based and deems it appropriate to support mineral reserve estimation.

SRK converted the resource model to a mine design model by including dilution and ore loss (Section 15.2.6), as well as estimates of net smelter return (Section 15.2.7).

15.2.3 Pricing and Off-Site Costs

SRK confirmed metal pricing assumptions with NGEx. The metal prices used in pit optimization are provided in Table 15.1.

Table 15.1: Metal price assumptions for pit optimization

Metal	Units	Value
Copper	US\$/lb	2.95
Gold	US\$/oz	1,225
Silver	US\$/oz	19.00

15.2.4 Geotechnical Slope Guidance

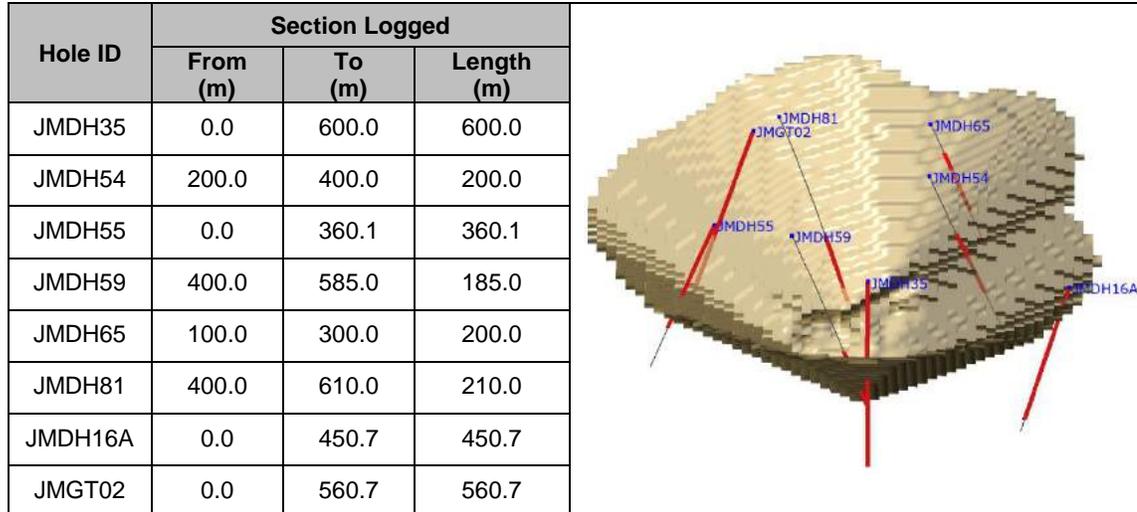
SRK conducted pit geotechnical studies for the PFS in two phases. Phase 1 was a gap analysis to evaluate the project geotechnical understanding and to inform the design of the Phase 2 (PFS) study. The latter is described herein.

Logging Program

SRK selected drillholes from previous drilling programs to undertake geotechnical logging. The selection was based on the findings of SRK’s gap analysis, to provide coverage of the previous pit shell ultimate pit walls (Ovalle, et al., 2016) and be representative of the range of lithology and alteration units. The program included seven resource drillholes and one geotechnical drillhole from the 2013-14 program.

The program was carried out between July 2 and July 13, 2018, at the NGEx core storage facility in San Juan, Argentina. Logging and testing was undertaken by two SRK geotechnical engineers and two NGEx geologists.

SRK prepared a site-specific core logging atlas and logging template for the project which was modified to account for the disturbed core condition. Logging was carried out on sections of the drillholes over the depths of most relevance to the ultimate pit shell geometry i.e. sections of drillholes near and behind the pit walls. Figure 15.1 shows the selected drillholes and details of the logged intervals. A total of 2,766 metres of core was logged.



Source: SRK, 2018

Figure 15.1: Field program drillholes and logged sections in red (view looking south)

Geotechnical logging focused on recording the parameters necessary for the classification of the rock mass according to the system by Bieniawski (1989) [RMR_B(89)]. Other geotechnically relevant parameters were recorded, such as intensity of alteration, intensity of microdefects and characteristics of the major structures.

Rock strength evaluation was conducted by empirical methods and point load testing on a basis of approximately one test per 3-m run, resulting in a total of 789 tests. Representative samples were taken for possible geotechnical laboratory testing. Test results were not available by the effective date of this study; therefore, strength conversion factors were developed using the laboratory results from the AMEC (2014) study.

A logging program was conducted on core photos from select historical resource drillholes with the objective of verifying the resource drilling RQD data set. SRK found that the values in the historical data set were generally too low. SRK attributed this to an incorrect method of accounting for mechanical drilling and handling-induced fractures. For SRK's rock mass classification in this study, the more accurate photo-logged values were used.

Design Domains

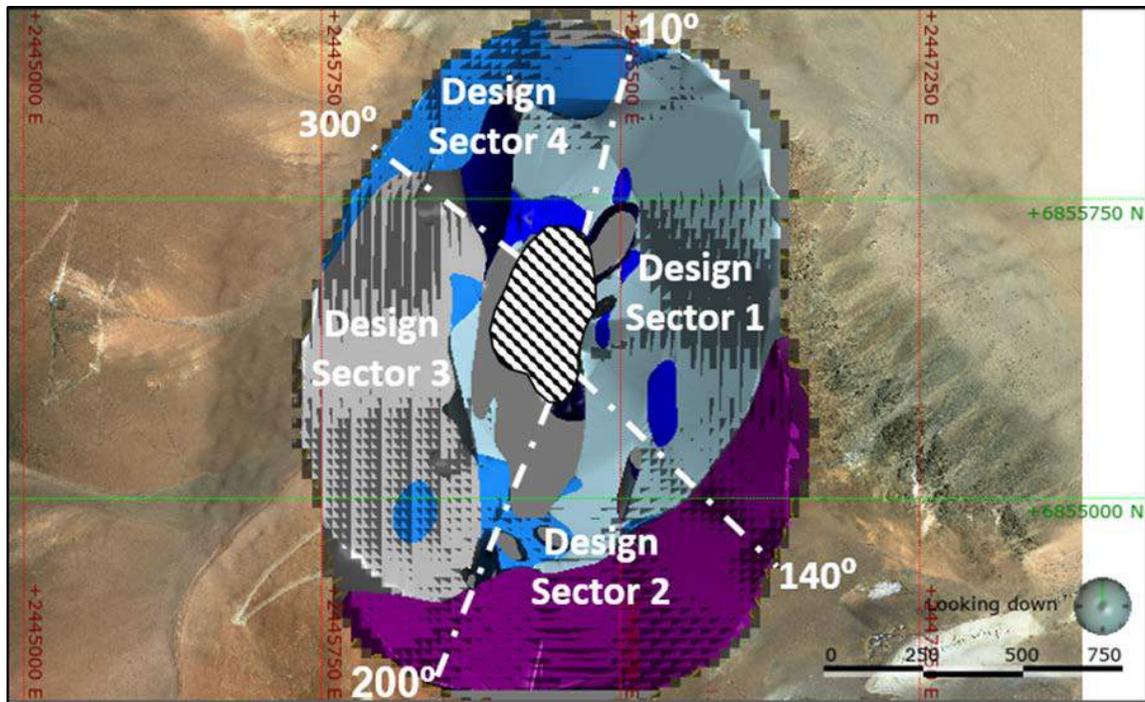
Preliminary geotechnical domains with similar properties were grouped into five design domains. Based on the findings of the field program, and with reference to the AMEC (2014) laboratory test results, using RocLab (RockScience Inc.), representative intact and rock mass properties for the five geotechnical domains were selected. Note that although fault zones were encountered in the field program and properties were defined, because their total length encountered/logged was not statistically significant and because the three-dimensional structural controls of the site are not well constrained, major structures were not able to be reliably included in this study.

Geotechnical Slope Design

Based on the spatial distribution of geotechnical domains in the pit shell and their geometry behind the pit walls, areas of the pit walls with similar domain composition were established as the design pit sectors (Figure 15.2). For each design sector, sections which were most representative of the sector domain geometry were selected for slope stability analysis.

Following the approach in Bieniawski (1989) adjustments were made to the rock mass rating, $RMR_B(89)$, values to arrive at modified rock mass ratings (MRMR) values which account for the effects of mining. In sectors that comprised multiple geotechnical domains, the domain that was most represented, or had the lower RMR value, was selected.

Using the calculated MRMR values and adopting an acceptability criterion (factor of safety) of $FoS=1.3$, design sectors were plotted on the industry-recognised empirical slope angle chart (Haines and Terbrugge, 1991) to select inter-ramp angles. Respective of the relatively sparse data set, project stage, and level of confidence associated with the data, an inter-ramp angle (IRA) acceptability criterion of $FoS=1.3$ was selected. Consistent with industry standard practice, SRK recommends a maximum inter-ramp slope height of 105 m, consisting of seven 15-m high benches.



Source: SRK, 2018

Figure 15.2: Pit shell cut-out showing geotechnical design domains and pit design sectors

Design IRAs were selected with guidance from the chart and based on the slope geometry by respecting the bench face angle (BFA) of 65° and bench width of 8 m (east wall) [Design Sector 1 and 2], and BFA of 70° and bench width of 8.5 m (west wall) [Design Sector 3 and 4] from AMEC (2014). With further consideration of haul roads (35 m wide) to determine the OSAs, the pit slope design criteria summarized in Table 15.2 were derived for pit optimization.

Table 15.2: Pit slope design criteria

Domain	Inter-Ramp Angle (°)	Stack Height (m)	Overall Slope Angle (°)
1	46	105	43
2	46	105	43
3	49	105	46
4	46	105	42

Open Pit Hydrogeology

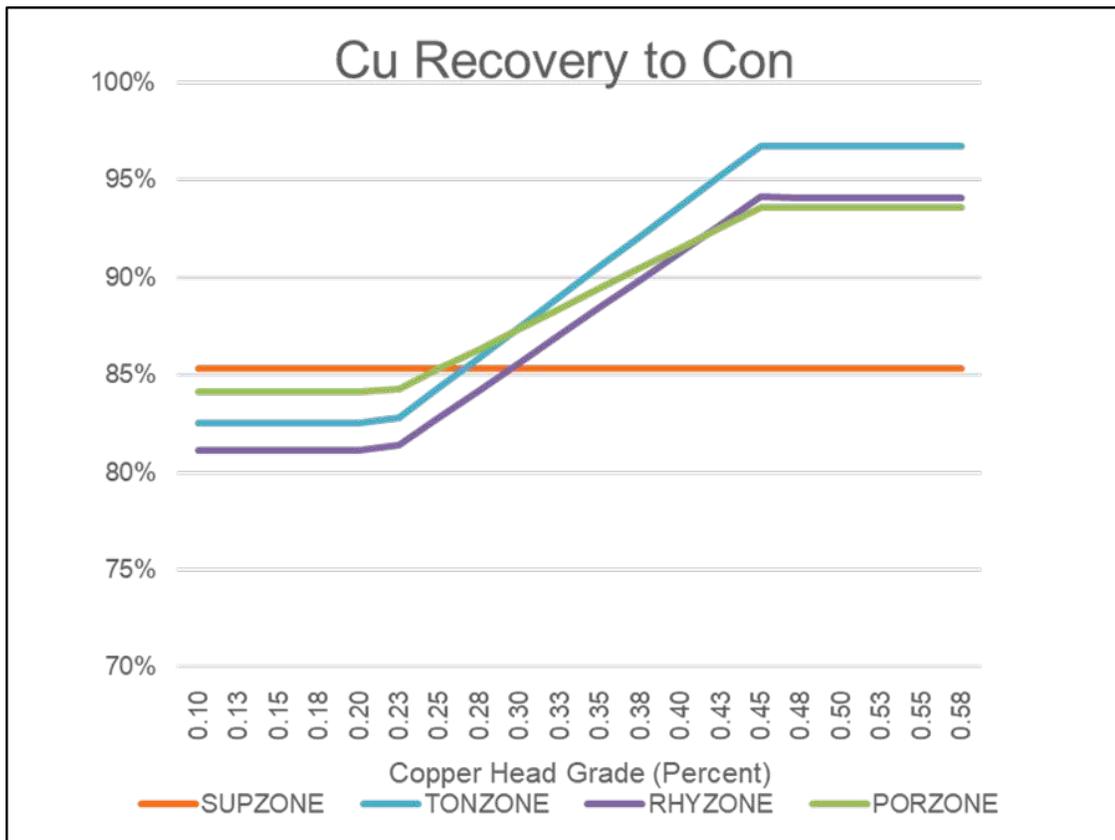
A conceptual hydrogeological model was developed by AMEC (2014), based on limited Lugeon tests and water probe measurements in selected drillholes as part of their investigations.

Key findings regarding the hydrogeological regime at the site in AMEC (2014) were as follows:

- Most of groundwater enters the system from infiltration from precipitation and snow melt, with lesser proportions from surface water infiltration from creeks and gullies
- The phreatic surface is a subdued replica of the surface topography with higher groundwater elevations in the higher topographic areas (to the south and west)
- The phreatic surface is anticipated to generally intersect the upper third of the preliminary ultimate pit, except for the western walls where it is likely to be higher
- There is a general trend of decreasing hydraulic conductivity with depth. The bedrock permeability is typically very low with very tight discontinuities, however localized fracture zones, likely associated with fault structures, have significantly higher permeabilities.

15.2.5 Metallurgical Recoveries Used for Mine Planning

The metallurgical recoveries for Josemaría are derived as a grade-recovery relationship for copper and as fixed values for gold and silver. The recoveries for copper as a function of grade and ore type are provided graphically in Figure 15.3. The fixed recoveries of gold and silver by ore type are provided in Table 15.3. At the time of mine planning, metallurgical testing was still in progress, and final metallurgical recoveries had not been determined. Following completion of the testwork, recovery assumptions changed (generally increasing), but it was determined this would not have a material impact on the outcome of the pit optimization process.



Source: SRK, 2018

Figure 15.3: Grade-recovery relationships by ore type for copper

Table 15.3: Gold and silver recoveries by ore type

Ore Type	Gold Recovery, %	Silver Recovery, %
Supergene	67.2	73.0
Tonalite	72.6	67.5
Rhyolite	61.7	74.9
Porphyry	59.2	52.9

15.2.6 Dilution and Ore Loss

Josemaría is a disseminated orebody lacking complexity in its ore/waste contacts. Based on experience with such deposits and engineering judgement, SRK has assumed constant dilution and ore loss values of 5% and 2% respectively. SRK does, however, recommend a bench by bench review of dilution values in future studies.

15.2.7 NSR Calculation

As there are multiple metals in the Josemaría project, with varying metallurgical recoveries, payable terms, and treatment and refining costs, SRK has used net smelter return (NSR) to assign values to the resource blocks for use in pit optimization.

In addition to the previously discussed metal prices, recoveries and dilution/ore loss, SRK used the off-site parameters provided in Table 15.4 to calculate NSR values for each of the blocks in the mine design model.

Table 15.4: Off-site pit optimization parameters

Parameter	Units	Value
Concentrate Transport		
Concentrate Moisture	%	8
Trucking	US\$/wmt	40
Ocean Freight	US\$/wmt	55
Port Handling	US\$/wmt	2
Copper Off-sites		
Cu Payable	%	96.5
Cu Deductions Concentrate Treatment	%	1
Cu Refining Cost	US\$/dmt	85
	US\$/pay lb	0.085
Gold Off-sites		
Au Payable	%	97
Au Refining Cost	US\$/oz	5
Silver Off-sites		
Ag Payable	%	90
Ag Refining	US\$/oz	0.3

15.2.8 Cost Inputs

For pit optimization studies, operating and sustaining capital costs are required inputs. Initial capital costs are considered sunk and do not play a role in the LG algorithms (though are considered in full economic analysis).

Mine Operating Costs

Based on projects of similar scale, SRK assumed a mining cost for waste and ore of \$1.80/t. This was at a reference elevation in the pit of 4,180 m. For each 10 m drop below this elevation in the pit, an incremental mining cost adjustment factor (MCAF) of \$0.025/t was added, and for each 10 m above this elevation, an MCAF of \$0.016/t was added.

Mill Operating Costs

Based on previous work on Josemaría, a mill operating cost of \$3.60/t milled was assumed for pit optimization.

General Site and Administration Costs

SRK has assumed a general site and administration (G&A) cost of \$0.80/t milled.

Sustaining Capital

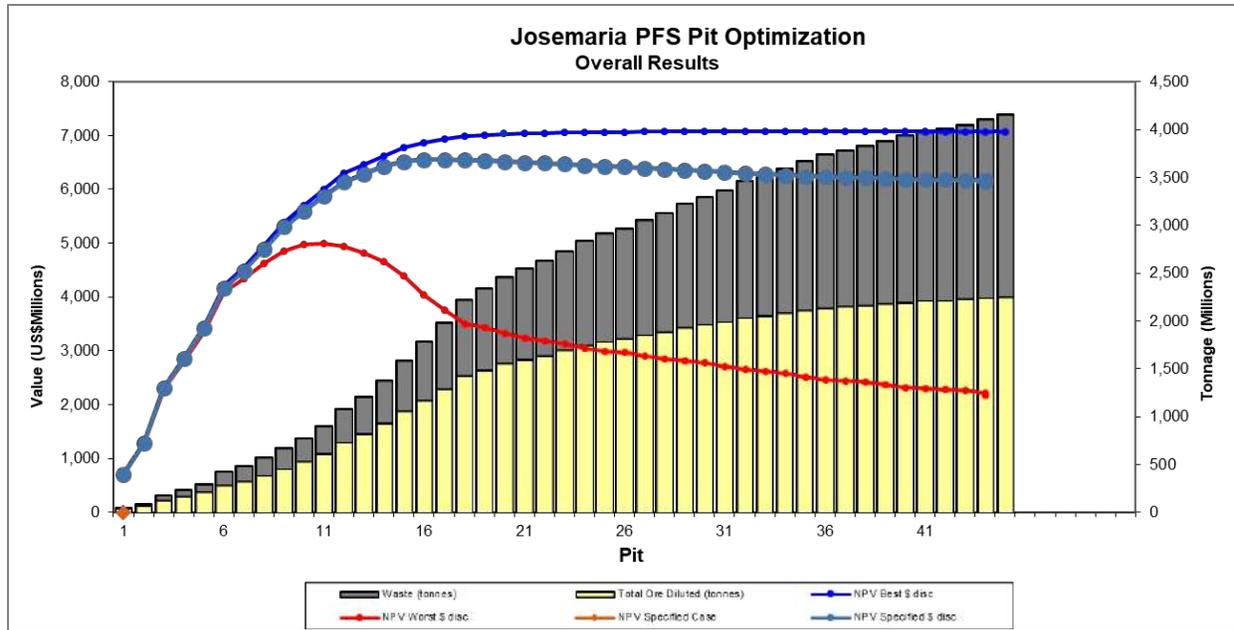
Sustaining capital is required to extend the life of mine and process equipment and to facilitate raising of the tailings facility embankments. For this, SRK assumed a sustaining capital cost of \$0.55/t milled.

15.3 Pit Optimization

15.3.1 Optimization Results

Using the foregoing input parameters, the pit optimization results in Figure 15.4 were derived. The chart shows the impact of increasing revenue factor (metal prices) on pit size and pit value. The base metal price assumptions represent the revenue factor 1.0, corresponding to pit shell #40. Each pit shell represents a 2% increment in revenue factor, from 0.22 to 1.10.

The histogram bar portion of the chart shows the pit shell quantities, with diluted ore in yellow and waste in grey. The lines represent variations of pit net present value (before initial capital cost) and base metal price assumptions. The Best Case line (blue) shows NPV if each pit shell is mined incrementally up to the current pit shell, while the Worst Case line (red) shows NPV if the current pit shell is mined a bench at a time. Typically, with appropriate pit phasing, mine planners are able to achieve value curves between the two, preferably closer to the Best Case. That was the objective of the third line, the Specified Case.



Source: SRK, 2018

Figure 15.4: Pit optimization results

The Specified Case here used a series of selected pit shells as phases to be mined sequentially to derive a mine schedule. The phase shells selected were 2, 3, 6, 8, 10 and 12. These may not be the shells ultimately selected for pit phasing, but nevertheless, give some idea of what NPV can be achieved.

15.3.2 Ultimate Pit Shell Selection

In assessing the pit optimization results, SRK selected pit shell 15 (revenue factor 0.50) as the ultimate pit and basis of mineral reserves. This represented the best balance of value and risk for pit sizing, approaching the maximum value for the Specified Case.

The quantities for this ultimate pit selection are provided in Table 15.5.

Table 15.5: Ultimate pit quantities

Parameter	Units	Value
Diluted Ore	M tonnes	1,040
Waste	M tonnes	652
Strip Ratio	w:o	0.62
Average Copper Grade	%	0.28
Average Gold Grade	g/t	0.21
Average Silver Grade	g/t	0.92
Average NSR	\$/t	19.8

15.4 Reserve Pit Design

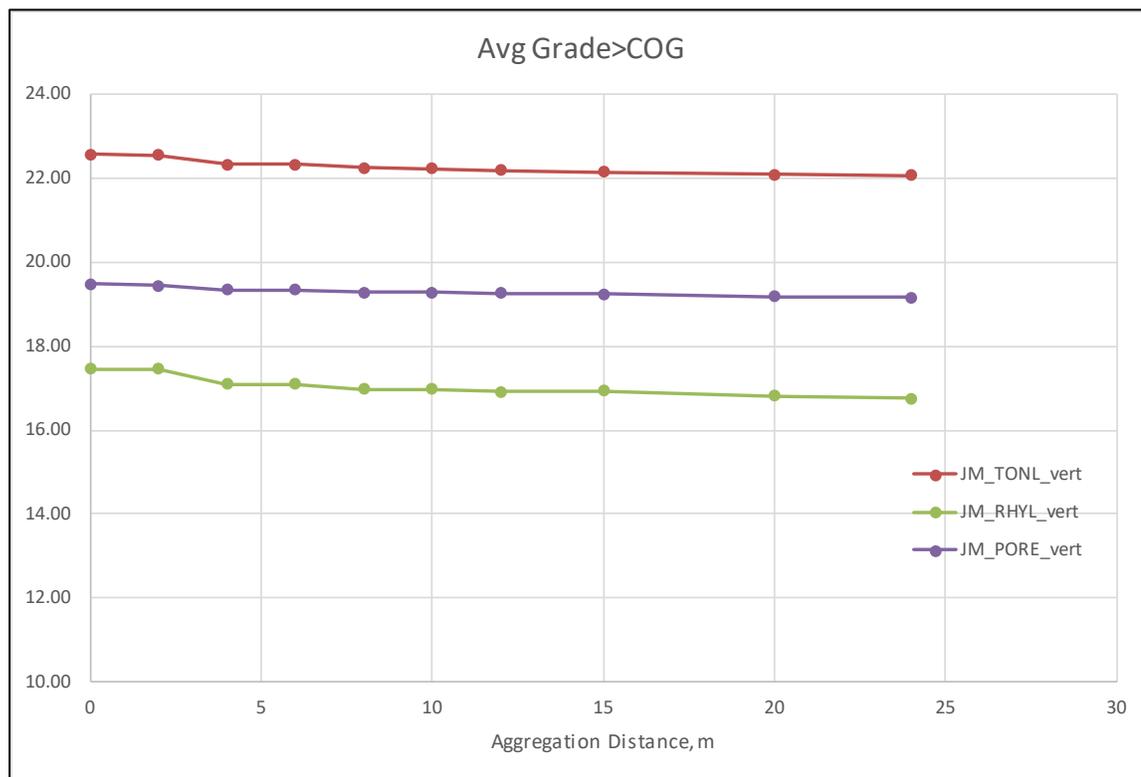
15.4.1 Parameters Relevant to Mine Design

Selective Mining Unit Size

Often, bench height and selective mining unit size are determined as a function of the mining equipment size that is perceived to meet a certain mining rate. This typically means selecting the largest equipment that matches the mining rate. However, this approach ignores the heterogeneity of the ore deposit, which in some instances, may point to requiring more selective mining with smaller mine equipment.

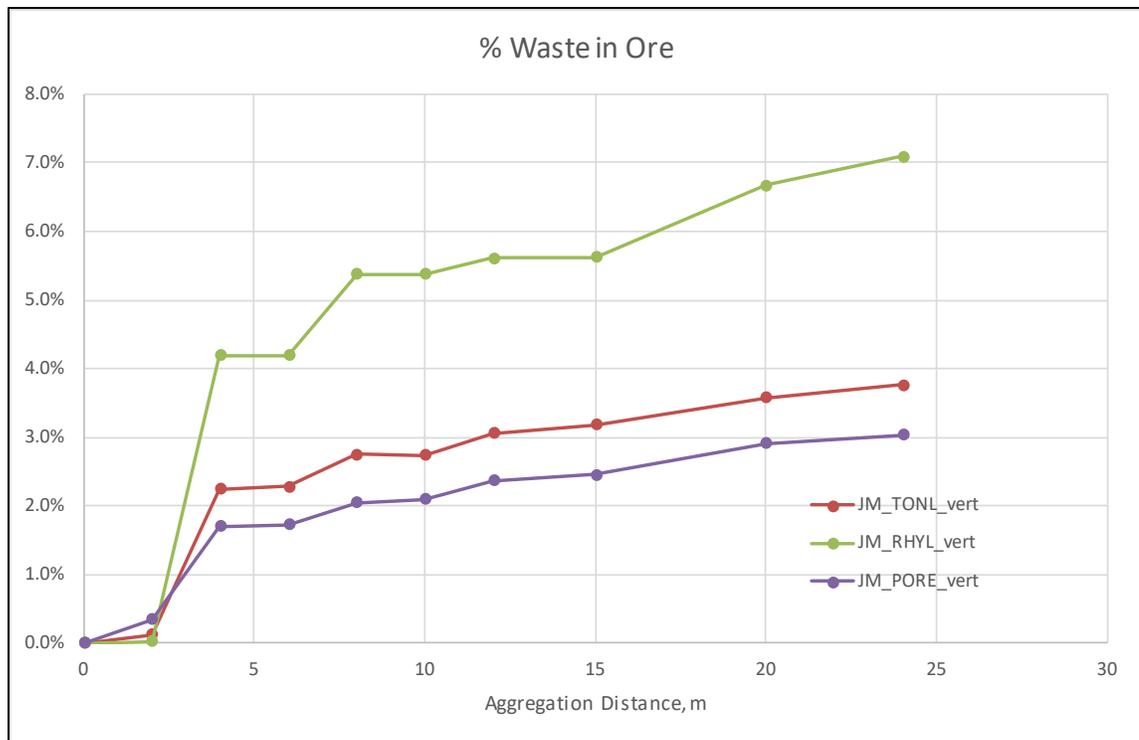
SRK has developed an approach that uses inherent characteristics of the orebody to guide the selection of mining scale and bench sizing. It requires the analysis of the heterogeneity of the deposit from exploration hole data. The results of this analysis for Josemaría are shown in Figure 15.5 and Figure 15.6.

Figure 15.5 shows that for the three ore types that make up 94% of the core intervals above cut-off (tonalite, rhyolite, and porphyry), there is very little impact on the average grade above cut-off (\$5/t NSR) with increasing mining scale.



Source: SRK, 2018

Figure 15.5: Impact on grade of increasing mining scale for Josemaría ore types



Source: SRK, 2018

Figure 15.6: Impact on waste in ore of increasing mining scale for Josemaría ore types

For the same three ore types, Figure 15.6 shows that while there is an increase in the amount of waste contained in aggregations of intervals that are above cut-off (i.e. “ore”), the levels of waste inclusion are very low – less than 6% for mining scales up to 15 m.

Thus, SRK concludes that Josemaría can proceed as a mass-mining operation, in both waste and ore, and that a bench height of 15 m for large scale mining equipment is appropriate for pit design.

Geotechnical Slope Design

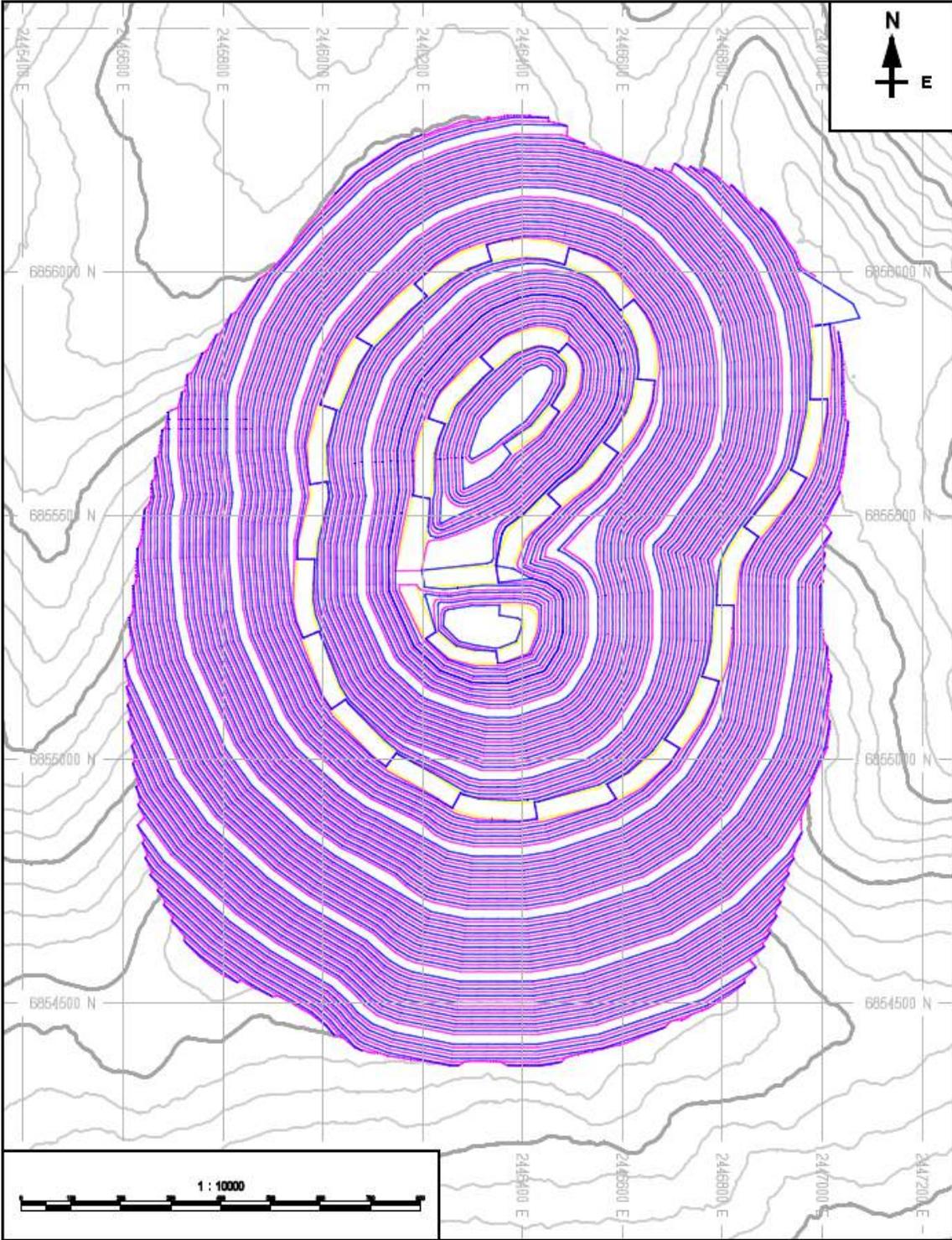
The pit wall design criteria for this study are shown in Table 15.6.

Table 15.6: Geotechnical inputs used in the study

Design Parameter	Design Sector			
	1	2	3	4
Bench Height (m)	15	15	15	15
Bench Width (m)	8	8	8.5	8.5
Bench Face Angle (°)	65	65	70	70
Stack Height (m)	105	105	105	105
Geotechnical Berm Width (m)	20	20	20	20
Inter-ramp Angle (°)	46	46	49	46

15.4.2 Ultimate Pit Design

SRK designed the ultimate pit design for Josemaría in alignment with pit shell 15 from the pit optimization analysis. The design is provided in Figure 15.7. Compared to pit shell 15, the ultimate pit design has 7% more waste and 3% less ore, which are acceptable variances in such designs.



Source: SRK, 2018

Figure 15.7: Josemaría ultimate pit design

15.5 Mineral Reserve Estimate

15.5.1 Cut-Off Grade

The cut-off grade for Josemaría determines what material is processed as ore versus what is disposed of as waste. The grade in this instance is the NSR of the material and the cut-off NSR is the value at which the value of the ore is more than the cost to process it and to pay for G&A and associated sustaining capital costs. That value for Josemaría is \$4.95/t; however, this is only for ore which is directly fed to the primary crusher and mill.

There will be times when there is more material being mined above this grade than the mill can handle. When this occurs, the material must be stockpiled. Stockpiling incurs a cost, and so this cost must be accounted for in determining what is ore. Consequently, SRK has added \$1.00/t to account for stockpiling and reclaiming costs. Thus, material which cannot be fed directly to the mill must have an NSR value of at least \$5.95/t to be placed in a long-term low-grade stockpile as ore.

15.5.2 Mineral Reserve Estimate

The mineral reserve estimate for Josemaría, provided in Table 15.7, is based on the resource model documented in the mineral resource estimate technical report (Ovalle et al., 2016). The mineral resources are inclusive of mineral reserves. The reserves are calculated using a combination of the ultimate pit design (Section 15.4.2), cut-off grade (Section 15.5.1), and production schedule (Section 16.3).

15.6 Relevant Factors

SRK is not aware of any existing environmental, permitting, legal, socio-economic, marketing, political, or other factors are likely to materially affect the mineral reserve estimate.

In addition to the mine and processing facility development, all infrastructure required to support the stated mineral reserve have been accounted for in this PFS.

Mineral reserves have been economically tested to ensure that they are economically viable (Section 22). The project remains economic across a range of key input parameters.

The pit design for establishing mineral reserves also encompassed inferred mineral resources (Section 22.12.3). Inferred mineral resources are too speculative to be the basis of mineral reserves. While there is the opportunity that future exploration may result in upgrading some of inferred mineral resources to indicated or measured, there is no guarantee that this may occur.

Table 15.7: Mineral reserve statement for the Josemaría project, San Juan province, Argentina, 20 November 2018

Category (all domains)	Tonnage	Grade				Contained Metal		
	(Mt)	Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)	Cu (B lbs)	Au (M oz)	Ag (Moz)
Proven	-	-	-	-	-	-	-	-
Probable	1,008	0.29	0.21	0.92	0.41	6.5	6.5	28.8
Total Proven and Probable	1,008	0.29	0.21	0.92	0.41	6.5	6.5	28.8

Notes to accompany Josemaría mineral reserve statement:

1. Mineral reserves have an effective date of 20 November 2018. The Qualified Person for the estimate is Mr. Robert McCarthy, P.Eng.
2. The mineral reserves were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), Definition Standards for Mineral Resources and Reserves, as prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.
3. The mineral reserves were based on a pit design which in turn aligned with an ultimate pit shell selected from a Whittle™ pit optimization exercise. Key inputs for that process are:
 - Metal prices of \$2.95/lb Cu, \$1,225/oz Au; \$19.00/oz Ag
 - Mining cost of \$1.80/t ore and waste at a reference elevation of 4180 m, plus cost adjustments of \$0.016/t/10m bench above reference and \$0.025/t/10 m bench below reference
 - Processing cost of \$3.60/t ore milled
 - General and administration cost of \$0.80/t milled
 - Sustaining capital costs of \$0.55/t
 - Pit slope angles varying from 42° to 46°
 - Process recoveries are based on grade. The average recovery is estimated to be 85% for Cu, 65% for Au and 66% for Ag
 - CuEq was calculated using total payable revenue from all metals in the mine plan, converting to payable copper, and back calculating for grade based on life of mine average copper recoveries and payables
4. Mining dilution is estimated to be 5%.
5. Ore loss is assumed to be 1%.
6. The mineral reserve has an economic cut-off, based on NSR, of \$4.95/t for direct mill feed.
7. There are 711 Mt of waste in the ultimate pit. The strip ratio is 0.71 (waste:ore).
8. All figures are rounded to reflect the relative accuracy of the estimate. Totals may not sum due to rounding as required by reporting guidelines.

16 Mining Methods

16.1 Open Pit Mining

16.1.1 Open Pit Mining

Being a large, near-surface orebody, the Josemaría project will be developed as an open pit mining operation. Waste and ore will be drilled and blasted, loaded by hydraulic shovels and loaders, and transported by haul trucks to external waste storage facilities (WSF), long-term stockpiles, or a primary crusher for mineral processing.

During a strategic mine planning study prior to the PFS, SRK established, along with NGEx, that an optimal processing throughput rate was 150 ktpd. This became the basis of all mine planning in the PFS.

The same strategic mine planning identified that the sole mineral processing route would be comminution-flotation, thus simplifying the ore selection process. Notwithstanding any potential future variations based on geometallurgical domaining, the current PFS calls for grade control to only differentiate waste from ore and direct feed ore from stockpiling ore.

16.1.2 Equipment Selection

In establishing a mill throughput rate of 150 ktpd, SRK has assumed that large scale mining would be appropriate for the Josemaría project. Consequently, SRK envisages 290-t (320-ton) haul trucks being loaded with 36-m³ (47 yd³) hydraulic front shovels with rock drilled by rotary blasthole drills capable of drilling 381-mm (15-inch) holes. Both the large hydraulic shovels and the rotary blasthole drills are to be electric-powered. The haul trucks have been specified to be autonomous, saving on labour costs while at the same time increasing equipment utilization.

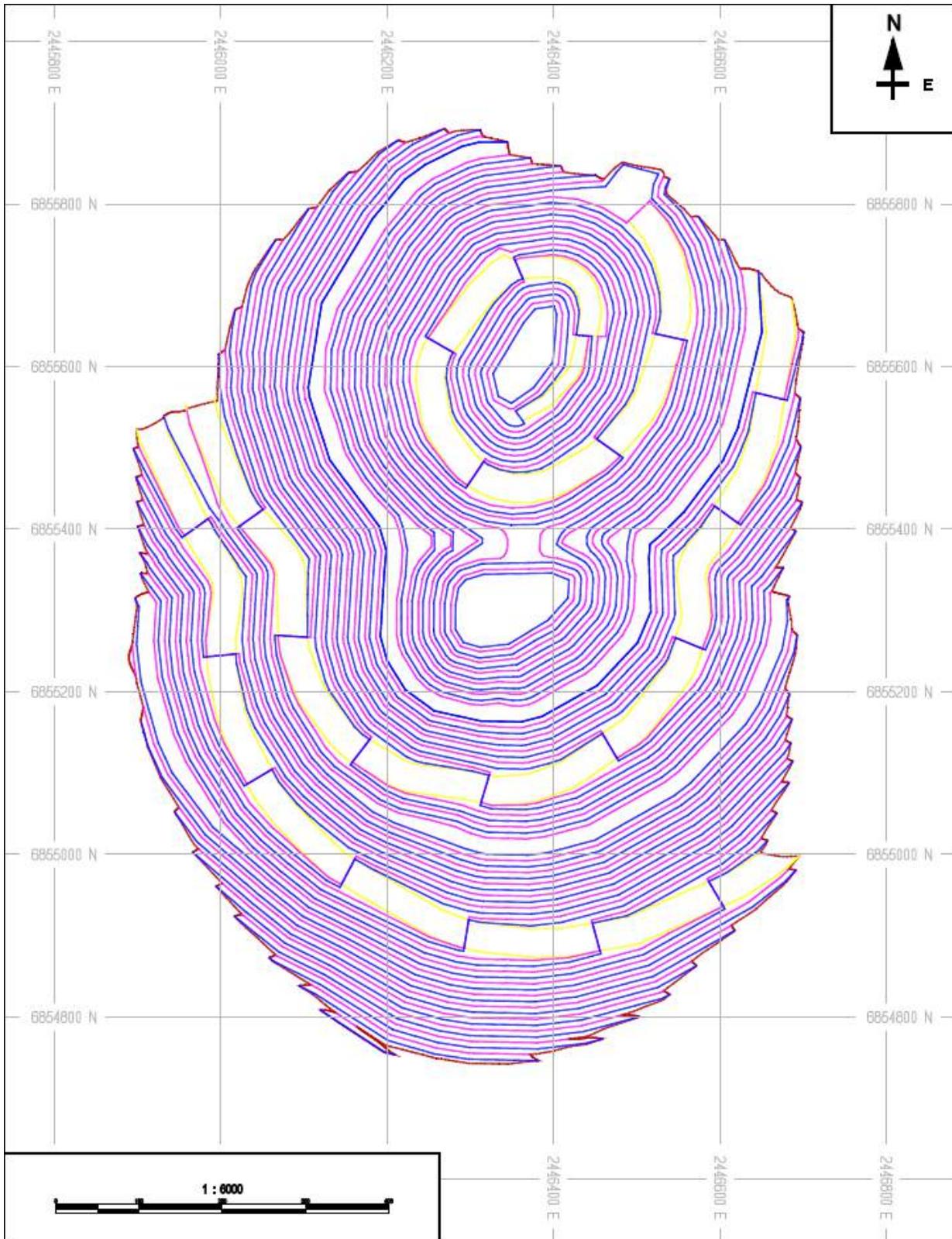
This primary production fleet is to be complemented by 20-m³ (26-yd³) front-end loaders for operation at the run-of-mine (ROM) pad and occasional truck loading in the mine and by small rotary drills for wall control blasting and supplemental blasting, 5.3-m (17.3-ft) blade track dozers, 4.6-m (15.2-ft) blade wheel dozers, 4.9-m (16-ft) blade graders, 135-t (150-ton) water trucks, and an assortment of smaller backhoes, dozers, utility dump trucks, and maintenance vehicles to support the operations.

Details of equipment fleet requirements are provided in Section 16.3.3.

16.2 Mine Design

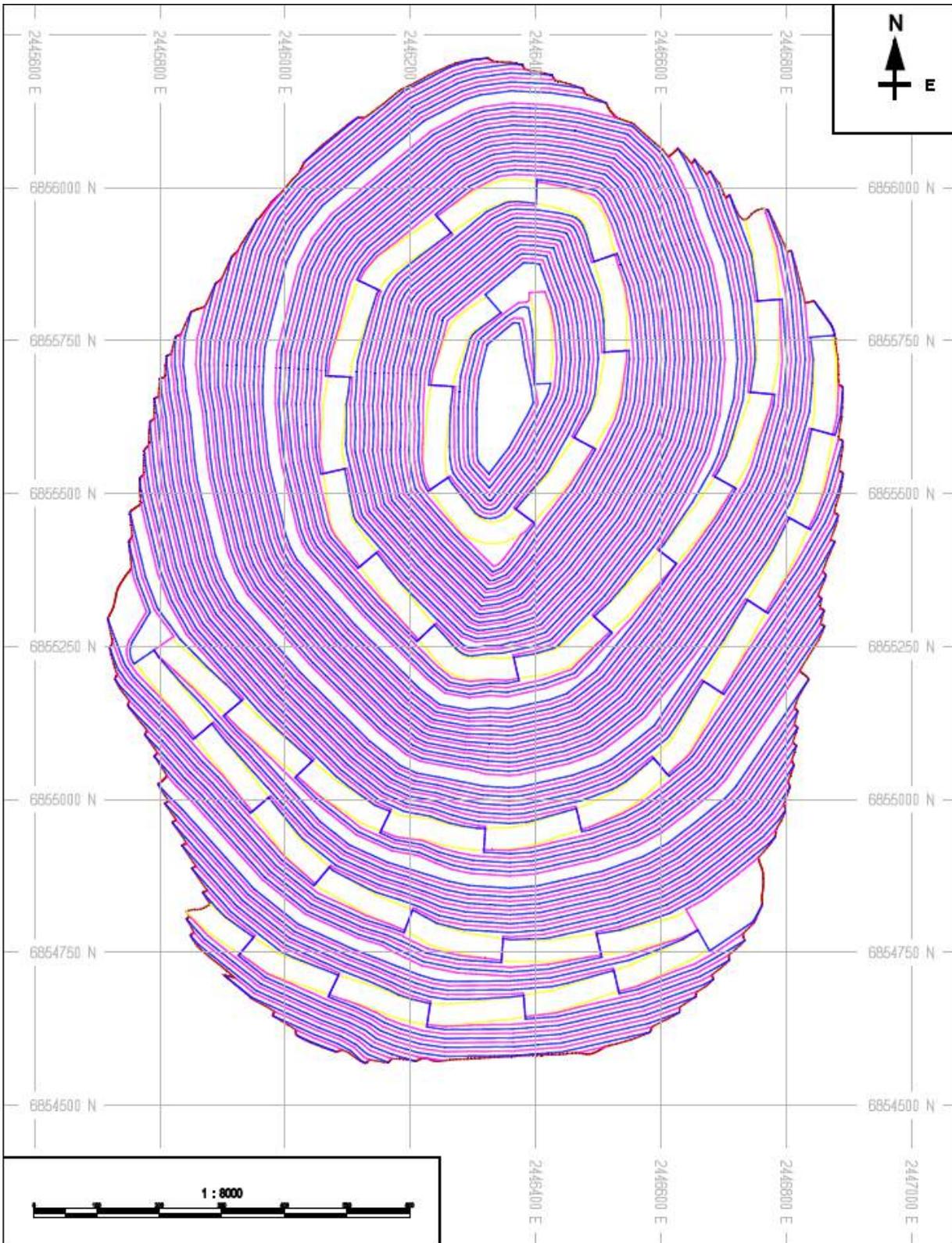
16.2.1 Pit Phase Designs

SRK has designed pit phases to facilitate the early and smooth release of higher grade ore material. There are to be four phases in all. These are illustrated in Figure 16.1 to Figure 16.5. A longitudinal section showing all four phases is provided in Figure 16.5 (location of cross-section is on Figure 16.4).



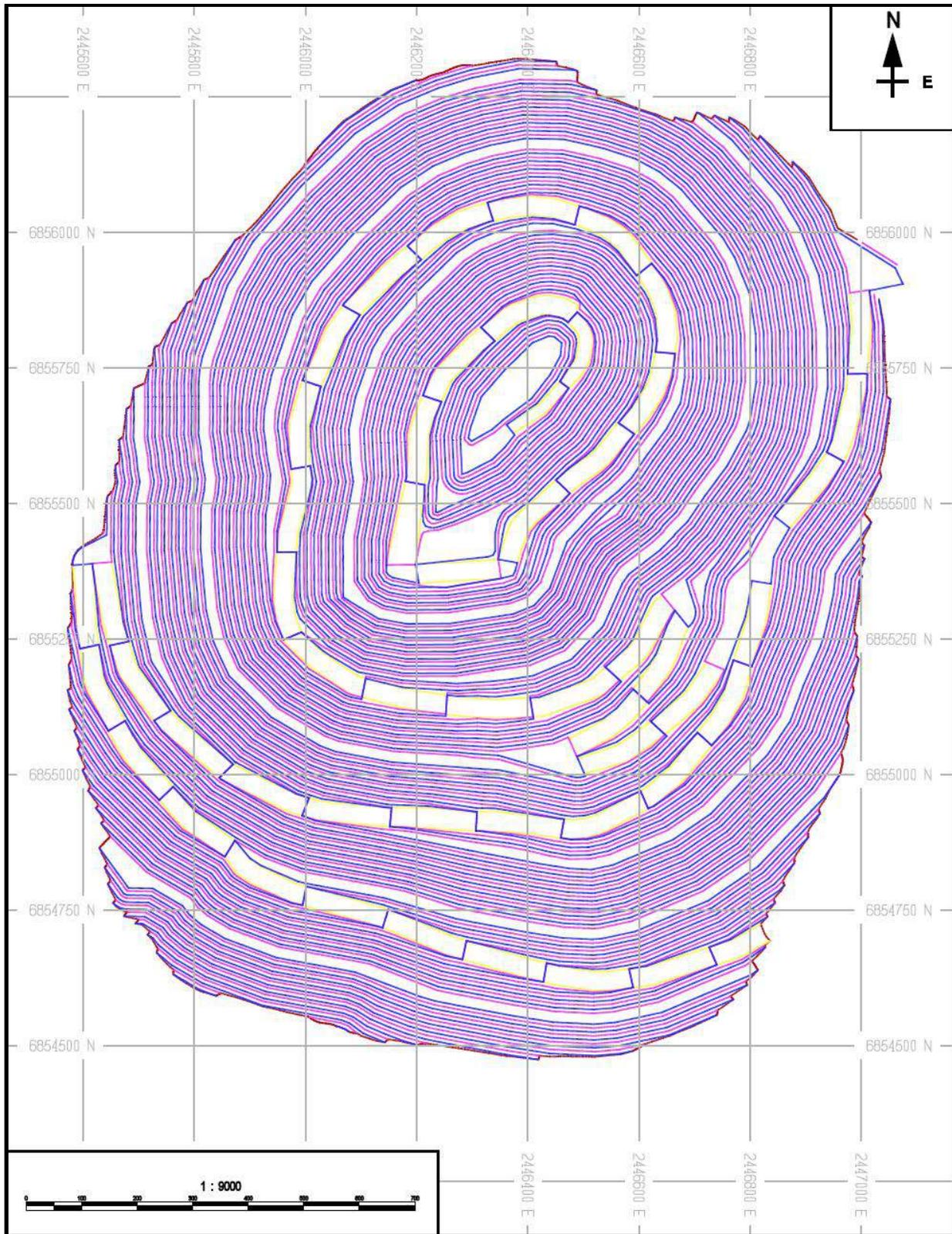
Source: SRK, 2018

Figure 16.1: Josemaría Phase 1 pit design



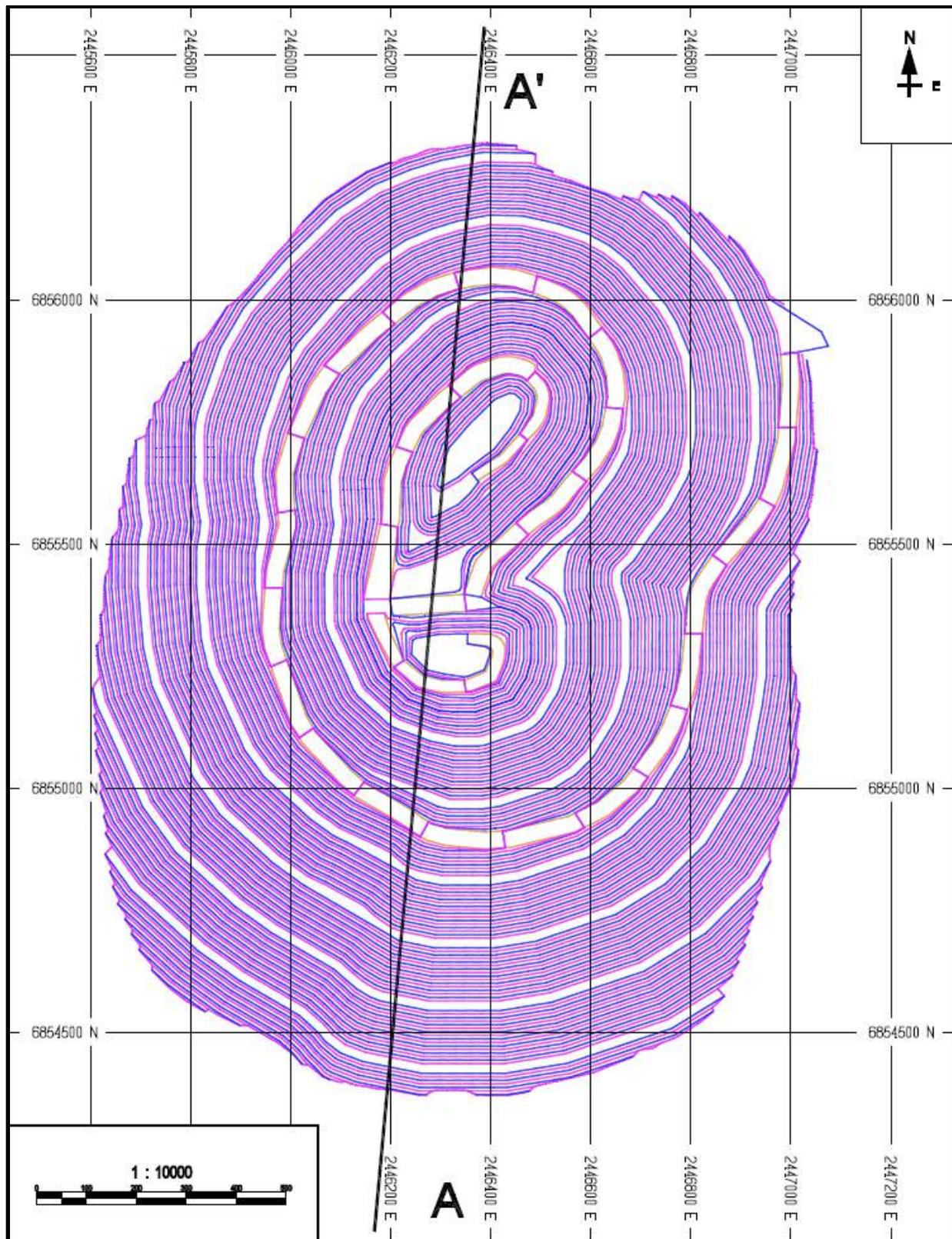
Source: SRK, 2018

Figure 16.2: Josemaría Phase 2 pit design



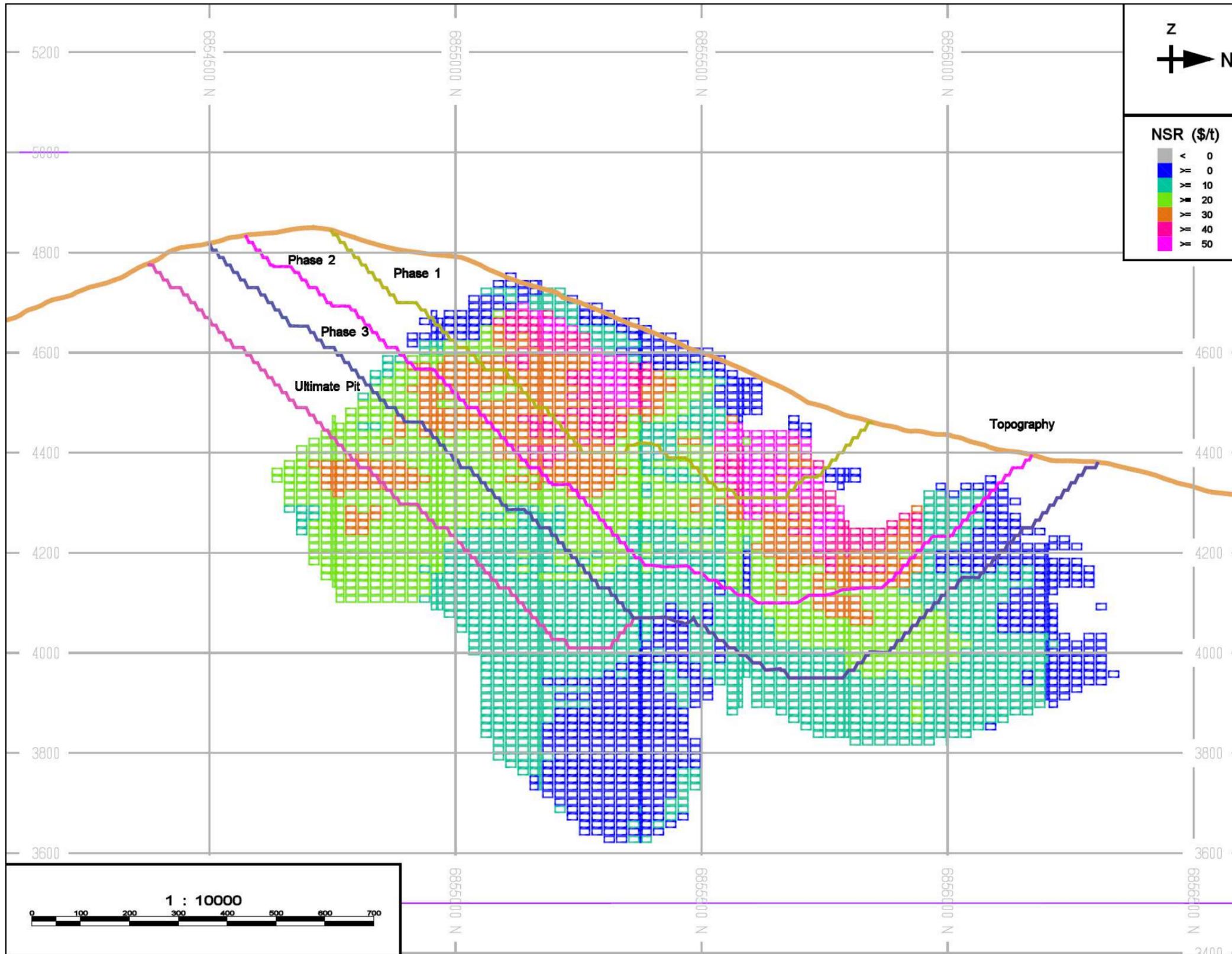
Source: SRK, 2018

Figure 16.3: Josemaría Phase 3 pit design



Source: SRK 2018

Figure 16.4: Josemaría Phase 4 pit design



Source: SRK, 2018

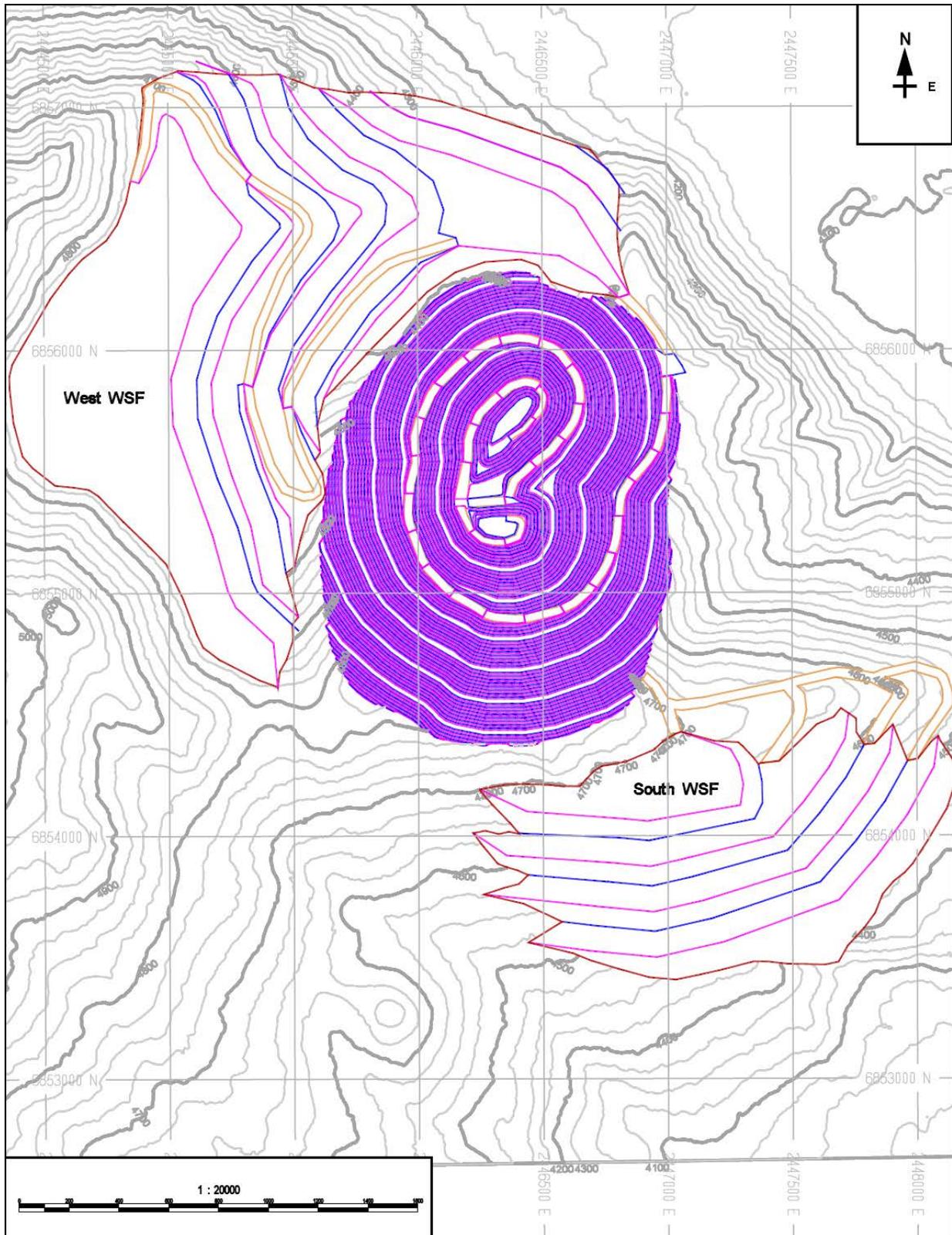
Figure 16.5: Josemaría longitudinal section (A-A') of pit phase designs

16.2.2 Waste Storage Facility Designs

SRK has designed two waste storage facilities (WSFs). These are shown in Figure 16.6.

The South WSF holds all the waste from the upper benches of the pit phases (down to 4700 m bench), while the West WSF holds all remaining waste. The West WSF is to be generally constructed as a top down, wrap-around lift facility in early years with bottom-up lifts expanding the facility in later years.

For the PFS, all waste is assumed to be potentially acid generating (PAG) and so no special waste handling or segregation has been contemplated. Should future work suggest specific waste packages need segregation due to more adverse PAG characterization, the South WSF, which drains towards the TSF, can be designed to store this material.



Source: SRK, 2018

Figure 16.6: Josemaría waste storage facilities

16.3 Mine Production Schedule

16.3.1 Pre-Production Development

SRK envisages pre-production development activities associated to the mining operations to consist of:

- Pioneering access/haulage roads to the top of the Phase 1 pit and from there up to the South and West WSF areas.
- Establishing initial access to the ROM pad for ore haulage
- Cutting and filling material as required to establish the ROM pad in front of the primary crusher
- Establishing the low-grade stockpile area by constructing a waste rock platform with hauled rock
- Establishing the accesses and pads for the explosives facility and powder magazine
- Pre-stripping 33 Mt of waste from the Phase 1 and Phase 2 pits to set up for consistent ore release

The duration of the above activities is anticipated to be 12 months.

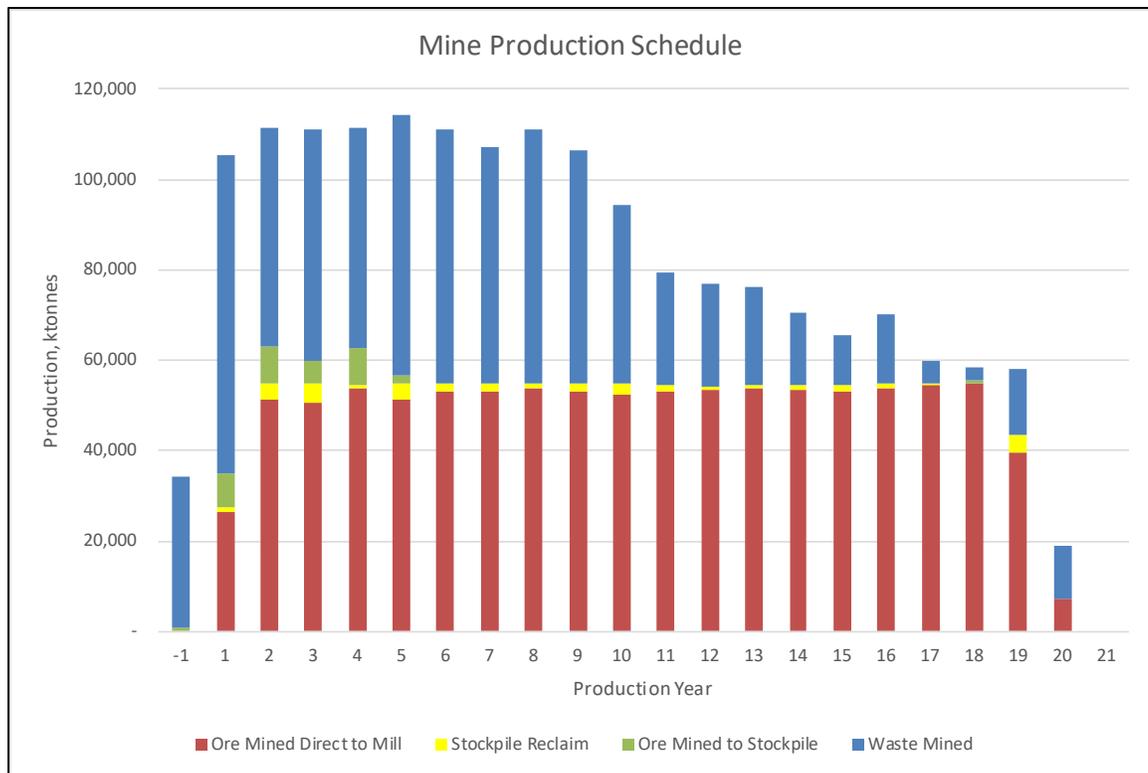
16.3.2 Production Mining

The production schedule for Josemaría is provided in Figure 16.7. After a ramp-up first year, the mine consistently delivers 54.75 Mt of ore to the mill each year. This is made up of both direct feed and stockpile reclaim. The last couple of years tail off as bench advance rates limit ore release.

The average strip ratio throughout the mine life is 0.7:1, and for the first 10 years, this is 1.0:1.

To ensure the best grade material is available at all times for mill feed, SRK has identified two cross-over grades in addition to the low-grade cut-off grade used to segregate ore from waste (Section 15.6.1). Ore greater than \$9.00/t NSR is to be considered medium-grade ore (implying that low-grade ore is from \$4.95/t [direct fed] or \$5.95/t [stockpiled] to \$9.00/t). It is to be stockpiled on the ROM pad, unless capacity is available in the mill for it to be direct fed. Ore greater than \$16.00/t NSR is to be considered high-grade ore and is intended for direct feed to the mill, though in some early periods in the schedule, an excess of high-grade ore is stockpiled on the ROM pad.

The mill is preferentially fed high-grade, medium-grade, and low-grade ores (in that order) to maintain the mill at full capacity.

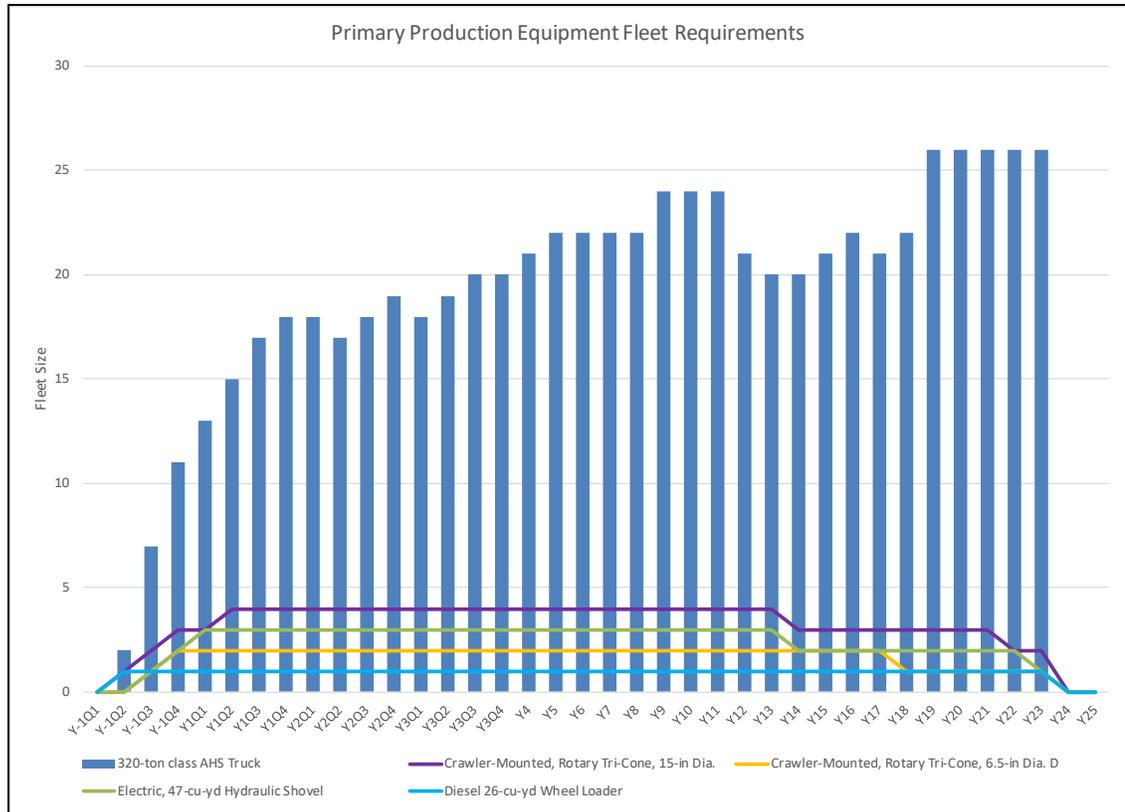


Source: SRK, 2018

Figure 16.7: Mine production schedule

16.3.3 Equipment Requirements

The equipment fleet sizes for key equipment classes are provided in Figure 16.8 and Table 16.1. Equipment requirements are estimated on a first principles basis, with haul truck requirements based on measured haul profiles. Equipment availability and utilization is based on SRK experience and judgement. Availability and utilization adjustments for the autonomous haulage fleet were provided by Komatsu S.A.



Source: SRK, 2018

Note: The ROM loader is not included in this chart

Figure 16.8: Production equipment fleet requirements

Table 16.1: Josemaría primary mine equipment

Equipment Type	Equipment Class	Maximum Fleet Size
Blast Hole Drill	150-t pulldown, 381-mm	4
Small Drill	171-mm	2
Hydraulic Shovel	36-m ³	3
Front-End Loader	20-m ³	2
Haul Truck	290-t	26
Track Dozers	5.3-m blade	4
Wheel Dozers	4.6-m blade	2
Grader	4.9-m blade	4
Water Truck	136-t	3
Backhoe	3.8-m ³	1

Other equipment specified in the PFS include:

- Small utility backhoe
- Two 24-t articulated dump trucks
- 3.3-m blade track dozer
- Portable crusher (for blast hole stemming)
- Fuel and service trucks
- Mechanic and welder trucks
- 55-t crane
- 6-t forklift
- Tire manipulator
- Crew busses
- Pick-up trucks

16.4 Mining Operations

16.4.1 Drilling

SRK envisages electric rotary drills capable of drilling 381-mm (15-inch) holes as the primary production drill. These would be supported by smaller rotary drills capable of drilling 171-mm (6.75-inch) holes for wall control holes and some make-up drilling in-pit.

16.4.2 Blasting

All rock at Josemaría is to be blasted. SRK has assumed that the supply and loading of explosives will be a contracted service. To this end, cost estimates were received from Orica (Argentina & Bolivia division) for the supply of all facilities, equipment, and personnel. All explosives and accessories were quoted on a unit basis. The cost of the personnel and construction and maintenance of facilities is recovered by a monthly service charge.

SRK has assumed a 50%/50% explosives blend of ammonium nitrate-fuel oil (ANFO) and emulsion. A powder factor of 260 kcal/t was targeted for blasting.

16.4.3 Loading

SRK has assumed all waste and most ore loading will be done with three 36-m³ (47-cu.yd.) electric hydraulic front shovels and one 20-m³ (26-cu.yd.) wheel loader. An additional 20-m³ wheel loader will be stationed at the ROM pad to facilitate stockpile reclaim, whether this is tramming medium and high-grade ore or loading trucks at the low-grade stockpile to transfer to the crusher.

16.4.4 Hauling

SRK has evaluated the adoption of autonomous haulage for the Josemaría project. In consideration of the increased availability, utilization and efficiency of autonomous haulage, there is a significant reduction in the number of trucks required for the haulage fleet. SRK further increased the operating hours per day of loading equipment, with overlapping operators, to allow for near-continuous loading of the trucks. Additional operating costs for the system were offset by tire and labour cost savings.

16.4.5 Support Activities

Roads and dumps will be maintained by a fleet of support equipment including 5.3-m blade track dozers, 4.6-m wheel dozers, and 4.9-m blade motor graders. Additionally, 135-t water trucks will be used to control dust.

16.4.6 Ancillary Equipment

SRK has accounted for ancillary support equipment including: light plants, crane, forklift, tire manipulator, field service and maintenance vehicles, and light vehicles.

16.4.7 Dewatering

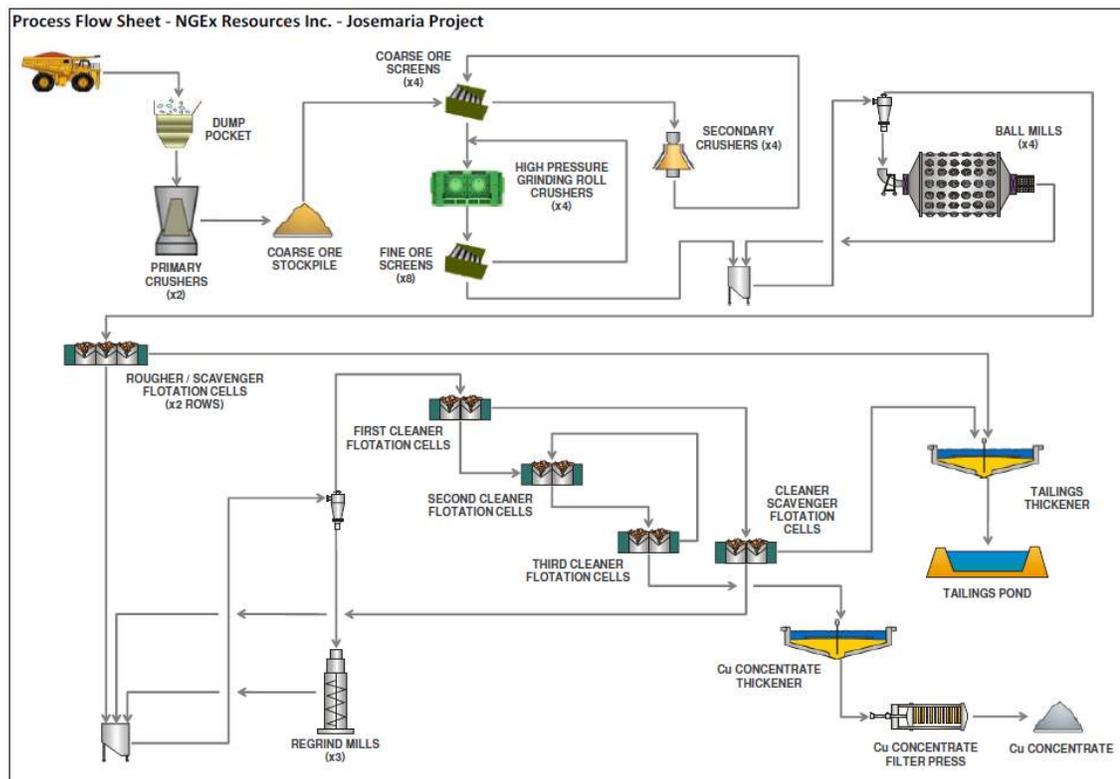
The Josemaría pit is expected to experience in-flows of groundwater to an average maximum of 72 L/s. SRK has provisioned for in-pit sumps and pumps capable of meeting peak in-flows of up to 100 L/s.

17 Recovery Methods

17.1 Introduction

17.1.1 General Facility Description

The process facilities are designed to treat a nominal rate of 150,000 tpd of sulphide ore and produce copper concentrate. The process design is based on existing technologies and the largest available and proven equipment sizes. A simplified process flow is provided in Figure 17.1.



Source: Ausenco, 2018

Figure 17.1: Simplified process flow diagram

17.1.2 Comminution Circuit Review

Ausenco undertook a comminution circuit review study for the Josemaría project as part of the PFS. This review included benchmarking the Josemaría ore against other operations, evaluating the previous comminution circuit design, providing a high-level design for SABC (semi-autogenous mill, ball mill, and pebble crushing) and HPGR-B plants, and updating the capital and operating cost estimates.

Ausenco adopted a conservative approach for sizing the HPGRs due to a limited number of HPGR laboratory and/or pilot trials completed and as such, the absence of parameters obtained therefrom such as the specific throughput rates (M-dot) and recycle rates. Consequently, an

HPGR motor power of 6,800 kW was considered to ensure that the installed power is sufficient to achieve nominal throughput rate.

The SABC circuit option is an established technological approach to comminution and generally results in smaller physical footprints than HPGR-based circuits. The capital cost estimate showed that installation of an SABC circuit resulted in 15% lower Capex than the HPGR-B circuit. However, the SABC circuit required higher energy and steel media consumption and the operating cost estimated for the HPGR-B circuit was 27% lower than that for the SABC circuit.

The ore characteristics, net present cost (NPC) calculations and benchmarking against similar ores strongly support the use of HPGR technology over the life of the Josemaría project. The cumulative NPC was calculated for a 20-year period using an 8% discount rate, and cost differences indicated that the HPGR circuit is a more financially attractive alternative.

17.2 Processing Plant General

17.2.1 Crushing

Primary Crushing

The crushing circuit will consist of two parallel trains of identical primary gyratory crushers (1,000 kW installed power), treating a total of 8,333 tph (dry). ROM material will be delivered to the primary crushers by mine trucks into a single dump pocket with a 500-t live capacity to the top of the primary crusher for each train. The primary crusher will receive ROM (F_{100} of 1,200 mm) and reduce it to a P_{80} of 137 mm. The primary crusher will discharge the crushed ore into a 500-t live capacity surge pocket where a variable speed apron feeder will deliver ore from the surge pocket to the coarse ore stockpile feed conveyor. The stockpile feed conveyor will deliver the ore to a coarse ore stockpile for reclaim.

Crushed Ore Stockpile and Reclaim

Dual reclaim tunnel systems will be used to house a total of six reclaim apron feeders (three per train) that draw ore from the coarse ore stockpile (454,000 wet t) to two coarse ore reclaim conveyors. The coarse ore stockpile will have a total live capacity of 12 hours of mill feed at the nominal feed rate (91,000 wet t). The stockpile material will be reclaimed from the stockpile at an average rate of 3,676 tph (dry) per train to the coarse ore surge bins. A magnet will be installed on each conveyor to remove tramp iron from the ore, as well as a metal detector to indicate any metal not picked up by the magnet. The operator will manually remove any remaining tramp metal to protect the downstream secondary crushers. The reclaim conveyor will discharge to a coarse ore surge bin feeder ahead of coarse ore surge bins. The coarse ore screen feeder will withdraw ore from the bin at a controlled rate to maintain a steady load and evenly distribute feed on the coarse ore screens. The coarse oversize from the screen will feed a common secondary crusher surge bin, while the finer screen undersize will be conveyed to the HPGR surge bins.

Secondary and Tertiary Crushing

There will be four secondary cone crushers (933 kW installed power), each of which will operate in an independent line. Each line will consist of a surge bin section, a belt feeder and a standard cone crusher. The crushed product from the secondary crusher will be returned to the coarse ore surge bin. The oversize ($P_{100} = 365$ mm) will be reduced to a product with a P_{80} of 41 mm, which will then be conveyed to the coarse ore surge bin for screening. An allocation of space for two additional crushers (one for each line) will allow for possible future expansion to provide processing flexibility.

Tertiary crushing will be performed by four independent HPGR lines (6,800 kW installed power). Each line will include an HPGR surge bin, a feeder, and a HPGR unit. Each line will share common feed distribution and product conveyors. The HPGR product will be transferred to a row of fine ore surge bins. The HPGRs will be in closed circuit with wet screens, with the downstream fine ore screens providing a control on the particle size to the grinding circuit ($F_{80} = 3.8$ mm). Oversize (> 6 mm) from these screens will return to the HPGR surge bin.

17.2.2 Grinding

There will be four independent grinding lines, each consisting of a fine ore surge bin, two reclaim feeders, two screens, a pump box and cyclone feed pump, a cyclone cluster and a ball mill (7.9 m diameter x 12.5 m effective grinding length with 19,000 kW installed power). Each feeder will discharge to a wet screen where the finely crushed ore will be slurried and washed, with the fine underflow slurry discharging to the ball mill cyclone feed box. The partly dewatered screen oversize will be conveyed back to the HPGR surge bin.

Cyclone overflow ($P_{80} = 130$ μ m) will be sampled and analyzed for size and metal content as it passes to two banks of the rougher scavenger flotation circuit. Two grinding lines will feed one bank of flotation cells for initial recovery of copper mineral from the ground ore.

Each grinding line will have a single (duty only) variable speed cyclone feed pump. Ball mill discharge slurry will be pumped to the cyclone cluster. The cyclone underflow will gravity flow to the ball mills with the cyclone overflow reporting to the rougher flotation section. A full spare assembly will be held on site and changed out when required.

17.2.3 Flotation

The flotation circuit is designed to receive feed material grading 0.29% Cu and 0.21 g/t Au over the LOM. Nominal copper recovery to the flotation concentrate is 86.4%, with a gold recovery of 70.9%. Nominal copper concentrate grade is expected to be 25.1% Cu with 14.5 g/t Au.

Flotation comprises rougher scavenger flotation, regrinding of the rougher concentrate and three stages of cleaner flotation to produce a final copper concentrate.

Rougher scavenger flotation will consist of two parallel lines of nine 630-m³ forced air tank flotation cells fed at a density of 35% w/w solids. The concentrate from the first two cells of each row, of higher copper grade, will be diverted directly to cleaner flotation. This will bypass the

regrind stage. The concentrate from the remaining seven cells will require regrind prior to further upgrading in the cleaner flotation section. The tailings from the rougher scavenger flotation section will report to the final tailings.

The regrind section will consist of three vertical stirred regrind mills (11,220 kW installed power), where size reduction to a P_{80} of 25 μm will be achieved in closed circuit with hydrocyclones.

The combined bypass rougher concentrate and reground rougher concentrate will feed the first cleaner flotation section. The first cleaner flotation section will consist of six 300- m^3 forced air tank flotation cells. The tailings from the first cleaner flotation will feed the cleaner scavenger flotation section consisting of seven 630- m^3 forced air tank flotation cells. The concentrate from the cleaner scavenger section will be directed back to the regrind section with the tailings reporting to the final tailings.

The second and third cleaner sections will upgrade the concentrate from the first cleaner cells to final concentrate grade. The second cleaner flotation section will consist of six 70- m^3 forced air tank flotation cells with the concentrate directed to the third cleaner stage and the tailings recycled to the first cleaner stage. The third cleaner section will consist of six 30- m^3 forced air tank flotation cells with the final concentrate directed to the dewatering section and the tailings recycled to feed the second cleaner section.

Copper flotation will be promoted using Potassium Amyl Xanthate (PAX), Sascol 95 and Matcol TC-123 collectors with Methyl Isobutyl Carbinol (MIBC) added as a frother. Lime will be used for pH modification, as required.

17.3 Copper Concentrate Thickening and Filtration

The final copper concentrate will be thickened with flocculant to 65% solids in a high-rate thickener (35-m diameter). The thickened concentrate will feed the copper filtration stock tank. Thickener overflow water will be recycled to the process.

An agitated tank will have capacity to store six hours production of thickened concentrate ahead of the filtration stage. One pressure filter (61 plates, 2.5 m by 2.5 m plates) will reduce moisture content of the concentrate to approximately 9% for shipment. Filtered concentrate will fall into a covered storage area and will be transferred to a covered copper concentrate stockpile area by front-end loader. There will also be an open emergency storage area in the event of shipment disruptions.

A front-end loader will be used to load concentrate from the storage area to copper concentrate trucks. Each concentrate truck will be weighed and sampled to ensure maximum filling without overloading.

17.4 Tailings Thickening and Storage

Flotation tailings will represent approximately 98% of the total plant feed tonnage and are designed to be safely stored in a dedicated facility. Tailings will initially be thickened in two high-

rate thickeners (85 m diameter) to approximately 62% solids and pumped to the TSF. The thickener overflow will be recycled to the process water system.

There will be two barges with pumps in the main pond to pump the water up to a process water tank. Reclaimed water will be pumped to the process water tank for distribution within the process plant or advanced for direct use in the cyclone feed dilution.

Any water that seeps under or through the TSF embankment, along with drainage from the embankment dam, will be recovered by seepage pumps. The overall facility has no discharge to the surrounding environment other than water evaporation to the atmosphere.

17.5 Fresh Water Supply

The fresh water system will supply water to parts of the process that require a clean water supply, as well as making up for water lost in processing. Losses will be mainly due to the physical entrainment of water with tailing solids in the TSF. Much smaller amounts will be lost to evaporation and the copper concentrate.

17.6 Process Design Criteria and Equipment Sizing

Process design criteria and equipment sizing are mainly based on the Phase II testwork completed by SGS. SGS conducted an extensive range of testwork including comminution and flotation tests on both composite and variability samples suitable for PFS level. Simplified flotation tests were conducted by ALS, Kamloops in the late stage of the PFS to verify or improve flotation results and support the criteria developed for the engineering design. These test results showed potential to reduce the flotation circuit size and simplify reagent schemes relative to earlier testwork. Further testwork is recommended to further evaluate the simplified flotation circuit and reagent schemes in the next phase of study.

The nominated process design criteria for the PFS are presented in Table 17.1.

Table 17.1: Process plant design criteria

Plant Area	Description	Rates	Comments
Process General	Plant Design Capacity	54.75 Mt/year	
	Plant Design Capacity	150 ktpd	
	Primary Crushing Availability	75%	Ausenco benchmark
	Secondary Crushing Availability	85%	Ausenco benchmark
	HPGR availability	90%	Ausenco benchmark
	Grinding and Flotation availability	90%	Ausenco benchmark
	Concentrate Filter availability	80%	Ausenco benchmark
	Coarse Ore Stockpile Live volume	12 hours	Ausenco benchmark
	Coarse Ore Stockpile Cover	None	
Ore Characteristics	Ore specific gravity	2.77	SGS Phase II testwork
	Ore Moisture content	3%	Ausenco benchmark
	JK SMC test parameters, A x b	32.1	SGS Phase II testwork (25 th percentile)
	Bond rod mill work index	13.3 kWh/t	SGS Phase II testwork (75 th percentile)
	Bond ball mill work index	14.7 kWh/t	SGS Phase II testwork (75 th percentile)
	Bond abrasion index	0.187 g	SGS Phase II testwork (75 th percentile)
Grinding	Target P ₈₀	130 µm	SGS Phase II testwork
	Recirculating load	300%	Ausenco benchmark
Flotation	Average copper head grade	0.29%	Mine plan
	Mass pull to final concentrate	1.39% new feed	SGS Phase II testwork

Plant Area	Description	Rates	Comments
	Rougher residence time (lab)	20 minutes	SGS Phase II testwork
	Cleaner 1 residence time (lab)	8 minutes	SGS Phase II testwork
	Cleaner 2 residence time (lab)	7 minutes	SGS Phase II testwork
	Cleaner 3 residence time (lab)	4 minutes	SGS Phase II testwork
	Cleaner scavenger residence time	27 minutes	SGS Phase II testwork
	Rougher scale-up factor	2	Ausenco benchmark
	Cleaner scale up factor	2.5	Ausenco benchmark
	Regrind target P ₈₀	25 µm	SGS Phase II testwork
Thickening Design	Settling Rate - Concentrate	0.15 t/h/m ²	Ausenco benchmark
	Settling Rate - Tailings	0.95 t/h/m ²	SGS Phase II testwork
	Concentrate thickener U/F	65%	SGS Phase I testwork
	Tailings thickener U/F	62%	SGS Phase II testwork
Concentrate Filter	Filtration rate	450 kg/h/m ²	Ausenco benchmark
	Final concentrate moisture content	9%	Ausenco benchmark

17.7 Major Process Requirements

A summary of the major requirements for the process plant at Josemaría is provided in Table 17.2.

Table 17.2: Selected major process plant requirements

Area	Description	Units	Value
Power	Total installed power	MW	188
	Peak demand power	MW	150
Water	Process make-up	m ³ /a	23,584,398
Consumables/ Reagents	Steel grinding media	t/a	4,271
	Lime	t/a	24,638
	Cu collector (SASCOL 95 equiv)	t/a	1,369
	Cu collector (MATCOL TC-123 equiv)	t/a	1,369
	MIBC (Frother)	t/a	548
	Antiscalant	t/a	274
	Flocculant (Tailings)	t/a	164
Labour	Plant labour	units	265
Diesel	Diesel (surface fleet)	L/a	2,740,599
Concentrate	Concentrate tonnage	t/a	761,025

18 Project Infrastructure

18.1 General Site Layout

The general site layout is shown in Figure 18.1.

18.2 Access Roads and Logistics

18.2.1 Road Logistics

Concentrate will be transported from the Josemaría mine site to the Port of Caldera using a standard combination of tandem drive tractor articulated with a tipper semi-trailer. The trucks will carry up to a gross weight of 52 t and a concentrate payload of 32.7 t. This will require special permitting under the national regulations to allow use on the public highway system.

Approximately 57 km of light vehicle road will require upgrading to a 9-m wide, two-lane, dirt road to connect the Josemaría mine site to the Argentina Ruta Nacional 76 highway. The route then enters Chile through the international border crossing at Pircas Negras and continues through Copiapó to the Port of Caldera using Chilean highways C-359, C-459, C-503, C-401, C-35, and Route 5. The travel distance between the site and the port is approximately 343 km.

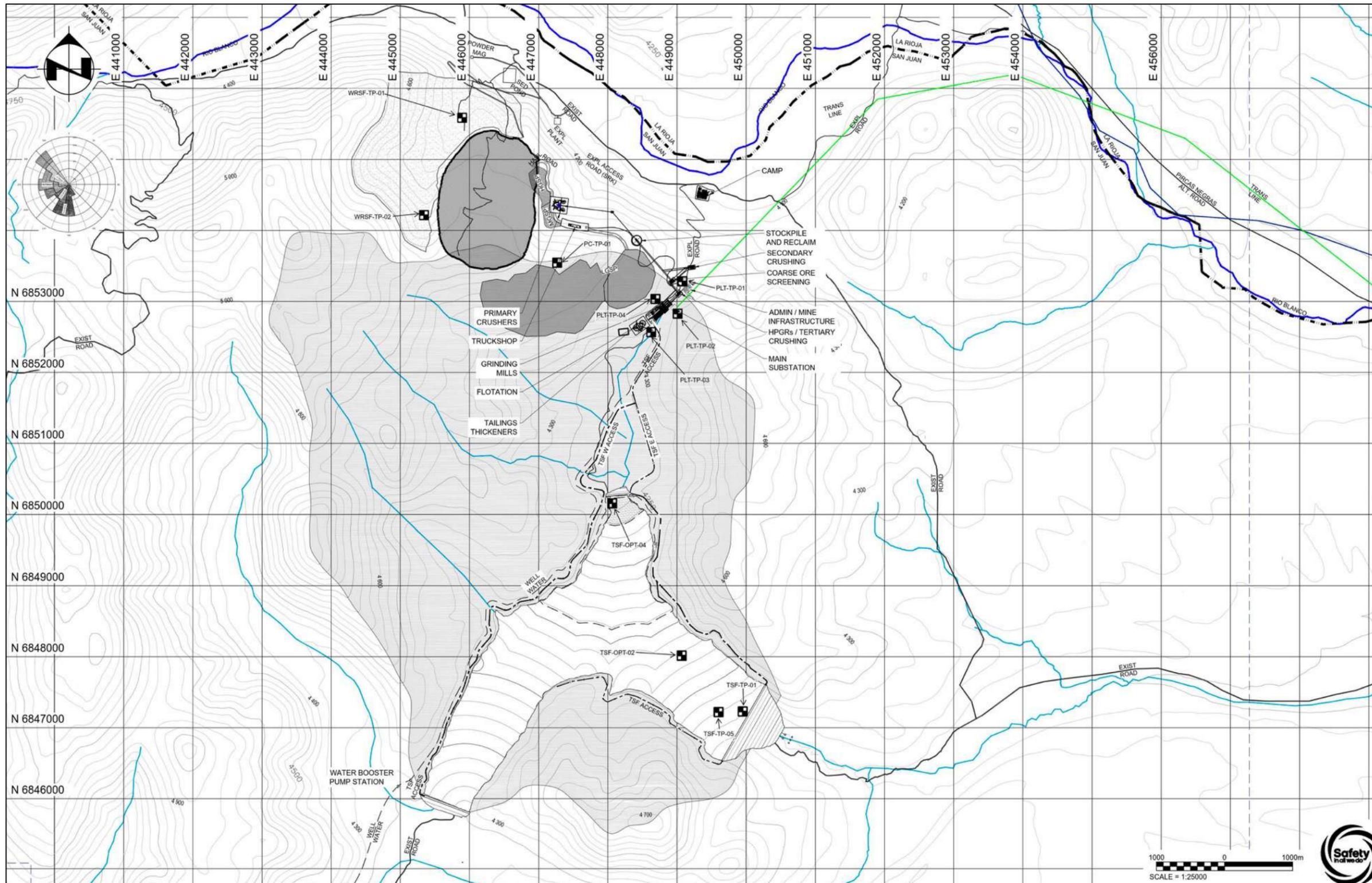
It is anticipated that the truck cycle time from the mine site to the port will be approximately 34 hours. Concentrate transport has been designed based on haulage schedule of 330 days per year. This allows for interruptions such as landslides, floods, road maintenance operations, civic holidays, labour disputes and day/night/climatic conditions affecting visibility.

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 concentrate shipments. Roads will connect various mine facilities, including the proposed open pit, truck shop, conveyor locations, process plant and crushers, electrical substations, and administrative buildings.

18.2.2 Port Logistics

The Port of Caldera has two suitable existing terminals for the export of copper concentrate and the import of consumables: Punta Padrones and Punta Caleta.

Export of concentrate will be from the Punta Padrones Terminal. The terminal is operated by Minera Contractual Candelaria, which is majority owned by another Lundin Group Company and currently ships copper concentrate. The total annual capacity of the terminal is estimated at 3.5 million wmt and currently operates at 600,000 wmt.



Source: Ausenco, 2018

Figure 18.1: General site layout - Josemaría project

New infrastructure required to accommodate the Josemaría throughput will consist of:

- A new storage building for ~50,000 wmt of storage, estimated as a 50 x 100 m building, with equipment to tie into the existing out-loading system, including:
 - Reclaim conveyor(s)
 - Reclaim hopper(s)
- A new truck dump system, including:
 - ~ 30 x 10 m building
 - Truck dump belt feeder
 - Elevating conveyor and tripper
- Administration and services (utilities, maintenance, etc.) is assumed to be provided by the existing operations and is not included in this study.

Import of consumables to Punta Caleta Terminal:

- Operated by Puerto Caldera SA for breakbulk and containerized products
- Capable of handling vessels up to 66,000 DWT, 222 m LOA and 11 m draft

18.3 Power

18.3.1 Transmission Line Connection

Given the site's close proximity to Chile, a preliminary power study was conducted to compare potential grid interconnections points in both Argentina and Chile. The recommendation of the power study was to connect in Argentina since costs were comparable to a connection in Chile, and this would also allow all project infrastructure to remain within Argentina.

Two substations in Argentina (Guanizuil and Bauchaceta) were identified with enough capacity to handle the Josemaría electrical load, with Guanizuil being in closer proximity to the project site. Upgrades will be required at the Guanizuil substation to accommodate the new interconnection. The transmission line connecting Guanizuil substation to the plant site will be a single-circuit 220-kV line that is approximately 250 km long.

Alternative options will be reviewed during the feasibility study to ensure that the most cost-effective option is selected for the final design. At that stage, local power utility will also be engaged to confirm interconnection and permitting requirements.

18.3.2 Electrical Load

The maximum demand load is estimated to be 150 MW based on the anticipated installed mechanical equipment and all non-process loads.

18.3.3 Main Substation

The incoming electrical power from the 220-kV transmission line will be stepped down at the Main Substation switchyard to 13.8-kV for in-plant distribution through two 220/13.8-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.

Within its perimeter fence, all required auxiliary services will also be housed including emergency generator, electrical room and control room for substation operation. 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 housed within the grinding facility.

18.3.4 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 TSF, water well pumps, and camp.

Prefabricated electrical rooms have been considered for the various crushing and processing areas including: primary crushing, secondary crushing, stockpile, HPGR, grinding, and tailings.

Variable frequency drives have been allowed for HPGR motors which will be fed from the main 13.8-kV switchgear located in the HPGR electrical room. 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 MCCs will be located in the electrical room and will include intelligent combination starters, with circuit breakers for instantaneous fault protection.

All critical loads at the process plant will be powered by a 1-MW emergency diesel generator adjacent to the grinding facility, and uninterruptable power supply systems will also be located at each electrical room, control room and operator cabin.

18.4 Plant Buildings

Plant and ancillary buildings will be pre-engineered and modular to the greatest extent possible, and will include:

- Gatehouse
- Mine infrastructure area building consisting of truck shop, truck wash, mine offices, mine dry and warehouse

- Laboratory
- Administration building
- Mill dry facility and plant change rooms
- Plant workshop and warehouse

18.5 Camp and Accommodations

Due to the remote location, the construction and operations workforce will be housed in an accommodation camp. The camp is planned to be located approximately 2.5 km east of the pit location and will be to the north of the main plant site. 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 requirements for a 4,000-man camp (this allows for an existing camp currently established at the site). 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 300 persons.

18.6 Fuel Supply and Storage

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

Knight Piésold has identified locations of three potential water supply sources that are under consideration for the Josemaría project and these are permitted for water well exploration and drilling in the 2018/2019 season. The locations have been identified based on regional geology and topography, and they range between 22.4 km to 24.4 km away from the plant site, and therefore 23.4 km was used as the average pipeline length.

The assumption at this phase of the project is that all wells will be located in relatively close proximity of each other and will produce sufficient water supply to meet the water demands of the project. The water supply capacity of nearby valley aquifers will need to be tested and confirmed during the next phase.

The maximum make-up water requirement for a mean annual precipitation cycle is estimated to be 555 L/s (during the dry season), while the overall average make-up water requirement on an annual basis is 353 L/s, based on 150 ktpd nominal feed rate. 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. Booster station(s) will be installed as needed to deliver water from the location of the wells to plant site. One vertical turbine pump and three booster pumps have been selected at this level of study. Due to the relatively high operating pressure, carbon steel piping has been included. There will be the opportunity to optimize the design of this system during the next phase. Construction water source will be confirmed during the next phase.

Due to the arid region, water recovery processes will be reviewed and further optimized during the feasibility study. Areas to assess will include consideration of the type of tailings thickening to reduce the make-up water requirement.

18.8 Infrastructure Geotechnical Investigation

18.8.1 Field Program

As part of the design of the TSF, primary crusher, processing facilities, waste rock storage facilities, and stockpiles, a geotechnical program was carried out at a PFS level. The field program included a test pit program to take samples of soil and rock from plant site location, primary crusher site, waste rock storage facility, stockpiles, and a new TSF site, 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. In addition, one geophysics line was performed along the longitudinal axis of the proposed TSF.

Surface geology was mapped by a qualified geologist in the area of the TSF, primary crusher, plant, waste rock storage facilities and the stockpiles. The mapping included:

- Surface stratigraphy
- Structural geology
- Glaciers, periglacial features (in accordance with BGC's requirements)
- Surface water corridors
- Vegetation (such as vegas)
- External geodynamic

The test pits program consisted of the excavation of 11 test pits (excluding the test pits excavated at the formerly proposed Heap Leach site) with an excavator to provide a detailed visual examination of near surface soil, groundwater, permafrost, and bedrock conditions (refer to Figure 18.2).



Source: Ausenco, 2018

Figure 18.2: Josemaría field test pit location map

As part of a hydrogeological investigation vertical electrical sounding (VES) were performed in July 2014 by TEOTOP. Several geophysical lines were performed, including along the Pirca de Los Bueyes stream that included VES points JM6, JM7, JM8 and JM15. Geophysics showed that the depth of the alluvium along the center of the tailings facility varied from 66 m to 172 m to bedrock, while at the Main tailings embankment, the depth to bedrock was approximately 70 m.

18.8.2 Infrastructure

The Josemaría project infrastructure is situated on alluvium and colluvium typically less than 5 m thick that is underlain by weathered bedrock, except in the bottom of the TSF valley where the alluvium ranges from 66 to 172 m along the center of the valley.

18.8.3 Groundwater

Groundwater was encountered during the excavation in three of the test pits within the footprint of the tailings dam footprint. During the excavation there was flow in the Pirca de Los Bueyes stream, therefore the alluvium in the base of the valley was fully saturated. As part of the plan to mitigate near surface water within the footprint of the TSF dam, a geomembrane cut-off wall tied into bedrock along with an underdrain and seepage collection well system shall be installed to reduce or eliminate upward seepage from bedrock into the base of the TSF embankment.

18.8.4 Permafrost

Based on the 11 test pits across the site ranging in elevation from 4,800 to 4,000 masl, no permafrost was encountered at this site. Therefore, based on the field investigation there should be no permafrost in the area of the waste rock storage facilities, TSF and other mine infrastructure.

18.9 Tailings Storage

18.9.1 Introduction

The principal objective of the TSF design is to ensure protection of the regional water during operations and closure, while containing solid waste materials within a geotechnically stable engineered facility. The anticipated tailings stream from the concentrator will be at a typical solids throughput of approximately 150 ktpd (after ramp-up and ramp-down) over 20 years. The tailings will be discharged as a thickened slurry with a solids content of about 62% by weight. The tailings stream to the TSF will be directly discharged to the impoundment at various locations around the facility to maximize the storage capacity of the TSF. The tailings will also be utilized to provide a low-permeability seal on the alluvium base through strategic deposition practices to ensure the supernatant pond is over tailings covering the alluvial basin.

The center of the TSF is located 4 km southeast of the concentrator. Due to the favourable topography in the TSF area providing a natural basin for the impoundment of tailings, only one embankment is initially required in the southeast end of Pirca de Los Bueyes Basin with two smaller embankments required later in the north and south to provide containment of the tailings

within this basin. The TSF has been sized to provide sufficient capacity to store approximately 1,008 Mt of tailings based on the mine and concentrator production schedules.

The ultimate facility occupies an area of approximately 8.7 Mm², including the tailings embankments. The TSF has additional capacity to expand to an estimated maximum storage capacity of approximately 1,500 Mt, with appropriate increases in embankment heights.

18.9.2 Site Characteristics

Topography and Surface Hydrology

See Section 5 for description of the topography and surface hydrology in the area of the TSF.

Rainfall and Evaporation

The information and analyses provided in Section 20.3.1 were used in the design of the TSF.

Geology

The TSF impoundment and embankments foundations consist of unconsolidated and medium-consolidated alluvial deposits, colluvial deposits and rock outcrop on the side slopes of the valley. The alluvial thickness ranges from 0 to 172 m within the impoundment and 0 to 60 m within the main TSF embankment. Approximately 100% of the alluvial layer is underlain by low-permeability bedrock. There is no evidence of karstification within the TSF footprint based on the site geology.

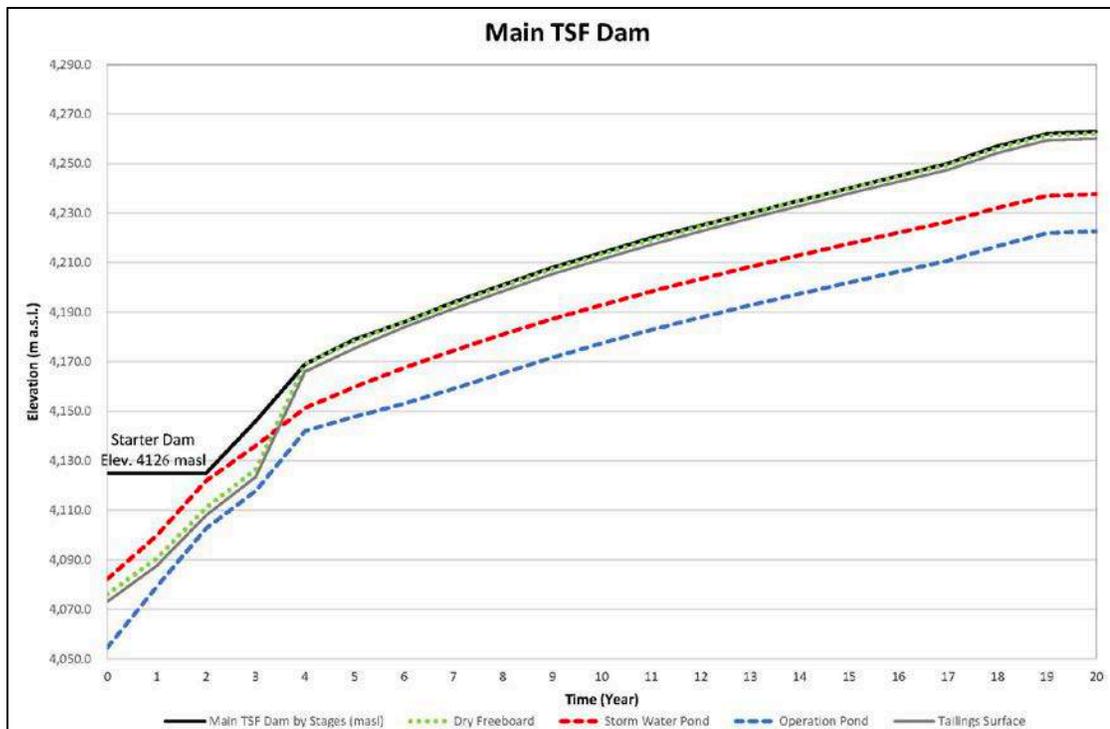
18.9.3 Tailings Characteristics

The ore will go through a grinding circuit and floatation process producing tailings (waste) with a P₈₀ equal to 150 microns. The specific gravity of the tailings is approximately 2.70 based on laboratory testing. The tailings will be thickened to approximately 62% solids and then the tailings will flow by gravity or be pumped to the TSF located approximately 4 km to the south. Tailings will be deposited from several points around the TSF. The in-situ density will range from 1.45 to 1.55 t/m³ during the life of project.

18.9.4 Design Basis and Assumptions

A TSF siting study was performed, and the chose site was selected as the preferred location because of its proximity to the plant and lower estimated capital costs. After the capital costs were evaluated in more detail during November 2018, Ausenco developed a high-density slurry tailings with centerline line raise construction of the main embankment after starter facility. Ausenco developed a TSF design that follows both international and national (Argentina) standards and was based on the following design and operating requirements:

- Production Rate
 - Mining and milling production at an average rate of 55 Mtpa, not including ramp-up
- TSF Capacity
 - Tailings will be stored in a TSF sized to contain 1,008 Mt and the design storm event of “one third between 1,000 years and the Probable Maximum Flood” event, and will be constructed continuously from Years 1 through 19
- Embankment
 - The TSF impoundment requires three embankments to contain tailings over the life of the project. The main tailings embankment will be constructed using centerline construction after the initial starter embankment to minimize initial capital and subsequent downstream raises, and using material borrowed from within the impoundment and near the plant (low permeability material). The smaller north and south embankments will be constructed using downstream construction methods.
- Stability
 - The tailings embankment is to be stable and designed to the standards consistent with the appropriate hazard classification (CDA 2014) and modern embankment (dam) engineering practice. The Earthquake Design Ground Motion (EDGM) has an annual exceedance probability (AEP) of 1:2,475.
- Density
 - For the development of the TSF filling schedule, an average tailings dry density of 1.45 t/m³ was assumed for the first two years of production and a conservative density of 1.55 t/m³ from Years 3 to 20 (end of operations). The stage storage curve is shown in Figure 18.3.
- Deposition
 - Conventional tailings slurry will be discharged from spigots located on the embankments and other strategic locations to develop a tailings beaches, and push the corresponding supernatant pool away from the main TSF embankment after the initial year
- Tailings beach slope
 - A beach slope of 1.0% away from the embankments was assumed for the design, based on a tailings percent solids of approximately 62%



Source: Ausenco, 2018

Figure 18.3: Stage storage curve

- ARD Potential
 - Based on Knight Piésold’s understanding of the tailings geochemistry, the tailings are likely to be PAG. To minimize seepage below the limits of the TSF, a liner system was considered down to bedrock along with a seepage collection well system installed downstream of the main embankment that will capture seepage above the bedrock/alluvial interface in the center of the valley. During deposition, the tailings will be alkaline due to the addition of lime to the process circuit. Additional testwork is required to better understand the short- and long-term geochemistry of the tailings and the effect of the addition of lime along with any additional mitigation measures that may be required during or after operations.
- Supernatant
 - Supernatant will be reclaimed and returned back to the mill using a barge system, which will provide a significant portion of the process water for the mill
- Spillway
 - A spillway will be constructed and sized to convey run-off from the attenuated PMP event

18.9.5 TSF Layout, Development and Operating Strategy

General

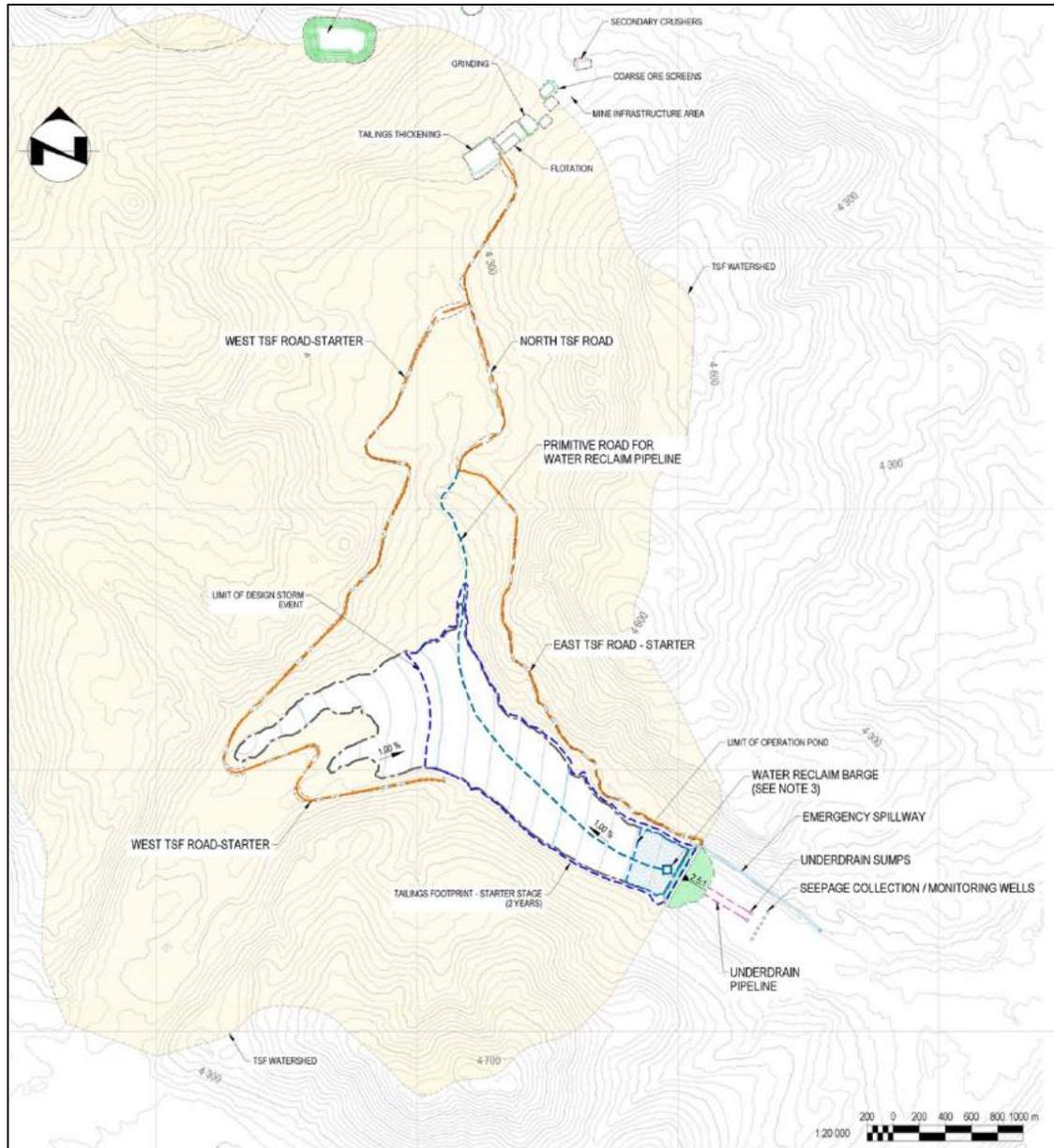
Tailings will be deposited in an impoundment located in the Pirca de Los Bueyes basin, southeast of the concentrator. The overall features of the TSF for Year 2 and ultimate mine life are reflected in Figure 18.4 and Figure 18.5.

- The three alluvium shell starter embankments, referred to as the Main, North and South embankments, include geomembrane, low permeability soil liner and filter zone on the upstream sides and the main shells constructed of alluvium
- Tailings distribution pipelines – spigot discharge points around the TSF
- Reclaim water systems – barge with supernatant pipeline to the plant
- Supernatant (surface water) pond
- Seepage management system – underdrain, and seepage/collection and monitoring wells located downstream of the main embankment

Embankments

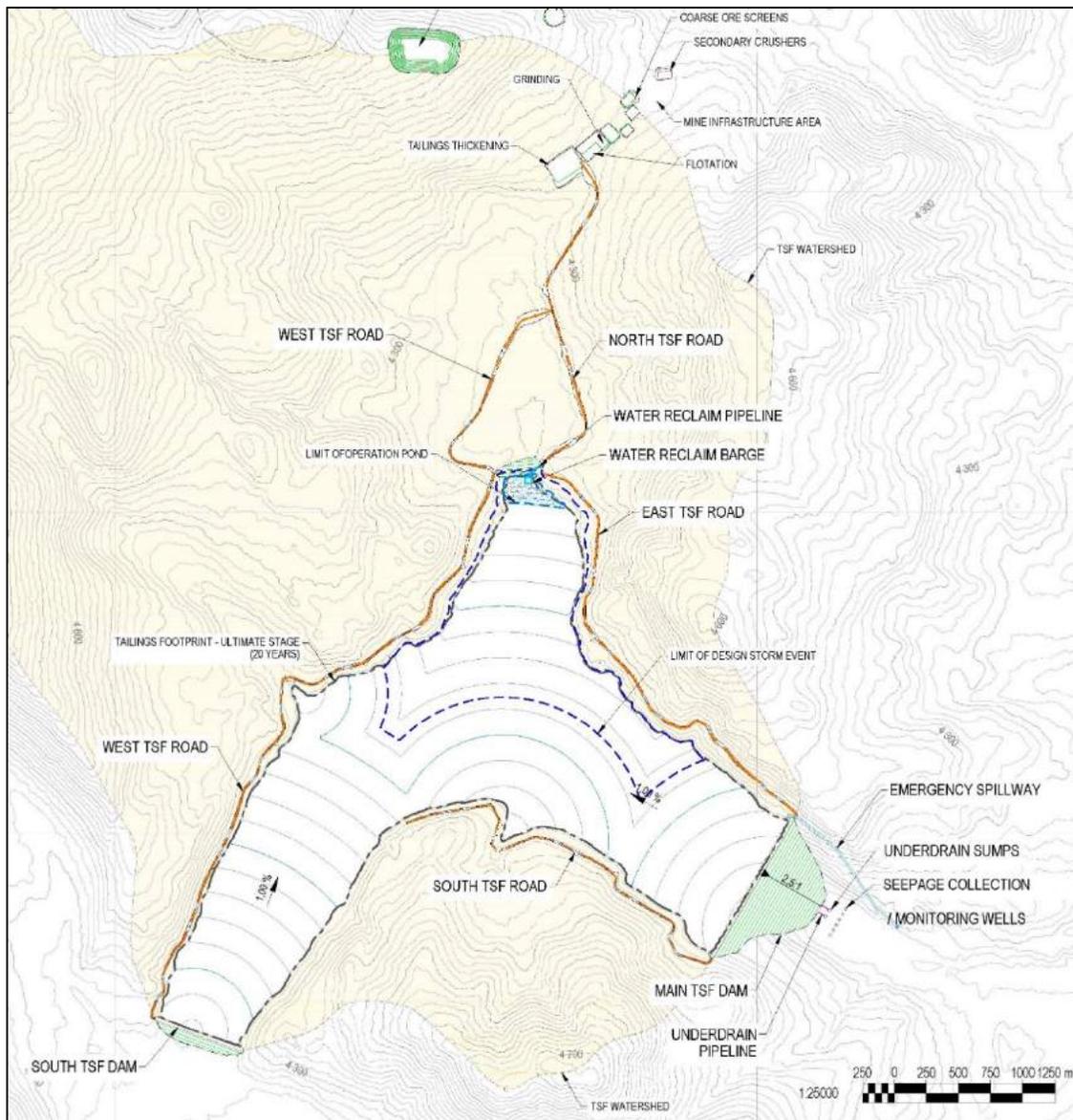
The TSF embankments will be developed in stages throughout the life of the TSF using a combination of suitable alluvium borrow materials from the impoundment and downstream of the embankments (Main and South) and upstream (North) of the embankment.

The Main embankment will be developed across Pirca de Los Bueyes stream. The alluvium-fill starter embankment is designed to accommodate tailings production, the maximum operating pond, and the design storm event, and two metres of freeboard for the first two years of operations and will be approximately 78 m high (crest to downstream toe). The embankment shell zones for the starter embankment will be constructed with random fill comprising suitable alluvium from local borrow sources. The upstream side will be covered with geomembrane, low permeability soil liner and a filter zone down to bedrock as a cut-off barrier to reduce seepage downstream.



Source: Ausenco, 2018

Figure 18.4: TSF tailings deposition Year 2



Source: Ausenco, 2018

Figure 18.5: Ultimate TSF tailings deposition with north and south embankments

Ongoing embankment raises will comprise centreline raises with alluvium along with a geomembrane liner, low permeability soil liner and filter zone. The maximum height of the final embankment (crest to downstream toe) will be approximately 215 m.

The South embankment will be required in Year 10 of operations with construction similar to that of the Main embankment, except that due to the depth to bedrock, it is not cost effective to create a seepage cut-off barrier. For the South embankment, in addition to the upstream geomembrane there will be a filter zone between the soil liner and alluvium with reliance on using tailings to seal the adjacent alluvium and walls of the impoundment with tailings. Ongoing embankment raises

will be developed similarly to the southwest embankment. The ultimate embankment is required to a maximum height of approximately 47 m.

The North embankment will be required in Year 12 of operations with construction similar to that of the South embankment. The ultimate embankment is required to a maximum height of approximately 51 m.

Borrow Materials

All embankment construction materials will be sourced from the TSF impoundment or downstream, except the low permeability soil. The alluvium is expected to be sourced relatively easily from the impoundment based on the field geotechnical and laboratory programs. Some mechanical separation may be required to remove oversized material (i.e. greater than 0.6 m) along with screening for the filter materials. The low permeability soil will be sourced from the plant site area, which has sufficient volumes for both the starter and ultimate embankments.

Water Management

A monthly operational mine site water balance, incorporating stochastic analysis for climate conditions, indicates that the TSF will be in a water deficit condition during the mine operating life. To satisfy mill requirements a make-up water supply system will be required. In addition, a spillway will be constructed for the starter facility and subsequent raises to safely pass storm events greater than the IDF of 1:2,475-year return period. However, the tailings facility should be able to store the PMF.

Surface water from the TSF watershed will be captured in diversion channels along the TSF access roads and conveyed to the TSF along with natural stream channel in the TSF basin to reduce water make-up requirements.

Tailings Distribution

A single stream of whole tailings will discharge into the TSF from multiple discharge points around the facility.

The discharged thickened tailings will settle rapidly, forming subaerial beaches with 1% slope, while below the supernatant pond, tailings beaches will form at steeper slopes, approximately 4 to 6% depending on the material. A deposition strategy will be developed and managed with the intent to seal off the alluvium with tailings, maximize storage efficiency in the TSF and to maintain the deeper, cleaner locations of the supernatant pond in the vicinity of the reclaim barges.

Water Reclaim System

Water will be reclaimed from the tailings decant and operations pond by floating, barge-mounted pump-stations, which will report to the process plant. The floating pump-stations will initially be located near the main embankment and gradually move west as the beach near the embankment is established. In the final years, the barge will be located near the north embankment.

Water reclaimed from the TSF will consist of supernatant from the settled tailings, as well as runoff from precipitation, snowmelt and ice melt, from within the TSF catchment area. Dedicated pipelines will convey reclaimed water to the process plant located up-gradient of the TSF.

18.9.6 TSF Seepage Management

Seepage within the foundation and along the main embankment of the TSF has been identified as a risk due to the permeable nature of the alluvium and lack of hydrodynamic containment. Engineered and operational measures to reduce seepage risks include geomembrane lining of the starter embankment and down to bedrock along with a primary and secondary seepage collection systems.

No primary and secondary seepage collection system are required for the South and North embankments.

TSF Embankment Stability Analysis

Stability analyses were carried out to investigate the stability of the main, south and north embankments under both static and seismic loading conditions. These comprised checking the stability of the embankment arrangement for each of the following cases:

- Static and pseudo-static conditions during operations and post-closure
- Earthquake loading from the MDE of 1:2,475 AEP

The stability analyses were based on a typical cross-section through the embankments for the final TSF with a supernatant pond elevation of 4,177 m and the main embankment crest elevation of 4,255 m. The results of the stability analyses satisfy the minimum requirements for factors of safety and indicate that the proposed design is adequate to maintain both short-term (operational) and long-term (post-closure) stability. The seismic analyses indicate that any embankment deformations during earthquake loading from the MDE would be minor and would not have any significant impact on embankment freeboard or result in any loss of embankment integrity. The results also indicate that the embankment is not dependent on tailings strength to maintain overall stability and integrity.

18.9.7 Instrumentation

Geotechnical instrumentation, comprising vibrating wire piezometers, slope inclinometers, internal settlement instrumentation, and exterior movement (survey) monuments will be installed at selected planes along the embankments to monitor embankment performance. Groundwater wells will also be installed at suitable locations downstream of the embankments to monitor groundwater quality.

18.9.8 Closure and Reclamation

The primary objective of the closure and reclamation initiatives will be to eventually return the TSF site to a self-sustaining facility with pre-mining usage and capability. The TSF will be required to maintain long-term stability, protect the downstream environment and manage surface water.

Upon mine closure, surface facilities will be removed in stages and full reclamation of the TSF will be initiated. General aspects of the closure plan include:

- Selective discharge of tailings around the facility during the final years of operations to establish a final tailings surface and water pond that will facilitate post closure surface water management and reclamation
- Dismantling and removal of the tailings and reclaim delivery systems and all pipelines, structures and equipment not required beyond mine closure
- Construction of an overflow spillway and channel to allow surface water discharge downstream of the TSF
- Removal and re-grading of all access roads, ditches and borrow areas not required beyond mine closure for the TSF if applicable
- Long-term stabilization of all exposed erodible materials
- During the closure phase of the project, suitable alluvium will be placed on the beach surface after tailings deposition ends to minimize dusting potential. Measures may also include tailings stabilization (e.g., adding an agent to create a trafficable crust) in the final year of operation.

18.10 Site Wide Water Balance

The site water balance used a deterministic model developed in GoldSim® on a monthly timestep basis for the life of mine operations to determine make-up water requirements. Three precipitation scenarios were considered in the model: dry, average and wet year cycles.

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, but integrates the flows between different mine components such as the open pit, process plant, TSF pond, underdrain sumps and seepage collection wells.

Due to the dry conditions inherent to the arid climate of the project area, the TSF will be used as an on-site water storage pond, which will also collect the watershed runoffs, including extreme events. The TSF, is therefore, designed with sufficient storage capacity to avoid any discharges to the environment.

Available information such as hydrogeology, hydrology, and the climate data were used in the model (PFS level). Production ramp-up, thickening improvement, accumulative storage, and tailings deposition plan were included in the model.

The main components for the site water balance are:

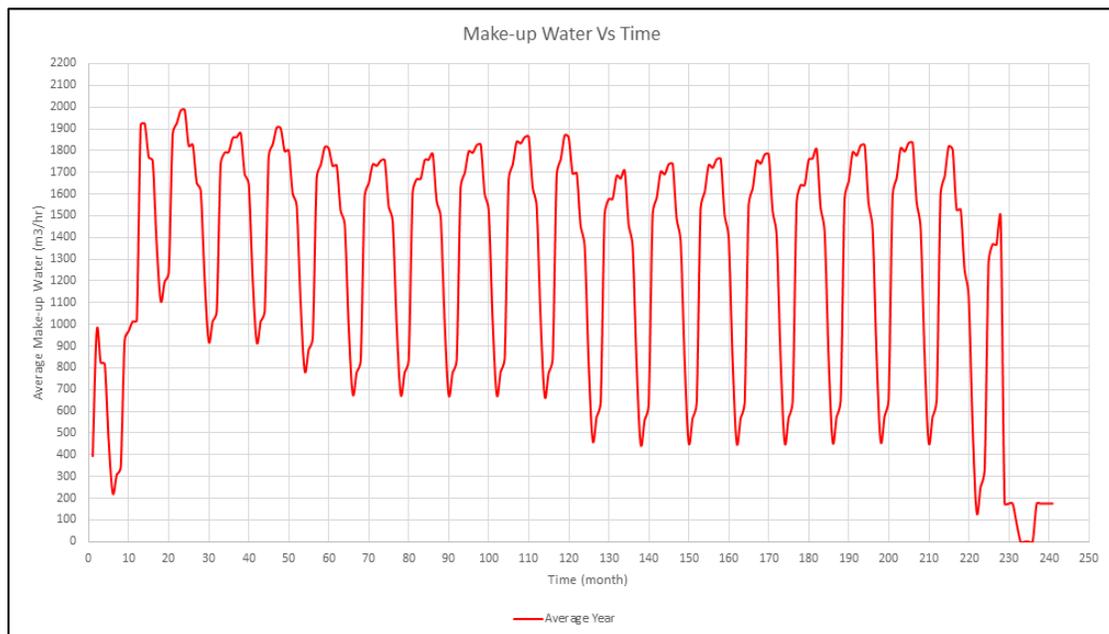
- Process plant
- Tailings storage facility

- Open pit dewatering
- Seepage collection wells below TSF
- Surface water diversion works

The following conclusions were derived from the site water balance:

- In the average and wet cycle wet season, the TSF is able to provide the entire amount of water required by the process plant. Therefore, the minimum pump capacity of 1,400 L/s is recommended for the TSF reclaim water barge.
- The make-up water requirements based on the site water balance are:
 - Dry yearly precipitation cycle
 - Dry season is 562 L/s, wet season is 48 L/s and average is 451 L/s
 - Wet yearly precipitation cycle
 - Dry season is 558 L/s, wet season is 138 L/s and average is 221 L/s
 - Average yearly precipitation cycle
 - Dry season is 555 L/s, wet season is 0 L/s and average is 353 L/s

The mine make-up water requirements for an average year are summarized in see Figure 18.6.



Source: Ausenco, 2018

Figure 18.6: Mine average year make-up water requirements

The seepage analysis estimated that the TSF seepage rate would be 15 L/s. However, the flow from the seepage collection wells was designed to capture 30 L/s in the model. If the flow rate exceeds this limit, it must be pumped directly back to the plant as make-up water to avoid losses.

In Years 1 and 20, the make-up requirements are significantly less than the other years due to ramp-up of production throughput and decline in production throughput at the end of the mine life. Fluctuations in maximum and minimum make-up water requirement from Years 2 through 19 are due to minor metrological and production throughput variations.

18.11 Surface Water Management

A number of water control structures have been designed for surface water management in 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 support water for process are:

- Recycling water used for processing ore to reduce the volume of water demand for process
- Intercepting and diverting surface water from entering the mine site by building diversion channels to reduce the potential for water contact with exposed ore and waste
- Capturing runoff and directing the water to the TSF to reduce off-site water supply requirements
- Impounding as much ephemeral runoff volume as possible in water retaining ponds
- Collect contact water from the WSF in a sediment collection pond as part of a zero-release program. The pond will be sized to collect and evaporate contact water or pump excess water to the plant or TSF.

It will be necessary to alter the current flow path of surface water flows to reduce the potential for harm to infrastructure or to minimize the potential for mixing clean water with runoff from disturbed sites.

Surface runoff that can be intercepted and directed by the diversion works will be considered to be non-contact water. Any water stream 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 and diverted to the TSF.

The surface runoff diversion works for the management of non-contact water consist of diversion channels, perimeter channels, 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 runoff from the road platform and the basins they transect and direct it to the nearest discharge points; transverse drainage enables the downstream discharge of flows intercepted by the channels, or unimpeded flows in the large stream drainages. Transverse drainage consists of culverts and low-water crossings facilities.

19 Market Studies and Contracts

The product of the mine will be a conventional copper concentrate with a concentrate grade forecast to average 25.1% Cu over the life-of-mine. This is generally considered to be marketable in a conventional manner, with fixed price per tonne of concentrate assumed for treatment. No deleterious elements are forecast to be present in the concentrate and no penalties have been modelled. On an annual basis, the grade varies between 20.4% and 27.0%. However, only 5% of the total concentrate shipped is expected to be below 23%. This occurs right at the end of the mine life, and any issues associated with marketability of this low concentration would not have a material impact on project value.

Gold grades are expected to average 14.5 grams per tonne of concentrate. Silver grades are expected to average 54.2 grams per tonne of concentrate.

The concentrate parameters have been forecast based on testwork described in Section 13.

No contracts in relation to concentrating, smelting, refining, transportation, handling, sales and hedging, and forward sales, nor any other marketing arrangements are currently in place. Pursuant to the terms of NGEx's acquisition of its previous partner's (JOGMEC) 40% interest in the Josemaría project announced on November 13, 2017, JOGMEC holds an option to purchase up to 40% of the material produced from any mine on the property based on the prevailing market price for the material.

The concentrate is to be trucked to the existing concentrate export port of Caldera on the Chilean coast and exported to smelters in Asia.

Logistics for concentrate transport are discussed in Section 18.2. Costs and commercial terms are shown in Table 19.1.

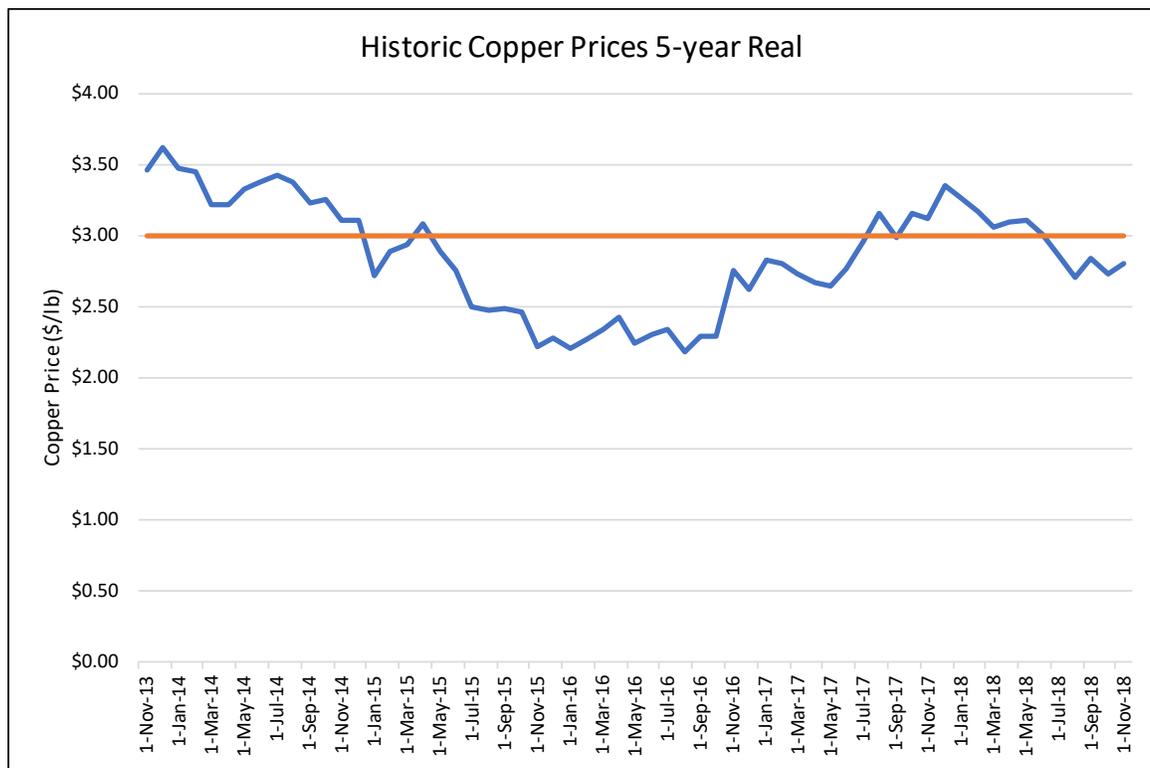
Table 19.1: Concentrate freight and TC/RC assumptions

Category	Units	Costs
Concentrate Freight		
Road Freight	\$/wmt	\$51.45
Port and Handling	\$/wmt	\$22.50
Ocean Freight	\$/wmt	\$52.50
Total Freight Charges	\$/wmt	\$126.45
Treatment and Refining Charges		
Treatment Charges	\$/dmt	\$75.00
Losses	%	0.25%
Copper Refining Charge	\$/lb	\$0.08
Gold Refining Charge	\$/oz	\$5.00
Silver Refining Charge	\$/oz	\$0.30
Penalties	\$/dmt	None modelled

The price assumptions used for this study are shown in Table 19.2. 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). SRK also considers the prices used in this study to be consistent with the range of prices being used for other project studies.

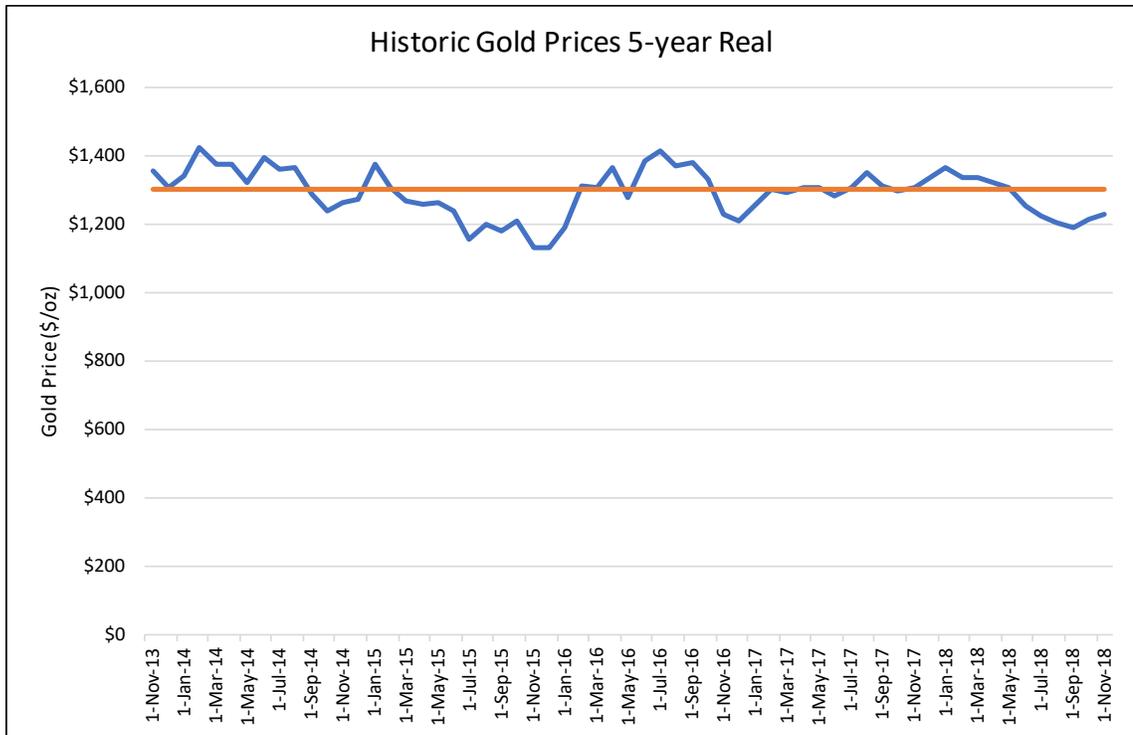
Table 19.2: Commodity price assumptions

Commodity	Units	Price
Copper Price	\$/lb	\$3.00
Gold Price	\$/oz	\$1,300
Silver Price	\$/oz	\$20.00



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

20 Environmental Studies, Permitting, and Social or Community Impact

20.1 Introduction

NGEx has made considerable efforts to undertake environmental studies and community engagement to facilitate the advancement of the Josemaría 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 legal framework for mine permitting in Argentina is derived mainly of 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 Josemaría 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 5 March 2018 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 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

20.3 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 Josemaría project in April 2014, located at an elevation of 4,448 masl; however, data for this station are only available starting in late January 2015. Additionally, two other climate stations were installed close to the project, at the neighboring Los Helados and Filo del Sol projects. The Los Helados climate station is located at an elevation of 4,974 masl and was installed in late January 2015. The Filo del Sol climate station is located at an elevation of 5,012 masl 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 Josemaría 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 Josemaría climate station.

There are several climate stations managed by Dirección General de Aguas (DGA) in Chile, as well as Servicio Meteorológico Nacional (SMN) and Instituto Nacional de Tecnología Agropecuaria (INTA) in Argentina, that either are operating or have operated in the regional vicinity of the project area. All of the regional stations are located at elevations in excess of 2,000 m lower than the project, and as such, have climate conditions considerably different. 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 Josemaría climate station. The Lautaro Embalse climate station is located approximately 66 km northwest of the project at an elevation of 1,110 masl.

Temperature

Mean, minimum, and maximum temperatures were recorded at the Josemaría station on an hourly basis from January 2015 to January 2018. 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 are 12.0 °C and -25.6 °C, respectively.

To develop a long-term temperature series for the project, concurrent temperature data for the Josemaría climate station and the most representative regional climate stations were analysed to assess the suitability of the regional climate data as predictors of climatic conditions. 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 -2.1 °C, with monthly mean temperatures ranging from a high of 7.3 °C in January 2017 to a low of -21.4 °C in June 1978.

Table 20.1: Monthly mean and annual mean temperature (°C) with synthesized data set

Station Name	Year	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
Josemaría	1973	0.2	-0.5	-0.8	-4.2	-6.5	-10.4	-12.4	-7.4	-1.8	-0.5	3.1	2.8	-3.2
	1974	3.9	2.3	1.6	-1.9	-3.6	-8.9	-6.9	-5.1	-7.2	-2.4	1.2	1.2	-2.1
	1975	3.6	3.1	0.9	-1.0	-4.6	-4.9	-7.7	-8.2	-6.6	-2.2	-2.3	1.2	-2.4
	1976	2.9	1.6	1.5	-2.2	-4.8	-10.1	-6.6	-8.6	-4.9	-1.6	0.5	4.2	-2.3
	1977	4.7	-2.7	3.9	-0.4	-2.3	-4.2	-8.8	-6.1	-2.5	-1.8	1.4	4.5	-1.2
	1978	4.1	2.3	0.9	-1.7	-6.0	-21.4	-10.8	-7.1	-5.9	0.3	2.1	3.2	-3.3
	1979	3.5	2.2	-0.9	-2.0	-3.0	-6.5	-5.1	-0.4	-3.6	1.5	2.3	6.0	-0.5
	1980	6.1	3.3	4.7	-0.4	-2.9	-6.7	-6.5	-4.2	-2.8	-3.5	0.9	3.9	-0.7
	1981	4.0	5.5	5.9	-1.2	-2.8	-4.7	-5.1	-2.9	-1.2	-0.4	1.9	4.7	0.3
	1982	2.8	2.8	0.5	-1.6	-5.1	-6.9	-3.5	-1.3	-3.4	-0.1	1.9	2.3	-1.0
	1983	4.4	4.7	3.6	1.2	-5.9	-11.8	-9.7	-6.7	-7.1	0.5	1.1	2.6	-1.9
	1984	3.1	3.3	2.0	-1.0	-6.1	-10.0	-7.0	-7.8	-4.2	-1.1	-0.4	1.9	-2.3
	1985	1.3	2.9	1.4	-1.3	-3.3	-3.4	-10.0	-7.0	-4.0	-0.3	0.0	1.5	-1.9
	1986	2.9	1.9	1.2	-0.9	-4.0	-6.6	-2.5	-4.1	-2.5	-0.3	1.4	4.1	-0.8
	1987	4.2	5.4	2.1	0.4	-8.1	-3.0	-11.2	-5.5	-3.3	-1.1	1.6	3.2	-1.3
	1988	3.2	4.1	3.1	0.3	-3.6	-7.1	-9.4	-4.5	-7.4	-1.0	0.5	0.8	-1.8
	1989	2.3	3.5	0.7	-3.1	-4.4	-5.9	-7.9	-4.5	-7.3	-0.7	0.2	1.8	-2.1
	1990	3.6	2.6	1.6	-2.3	-3.2	-3.2	-6.8	-4.9	-3.9	-3.5	0.2	1.0	-1.6
	1991	1.6	2.2	1.5	-0.4	-2.1	-6.5	-6.6	-7.1	-2.8	-3.1	0.9	0.4	-1.8
	1992	3.8	2.8	1.2	-2.6	-5.1	-7.7	-6.3	-5.5	-4.1	-1.0	0.3	1.6	-1.9
1993	2.8	1.6	1.1	-0.8	-5.1	-4.4	-8.0	-4.0	-6.2	-2.8	-0.7	0.8	-2.1	
1994	2.0	1.9	0.7	-0.3	-3.6	-5.8	-8.2	-4.9	-0.7	-3.4	-1.4	-0.1	-2.0	
1995	-0.1	0.3	-0.4	-2.3	-3.5	-4.2	-9.7	-7.7	-4.9	-3.3	-1.5	-0.1	-3.1	
1996	-0.5	1.2	-0.5	-6.1	-6.3	-7.8	-5.6	-6.9	-4.0	-2.9	0.0	-1.4	-3.4	
1997	2.1	2.8	0.9	-2.0	-4.0	-9.8	-5.0	-3.7	-2.2	-3.2		3.2		
1998	5.7	4.0	2.2	-2.8	-4.6	-6.6	-5.7	-7.6	-6.9	-2.3	-1.5	0.2	-2.2	
1999	0.3	3.2	0.7	-2.9	-5.0	-7.4	-8.5	-5.3	-5.3	-4.3	-2.6	-0.5	-3.1	

Station Name	Year	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
	2000	1.3	1.6	0.0	-4.0	-6.3	-9.1	-6.5	-5.9	-5.7	-1.7	-2.1	1.2	-3.1
	2001	-0.1	1.7	-0.3	-3.1	-7.7	-8.3	-5.0	-4.2	-7.1	-2.1	-1.8	1.8	-3.0
	2002	2.9	3.8	3.8	-2.1	-3.8	-6.6	-4.9	-2.5	-3.0	-1.5	0.0	0.0	-1.2
	2003	2.2	3.2	1.3	-2.3	-1.6	-5.2	-5.6	-3.1	-3.6	-1.1	-1.5	-0.5	-1.5
	2004	-0.8	-0.9	-1.6	-4.9	-8.0	-5.3	-5.5	-7.6	-2.8	-4.0	-2.1	-1.5	-3.7
	2005	-2.4	-1.8	-3.3	-2.9	-9.6	-3.4	-7.4	-1.8	-5.8	-3.6	0.6	1.2	-3.4
	2006	3.6	4.3	2.1	-1.0	-2.3	-3.4	-3.7	-2.8	-3.1	-2.5	-1.0	0.5	-0.8
	2007	0.2	-0.8	-2.2	-3.0	-9.6	-12.0	-8.2	-13.0	-6.4	-5.4	-4.1	-2.0	-5.5
	2008	0.3	-0.6	-0.1	-4.2	-6.6	-8.2	-8.4	-6.3	-6.1	-5.0	-3.4	1.3	-3.9
	2009	2.8	3.8	2.0	-0.1	-3.2	-5.1	-7.8	-3.3	-5.5	-1.3	0.5	1.8	-1.3
	2010	3.3	2.9	1.2	-1.2	-5.5	-8.4	-11.0	-4.5	-4.2	-3.4	-1.7	-0.9	-2.8
	2011	1.3	2.4	-0.6	-2.7	-3.7	-9.8	-9.7	-8.1	-2.2	-6.5	-0.9	0.8	-3.3
	2012	1.7	3.3	3.2	-2.8	-3.4	-5.9	-7.7	-8.0	-3.5	-3.7	-1.2	0.8	-2.3
	2013	3.1	2.7	1.8	-2.5	-6.7	-6.5	-5.8	-4.5	-4.5	-2.5	-1.7	0.5	-2.2
	2014	2.0	2.0	-0.6	-2.4	-6.8	-8.5	-6.1	-3.3	-5.5	0.4	-1.2	0.7	-2.4
	2015	2.2	5.0	4.1	1.0	-2.0	-1.5	-5.4	-5.2	-4.0	-3.4	-1.9	2.8	-0.7
	2016	5.6	6.9	4.1	-0.6	-3.9	-6.3	-6.6	-3.0	0.3	-1.9	1.1	4.2	0.0
	2017	7.3	3.4	3.2	0.3	-5.8	-5.0	-2.4	-5.0	-2.4	-2.5	1.0	4.4	-0.3
	2018	3.8												
	Mean	2.6	2.5	1.3	-1.8	-4.8	-7.0	-7.1	-5.4	-4.3	-2.1	-0.2	1.6	-2.1
	Minimum	-2.4	-2.7	-3.3	-6.1	-9.6	-21.4	-12.4	-13.0	-7.4	-6.5	-4.1	-2.0	-5.5
	Maximum	7.3	6.9	5.9	1.2	-1.6	-1.5	-2.4	-0.4	0.3	1.5	3.1	6.0	0.3

Wind

The mean annual wind speed calculated from the three years of record (2015-2017) at the site is 4.7 m/s. An average monthly low wind speed of 2.1 m/s was measured at Josemaría in January 2017, and an average monthly high wind speed of 7.3 m/s was measured in May 2016. The wind was calm (less than 1 m/s) for only 3.5% of the time, while wind speeds exceeded 7.5 m/s approximately 16% of the time. Winds are just as likely to occur at any time of day, and that wind speeds are fairly consistent throughout the day, although tend to be highest in the late afternoon and evening. The prevailing wind direction throughout all seasons is from the south, the west, and the northwest, with the strongest winds generally from the northwest and the weakest winds generally from the south. The site is consistently windy, both in terms of the frequency and the intensity of the wind. The maximum instantaneous wind speed was 18.7 m/s. The monthly mean wind speeds are typically greatest during the winter months (May to October) and lowest during the summer months (December to March).

Evaporation

Monthly Potential Evapotranspiration (PET) values were estimated for Josemaría 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

Meteorological Station	Elevation	Method	Year	Evapotranspiration (mm)														
				Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual		
Josemaría	4448 m	Hargreaves equation	2015		82	79	45	23	19	10	14	19	33	51	81			
			2016	98	93	70	33	12	6	4	27	49	46	69	92	598		
			2017	114	74	64	42	13	10	21	17	36	42	68	98	597		
			2018	92														
			Mean	101	83	71	40	16	12	12	19	34	40	63	91	582		
		Thornthwaite equation	2015		95	77	34	0	0	0	0	0	0	0	0	71		
			2016	81	87	58	0	0	0	0	0	14	0	32	67	339		
			2017	100	62	54	13	0	0	0	0	0	0	33	74	336		
			2018															
			Mean	91	81	63	16	0	0	0	0	5	0	22	71	347		
		Penman-Monteith equation	2015		101	89	68	56	62	47	50	62	78	96	119			
			2016	122	112	100	66	50	34	45	58	82	85	103	121	979		
			2017	123	94	98	75	41	43	61	60	73	88	104	-	861		
			2018	51														
			Mean	99	102	96	70	49	46	51	56	72	84	101	120	946		
		Average of All Three Methods				97	89	77	42	22	19	21	25	37	41	62	94	625

The PET values presented in Table 20.2 vary considerably, particularly in terms of the mean annual values, with a range of 347 to 946 mm. The average values of all three sets of estimates indicate a mean annual PET of 625 mm, with an average monthly low of 19 mm during the month of June, and an average monthly high of 97 mm during the month of January. It is important to note that both the Hargreaves and Penman-Monteith PET annual totals include values during months when temperatures are well below zero, so these are essentially equal to potential sublimation values.

Precipitation

Precipitation at Josemaría is an infrequent occurrence, with very little occurring in dry years, and only a few substantial events providing the majority of the total in wet years. Precipitation data are available for the Josemaría, Filo del Sol, and Los Helados climate stations. The precipitation record for the Josemaría station demonstrates an average annual value of the three-year precipitation record for Josemaría of 252 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 109 mm, annual values ranging from a low of 0 mm to a high of 612 mm, and monthly values ranging from 0 mm (many occurrences) to 316 mm (June 1997). 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)

Station Name	Year	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
Josemaría	1967	0	0	0	0	0	0	0	0	0	0	0	9	9
	1968	0	0	0	0	0	0	0	0	0	0	0	0	0
	1969	0	0	0	0	0	1	0	63	0	0	0	0	63
	1970	0	0	0	0	1	0	0	0	0	0	0	0	1
	1971	25	0	0	0	0	66	0	12	0	0	0	0	103
	1972	0	0	0	0	0	0	0	0	0	0	0	0	0
	1973	0	0	0	6	0	46	0	0	0	0	0	0	52
	1974	0	0	0	0	0	0	0	0	24	0	0	0	24
	1975	0	0	21	0	60	36	0	0	0	0	0	0	116
	1976	0	0	0	0	48	0	0	13	3	0	0	0	64
	1977	0	0	0	46	0	0	36	0	0	0	0	0	82
	1978	0	0	0	0	0	0	0	0	0	0	0	0	0
	1979	0	0	6	0	0	0	0	0	7	0	0	0	13
	1980	0	0	0	66	0	0	57	21	25	12	4	0	185
	1981	0	0	0	0	0	0	0	0	60	0	0	0	60
	1982	0	0	0	0	0	0	15	0	0	4	0	0	19
	1983	0	0	0	9	45	179	30	118	0	0	0	0	380
	1984	0	0	39	0	0	22	204	0	0	0	0	0	265
	1985	0	0	0	0	0	0	24	1	0	0	0	0	25
	1986	0	0	0	0	0	30	0	7	45	0	1	0	83
1987	0	4	0	0	0	27	0	280	60	1	6	0	378	
1988	0	0	0	0	0	9	0	0	0	7	0	0	16	

Table 20.3: Long-term synthetic monthly and annual total precipitation (mm) - continued

Station Name	Year	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
Josemaría	1989	0	0	0	1	0	0	7	121	0	0	0	0	130
	1990	0	0	0	0	0	0	31	0	0	0	0	0	31
	1991	0	0	0	0	0	240	31	0	0	0	0	21	292
	1992	0	0	25	24	136	82	0	0	0	0	0	0	267
	1993	0	49	0	0	0	0	4	33	0	0	0	0	86
	1994	0	10	0	0	0	0	0	0	0	0	0	0	10
	1995	0	0	0	0	0	0	0	0	3	0	0	0	3
	1996	0	0	0	0	0	0	0	4	0	0	0	0	4
	1997	0	0	1	0	0	316	0	292	3	0	0	0	612
	1998	0	0	0	0	0	16	3	0	0	0	0	0	19
	1999	0	0	55	0	0	3	1	0	0	25	0	0	85
	2000	0	0	0	1	71	144	16	0	0	0	0	0	234
	2001	0	0	22	0	0	0	0	1	0	0	0	0	24
	2002	0	0	0	36	60	0	83	77	0	0	0	0	256
	2003	0	0	0	0	0	0	0	0	0	0	0	0	0
	2004	0	0	13	0	0	0	77	0	0	0	0	0	91
	2005	0	0	0	46	0	0	30	18	0	0	0	0	94
	2006	0	0	0	0	0	0	3	0	0	0	0	0	3
	2007	0	0	0	0	0	15	4	0	0	0	0	0	19
	2008	0	0	0	0	0	13	1	39	0	18	0	0	71
2009	0	0	0	0	0	9	0	31	0	0	0	0	40	
2010	0	0	0	0	0	136	0	0	4	34	0	0	174	

Table 20.3: Long-term synthetic monthly and annual total precipitation (mm) – continued

Station Name	Year	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
Josemaría	2011	0	0	0	1	0	15	93	0	0	0	0	0	110
	2012	0	0	0	45	0	0	0	0	0	0	0	0	45
	2013	0	0	0	0	70	0	22	0	0	0	0	0	92
	2014	0	0	0	0	70	3	0	0	18	0	0	0	91
	2015	0	0	108	0	12	0	51	138	10	30	0	0	349
	2016	0	0	2	16	10	65	36	19	0	1	2	8	158
	2017	9	13	0	3	77	36	0	10	3	24	66	2	242
	2018	6												
	Mean	1	2	6	6	18	25	23	22	3	2	1	1	109
	Minimum	0	0	0	0	0	0	0	0	0	0	0	0	0
	Maximum	25	49	108	66	136	316	280	292	34	30	66	21	612

A set of 24-hour extreme precipitation estimates were completed. Since extreme precipitation events generally occur during El Niño years, and these events have a periodicity of approximately five years (McPhaden, 2003), the daily maximum precipitation (DMP) recorded at Josemaría between 2015 and 2017, which is 47 mm, was assumed to be a reasonable estimate of the 5-year return period daily extreme precipitation event. This value was converted to an equivalent 24-hour extreme value, and the values for the 24-hour extreme events having return periods of 10, 20, 50, 100, and 200 were estimated. The 50-year, 100-year and 200-year 24-hour values for the site are estimated to be 112 mm, 129 mm, and 146 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 were 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.

To understand the cryosphere appropriately, NGEx contracted BGC Ingeniería 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.

20.3.4 Hydrology

The project sits at the upper boundaries of both the Upper Rio Blanco and the Upper Arroyo Pircas de los Bueyes watersheds. The Arroyo Pircas de los Bueyes watershed flows into the Macho Muerto River, which ultimately feeds into the Rio Blanco, which in turn drains to the Rio Jáchal, one of the principal rivers of San Juan province in Argentina.

A summary of streamflow studies is provided in Knight Piésold (2018a). The mean unit runoff varies substantially in the region. 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 climate events.

The measured average runoff in the Upper Rio Blanco River is 0.293 m³/s, which corresponds to a unit runoff of 5.05 L/s/km² (160 mm). This value is based on a limited number of point discharge measurements, so it is not representative of average annual conditions, but it is generally consistent with the limited regional data, and it supports the conclusion that runoff in the project area is low.

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

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. Three hundred drill core samples were retrieved from site and sent to the SGS Canada Inc. laboratory in Burnaby Canada.

The analytical test program consisted of standard static testing that to provide for an assessment of the potential for acid generation (via acid-base accounting) and metal content (via ICP-MS). The potential for metal leaching of specific metals was examined through leach extractions.

The static test program identified that total sulphur is variable in the mine rock samples, with values ranging from <0.0050% to 8.8%. For the majority of samples, sulphide sulphur is the primary form of sulphur. However, sulphate sulphur was present in some lithologies.

The neutralizing potential (NP) is generally low for all mine rock samples. Those with higher NP were derived from total inorganic carbon (e.g., calcite). At lower NP values, a significant portion of the NP is associated with other mineral (e.g., aluminosilicates).

The majority of the mine rock samples (88%) were classified as Potentially Acid Generating (PAG). This is due to both the high sulphur content and the low NP of the mine rock. The PAG samples were identified in each of the units tested, with few exceptions.

A sub-set of samples have also been submitted for mineralogy (QEMSCAN and petrography) and kinetic testing to quantify the rates of sulphide oxidation, neutralization potential depletion and metal release rates of wastes. The kinetic testing program includes a laboratory-based humidity cell test program and shake flask extraction tests. Elevated trace elements that were identified in the static program include Ag, As, Cd, Cu, Mn, Mo, and Se. These elements may be of potential concern in leachate and will be considered further when the shake flask extraction and the humidity cell results become available, which is expected in 2019.

Based on the geochemical program to date, all waste is assumed to be PAG, so water management, waste rock handling, and the tailings storage facility have been designed accordingly. 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.

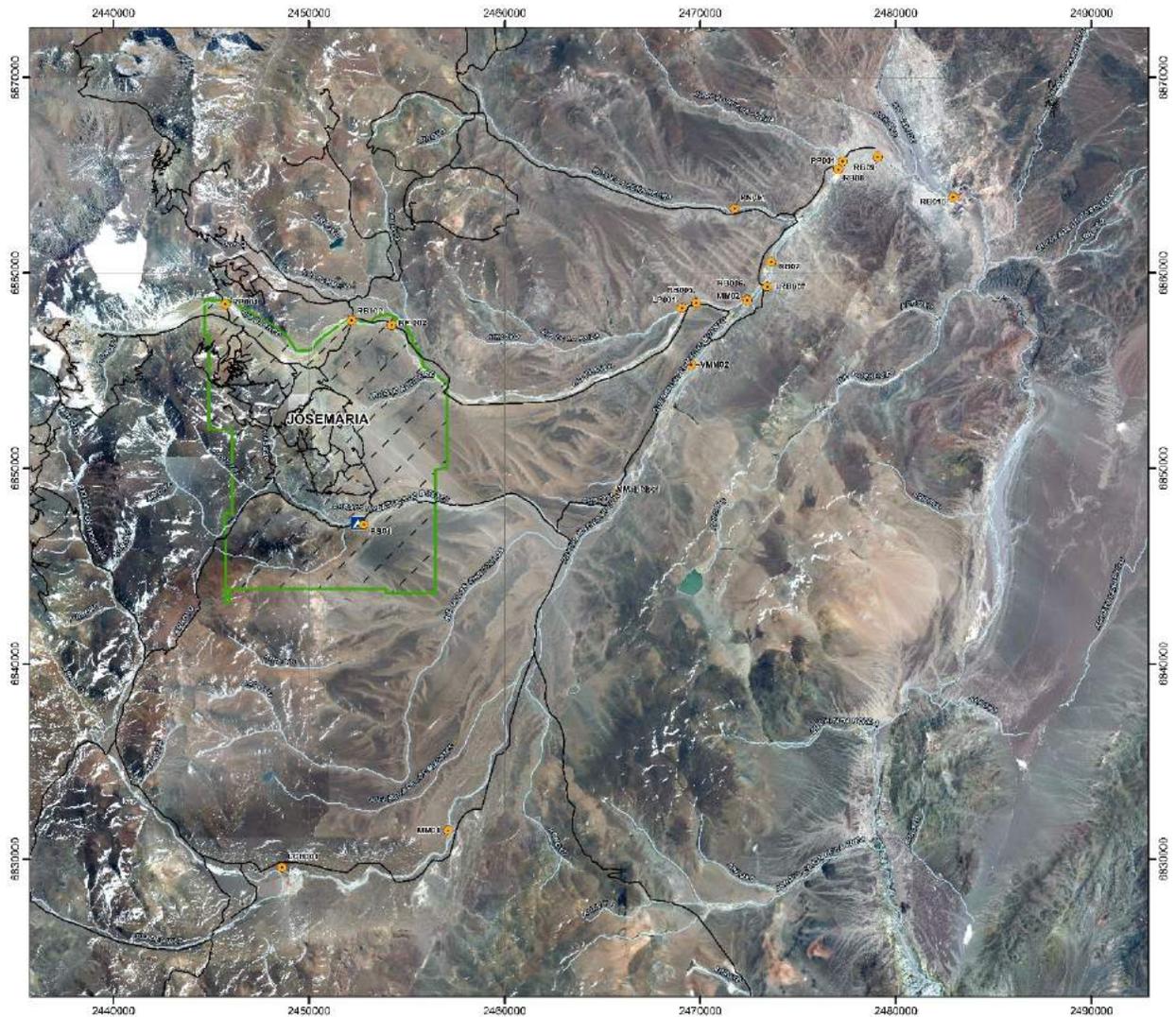
20.3.6 Water Quality and Aquatic Biota

A focused study program for the Josemaría project was carried out by Knight Piésold (2018b), 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 are shown on Figure 20.1.

Results indicate that waters in the upper Rio Blanco are acidic, with pH values averaging 4.5. The pH values increased at lower elevations, becoming alkaline (up to pH 8.4) downstream of its confluence with the Rio Macho Muerto. Elevated metals were similarly found in the upper watershed, including aluminum, arsenic, barium, beryllium, boron, cadmium, zinc, cobalt, cadmium, iron, manganese, and vanadium. Metals concentrations decreased downstream.

Water samples from Arroyo Pircas de Bueyes and Rio Macho Muerto had neutral pH, and generally low concentrations of metals, with the exception of arsenic and iron.

Species richness of invertebrates was very low in the upper Rio Blanco. Invertebrates were found in much higher abundance in the Arroyo Pircas de Bueyes, with highest densities identified in the lower Rio Macho Muerto. As would be expected, higher densities were correlated with lower elevations.



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. Soils were very acidic in the upper Rio Blanco basin, while those within the Pirca de Bueyes catchment were moderately basic. 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 (2018c) conducted surveys at the project for vegetation and wildlife, which complemented an earlier study from 2013, which included several adjoining mineral concessions, including the Josemaría project (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 was observed above 4,700 masl. Wetlands or vegas are found in valley bottoms 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*.



Credit: Knight Piésold, 2018

Figure 20.2: Typical steppe habitat dominated by *Stipa spp* grasses



Credit: Knight Piésold, 2018

Figure 20.3: Vega in the upper Río Pirca de Los Bueyes dominated by *Oxychloe Castellanosii*

Faunal diversity was represented by 31 bird species, 7 mammal species, and 1 species of reptile. The highest abundance of wildlife was associated with vega habitat. This included several waterfowl species, passerine birds, and small mammals. Groups of guanaco and vicuña were noted along the access road corridors.

Several 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 federal Ministry of the Environment and Sustainable Development (Secretaría de Ambiente y Desarrollo Sustentable). All plants present in the project area that are tracked by “PlanEARr” are considered abundant, although restricted in their distribution.

The 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 (*Bute polyosoma*)
 - Aplomad falcon (*Falco femoralis*)
 - Peregrine falcon (*Falco peregrinus*)
 - Darwin's Rhea (*Pterocnemia pennata garleppi*)
 - Andean Condor (*Vultur gryphus*)
- Mammals:
 - Vicuña (*Vicugna vicugna*)
 - Puma (*Puma concolor*)

These species are restricted in their trade and are a focus for protection.

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, Manzanares, 2015, Knight Piésold Consulting, 2018d). 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.

Forty-nine archaeological sites were identified within the project area, of which 35 were within the concession. The sites were generally composed of rock formations (circles, semi-circles, or walls), with some associated with lithic material. Almost 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 4,300 masl. A smaller percentage is located in hills and plateaus from where there is a high visibility of the environment and watersheds. These strategic locations could be related to hunting of camelids.

Project design will avoid direct impacts to archaeological sites where possible. Where impacts cannot be avoided, the identified site will be studied 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, NGEx 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

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.

Access from within Argentina will also be required for the project, which transits remote mountain roads before rejoining the highway network at the town of Guandacol. This area is very sparsely populated, with no settlements or homesteads for more than 100 km from the project to the nearest property. 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.

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 NGEx 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 Guandaco, located approximately 150 km from the project area, and have focused on road maintenance contracts and employment.

Increased interaction with the communities and implementation of formalized engagement is planned to be concomitant with feasibility level studies.

20.4.3 Indigenous Populations

No indigenous people have been identified in the Argentine project area, including along its electric transmission or access corridors.

20.5 Waste and Tailings Disposal

Waste rock storage designs were developed by SRK, while tailings disposal designs were developed by Ausenco. These are more fully described in Sections 16 and 18, respectively, of this report. A summary of the approach to tailings management as provided by Ausenco is summarized below.

20.5.1 Tailings Storage Facility

Tailings will be stored as a thickened slurry in a purpose-built TSF within the upper Pircas de Los Bueyes catchment. Tailings are expected to be delivered to the TSF with a solids content of 65% by weight, at a rate of 150 ktpd. Tailings will be discharged to the TSF at various locations around the facility. Containment will be achieved by one main embankment, with two smaller embankments placed at the upstream limits.

The TSF has been sized to provide sufficient capacity to store approximately 1,008 Mt of tailings within a footprint area of approximately 8.7 Mm², including the tailings embankments. The TSF has additional capacity to expand to an estimated maximum storage capacity of approximately 1,500 Mt, with appropriate increases in embankment heights.

20.5.2 Site Selection

Satellite imagery was reviewed to identify potential locations for a TSF in the vicinity of the proposed process plant location at Josemaría. Six potential sites were identified for the PFS. The potential site selected for the mine design took into account the tailings transport distance, pumping elevations, resulting basin surface area and environmental risks. Each parameter was assessed in a matrix ranking to determine the preferred alternative.

20.5.3 Design Parameters

The TSF has a design capacity of 1,008 Mt of tailings with an average dry density of 1.45 t/m³ for the first two years of production and 1.55 t/m³ from thereon. The TSF will be able to store or safely pass an IDF of one third between a 1,000-year return period and the Probable Maximum Flood event, as per CDA Guidelines. The embankment design will incorporate protection from ground movement of a MDE with an exceedance probability of 1:2,500 years.

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 and 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 tailings storage facility 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; however, responsible closure planning has been considered. A provisional closure plan will be included with the Mine EIA submission. 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 tailings facility 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

- Selective discharge of tailings around the facility during the final years of operations to establish a final tailings surface and water pond that will facilitate post closure surface water management and reclamation.
- Placement of suitable alluvium on the tailings beach surface after tailings deposition ends to minimize dusting potential
- Dismantling and removal of the tailings and reclaim delivery systems, and all pipelines, structures and equipment not required beyond mine closure.
- Construction of an overflow spillway and channel to allow surface water discharge downstream of the TSF.
- 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 five years, with a further five years of monitoring for a total 10-year closure period.

A detailed closure cost will be developed to support the Mine EIA submission. Based on the foregoing, a preliminary estimate of approximately \$150 million 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 (USD)	Total (USD 000)
Direct Costs				
Dismantling of equipment and structures, demolition of structure and foundations	ha	38	1,500,000	57,000
Access control / safety berm around pit	m	3,540	200	708
Retraining waste rock storage diversion ditches to the pit	km	1	500,000	500
Contouring and placement of sand / alluvium on TSF surface	ha	900	18,000	16,200
Scarification and contouring of the footprint	km	50	6,000	300
Dismantling of electrical transmission line	km	250	5,000	1,250
Detailed closure engineering and planning	study	1	900,000	900
Active closure monitoring	year	5	200,000	1,000
Post closure monitoring	year	5	200,000	1,000
Misc. (Waste management and disposal, specialist contracts, etc.)	lump sum	1	3,800,000	3,800
SUBTOTAL DIRECT COSTS				82,658
Indirect Costs				
Contractor Fees (25% of Direct Costs)			25%	20,665
Administration (15% of Direct Costs)			15%	12,399
SUBTOTAL OF INDIRECT COSTS				33,063
SUBTOTAL				115,721
Contingency at 30%				34,716
TOTAL CLOSURE COST (USD)				150,438

21 Capital and Operating Costs

21.1 Capital Cost Estimate

21.1.1 Estimate Classification

The estimate has been prepared in accordance with Ausenco's PFS standards. These are consistent with the recommended practices of the American Association of Cost Engineers (AACE) for a Class 4 estimate and have an accuracy range of +/-25%. This estimate includes the cost to complete the detailed design, procurement, construction and commissioning of all the required facilities.

This estimate collectively presents the entire costs for Josemaría project including SRK's mining scope, Ausenco's process and infrastructure scope and Owner's (NGEx) scope. The physical facilities and utilities for the estimate include, but are not limited to:

- SRK:
 - mine development
 - mine dewatering
 - mining equipment
 - mine explosives magazine
- Ausenco:
 - mine workshop/administration/warehouse building (including change rooms)
 - process plant
 - on-site infrastructure
 - off-site infrastructure
 - tailing storage facilities
 - port upgrades
- NGEx:
 - Owner's costs

21.1.2 Estimating Methodology

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, including the installation/construction hours, unit labour rates and contractor distributable costs,

bulk and miscellaneous material and equipment costs, any subcontractor costs, freight and growth.

Mining

Summary of Assumptions for Estimate

SRK's derivation of the mine operations capital cost estimate is based on the following information sources:

- Equipment quotations (Komatsu S.A.)
- SRK experience and benchmark costs

Pre-Production Capitalized Operating Costs

Pre-production mining consists of the following activities:

- Establishing main haul roads – to ROM pad, low-grade stockpile, and waste dumps
- Pre-strip mining to release first ore – some waste is used to build platform for low-grade stockpile, rest initiates the South and West WSFs
- Incidental ore mining and stockpiling – minor amounts of ore encountered prior to mill start-up

Process and Infrastructure

The estimate has been based on the traditional EPCM approach where the EPCM contractor will oversee the delivery of the completed project from detailed engineering and procurement to handover of working facility. The EPCM contractor shall engage and coordinate several subcontractors to complete all work within the given scopes. Typical vertical and/or horizontal contract packages are identified and aligned with different pricing models such as, but not limited to:

Schedule of Rates (unit price)

The contract pricing model is based on estimated quantities of items included in the scope and their unit prices. The final contract price is dependent on the quantities needed to complete the work under the contract

Time & Materials

Time and material fixes rates for labour and material expenditures, with the contractor paid on the basis of actual labour hours (time), usually at specified hourly rates, actual cost of materials and equipment usage, and an agreed upon fixed add-on to cover the contractor's overheads and profit

Design and Construct

With this option, one entity will provide design and construction services for an awarded scope of work. A higher degree of price certainty can be achieved when a lump sum arrangement is used. This method also provides a single point of accountability and an improved integration of the design with construction.

Work Breakdown Structure (WBS)

The estimate has been arranged by major area, area, major facility, and facility. Each sub-area has been further broken down into disciplines such as earthworks, concrete, etc. Each discipline line item is defined into resources such as labour, materials, equipment, etc., so that each line comprises all the elements required in each task.

The WBS has been developed in sufficient detail to provide the required level of confidence and accuracy and also to provide the basis for further development as the project moves into execution phase.

21.1.3 Definition of Costs

The estimate is broken out into 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.

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. Contractor's indirect costs are contained within each discipline's all-in rates.

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.

21.1.4 Exchange Rates and Foreign Content

The exchange rates used in the estimate are shown in Table 21.1 and have been determined from the XE website as of 14 September 2018 and are applied to foreign currency data.

Table 21.1: 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 = 0.89 EUR
GPB	Great Britain Pound	1 USD = 0.77 GBP
ARS	Argentine Peso	1 USD = 39.90 ARS
AUD	Australian Dollar	1 USD = 1.40 AUD
CLP	Chilean Peso	1 USD = 684.93 CLP

Table 21.2 identifies the foreign priced content and USD priced content.

Table 21.2: Foreign and USD content

Country	Initial Capex (\$M) (excl. contingency)	% of Costs (excl. contingency)
United States priced content	2,096	91.7%
Canadian priced content	13	0.6%
European priced content	69	3.0%
Great Britain priced content	0	0.0%
Argentine priced content	70	3.0%
Australian priced content	38	1.7%
Chilean priced content	0	0.0%
Total – Directs and Indirects (less contingency)	2,286	100.0%

21.1.5 Market Availability

The pricing and delivery information for quoted equipment, material and services was provided by suppliers based on the market conditions and expectations applicable at the time of developing the estimate.

The market conditions are susceptible to the impact of demand and availability at the time of purchase and could result in variations in the supply conditions. The estimate in this report is

based on information provided by suppliers and assumes there are no problems associated with the supply and availability of equipment and services during the execution phase.

21.1.6 Estimate Summary

The estimate is derived from a number of fundamental assumptions as shown on the process flow diagrams, drawings, scope definition and WBS. It includes all associated infrastructure as defined within the scope of works.

The capital cost estimate has been summarized in Table 21.3 and is stated in United States dollars (USD) with a base date of Q4 2018 and with no provision for forward escalation. Please note that Mine related costs (WBS 1000) in Table 21.3 also include approximately \$49 million in capitalized Opex (pre-production) and \$62 million for the construction of a truck shop.

Table 21.3: Capital cost estimate summary by major area (WBS Level 1)

Cost Type	WBS LVL 1	LVL 1 Description	Initial (USD \$000)	Sustaining (USD \$000)	Total (USD \$000)
Direct	1000	Mine	262,990	254,917	517,907
	2000	Crushing	485,806	0	485,806
	3000	Process	456,618	0	456,618
	4000	On-site infrastructure	162,654	513,634	676,288
	5000	Off-site infrastructure	279,641	0	279,641
Direct Subtotal			1,647,709	768,551	2,416,260
Indirect	6000	Indirects	309,557	66,389	375,946
	7000	Project delivery	245,228	0	245,228
	8000	Owners costs	83,494	0	83,494
	9000	Provisions	474,657	25,492	500,149
		Indirect Total	1,112,937	91,881	1,204,818
PROJECT TOTAL			2,760,646	860,432	3,621,078

A more detailed breakdown of the initial capital cost estimate is provided in Table 21.4.

Table 21.4: Initial capital cost estimate Summary Level 2

Cost Type	WBS LVL 2	LVL 2 Description	Total (\$000)
Direct	1100	Pre-production (Capitalized Opex)	49,334
	1300	Mining equipment	151,592
	1400	Ancillary services	62,063
	2100	Primary crushing	110,858
	2200	Course ore stockpile & reclaim	97,987
	2300	Secondary crushing	61,690
	2400	Tertiary crushing	137,727
	2500	Tertiary screening	77,544
	3100	Grinding	211,190
	3200	Flotation & regrind	129,771
	3300	Concentrate thickening	6,019
	3400	Concentrate filtration, storage, loadout	19,603
	3500	Tailings thickening	39,210
	3600	Reagents	9,455
	3700	Plant services	30,676
	3800	Plant common	10,694
	4100	Site development	239
	4200	Tailings storage facility	110,581
	4300	Power supply & distribution	38,531
	4400	Yard utilities	2,805
	4500	Administration buildings	1,194
	4600	Plant buildings	4,408
	4700	Mobile equipment	4,896
	5100	Off-site roads	4,322
	5200	Power supply	138,029
	5300	Water supply	62,291
	5400	Port logistics	75,000
Subtotal Direct Costs			1,647,709

Cost Type	WBS LVL 2	LVL 2 Description	Total (\$000)
Indirect	6100	Field indirects	98,306
	6200	Heavy lift cranes	6,000
	6300	Construction accommodation & messing	143,066
	6400	Vendor installation assistance	6,051
	6500	Pre-commissioning & commissioning	16,547
	6600	Spares and first fills	39,588
	7100	EPCM	245,228
	8100	Owner's costs	83,494
	9100	Contingency	474,657
Subtotal Indirect Costs			1,112,937
PROJECT TOTAL			2,760,646

Mine Equipment Capital Costs

The mine equipment capital cost is estimated for both primary and ancillary equipment. The primary equipment includes items such as drills, loading equipment, haul trucks, track dozers, and graders. The ancillary equipment includes light vehicles, service vehicles, and dewatering/lighting equipment.

The primary equipment requirement estimate is based on the mine schedule quantities, determinations of productivities and therefore equipment requirements. Costs are derived from vendor quotations.

The ancillary equipment capital cost estimate is based on assessed requirements and SRK benchmark cost information.

The capital cost estimate for mine equipment is summarized in Table 21.5. New equipment costing has been assumed.

Erection, commissioning and training costs for the primary mine equipment was estimated at 6.5%, and spare parts for the same was estimated at 3.0% of initial unit purchases. Freight on all mine equipment was estimated at 4.0%. Contingency on the equipment costs was assigned at 10%.

Table 21.5: Mine equipment capital cost summary with no contingency

Item	Total Cost (\$000)	Initial Cost (\$000)
Primary		
<u>Production</u>	<u>329,383</u>	<u>115,239</u>
Crawler-Mounted, Rotary, 381-mm Dia.	22,014	9,434
Crawler-Mounted, Rotary, 171-mm Dia.	2,274	2,274
Electric, 36-m ³ Hydraulic Shovel	26,842	26,842
Diesel 20-m ³ Wheel Loader	18,096	9,048
290-t class AHS Truck	260,157	67,641
<u>Support</u>	<u>32,625</u>	<u>15,203</u>
Track Dozer, 5.3-m blade	15,444	5,940
Backhoe, 3.8-m ³ bucket	553	553
834H-class 4.6-m blade	1,909	1,909
Grader, 4.9-m blade	8,212	2,464
Water Truck, 135-t class	6,506	4,337
Subtotal Primary	362,007	130,442
Ancillary		
Dewatering and Lighting	893	319
Small Earthmoving	1,806	1,806
Portable Crusher/Screening	686	686
Moving Equipment	1,962	1,962
Service/Maintenance	6,391	3,196
Light Vehicles	13,951	2,790
Communications and Control	4,736	4,736
Subtotal Ancillary	30,425	15,495
Total Equipment Purchase	392,432	145,937
Erection/Commissioning	23,530	8,479
<u>Equipment Indirects</u>		
Freight	14,480	5,218
Spares	1,222	1,222
Total Equipment	431,665	160,856

Mine Infrastructure Capital Costs

Mine infrastructure is limited for the Josemaría project. The explosives plant and associated garage facilities will be required; however, explosive loading is a contracted service, and the explosives provider will be responsible for the construction of these facilities. NGEx would be responsible for preparatory earthworks (see Miscellaneous below).

Refueling stations are to be included in the maintenance area adjacent the pit and are included in the maintenance area capital cost estimate.

Miscellaneous and Other Indirect Mine Capital Costs

Miscellaneous mine capital expenditures include:

- Earthworks for the explosives plant and related accesses - \$750,000
- Survey equipment and software - \$60,000
- Geology/Mine Planning software - \$100,000

Indirect costs related to future mining studies, including detail implementation design, have not been considered, presumed to be sunk costs at the time of project commencement.

Contingency for miscellaneous expenditures is estimated 20%.

Summary of Mining Capital Costs

The summary of mining capital costs is provided in Table 21.6.

“Operating costs” are estimated for pre-production activities in the SRK mine cost model, but because they occur before mill start-up, they are capitalized in the economic model. The pre-production activities take place over 12 months at a cost of \$49.3 million, before contingency.

Table 21.6: Summary of mining capital costs without contingency

Item	Initial Purchase (# units)	Sustaining Capital (# units)	Total (# units)	Total Cost (\$000)	Initial Cost (\$000)
Pre-Production Mining				49,334	49,334
Mine Equipment	91	208	299	431,665	160,856
Miscellaneous				910	910
Mining Indirects				0	0
Total Mining Capital				481,909	211,100

21.2 Operating Cost Estimate

The processing plant throughput is designed to operate at approximately 150 ktpd. The annual throughput is expected to be 54.75 Mt over a 20-year mine life.

An average operating cost was estimated based on the proposed mine schedule. This cost includes mining, processing, G&A, surface services and shipping of product. The average LOM operating cost is estimated to be \$6.74/t milled.

21.2.1 Common Cost Inputs

The following common cost inputs were used in estimating operating costs for the Josemaría project:

- Diesel fuel costs of \$0.77/L
- Electricity cost of \$0.075/kWh
- 32.5% benefits burden on wages

All operating cost estimates are deemed to be accurate to within +/-25%.

21.2.2 Mining Operating Cost Estimate

As with the capital cost estimate, SRK's derivation of the mine operations operating cost estimate is based on the following information sources:

- Equipment quotations (Komatsu S.A)
- Service quotations (Orica-Argentina/Bolivia)
- SRK experience and benchmark costs

Where appropriate, equipment costs were aligned with other contributors to the PFS.

Mine Operating Input Data

The following key inputs were used to develop the mine operating costs:

- For mine operations, a two week on, one week off, 12-hour shift roster was assumed, with no scheduled shutdown time (i.e. 365-day year)
- Three crews fulfill the shift roster
- 5% freight on parts & consumables
- Exchange rate - 39.9 Argentine pesos to one USD

Labour Rates

The labour rates used for mine and maintenance hourly personnel were derived from previous Argentine project experience, which was current up to the end of 2016. In alignment with the Argentine Consumer Price Index, the labour rates from that time were escalated 65% to Q3 2018. The project labour rates, in USD, are provided in Table 21.7 and Table 21.8.

Table 21.7: Labour rates for hourly personnel

Mine Operations		Mine Maintenance	
Role	Base Wage (\$ / hr)	Role	Base Wage (\$ / hr)
Driller, blasthole	11.9	Heavy Eqmt Mechanic	11.9
Driller Helper, blasthole	7.39	Welder/Mechanic	11.9
Shovel/Loader Operator	11.9	Electrician/Instrument	13.41
Track/Wheel Dozer Operator	10.28	Lubeman/PM Mechanic	8.77
Backhoe Operator	10.28	Tireman	11.9
Grader Operator	10.28	Light Vehicle Mechanic	11.9
Water/Tow Truck Driver	8.77	Labourer/Trainee	5.64
Labourer/Trainee	7.39	VSA Trades	11.9
VSA* Operator	10.28	VSA Labourer	5.64
VSA* Labourer/Trainee	7.39		

* VSA – “vacation-sickness-absenteeism” allowance of 15%

The unburdened salaries for mine management and supervisory staff were also based on previous Argentine project experience. The estimated salaries are provided in Table 21.8.

Table 21.8: Salaries for staff (unburdened)

Role	Annual Salary (\$)
Mine Manager	138,000
Mine General Foreman	126,000
Mine Foremen	110,000
Trainer	48,000
Maintenance Superintendent	126,000
Maintenance General Foreman	91,000
Senior Maintenance Planner	37,000
Shift Maintenance Planner	32,000
Maintenance Shift Supervisors	73,000
Mechanical Engineer	37,000
Warehouse Supervisor	39,000
Mine Buyer	26,000
Mtc/Purchasing Clerk	20,000
Chief Engineer	126,000
Senior Mining Engineer	91,000
Mining Engineer	73,000
Drill & Blast Engineer	73,000
Mine Geologist	56,000
Technician/Ore Control	32,000
Surveyor	37,000
Secretary/Clerk	20,000

Equipment Costs

Equipment costs, based on parts and consumables, fuel and maintenance labour are provided in Table 21.9.

Table 21.9: Mine equipment unit costs

Description	Cost (\$/hour)
Crawler-Mounted, Rotary, 381-mm Dia.	193.15
Crawler-Mounted, Rotary, 171-mm Dia.	165.6
Diesel, 36-m ³ Front Shovel	741.86
Diesel 20-m ³ Wheel Loader	398.15
Diesel, 3.8-m ³ Backhoe	193.15
290-t class Haul Truck	326.49
Track Dozer, 5.3-m blade	143.01
Wheel Dozer, 4.6-m blade	118.43
Grader, 4.9-m blade	99.25
Backhoe, 3.8-m ³ bucket	55.65
Water Truck, 135-t	236.77

Mine Operating Costs by Activity

The mine operating costs are categorized by mining activity and are listed in Table 21.10 in terms of total dollars over the life-of-mine plan (after pre-production) and relevant unit costs (also after pre-production).

Table 21.10: Mine operating costs by activity

Activity	Total Cost (\$000)	Unit Cost (\$/t)
Drilling	152,218	0.09
Blasting	553,695	0.33
Loading	303,773	0.18
Hauling	1,364,687	0.81
Support	239,920	0.14
Mine General	43,592	0.03
Mine Administration	94,923	0.06
Total	2,752,807	1.63

Note that the operating costs include a 7% contingency.

The mine operating costs by year are provided in Table 21.11.

Table 21.11: Mining operating costs by year

Item	Units	Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
		Total																				
Direct Mill Feed	kt	975,872	26,552	51,201	50,526	53,992	51,258	53,126	53,203	53,809	53,260	52,357	53,014	53,612	53,698	53,353	53,149	53,890	54,414	54,750	39,647	7,061
To Stockpile	kt	32,137	7,444	8,165	5,173	8,147	1,610	0	0	0	0	0	0	0	0	0	0	0	0	739	0	0
Waste	kt	711,556	70,302	48,364	51,205	48,644	57,631	56,383	52,480	56,236	51,650	39,617	24,656	22,599	21,816	16,066	10,930	15,369	4,964	2,899	14,427	12,014
Total Mined	kt	1,719,565	104,298	107,730	106,904	110,783	110,499	109,509	105,683	110,045	104,911	91,974	77,670	76,210	75,514	69,419	64,080	69,259	59,378	58,388	54,074	19,074
Strip Ratio	w:t	0.7	2.1	0.8	0.9	0.8	1.1	1.1	1.0	1.0	1.0	0.8	0.5	0.4	0.4	0.3	0.2	0.3	0.1	0.1	0.4	1.7
Operating Costs																						
Drilling	\$000s	152,218	2,759	8,681	9,482	9,338	9,784	9,478	9,527	8,797	9,579	9,162	8,370	7,189	7,282	7,046	6,673	6,336	6,466	5,926	5,888	5,076
Blasting	\$000s	553,695	12,427	33,443	34,397	34,167	35,245	35,166	34,891	33,827	35,040	33,613	30,016	26,040	25,634	25,441	23,746	22,262	23,701	20,955	20,679	19,480
Loading	\$000s	303,773	6,467	18,380	19,464	19,443	19,489	19,948	19,507	18,837	19,453	18,668	16,709	14,061	13,613	13,536	12,613	11,796	12,514	10,843	10,601	10,519
Hauling	\$000s	1,364,687	18,086	52,584	59,392	64,701	72,460	66,154	73,059	73,334	72,908	64,213	60,386	60,906	66,113	69,110	67,083	69,721	82,911	82,898	83,699	82,490
Support	\$000s	239,920	5,798	12,509	12,972	13,184	13,473	12,816	13,228	12,950	13,010	12,324	11,895	11,482	11,620	11,733	11,522	11,503	12,132	11,882	11,871	11,747
General Mine/Mtce	\$000s	43,592	1,496	2,208	2,203	2,209	2,225	2,230	2,312	2,312	2,312	2,230	2,225	2,224	2,224	2,251	2,246	2,246	2,246	2,252	2,246	2,205
Supervision & Technical	\$000s	94,923	2,302	5,117	5,161	5,151	5,201	5,197	5,184	5,135	5,191	5,125	4,959	4,775	4,757	4,748	4,669	4,601	4,667	4,541	4,528	4,100
Total Operating Costs	\$000s	2,752,807	49,334	132,923	143,071	148,192	157,877	150,989	157,709	155,193	157,493	145,334	134,561	126,678	131,243	133,865	128,552	128,465	144,638	139,296	139,513	135,617

21.2.3 Processing Operating Cost Estimate

The LOM average process operating cost was estimated at \$3.23/t milled, based on expense types associated with a fixed throughput rate of 150 ktpd.

Basis of Estimate

Processing operating costs were estimated based on:

- Process flow diagram described in Section 17
- NGEx recommendations for fuel costs, electricity, and miscellaneous expenses, reviewed against Ausenco's database for reasonableness. Labour rates were established based on Ausenco's Argentina office information.
- Operating consumables are based on benchmarks from similar operations and from vendor information, and reagent usages are calculated based on SGS Phase II 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 the concentrator, TSF and water supply
- Labour for operating and maintaining concentrator, TSF and water supply including mobile equipment and light vehicles
- Fuel, reagents, consumables and maintenance materials for concentrator, TSF and water supply
- Fuel, lubricants, tyres and maintenance materials used in operating and maintaining the concentrator, TSF and water supply mobile equipment and light vehicles
- Operation of the TSF, including tailings and whole tailings slurry discharge management: dam raises, supernatant water reclaim, seepage water recovery and recycle of return water to concentrator process water
- 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 21.12 provides a summary of the source of cost data for each cost category.

Table 21.12: Processing operating cost data sources

Cost Category	Source of Cost Data
Processing labour	Salaries, wages and labour roster for processing were provided by both NGEx and Ausenco's Argentina office.
Reagents	Unit costs provided by suppliers, with estimated freight costs as percentage of the unit cost. Consumption rates based on SGS Phase II testwork.
Consumables	Unit prices provided by suppliers. Grinding media consumption rates were calculated based on the annual average abrasion index and 75 th percentile mill power for the scheduled ore. Mills and primary crusher liner consumption rates were estimated based on benchmarks from similar operations and from vendor information.
Power	Power costs for the mills were calculated using the 75 th percentile specific energy for each year of production. All other costs were calculated using load factors, operating hours per year and installed equipment power taken directly from the Mechanical Equipment List. The grid power cost of US\$0.075/kWh was supplied by NGEx.
Maintenance spares and consumables	Estimated at 4% of the mechanical plate work and equipment costs, and electrical and instrumentation 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.
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.

Summary

Operating cost estimates were prepared for processing by the following expense types:

- Power
- Reagents
- Consumables

- Maintenance
- Labour
- Mobile equipment

The main battery limits in the concentrator are from ore feed into the primary crusher dump pocket to tailings impounded in the TSF and concentrate load-out into road transport contractor trucks and raw water supply from the wells.

Table 21.13 provides summary processing costs per tonne.

Table 21.13: Processing operating cost summary

Areas	\$/t milled
Power	1.52
Consumables	0.82
Maintenance	0.66
Labour	0.23
Total Operating Cost	3.23

Power

Power was calculated by area, with installed power taken directly from the mechanical equipment list. Operating hours per year and equipment utilisation and load factors were used to calculate the total power usage in kWh per year. The installed power and average annual power consumption for each area is summarized in Table 21.14.

Table 21.14: Installed power and consumed annual power by area

WBS Code	Area Description	Total Installed Power (kW)	Total Consumed Power (kW)	Total Consumed Power (kWh/y)
2100	Primary Crushing	4,670	2,999	26,268,787
2200	Coarse Ore Stockpile & Reclaim	1,956	1,345	11,783,251
2300	Secondary Crushing	3,857	2,612	22,884,974
2400	Tertiary Crushing	30,661	21,413	187,581,034
2500	Tertiary Screening	665	449	3,936,078
3100	Grinding	84,166	58,838	515,424,866
3200	Flotation & Re grind	27,386	19,129	167,573,544
3300	Concentrate Thickening	108	74	648,240
3400	Concentrate Filtration, Storage, Loadout	775	453	3,970,251
3500	Tailings Thickening	2,661	982	8,600,393
3600	Reagents	547	220	1,928,225
3700	Plant Services	10,731	4,531	39,694,714
4200	Tailings Storage Facility	3,199	1,851	16,211,081
5310	Fresh Water supply (well pumps)	12,778	11,500	100,741,752
4400	Yard Utilities	5	3	27,594
4600	Plant Buildings	10	3	24,966
TOTAL		184,174	126,404	1,107,299,750

Reagents and Consumables

Concentrator reagent and consumable costs were estimated based on the throughput. The costs were based on calculated consumption rates and unit costs supplied by vendors (Table 21.15). Reagents costs include estimated transport to site.

Table 21.15: Reagent and consumable costs

Description	Units	Cost per unit (\$)	Units/yr	Annual Cost (\$)	Unit Cost (\$/t feed)
Crushing & Stockpiling					
Primary crusher bowl/mantles/etc.	units	615,565	2	1,231,131	0.02
Coarse Screen Top deck	units	30,761	16	492,173	0.01
Coarse Screen Bottom deck	units	30,761	16	492,173	0.01
Secondary crusher bowl/mantles/etc.	units	448,666	4	1,794,666	0.03
Wet Fine Screen	units	40,261	32	1,288,340	0.02
HPGR wear parts	units	2,007,385	4	8,029,538	0.15
Grinding					
50mm steel grinding media	t	1,255	4,271	5,359,478	0.10
Mill liners	units	1,142,400	4	4,569,600	0.08
Flotation					
Lime	t	179.2	24,638	4,415,040	0.08
Cu Collector (SASCOL 95 equiv)	t	5.25	1,369	7,189,770	0.13
Cu Collector (MATCOL TC-123 equiv)	t	2.77	1,369	3,786,510	0.07
MIBC (Frother)	t	3.23	548	1,766,016	0.03
Antiscalant	t	5.67	274	1,551,089	0.03
Flocculant (Tailings)	t	1.48	164	243,090	0.00
Filter Cloth	units	358	1,464	524,316	0.01
Vehicles					
Diesel	L	0.77	2,740,599	2,110,261	0.04
Total Consumables				44,843,191	0.82

Ball mill media consumption was calculated based on the ball mill power and abrasion index.

The consumption of reagents was calculated based on SGS Phase II testwork.

Mill and crusher liners, and screen deck consumption rates were estimated based on vendor information and benchmarking similar plants.

Costs for filter cloths were based on the vendor information.

Total reagents and consumables costs are estimated at US\$0.82/t.

Labour Costs

Labour costs were based on salaries and labour rosters provided by Ausenco's Argentina office and NGEx.

- All personnel are on shift schedule working 12 hours per day, two weeks on and one week off
- Regular pay for the first 2080 hours per year
- 1.5 x regular pay for the remaining 832.28 hours per year
- No annual leave and public holidays are accounted for

Transportation, recruitment and training costs are included as separate cost items in the G&A costs.

A breakdown of processing labour schedules and costs are summarized in Table 21.16.

Table 21.16: Process operating labour summary by function

Area	Number	Annual cost (\$/year)
Plant management	38	2,271,553
Shift Crew	72	3,018,282
Laboratory	22	1,042,071
San Juan Office	3	179,616
G&A	44	2,044,410
Maintenance	85	3,968,303
Total	265	12,524,233

Light Vehicle and Mobile Equipment

Costs were estimated for the concentrator and infrastructure light vehicle and mobile equipment fleet. These costs were developed from a schedule of light vehicles and mobile equipment for personnel transport and maintenance. This was based on the labour roster and organizational requirements. Other mobile equipment such as cranes and front-end loaders are required for operations, maintenance and support activities.

The fuel consumption rate was calculated for vehicles in this list. An average maintenance rate per engine hour was calculated and applied to the vehicles in each category to generate maintenance costs.

Costs for transport of personnel to site are included in the G&A cost. Costs associated with the mining mobile equipment and light vehicles are included in the mining cost estimate.

Maintenance Cost

Annual maintenance spares and consumables costs were estimated at 4% 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 2% of architectural capital costs per year.

The annual maintenance cost is estimated at US\$0.65/t for the concentrator and infrastructure combined.

21.2.4 General and Administrative

Operating cost estimates for G&A were prepared by Ausenco and reviewed by NGEx. The estimate for G&A was broken down by department in which the cost was incurred:

- Processing/operations
- General manager
- Finance, logistics and information technology
- HR, camp and security
- External affairs
- Health, safety and environment
- Technical services

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 \$32.2 million was estimated (\$0.59/t milled).

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

The processing/operations department cost is made up of testwork, 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 summarized in Table 21.17.

Table 21.17: Process/operations department cost summary

Item	Cost (\$/year)
Metallurgical Testwork	150,000
Training Costs	77,432
Safety Equipment	43,600
Environmental Testwork	200,000
Laboratory Equipment Maintenance	100,000
Total	\$571,032

Both the general manager's rate, San Juan representative office lease and maintenance costs are based on NGEx estimates.

An allowance of \$750,000 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. The administration cost summary is shown in Table 21.18.

Table 21.18: Administration cost summary

Item	Cost (\$/year)
General manager	390,000
San Juan Representative Office	400,000
Travel Expenses	750,000
Recruitment	54,500
Total	\$1,594,500

Other G&A Costs

Other G&A costs include camp cost, insurance, 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 & alcohol tests, lab consumables including reagents and chemicals, and recreational costs. These allowances were estimated based on NGEx's guidance. Costs for camp lodging and catering were US\$40 per person per day and were benchmarked against other projects. These allowance costs are summarized in Table 21.19.

Table 21.19: Other G&A cost summary

Item	Cost (\$/year)
Camp (assuming 300 people camp)	4,380,000
Insurance	3,000,000
IT (includes telephone)	100,000
Mobile phones	48,000
Couriers/Post	50,000
Legal and other Fees	1,000,000
Government charges	1,000,000
Conferences	20,000
Community Relations	300,000
Community Development	1,000,000
Education/Scholarships	100,000
Office Supplies	24,000
Office Furnitures	50,000
External Consultants	2,000,000
Software	100,000
Medical equipment/consumables	100,000
Lab consumables (including chemicals)	50,000
Recreation	100,000
Total	\$13,422,000

Contracts

Road maintenance costs were calculated based on an allowance of \$70,000/km/y for 180 km of gravel roads for transportation of shift workers to the nearest city. Labour, equipment and material costs are included in road maintenance cost, while reconstruction and improvements are excluded. Employee transport cost is estimated at \$30/return trip/person for an estimated 800 people. Security, cleaning service, maintenance contractors, and effluent handling/garbage removal allowances were estimated based on NGEx’s guidance. Security contract costs includes security personnel, management, camp lodging and catering as well as transport of security personnel to site. Contract costs are summarized in Table 21.20.

Table 21.20: Contracts cost summary

Item	Cost (\$/year)
Road maintenance (~180km)	12,600,000
Security (contract)	200,000
Cleaning Service	200,000
Maintenance Contractors	3,000,000
Effluent handling/garbage removal	200,000
Employee Transport (assuming 800 total)	416,000
Total	\$16,616,000

Operating Cost Contingency

Contingency was not included in the operating cost estimate, other than for mining.

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 significant disruptions affecting the development and operation of the project
- Exchange rate assumptions being approximately consistent with the assumptions in the Report
- The availability of certain consumables and services and the prices for power and other key supplies being approximately consistent with assumptions in the Report
- Labour and materials costs being approximately consistent with assumptions in the Report
- Assumptions made in mineral resource and Reserve estimates, including, but not limited to, geological interpretation, grades, metal price assumptions, metallurgical and mining recovery rates, geotechnical and hydrogeological assumptions, capital and operating cost estimates, and general marketing, political, business and economic conditions.

22.2 General

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 quarterly periods.

The model assumes a 30-month physical construction period.

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 22.1 shows a summary of key project parameters and project economics. LOM project annual cash flow is shown in Table 22.2.

Table 22.1: Summary of project economics

Project Metric	Units	Value
Pre-Tax NPV @ 8%	\$ B	2.91
Pre-tax IRR	%	21.4
After-Tax NPV @ 8%	\$ B	2.03
After Tax IRR	%	18.7
Undiscounted After-Tax Cash Flow (LOM)	\$ B	6.58
Payback Period from start of processing (undiscounted, after-tax cash flow)	years	3.4
Initial Capital Expenditure	\$ M	2,760
LOM Sustaining Capital Expenditure (excluding closure)	\$ M	860
LOM C-1 Cash Costs (Co-Product exc. royalty)	\$/lb CuEq.	1.26
Nominal Process Capacity	ktpd	150
Mine Life	years	20
LOM Mill Feed	kt	1,008,078
LOM Grades		
Copper	%	0.29
Gold	grams per tonne	0.208
Silver	grams per tonne	0.920
LOM Waste Volume	kt	711,556
LOM Strip Ratio (Waste:Ore)	ratio	0.71
First Three Years Average Annual Metal Production		
Copper	tonnes	170,000
Gold	ounces	352,000
Silver	ounces	1,026,000
LOM Average Annual Metal Production		
Copper	tonnes	123,000
Gold	ounces	232,000
Silver	ounces	791,000
LOM Average Process Recovery		
Copper	% contained metal	86
Gold	% contained metal	71
Silver	% contained metal	59

Table 22.2: LOM annual project cash flow

PREFINANCE SUMMARY CASH FLOW	Units	LOM Total	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Payable Revenue																	
Payable Revenue from Copper	\$M	16,316	0.0	0.0	0.0	571.4	1,219.0	1,097.6	1,058.7	842.3	725.0	791.3	898.7	862.7	820.6	862.8	864.3
Payable Revenue from Gold	\$M	6,023	0.0	0.0	0.0	285.8	542.7	440.1	390.2	349.5	293.9	311.2	310.0	309.8	304.4	308.1	287.9
Payable Revenue from Silver	\$M	316	0.0	0.0	0.0	9.9	20.5	22.5	18.5	15.3	14.3	15.0	16.8	16.6	15.2	15.7	16.2
Total Revenue from Payable Metal	\$M	22,655	0.0	0.0	0.0	867.1	1,782.3	1,560.2	1,467.4	1,207.1	1,033.2	1,117.5	1,225.5	1,189.0	1,140.2	1,186.6	1,168.3
<i>Copper Equivalent Payable Pounds</i>	<i>Mlbs</i>	7,552	<i>0.0</i>	<i>0.0</i>	<i>0.0</i>	<i>289.0</i>	<i>594.1</i>	<i>520.1</i>	<i>489.1</i>	<i>402.4</i>	<i>344.4</i>	<i>372.5</i>	<i>408.5</i>	<i>396.3</i>	<i>380.1</i>	<i>395.5</i>	389.4
Total TCRC Freight	\$M	2,670	0.0	0.0	0.0	92.1	198.4	174.9	164.6	137.7	118.9	130.1	146.6	138.2	132.4	140.1	139.8
Total Royalty	\$M	508	0.0	0.0	0.0	19.2	42.8	40.2	36.3	28.9	23.4	26.0	29.2	28.2	26.7	24.9	24.5
Total Minesite Revenue	\$M	19,477	0.0	0.0	0.0	755.9	1,541.0	1,345.1	1,266.5	1,040.5	890.9	961.4	1,049.7	1,022.6	981.1	1,021.5	1,004.1
OPERATING COSTS																	
Mining Opex	\$M	2,802	0.0	0.0	49.3	132.9	143.1	148.2	157.9	151.0	157.7	155.2	157.5	145.3	134.6	126.7	131.2
Processing (Mill)	\$M	3,271	0.0	0.0	0.0	96.7	177.0	177.0	177.0	177.5	177.0	177.0	177.0	177.5	177.0	176.9	175.4
Processing (Leach)	\$M	0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Site G&A	\$M	712	0.0	0.0	0.0	37.2	37.2	37.2	37.2	37.2	37.2	37.2	37.2	37.2	37.2	37.2	37.2
Offsite Infrastructure Opex (excl. roads)	\$M	19	0.0	0.0	0.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Less capitalized Opex	\$M	-49	0.0	0.0	-49.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total Operating Costs	\$M	6,754	0.0	0.0	0.0	267.8	358.3	363.4	373.1	366.7	372.9	370.4	372.7	361.0	349.8	341.7	344.8
Operating Cashflow	\$M	12,723	0.0	0.0	0.0	488.0	1,182.7	981.7	893.4	673.8	517.9	591.0	677.0	661.6	631.3	679.8	659.3
Summary Capex by Project Phase																	
Study Costs	\$M	0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Construction Costs	\$M	2,761	336.7	1,443.2	980.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Sustaining Capital Costs	\$M	860	0.0	0.0	0.0	47.2	25.9	29.8	28.5	32.8	24.7	33.9	28.8	96.3	47.4	50.6	60.3
Closure Costs	\$M	150	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Grand Total Capex (Including Closure)	\$M	3,772	336.7	1,443.2	980.7	47.2	25.9	29.8	28.5	32.8	24.7	33.9	28.8	96.3	47.4	50.6	60.3
Working Capital	\$M	42	0.0	0.0	0.0	18.0	4.6	1.8	-1.1	1.6	0.8	3.0	3.2	1.8	1.6	2.4	2.0
PRE-TAX CASHFLOW	\$M	8,910	-336.7	-1,443.2	-980.7	422.8	1,152.2	950.2	866.1	639.4	492.4	554.1	645.0	563.5	582.3	626.7	597.0
Tax																	
VAT	\$M	269	0.0	0.0	0.0	4.9	6.7	2.0	0.5	0.3	0.2	0.3	0.2	0.1	-0.2	-0.1	0.1
Corporate Income Tax (real)	\$M	2,323	0.0	0.0	0.0	0.0	0.0	0.0	103.1	183.6	141.7	131.3	151.2	157.2	148.0	148.9	153.0
Total Tax	\$M	2,327	0.0	0.0	0.0	4.9	6.7	2.0	103.6	183.9	142.0	131.7	151.4	157.2	147.8	148.8	153.1
After-tax Net Cash Flow (Real)	\$M	6,583	-336.7	-1,443.2	-980.7	417.8	1,145.5	948.2	762.4	455.5	350.5	422.4	493.6	406.2	434.5	478.0	443.9

Table 22.2: LOM annual project cash flow – continued

PREFINANCE SUMMARY CASH FLOW	Units	LOM Total	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27
Payable Revenue																	
Payable Revenue from Copper	\$M	16,316	827.4	802.1	741.9	751.0	890.2	918.4	681.1	89.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Payable Revenue from Gold	\$M	6,023	281.9	279.5	255.5	232.9	239.3	322.8	251.0	26.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Payable Revenue from Silver	\$M	316	16.9	17.6	16.7	15.4	19.1	19.1	12.9	2.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total Revenue from Payable Metal	\$M	22,655	1,126.2	1,099.3	1,014.1	999.2	1,148.7	1,260.3	945.0	117.7	0.0						
<i>Copper Equivalent Payable Pounds</i>	<i>Mlbs</i>	<i>7,552</i>	<i>375.4</i>	<i>366.4</i>	<i>338.0</i>	<i>333.1</i>	<i>382.9</i>	<i>420.1</i>	<i>315.0</i>	<i>39.2</i>	<i>0.0</i>						
Total TCRC Freight	\$M	2,670	135.3	132.5	123.7	124.5	145.8	157.5	119.5	17.2	0.0						
Total Royalty	\$M	508	23.3	22.6	20.3	19.8	23.6	26.6	19.6	1.9	0.0						
Total Minesite Revenue	\$M	19,477	967.6	944.2	870.2	855.0	979.2	1,076.2	805.9	98.6	0.0						
OPERATING COSTS																	
Mining Opex	\$M	2,802	133.9	128.6	128.5	144.6	139.3	139.5	135.6	61.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Processing (Mill)	\$M	3,271	176.3	176.3	176.6	177.0	177.5	177.0	141.2	25.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Processing (Leach)	\$M	0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Site G&A	\$M	712	37.2	37.2	37.2	37.2	37.2	37.2	30.2	11.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Offsite Infrastructure Opex (excl. roads)	\$M	19	1.0	1.0	1.0	1.0	1.0	1.0	0.8	0.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Less capitalized Opex	\$M	-49	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total Operating Costs	\$M	6,754	348.3	343.0	343.3	359.9	355.0	354.8	307.9	99.2	0.0						
Operating Cashflow	\$M	12,723	619.3	601.1	526.9	495.1	624.3	721.4	498.0	-0.6	0.0						
Summary Capex by Project Phase																	
Study Costs	\$M	0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Construction Costs	\$M	2,761	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Sustaining Capital Costs	\$M	860	37.3	57.8	67.8	44.0	47.8	48.6	50.2	0.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Closure Costs	\$M	150	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	15.0	30.1	30.1	30.1	30.1	15.0	0.0
Grand Total Capex (Including Closure)	\$M	3,772	37.3	57.8	67.8	44.0	47.8	48.6	50.2	0.6	15.0	30.1	30.1	30.1	30.1	15.0	0.0
Working Capital	\$M	42	1.8	1.8	1.3	2.2	3.4	3.2	-1.4	-8.6	-1.9	0.0	0.0	0.0	0.0	0.0	0.0
PRE-TAX CASHFLOW	\$M	8,910	580.2	541.5	457.8	448.9	573.1	669.6	449.2	7.4	-13.2	-30.1	-30.1	-30.1	-30.1	-15.0	0.0
Tax																	
VAT	\$M	269	0.3	0.2	0.1	0.5	0.4	0.1	-0.6	-4.5	-5.5	-1.8	0.0	0.0	0.0	0.0	0.0
Corporate Income Tax (real)	\$M	2,323	147.3	140.1	127.9	114.1	126.5	155.9	140.7	56.3	-2.0	-2.0	0.0	0.0	0.0	0.0	0.0
Total Tax	\$M	2,327	147.6	140.3	128.1	114.7	126.9	156.1	140.1	51.8	-7.5	-3.8	0.0	0.0	0.0	0.0	0.0
After-tax Net Cash Flow (Real)	\$M	6,583	432.5	401.2	329.7	334.2	446.2	513.5	309.2	-44.4	-5.7	-26.3	-30.1	-30.1	-30.1	-15.0	0.0

22.3 Production Schedule

The production schedule evaluated is summarized in Table 22.3 Metal production and mine physicals are shown graphically in Figure 22.1 and Figure 22.2.

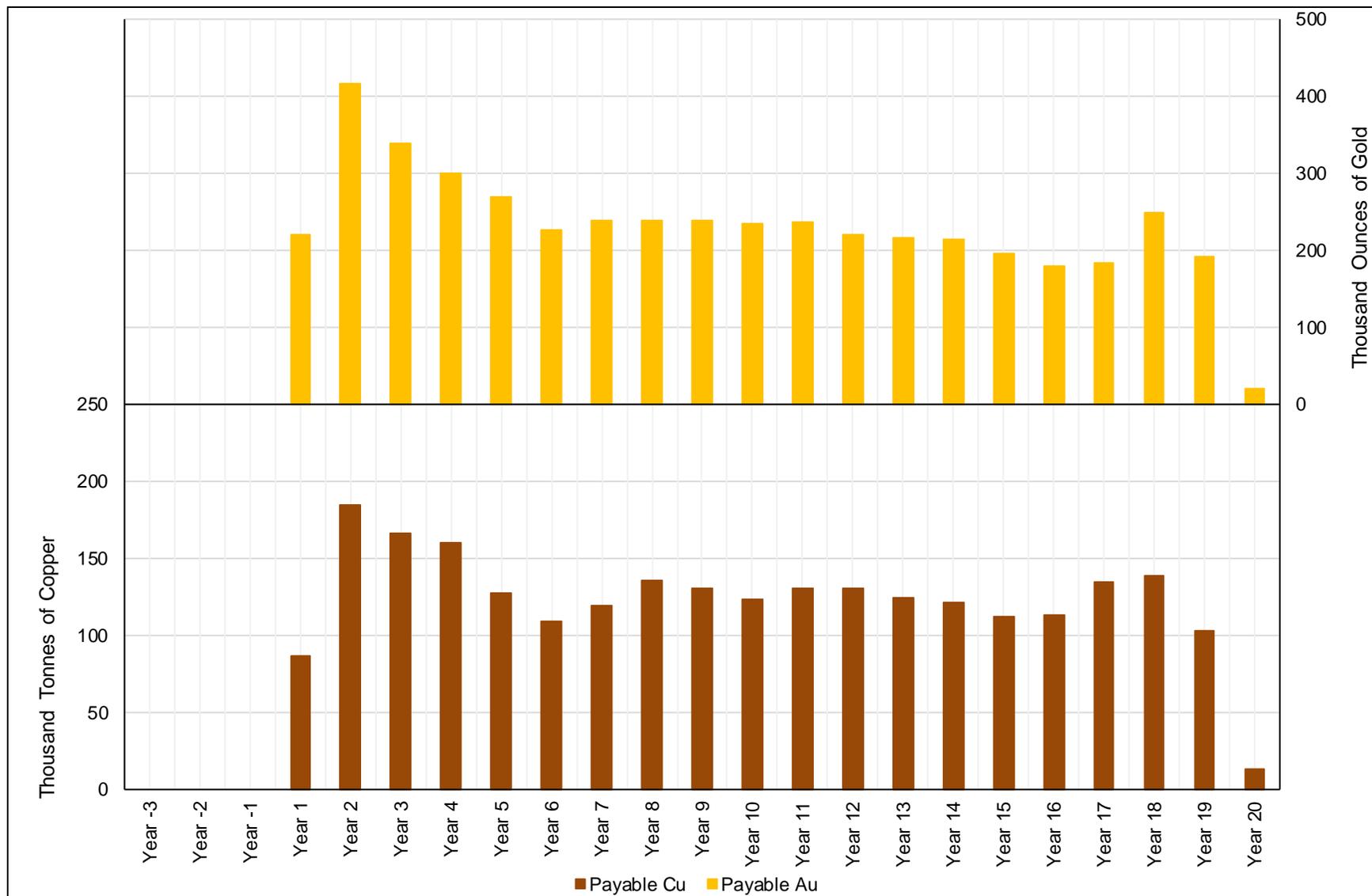
Operating margin is defined as the % of Net Revenue (after all offsite costs, payables and royalties) that reports to operating cashflow before capital expenditures, working capital and taxes.

Operating Margin = (Net Revenue-Operating Costs)/Net Revenue.

Table 22.3: Production schedule summary

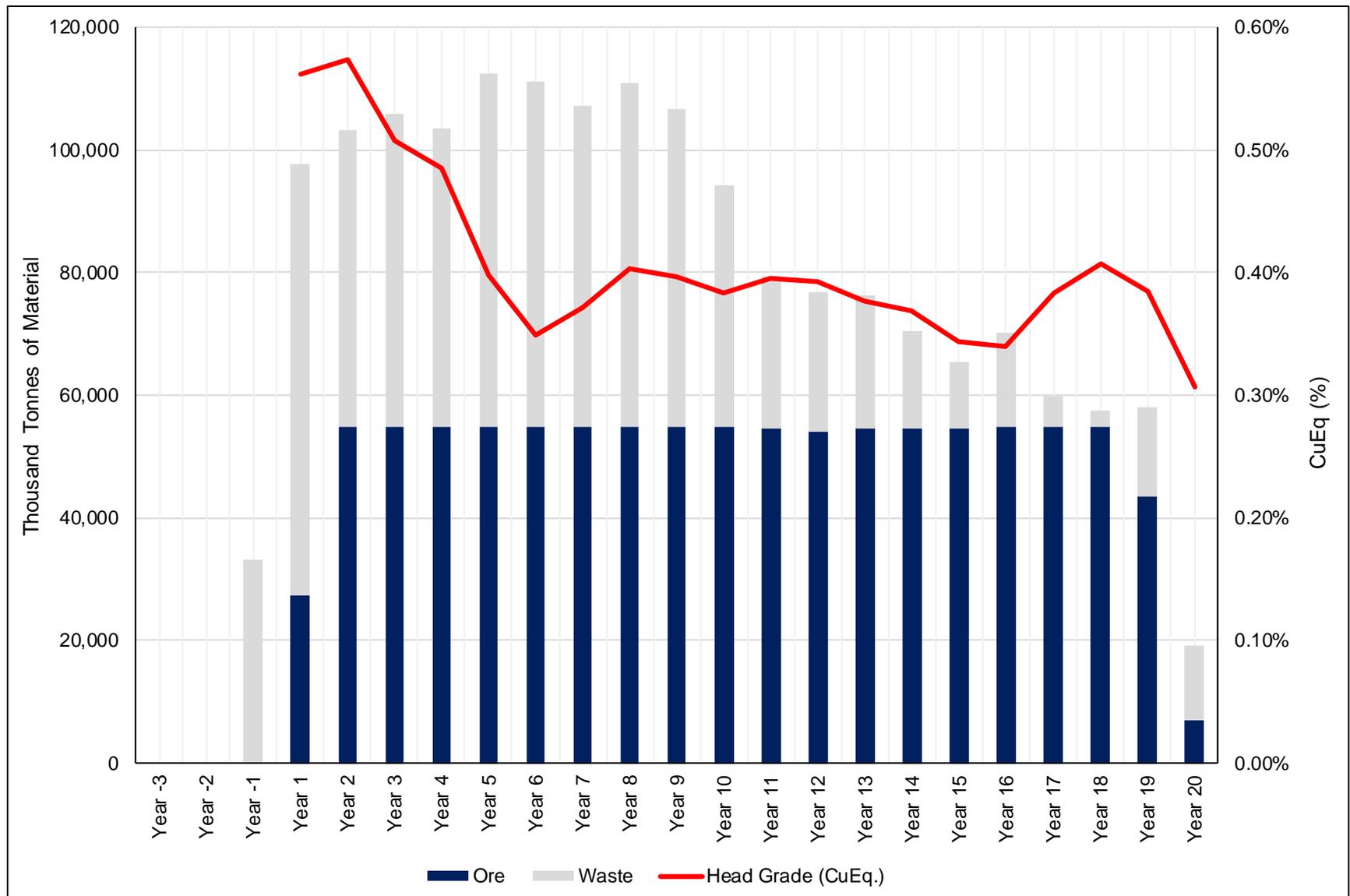
Item	Units	LOM Totals	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Ore	kt	1,008,078	0	27,510	54,750	54,750	54,750	54,900	54,750	54,750	54,750	54,900
Cu	%	0.2934	0.00	0.37	0.39	0.36	0.35	0.28	0.25	0.26	0.30	0.29
Au	gpt	0.208	0.000	0.358	0.338	0.278	0.249	0.225	0.190	0.198	0.196	0.197
Ag	gpt	0.920	0.000	1.004	1.099	1.172	0.919	0.820	0.772	0.826	0.906	0.856
Waste	kt	711,556	33,304	70,302	48,364	51,205	48,644	57,631	56,383	52,480	56,236	51,650
Strip Ratio	waste:ore	0.71	N/A	2.556	0.883	0.935	0.888	1.050	1.030	0.959	1.027	0.941
Head Grade (CuEq.)	% CuEq.	0.407	0.00	0.56	0.57	0.51	0.48	0.40	0.35	0.37	0.40	0.40
Payable Cu	kt	2,467		86	184	166	160	127	110	120	136	130
Payable Au	koz	4,633		220	417	339	300	269	226	239	238	238
Payable Ag	koz	15,820		495	1,025	1,127	927	764	715	751	838	828
CuEq lbs	Mlbs	7,552		289	594	520	489	402	344	373	408	396
CuEq tonnes	kt	3,425		131	269	236	222	183	156	169	185	180

Item	Units	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20
Ore	kt	54,750	54,693	54,181	54,491	54,498	54,601	54,750	54,900	54,750	43,593	7,061
Cu	%	0.28	0.29	0.29	0.28	0.27	0.25	0.26	0.30	0.30	0.28	0.23
Au	gpt	0.194	0.195	0.187	0.181	0.179	0.163	0.148	0.152	0.202	0.199	0.144
Ag	gpt	0.798	0.842	0.867	0.911	0.957	0.921	0.840	1.002	1.080	0.933	0.916
Waste	kt	39,617	24,656	22,599	21,816	16,066	10,930	15,369	4,964	2,899	14,427	12,014
Strip Ratio	waste:ore	0.724	0.451	0.417	0.400	0.295	0.200	0.281	0.090	0.053	0.331	1.702
Head Grade (CuEq.)	% CuEq.	0.38	0.40	0.39	0.38	0.37	0.34	0.34	0.38	0.41	0.38	0.31
Payable Cu	kt	124	130	131	125	121	112	114	135	139	103	14
Payable Au	koz	234	237	221	217	215	197	179	184	248	193	20
Payable Ag	koz	759	787	809	843	882	835	769	956	954	643	112
CuEq lbs	Mlbs	380	396	389	375	366	338	333	383	420	315	39
CuEq tonnes	kt	172	179	177	170	166	153	151	174	191	143	18



Source: SRK, 2018

Figure 22.1: Metal production schedule graph



Source: SRK, 2018

Figure 22.2: Mine production schedule graph

22.4 Pricing Assumptions

Flat real prices were assumed for the life of the project. Table 22.4 shows the price assumptions used.

Table 22.4: Pricing assumptions for economic analysis

Commodity	Units	Price
Copper Price	\$/lb	\$3.00
Gold Price	\$/oz	\$1,300
Silver Price	\$/oz	\$20.00

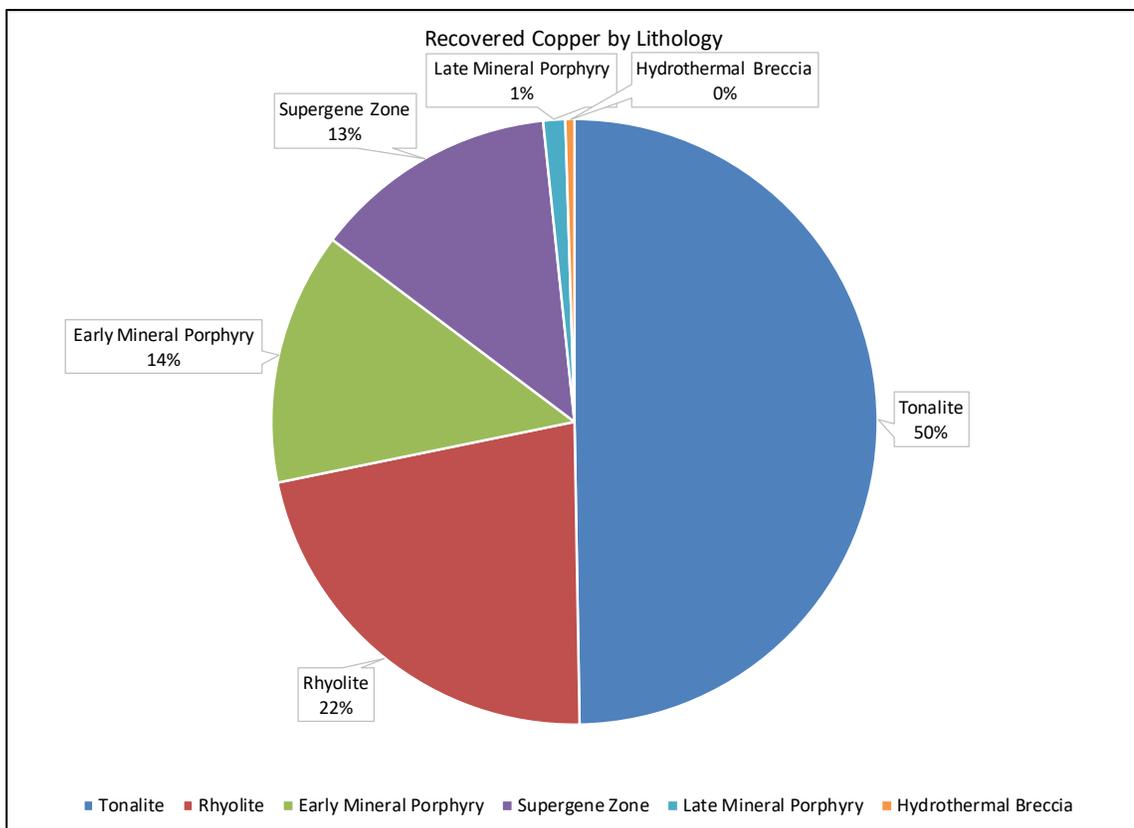
22.5 Processing Recovery Assumptions

The estimated processing recoveries were supplied in the form of algorithms that allowed for estimation of the process recoveries by ore-type. Table 22.5 summarizes the recovery assumptions used for the economic analysis. Additional detail on the development of the algorithms is contained in Section 13.

Table 22.5: Processing recovery assumptions used for economic analysis

Lithology	Copper Recovery (%)	Gold Recovery (%)	Silver Recovery (%)
Tonalite	88.3	77.0	51.1
Rhyolite	82.2	64.1	68.1
Early Mineral Porphyry	88.7	62.6	60.3
Supergene Zone	85.3	72.2	78.3
Late Mineral Porphyry	80.5	62.6	60.3
Hydrothermal Breccia	88.5	62.6	60.3
All Lithologies	86.4	70.9	59.0

Figure 22.3 shows the relative contribution to total recovered copper by lithology. It can be seen that the late mineral porphyry and hydrothermal breccia do not contribute significant value in the current mine plan.



Source: SRK 2018

Figure 22.3: Recovered copper by lithology

22.6 Capital Costs

Capital costs used for the evaluation are summarized in Table 22.6. Additional detail regarding the estimation of the capital costs is contained in Section 21. Note that the capital costs presented do not include any costs prior to construction commencement. Please refer to Section 26 for an estimate of the feasibility study work program and costs.

Table 22.6: Capital cost summary

Capital Costs	Construction (\$M)	Production (\$M)	LOM (\$M)
1000 - Mine	263	255	518
2000 - Crushing	486	0	486
3000 - Process	457	0	457
4000 - Onsite Infrastructure	163	514	676
5000 - Offsite Infrastructure	280	0	280
Sub-Total Direct Costs	1,648	769	2,416
6000 - Indirects	310	66	376
7000 - Project Delivery	245	0	245
8000 - Owner's Costs	83	0	83
9000 - Provisions	475	25	500
Sub-Total Indirect Costs	1,113	92	1,205
Project Total	2,761	860	3,621
Closure Costs			150
Grand Total Capex (Including Closure)			3,772

22.7 Operating Costs

Operating costs (Opex) are summarized in Table 22.7. The capitalized Opex is pre-stripping, which has been re-allocated and included in the mining capital costs shown in Table 22.7. The unit costs are expressed as total operating costs (before re-allocation) divided by total tonnage.

Table 22.7: Operating costs summary

Operating Costs	LOM (\$M)	Unit Rates (\$/t milled)	Unit Rates (\$/t material moved, incl. rehandle)
Mining Opex	2,802	2.78	1.63
Processing (Mill)	3,271	3.23	N/A
Site G&A*	712	0.71	N/A
Offsite Infrastructure Opex (excl. roads)	19	0.02	N/A
Less Capitalized Opex	-49	N/A	N/A
Total Operating Costs	6,754	6.74	N/A

* Site G&A includes Processing G&A of \$0.59/t milled (Section 21.2.3) plus Mining and other site G&A of \$0.12/t milled

22.8 Royalties

A San Juan Province government royalty of 3% (NGEx agreement) was applied to the "mine-head" revenue. The mine head revenue is the Net revenue of the project with all site operating costs (processing, infrastructure and G&A), except mining operating costs, deducted.

One further private royalty was considered in accordance with advice received from NGEx. This royalty, Lirio DPMA, is applicable to a discrete portion of the mine lease and PFS mine plan.

These royalties allow for deduction of most operating costs (and are applied after deduction of the San Juan provincial royalty) and are minor with respect to overall project economics. The estimated royalties payable (LOM) are summarized in Table 22.8.

Table 22.8: Royalty summary

Category	Units	LOM Totals
Total Revenue from Payable Metal	\$M	22,655
Total TCRC Freight	\$M	2,670
San Juan Provincial Royalty (NGEx agreement)	\$M	480
Lirio DPMA Royalty	\$M	28
Total Royalty	\$M	508
Total Minesite Revenue	\$M	19,477

For the feasibility study, SRK recommends that the ore attributable to the Lirio DPMA royalty be identified in the block model by code. This will allow for precise royalty calculations over time in the mine schedule.

22.9 Taxation

The taxation was modelled in a simplified manner, as is appropriate for PFS level of study. A corporate tax rate of 25% was applied, consistent with the newly revised standard rate for Argentina. Tax losses were carried forward. An opening balance of \$38.3M was assumed for tax losses on the advice of NGEx (The valuation is insensitive to this assumption, and no audit was carried out nor warranted).

Depreciation was modelled in a simplified fashion, suitable for a PFS evaluation. One-third (33%) of the capital expenditure was depreciated in a 3-year straight-line model and the remaining 67% was modelled in a 3-year 60:20:20 percent accelerated model. As a sensitivity, allocating all capital to the slower depreciation model made approximately \$1M difference to the project valuation. The project valuation is insensitive to minor variations in depreciation treatment.

22.10 Off-Site Costs

Off-site costs (concentrate freight, port handling, treatment charges and refining charges) were deducted from payable revenue. The basis for the charges is summarized in Section 19.

22.11 Sensitivity Analysis

Table 22.9 to Table 22.13 summarize the sensitivity of the project NPV (\$B at 8% discount rate) to variations in key input assumptions across a change of +/-20%.

Table 22.9: Two-factor sensitivity (NPV in \$B) – Capex and Opex

After-tax NPV at 8%		Operating Costs				
		-20.00%	-10.00%	0.00%	10.00%	20.00%
Capital Costs	-20.00%	\$2.94	\$2.72	\$2.50	\$2.28	\$2.06
	-10.00%	\$2.71	\$2.49	\$2.26	\$2.04	\$1.82
	0.00%	\$2.47	\$2.25	\$2.03	\$1.81	\$1.59
	10.00%	\$2.23	\$2.01	\$1.79	\$1.57	\$1.35
	20.00%	\$1.99	\$1.77	\$1.55	\$1.33	\$1.11

Table 22.10: Two-factor sensitivity (NPV in \$B) – Prices and discount rate

After-tax NPV at 8%		Discount Rate				
		6.00%	7.00%	8.00%	9.00%	10.00%
Metal Prices	-20.00%	\$1.24	\$0.73	\$0.52	\$0.33	\$0.17
	-10.00%	\$2.23	\$1.56	\$1.28	\$1.03	\$0.81
	0.00%	\$3.21	\$2.37	\$2.03	\$1.72	\$1.45
	10.00%	\$4.19	\$3.19	\$2.77	\$2.41	\$2.08
	20.00%	\$5.16	\$4.00	\$3.52	\$3.09	\$2.72

Table 22.11: Two-factor sensitivity (NPV in \$B) – Capex and metal prices

After-tax NPV at 8%		Metal Prices				
		-20.00%	-10.00%	0.00%	10.00%	20.00%
Capex	-20.00%	\$1.00	\$1.75	\$2.50	\$3.25	\$3.99
	-10.00%	\$0.76	\$1.52	\$2.26	\$3.01	\$3.76
	0.00%	\$0.52	\$1.28	\$2.03	\$2.77	\$3.52
	10.00%	\$0.28	\$1.04	\$1.79	\$2.54	\$3.28
	20.00%	\$0.02	\$0.79	\$1.55	\$2.30	\$3.05

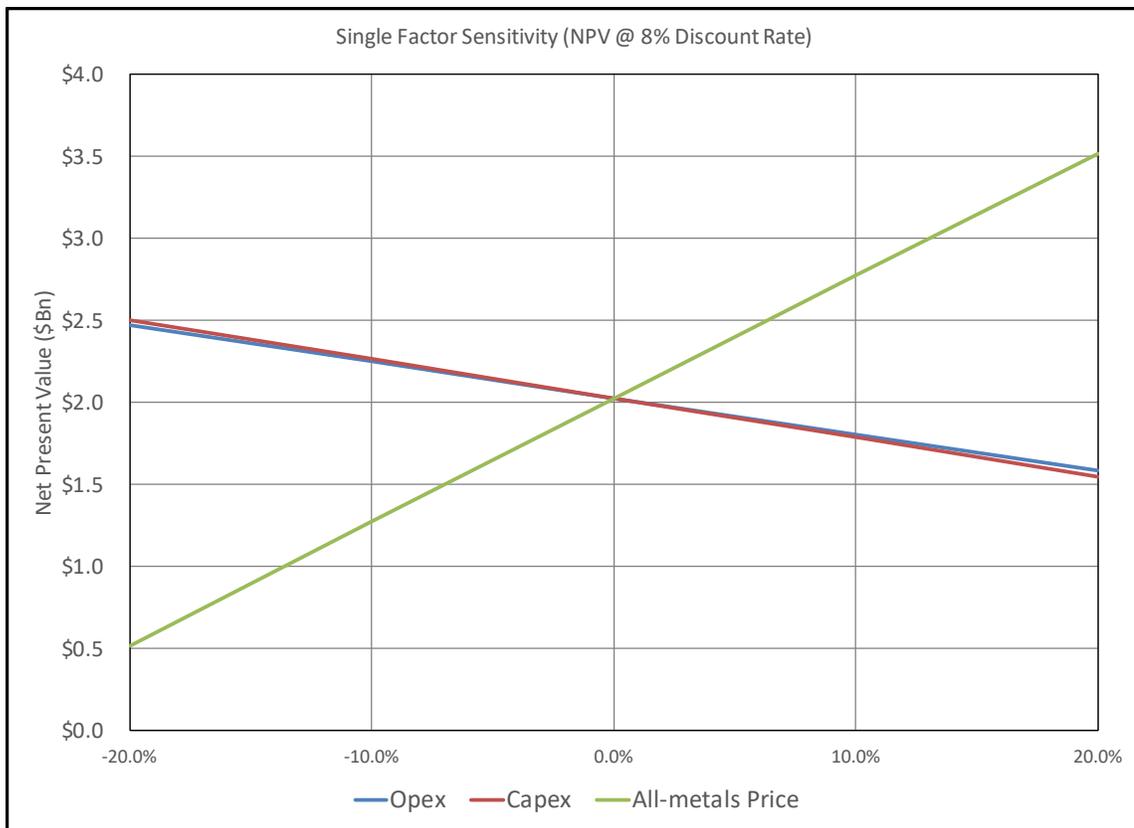
Table 22.12: Two-factor sensitivity (NPV in \$B) – Opex and metal prices

After-tax NPV at 8%		Metal Prices				
		-20.00%	-10.00%	0.00%	10.00%	20.00%
Opex	-20.00%	\$0.97	\$1.72	\$2.47	\$3.22	\$3.96
	-10.00%	\$0.75	\$1.50	\$2.25	\$3.00	\$3.74
	0.00%	\$0.52	\$1.28	\$2.03	\$2.77	\$3.52
	10.00%	\$0.30	\$1.06	\$1.81	\$2.55	\$3.30
	20.00%	\$0.07	\$0.83	\$1.59	\$2.33	\$3.08

Table 22.13: Sensitivity (NPV in \$B) – Individual metal prices

After-tax NPV at 8%		Metal Prices				
		-20.00%	-10.00%	0.00%	10.00%	20.00%
Copper Price (\$/lb)		\$2.40	\$2.70	\$3.00	\$3.30	\$3.60
After-tax NPV at 8%		\$1.00	\$1.49	\$2.03	\$2.56	\$3.09
Gold Price (\$/oz)		\$1,040	\$1,170	\$1,300	\$1,430	\$1,560
After-tax NPV at 8%		\$1.60	\$1.82	\$2.03	\$2.23	\$2.44
Silver Price (\$/oz)		\$16.00	\$18.00	\$20.00	\$22.00	\$24.00
After-tax NPV at 8%		\$2.00	\$2.02	\$2.03	\$2.04	\$2.05

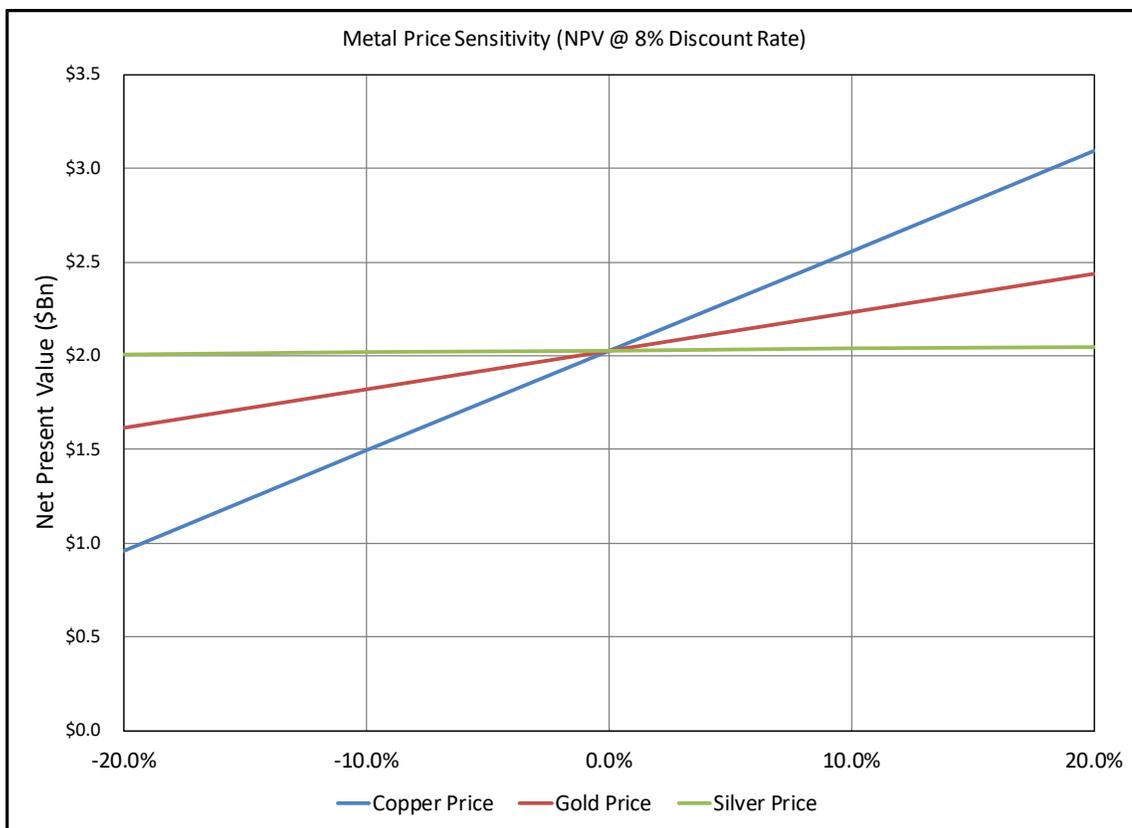
Figure 22.4 shows how the project NPV varies as price, capital costs and operating costs are varied across a range of +/-20%. As is common to all minerals industry projects, commodity price is a highly significant driver of value.



Source: SRK 2018

Figure 22.4: Single factor sensitivity – net present value

Figure 22.5 shows how the project NPV varies as individual commodity prices are varied across a range of +/-20%. Copper, being the main source of revenue, demonstrates greater sensitivity.



Source: SRK, 2018

Figure 22.5: Metals price sensitivity – net present value

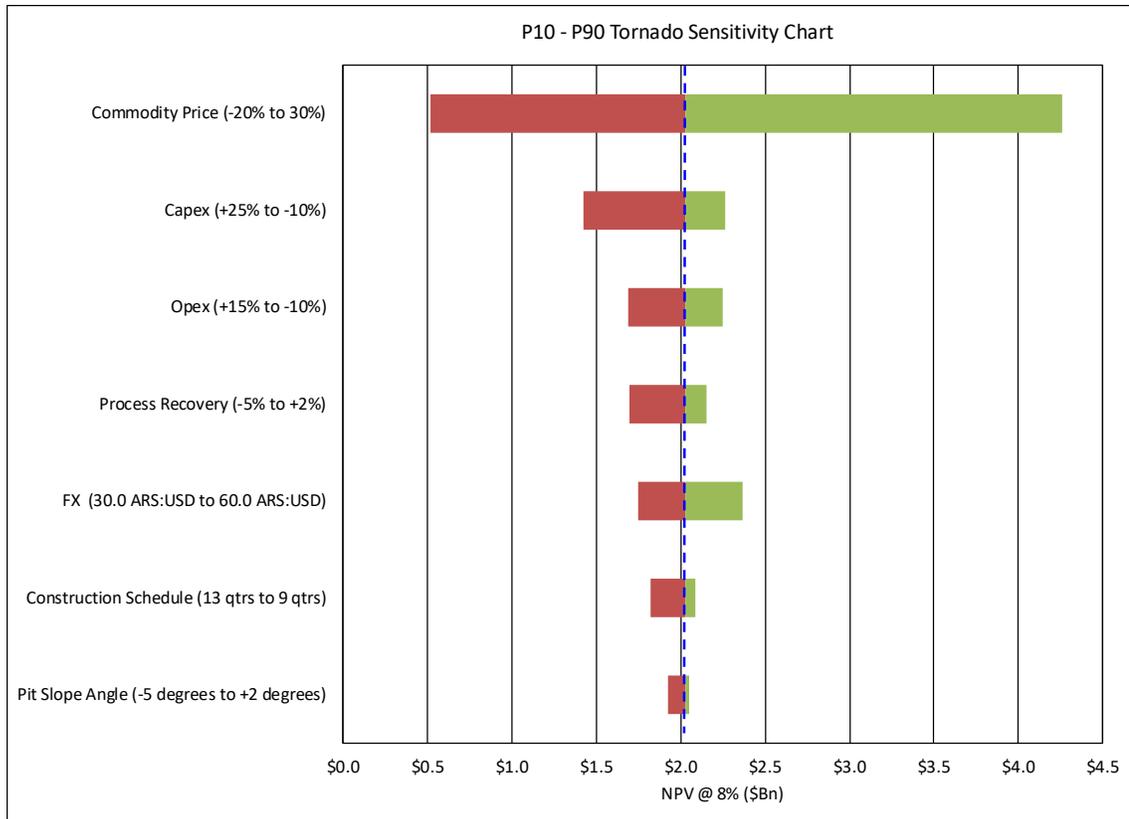
Figure 22.6 illustrates the response of project NPV to variations in assumptions regarding key value-drivers. The general approach was to estimate P₁₀ and P₉₀ values for each key driver.

A P₁₀ defines the parameter value that has only a 10% probability of not being realized on the downside. Conversely it is estimated that there is a 90% chance of that value being exceeded.

A P₉₀ defines the parameter value that is estimated to have only a 10% chance of being exceeded on the upside (or conversely a 90% chance of not being exceeded). Another way to look at it is that the parameter has an 80% chance of lying between the P₁₀ and P₉₀ values.

Commodity Price was estimated to have a moderately asymmetric risk with the range being defined at -20% and +30% compared to base-case. Across this range, commodity prices are the largest single uncertainty with respect to project value.

Capital Expenditure was estimated to have an asymmetric risk with a +25% to -10% range. That is, there is a 10% chance that the capital costs will be 25% higher than base case and a 10% chance that a saving of 10% will be realized.



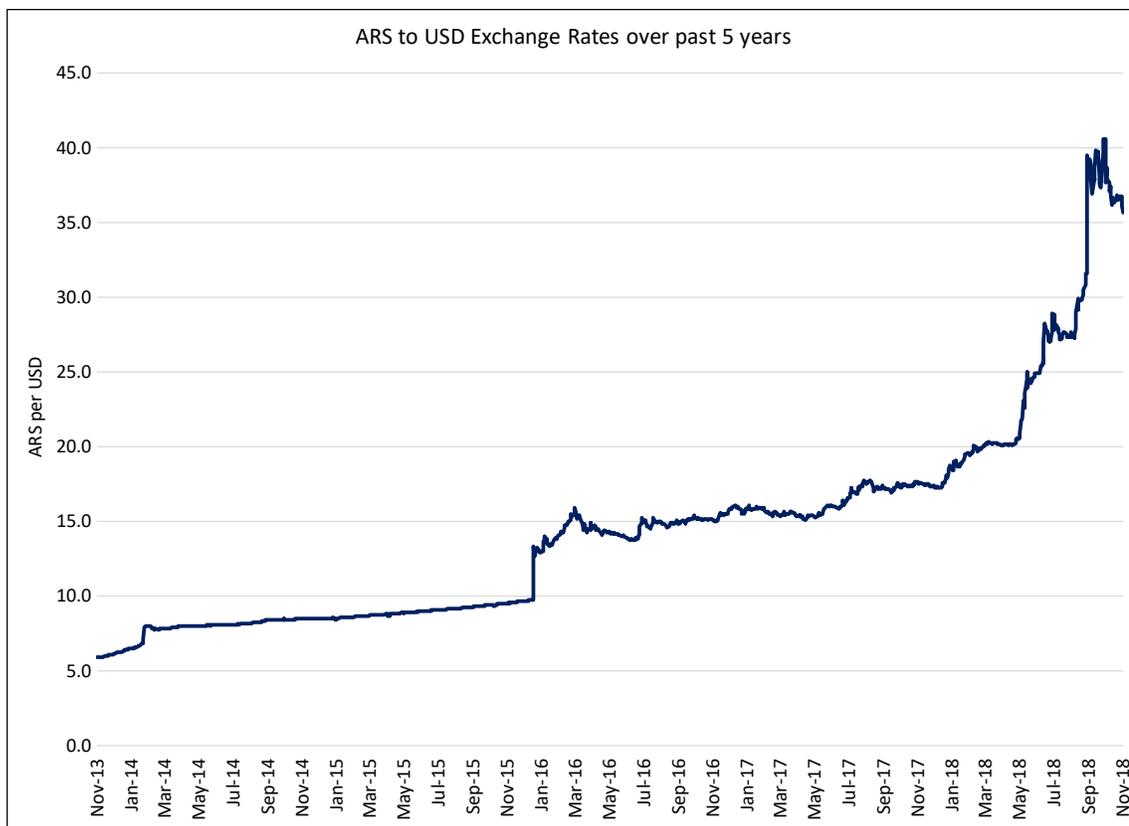
Source: SRK, 2018

Figure 22.6: Tornado diagram of key risk sensitivity

Operating Costs were estimated to have moderately asymmetric risk across the range of +15% downside to -10% upside.

Process Recovery is also considered as an asymmetric risk. A 5% reduction in recovery downside value and a 2% upside value was considered. Note that the flex is expressed as relative percent, and not percentage points. The flex is applied to all metals under all processing streams.

Foreign Exchange (FX) rates (ARS per USD) were flexed to demonstrate the effect of rapid FX fluctuations on the USD cost of building the project. The Argentine Peso (ARS) has suffered significant devaluation in USD terms over the past few years as shown in Figure 22.7. The flex was applied only to costs considered to be “local” to Argentina such as local labour, earthworks and concrete supply. Imported materials and equipment were not considered to be subject to this risk. On a weighted average basis, the overall % of costs for the project considered to be local was estimated as 48%. In the longer-term it is important to realise that locally denominated cost become subject to inflationary pressure as a consequence of currency devaluation and that a reversion to constant real costs occurs over time. For the purposes of modelling for Josemaría, no reversion was modelled, but the effect of FX fluctuations was limited to the construction costs only.



Source SRK, 2018

Figure 22.7: Argentine peso to US dollar historic exchange rate

Pit Slope Angle was flexed to assess the potential impact of variation in overall pit slope angles that may arise due to different geotechnical conditions to those modelled. Variation in volumes for waste for both 5° shallower and 2° steeper sensitivities were estimated from the 3-D models and topography and used to flex the mine operating costs accordingly.

Construction Schedule was flexed (downside) by adding an additional three quarters to the assumed 10-quarter base case. The upside case considers an acceleration of the assumed construction period by one quarter. To reflect the likely costs and savings associated with these potential outcomes, costs comprising construction indirects were pro-rated to the upside and downside schedules.

22.12 Risks and Opportunities

22.12.1 Risks

Project Strategy Risk

SRK undertook an analysis as part of this study to determine the optimum project strategy across a range of commodity prices, and the recommendation for the current 150 ktpd throughput

assumption remained valid. Overall, SRK considers that the likelihood of a major revision to project strategy emerging from the feasibility study to be low.

Commodity Price Risk

There is a risk that commodity prices may not be consistent with assumptions made in this study.

Capital Cost Risk

There is a risk that the capital required to build and operate the project may be higher than that forecast in this study. SRK recommends that the precision of the estimates be refined at feasibility study before commitment to project construction is made.

Operating Cost Risk

There is a risk that the operating costs incurred to operate the project may be higher than that forecast in this study. SRK notes that variability in the operating cost drivers (productivity, input costs and labour costs) over time is expected. The analysis assumes constant conditions but is best thought of as reflecting an expectation of average costs. SRK recommends that the precision of the estimates be refined at feasibility study before commitment to project construction is made.

Schedule Risk

There is a risk that the schedule to build the project may vary from that assumed in the study. This is an asymmetrical risk, with significantly more downside scope than upside. This risk is exacerbated by the seasonality of the location, with difficult construction conditions occurring in winter months. Small delays have the potential to be more significant than might otherwise be the case if they push critical path activities into winter months, thereby incurring a much longer delay.

Process Recovery Risk

There is a risk that achieved recoveries could be lower than estimated. The process recovery estimates will be refined as part of the feasibility study.

Pit Slope Angle

It can be seen from the analysis in Section 22.11 that the project economics are relatively insensitive to the pit-slope angle assumptions across the range tested. This is due to a combination of low strip ratio and favourable topography. Pit-slope stability will be subject to further analysis as part of the feasibility study. A geotechnical data-acquisition program has been developed and is being implemented with a view to informing more precise slope estimates and more refined mine designs as part of the feasibility study.

Permitting and Pre-construction Schedule Risk

This was not explicitly considered for the purposes of this study in the economic analysis as the analysis is conducted only from the commencement of construction. Nevertheless, the risk of

longer-than-anticipated permitting timeline will reduce the project value is considered from “today” forward.

22.12.2 Opportunities

Real Option Value

In the case of a large, long-life open-pit mine such as is contemplated for Josemaría, there exists significant optionality that can be leveraged to improve project cashflows and values. The simple sensitivity analysis conducted in Section 22.12 assumes a constant operating strategy, even as assumptions are varied. In practice, management has the option to alter strategy in response to those variations. Downsides can be mitigated, and upsides can be leveraged for greater returns.

It is also expected that the mine would be run using a dynamic cut-off policy where mill cut-offs, stockpiling strategies and mining rates will all be varied in real time to maximise returns as prices and costs vary. The benefits of this strategy are not reflected in the central estimate approach to valuation summarized in this report.

Project Strategy Opportunity

The probability of a major revision to project strategy is considered low, but nevertheless, careful consideration and revision of the strategic decisions should be a feature of studies going forward. In particular, effort should be made to enhance the optionality of the project, particularly where this is low cost.

Commodity Price Opportunity

There is a risk that commodity prices may not be consistent with assumptions made in this study. Higher prices, both realised and forecast would lead to re-optimisation of the mine and processing plans with a potential to create additional value beyond that shown by the sensitivity analysis summarized in Section 22.11.

Capital Cost Opportunity

Opportunities to reduce or defer capital expenditure may be realised in future studies. Care should be taken when considering the relationship between lower capital opportunities and technical risk to the project.

Operating Cost Opportunity

Operating costs may be lower than forecast for the purposes of this study. Lower costs should feed into both strategic and short-term mine planning, to allow optimisation of stockpiling and mill feed strategies.

Schedule Opportunity

This risk is highly asymmetric. SRK considers that the opportunity to execute a significantly shorter construction program is low. SRK cautions that optimised schedules with multiple critical or near-critical path activities will contain additional embedded risks.

Process Recovery Opportunity

Further metallurgical testwork will allow for optimisation of the process flow sheet and plant design in the Feasibility Study. Better than planned recoveries are possible.

Pit Slope Angle Opportunity

This is not considered to be a significant opportunity from an economic perspective.

22.12.3 Note on Inferred Resources

Disclaimer: Material based on inferred resources is considered too speculative geologically to have the economic considerations applied to it that would enable it to be categorized as mineral reserves. Material based in inferred resources cannot be considered within the mine plan and economic analysis for the purposes of this study and should not be added to the reserves for any purposes.

Within the economic pit shell used for the design and scheduling of the mine plan in this PFS, there is estimated to be 120 Mt of inferred resources above the cut-off grade used for this study, grading 0.28% Cu 0.20 g/t Au and 0.91 g/t Ag. This material is included in the current mine plan as waste, due to its speculative nature.

23 Adjacent Properties

Not applicable

24 Other Relevant Data and Information

Not applicable

25 Interpretations and Conclusions

25.1 Mineral Resource Estimates

Mineral resource estimates presented in this report represent the global mineral resources located at Josemaría as of 7 August 2015. 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
- Concentrate grade and smelting/refining terms
- Confidence in the modifying factors, including assumptions that surface rights to allow infrastructure such as tailings storage facilities and desalination plants to be constructed will be forthcoming.
- Delays or other issues in reaching agreements with local or regulatory authorities and stakeholders.
- Changes in land tenure requirements or in permitting requirements from those discussed in this Report.

25.2 Mineral Reserve Estimate

SRK confirms that the Josemaría mineral resource can be converted to a sizeable mineral reserve through application of open pit mining methods.

SRK perceives little risk in the mineral reserves as stated. However, there may be room for further optimization of the reserve.

As part of the initial evaluation of processing options for Josemaría, SRK considered feeding two processes – conventional comminution and flotation for copper mineralization and run-of-mine heap leaching for a gold-rich oxidized zone. At the level of study, the heap leaching option had marginal economics and it was decided not to include it in the PFS. Thus, the associated mineral resource was treated as waste in the current mine plant. However, the opportunity remains and should be investigated further as part of future work.

Presently, the waste storage facilities to the west of the pit are offset from the pit crest to ensure stability of the pit slopes. This places a practical constraint on the capacity of the waste facility. While there is additional waste storage capacity availability on the opposite side of the ridge to the pit (i.e. to the south), this comes with a haulage penalty. There may be opportunities to consider additional mineral resources for conversion to reserves if waste haulage costs allow waste to be transferred from west of the pit to south of the ridge. Conversely, the current plan of hauling the bottom-most waste in the final phase to the highest levels of the waste dump may not be optimal. While SRK validated the economics of the final pit phase, there may room for improvement to remove high-cost ore from the plan.

There may also be opportunities to refine the mineral reserves by considering flexible cut-off grade strategies that fully consider the balance of direct mill feed and long-term stockpiling. Optimization of the site general layout could result in additional low-grade stockpile capacity which could allow for an increase in the mid-grade ore cut-off grade. This would mean higher grade mill feed in early years and thus improved economics. However, this same measure could decrease the overall mineral reserve as more material will need to pass the higher stockpiling cut-off grade versus the direct mill feed cut-off grade.

Finally, as alternate technology continues to be considered for the Josemaría project to reduce costs (e.g. autonomous drilling), these costs savings, as well as those realized in this PFS, should be reflected in updated optimization analyses. Alternate pit shell options, in addition to the waste storage considerations above, could have an impact on the mineral reserves.

25.3 Pit Geotechnical

A key conclusion of the PFS gap analysis was that the stability of the Josemaría pit walls will be dictated by rock mass strength and major structures. In this study, geotechnical data was collected from historical resource and geotechnical drillholes. The core used to collect the new data was of poor quality because it had been handled, transported and cut for resource testing. This shortcoming was somewhat addressed by accompanying this core logging with semi-qualitative logging of the full core photos.

Due to the paucity of structural data and PFS timing constraints, the historical large-scale structures data was not included in the assessment. SRK recognises that these input data limitations mean that the recommended design pit walls in this study may be steeper than achievable. Major structures can have a significant influence on the stability of pit walls and may require the flattening of wall angles to achieve the project acceptability criterion. Such wall angles would have obvious ramifications on the project economics for the Josemaría site and therefore more thorough investigations and analysis are required for the project to advance.

25.4 Mining Methods

The Josemaría project is amenable to open pit mining methods. However, as the project is in a high elevation operating environment, where labour productivity and equipment utilization can be impacted, SRK has evaluated 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, it is SRK's opinion that autonomous haulage has sufficiently matured to be considered for Josemaría. There could be further opportunities to remove labour from the harsh environment at Josemaría by implementing autonomous drilling as well.

Though SRK has endeavoured to implement a cut-off grade strategy to optimize the ore feed to the mill, there remain opportunities to further optimize this strategy. By Year 10, when low-grade stockpile levels are depleted and mining rates begin to drop, it should be possible to increase the medium-grade cross-over grade, continue to mine at higher rates and enhance the ore quality to the mill.

25.5 Metallurgical Testwork

Metallurgical studies to support the design of the Josemaría project were completed in two tranches between 2015 and 2018.

The testwork completed in 2015 by SGS as Phase II is considered most relevant to the project and predictive metallurgical results are best estimated from this testwork. Subsequent testing by ALS in 2018, completed to validate earlier results, is also relevant. However, the age and extent of oxidation of the ALS samples is considered to have potentially influenced the metallurgical results.

To the extent known, the metallurgical samples tested as part of the SGS 2015 Phase II and ALS 2018 test programs are representative of the lithologies and types of mineralization and the Josemaría deposit as a whole.

The LOM average mill feed at Josemaría is expected to contain 0.29% Cu and 0.21 g/t Au. LOM concentrate grade is expected to be 25.1% Cu and 14.5 g/t Au.

In general, the flowsheet developed in the SGS 2015 test program, and further tested in the ALS 2018 testwork, is technically feasible for the Josemaría deposit mineralization.

There are no outstanding metallurgical issues related to the production of a copper concentrate from the lithology-based geometallurgical domains as tested.

25.6 Recovery Plan

The recovery plan for metal values at Josemaría is conventional and well-proven in practice around the world.

The process facilities are designed to treat ore from the open pit at a nominal rate of 150 ktpd. This can be considered a large copper mine and concentrator facility, but there are several similarly sized copper mine and concentrators operating successfully.

The process design is based on established technologies and the largest available, commercially proven equipment sizes. The comminution circuit employs three-stage crushing, including a HPGR-based tertiary crushing stage, followed by closed circuit ball milling. Ground ore is treated

in a conventional flotation circuit, consisting of rougher-scavenger flotation, regrinding of the rougher concentrate, and three stages of cleaner flotation,

The process plant will produce a single copper concentrate. Gold and silver values will be recovered to the copper concentrate and are expected to be payable at smelter rates. Overall recoveries from ore to concentrate are expected to be 86% for copper, 71% for gold and 59% for silver.

25.7 Tailings Management Facility

Ausenco developed a phased TSF design that has a storage capacity of 1,008 Mt of tailings, corresponding to approximately 20 years at a rate of 54 Mt/y and was designed in accordance to international and national (Argentina) best available technology (BAT). The TSF design considered the following:

- The TSF location was selected after a trade-off study evaluated six alternative sites to the preferred site. In an earlier assessment of the preferred site, no seismic or geotechnical fatal flaws were identified.
- The design considers construction, operations and closure
- The TSF main embankment will be expanded in stages using the centerline construction method while the two small embankments (north and south) will be constructed in stages using downstream method
- A composite liner system (geomembrane liner, low permeability zone and filter zone) along the upstream sides of the three embankments
- Slurried tailings will be discharged from the perimeter of the TSF embankments and other critical discharge points around the facility, creating supernatant pond that moves over time from the main embankment to the north embankment
- A site-specific water balance and hydrology (surface water and spillway) design was developed for the project that included the TSF
- The TSF has been designed with a minimum stability FoS of 1.5 static and 1.2 post-earthquake
- Diversion channel will be constructed to capture water within the TSF watershed and convey it to the TSF to reduce water make-up requirements, sized to convey the 10-year, 24-hour storm event
- The TSF spillway has been sized to convey the PMP event
- Instrumentation, such as piezometers and inclinometers, will be installed in the TSF embankments and foundation during construction of the starter embankments to monitor and assess the performance of the embankments

- The reclamation and closure activities for the TSF will focus on stabilization of the exposed tailings surfaces and limiting dust generation by selective placement of erosion resistant alluvium and establishing a permanent surface flow structure through the facility to the proposed spillway located next to the northwest embankment.

25.8 Environmental Studies, Permitting, and Social or Community Impact

NGEx 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 the Mining Code – Environmental Protection for Mining Activity. This is a recognized process with successful precedent in the San Juan province of Argentina. There are no known environmental issues that could materially impact the ability of NGEx to extract the mineral resources at the Josemaría project.

26 Recommendations

26.1 Mineral 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 recommended in Section 26.3, as it can share many of the same infrastructure, support and personnel costs. A total of 22 drill holes comprising 6,570 m of drilling is recommended for this phase of work. Drill core should be HQ diameter as much as possible.

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 metallurgical testwork program recommended in Section 26.5 of this report. 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 two 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. The budget for this work program is \$5.2M, including drilling costs, support costs, labour, assaying, geological interpretation and resource modelling.

26.2 Mineral Reserve Estimate

As outlined in Section 15.8, there are multiple opportunities to optimize the Josemaría project. These could have an impact on the mineral reserve estimate – either increasing it or decreasing it – but increasing project value in any case. To realize these opportunities, SRK recommends to:

- Consider if there is any scenario under which the gold oxide zone may be economically processed to become mineral reserve
- Conduct a trade-off study on waste storage facility location and capacity versus haulage. This could allow for expansion of the ultimate pit and reserves, or conversely, a reduction.
- Conduct a trade-off study of low-grade ore stockpile capacity versus cut-off grade strategy to optimize mill head grades and project economics

- Account for the latest mining costs, including addition advantages of new technologies in update pit optimizations
- Update mine plan based on new mineral resource estimate to allow for the declaration of a Proven mineral reserve

26.3 Pit Geotechnical

SRK recommends that an FS-level investigation program include:

- A comprehensive geotechnical photo logging program of the historical resource drillholes to improve the project understanding of the major structures characteristics and their spatial constraints, and to 'engineer' the RQD values.
- Targeted geotechnical drillholes in all design domains, pit sectors and lithology units including early porphyry unit which was not encountered in the PFS field program drillholes.
- Geotechnical drillholes, or geotechnical processing of resource drillholes, to investigate the north-south structural corridor major structures.
- Full geotechnical logging, including alteration and micro-defect intensity logging, ideally to be also conducted in the resource drillholes.
- Develop a larger (statistically significant) database of field and laboratory strength test results.
- ATV/OTV survey of all completed geotechnical drillholes and targeted resource drillholes for structural characterization.
- Packer tests in all design domains, lithology units, focusing on major structures encountered, to develop a larger (statistically significant) hydrogeological database.
- Install Vibrating Wire Piezometers (VWPs) to monitor groundwater levels and the hydrogeological regime. Ideally these would be installed prior to drilling nearby drillholes to measure hydraulic response due to drilling.
- Install standpipe piezometers for groundwater monitoring and baseline quality sampling.
- Comparison of historic core with photos for deterioration/weathering assessment and accelerated weathering test.

The results of these investigations would allow update of the project geotechnical model and refinement of the geotechnical design sectors for further slope stability analyses and improved confidence in the pit wall geotechnical design inputs. The cost for this program is estimated to be \$4,100,000, including direct drilling and support costs.

26.4 Mining Method

Commensurate with SRK's conclusions on the mining method for Josemaría, SRK recommends that additional autonomous technologies, such as drilling, be considered for the project. As well, SRK recommends that the cut-off grade strategy be further optimized to sustain higher grade feed

to the mill by maintaining higher mining rates when the mine plan currently calls for these to decrease.

26.5 Metallurgy and Processing

There are several opportunities with respect to processing and engineering design that could further improve the viability of the Josemaría project and may be evaluated as part of future work.

Examples of potential processing opportunities to add value include:

- The Josemaría deposit is endowed with a near surface, oxidized mineralization that contains appreciable gold. A high-level evaluation of the option to recover and treat this ore separately suggested that there was no economic benefit. However, because of the apparent ability to easily segregate this oxidized gold-bearing material and the material's amenability to leaching for recovery of the gold values, this opportunity could prove viable should the tonnage and ore of the oxide gold ore be increased through additional drilling/sampling and definition.
- Flotation recoveries concluded as part of the PFS are somewhat conservative, owing to the limited number of bankable test results generated in early testwork and to the potential for age and oxidation to have affected results in the recent testwork. There is an opportunity to improve recovery predictions applied to the economic model in the presence of additional, bankable testwork.
- Sizing of the high pressure grinding rolls (HPGR) in the tertiary crushing circuit was likewise conservative as a result of the limited design information that could be reliably derived from the available test results. With additional, dedicated testwork, the HPGR equipment sizing could potentially be reduced with an obvious positive influence on project economics.

Further testwork to support grinding and flotation design, specifically including variability testing with ore samples representing the distinct lithologies and their relative distribution as predicted by the mining plan, is recommended to establish the variances in metallurgical performance across the deposit. Specific characterization and testing should include:

- Mineralogical analysis with estimates of sulphide mineral liberation
- Testwork to support use of High Pressure Grinding Rolls (HPGR) and to further define equipment sizing and requirements
- Comminution tests on variability samples
- Open and locked-cycle flotation tests on variability tests
- Cleaner concentrate dewatering (settling and filtration) tests
- Waste rock and flotation tailings geochemical characterization, including static and kinetic testing as appropriate for geometallurgical domain composite and variability samples
- Tailings dewatering and consolidation testing, as well as slurry rheology tests

This additional testwork to be incorporated into the process design is estimated at a cost of about \$1,200,000.

26.6 Tailings Management Facility

To achieve feasibility study level design of the TSF, the following are recommended:

- Geotechnical investigations and laboratory testwork:
 - Additional geotechnical and hydrogeological drilling and test pits in the TSF area to investigate and confirm foundation conditions (specifically the extent of the colluvial apron), characteristics and depth to bedrock
 - Additional test pits and drilling to confirm suitability and availability of borrow materials for embankment construction
 - Additional laboratory index testing (including compaction tests) and strength and permeability tests on potential borrow materials for embankment construction
 - Laboratory testing to confirm the geochemical characteristics of the tailings to be utilized in the design of the TSF
 - Laboratory testing to confirm the physical characteristics of the tailings, such as material classification, strength, permeability and consolidation testing of the different tailings streams to determine settlement and permeability characteristics, and beach slope testing
- Design studies and analyses:
 - Confirmation of geochemical characterization and potential reactivity of tailings and from on-going waste characterization studies
 - Development of a tailings deposition strategy and consolidation of tailings to optimize material handling, and tailings discharge line and reclaim barge locations throughout operations using industry standard software
 - Evaluation of the potential impact of sloping sub-aerial and sub-aqueous tailings beach areas. Consideration of tailings beach configurations may result in modifications to staged embankment heights and the tailings deposition strategy.
 - Stability analyses for the TSF, including static and pseudo-static. Perform simplified deformation analysis for the TSF or more rigorous deformation analysis using FLAC or similar software if simplified method shows significant deformation.
 - Stability analyses for the TSF, including static and pseudo-static. Perform a rigorous deformation analysis for the TSF using FLAC or similar software along with stability analyses for the TSF.
 - Perform a failure mode and effect analysis (FMEA) on the TSF where there is potential to impact the environment or external parties in the event of a failure

- Look at potentially injecting a concreting agent into the tailings stream in the last couple of tailings deposition layers in the TSF. This will crust the top of the TSF and allow for the placement of the alluvium with machinery as well as reducing the potential for aeolian erosion and capital costs.

26.7 Infrastructure

The following engineering study work is recommended in the next phase of the project for infrastructure scope (estimated cost of approximately \$3,000,000):

- Optimization of general arrangement and locations of mining and process plant-site facilities such as the camp and other on-site infrastructure, including:
 - Separation of primary crushers to two work fronts
 - Elimination of mid-point transfer towers
 - Relocation of coarse ore stockpile closer to process plant
 - Densification of process area facilities to be in closer proximity
 - Revise the process plant location and layout further to realize cost benefits. Specifically, there are opportunities to revise the primary crusher location in connection with the haul road from the pit to the crusher and the overland conveyor from the primary crusher to the main process plant to reduce total costs. There are also further potential savings to be gained with further detailed engineering around the crushing circuit and the grinding and flotation circuits.
- Design of power supply to pit operations
- Optimization of off-site infrastructure including:
 - Additional engineering to identify the most cost-effective transmission line routing and utility connection point. The power requirement for the Josemaría project is considerable and although the identified connection point in Argentina is viable, there is an opportunity to further evaluate options for power supply. These include a more proximate connection, site power generation or co-generation at site, or alternative arrangements with utilities suppliers with a view to reducing costs associated with upgrading the existing infrastructure to the connection point and/or costs associated with power transmission to site.
 - Optimization of water supply wells and pipeline design based on new data from 2018/2019 exploration and drilling program
 - Further design development of main access road and required port upgrades
- Hydrogeological studies including monitoring to substantiate that there is adequate ground water flow to support commercial operations
- Geotechnical studies, including geotechnical investigations of the plant site, TSF and WSF sites, and other project related locations including:

- Conduct further geotechnical engineering studies, which are currently underway, for the foundations of the TSF and WSF, including borehole drilling and test pit excavations to test all assumptions made in this report and determine the foundation, borrow and fill placement conditions for design
- Complete detailed geotechnical characterization of the materials that will be used to construct the TSF and WSF through field investigation and laboratory programs
- Engineering studies, including water management and treatment, TSF and WSF design, including
 - More detailed analysis of the water and load balance to predict the accumulation of mill reagents and their degradation products in the process water circuit
 - Updated pit groundwater numerical model and inflow estimates, and incorporating seasonal effects to assess high and low water conditions
 - Updated TSF and WSF design based on additional field investigation results
 - Optimization of tailings deposition plan and waste rock placement sequence to match pit development and mill output
 - Confirmation of geochemical characterisation and potential reactivity of tailings and waste rock from on-going waste characterisation studies
 - Review and confirmation of TSF seepage predictions and seepage control measures and requirements
 - Review and reassessment of trade-off study for embankment construction material options based on the most current design requirements and cost estimate information
 - Water balance needs to be updated in the feasibility study using updated meteorological data, stream flow data along with open pit dewater requirements, and update of TSF water balance once the feasibility field investigation, laboratory programs are completed.

26.8 Hydrology and Surface Water Management

The following is recommended for water management:

- Design of water control structures should be considered for Years -1 (pre-stripping), 1, 3, 5, 10, 15 and end of mine life along with estimations of the potential for acid generation from the waste dumps, pit and TSF to refine on-site control measures.
- The designs proposed for surface water management have been carried out on topography with 5 m spacing. For the next level of study that 1-2 metre topography should be used.
- The development of a maintenance strategy with the purpose of confirming the efficacy and operation of hydraulic structures is required.

26.9 Environmental Studies, Permitting, and Social or Community Impact

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.

26.10 2019 Work Program

The work program for the upcoming year consists 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
- Pit geotechnical field work and data analysis
- Metallurgical testing
- Infrastructure studies, including:
 - Geotechnical studies of the plant site, TSF and WSF sites, and other project related locations
 - Hydrological and hydrogeological studies
 - Tailings and TSF characterization studies (geotechnical investigations and laboratory testwork)
- Continuation of existing environmental baseline studies and social programs
- Initiation of a Feasibility Study

The estimated cost for completing this work is summarized in Table 26.1.

Table 26.1: Josemaría work program cost estimate

Program Component	Cost Estimate (\$000)
Resource drilling	5,200
Pit geotechnical - drilling and engineering	4,100
Metallurgical testing	1,200
Infrastructure studies - field program and laboratory work	3,000
Environmental baseline studies and social programs	1,000
Feasibility study	3,000
Total Cost	\$17,500

27 References

- BGC Ingeniería Ltda., 2015. Informe de Clima y Meteorología – Proyectos Josemaría y Filo del Sol. Report prepared for MFDO and DEPROMINSA.
- Durán, V., Lucero, G., Estrella, D., Castro, S., y L. Yerba, 2014. Prospección arqueológica preliminar. Proyectos: Josemaría y Filo del Sol. Dpto. Iglesia, provincia de San Juan. Temporada 2013-2014. Informe presentado a Desarrollo de Prospectos Mineros S.A. (DEPROMINSA). Filo del Sol Exploraciones S.A.
- ILO, 1989. C169 - Indigenous and Tribal Peoples Convention, 1989 (No. 169). International Labour Organization. www.ilo.org
- Knight Piésold Consulting, 2018a. Josemaría Project Hydrometeorology Assessment.
- Knight Piésold Consulting, 2018b. Estudio de Línea de Base Ambiental Acotada. Estudio de Línea de Base Limnológica. VA201-00605/01-02-INF-B
- Knight Piésold Consulting, 2018c. Estudio de Línea de Base Ambiental Acotada. Estudio de Línea de Base de Flora y Fauna Verano de 2018. VA201-00605/01-03-INF-B
- Knight Piésold Consulting, 2018d. Estudio de Línea Base Ambiental Acotada. Estudio de Línea Base Arqueológica. VA201-00605.01-02-INF-B. 6 junio 2018.
- Maidment, D. R., 1993. Handbook of Hydrology, McGraw-Hill Inc.
- Manzanares, B., 2015. Resumen Ejecutivo. Estudios para la Línea de Base Ambiental. Proyecto José María. Octubre de 2015.
- McPhaden, M. J., 2003. Tropical Pacific Ocean heat content variations and ENSO persistence barriers. *Geophysical research letters*, 30(9).
- Métodos Consultores Asociados, 2014a. Estudio de Ruido Ambiental, Proyectos Josemaría y Filo del Sol.
- Métodos Consultores Asociados, 2014b. Medición de Vibraciones Terrestres, Proyectos Josemaría y Filo del Sol.
- Molina, A., 2013. Caracterización de la Diversidad Biológica. Proyectos: Caballo, José Maria, La Chola, Solitario 17 y Potro.
- Mpodozis, C., and Kay, S.M., 2003, Neogene tectonics, ages and mineralization along the transition zone between the El Indio and Maricunga mineral belts (Argentina and Chile 28°-29°S) [abs.]: Congreso Geológico Chileno, 10th, Concepción, 2003, CD-ROM, 1p.

- Ovalle, A., Quiñones, C., Quezada, C., Frost, D., Khera, V., Zandonai, G., 2016. Constellation Project incorporating the Los Helados Deposit, Chile and the Josemaría Deposit, Argentina NI 43-101 Technical Report on Preliminary Economic Assessment, 31 March 2016.
- Pittaluga, M., 2015. Estudio de Suelos en la Zonas de Influencia de Los Proyectos Josemaría y Filo del Sol Departamento Iglesia, Provincia de San Juan.
- Seedorf, E., Dilles, J.H., Proffett, J.M.Jr., Einaudi, M.T., et al., 2005, Porphyry deposits: Characteristics and origin of hypogene features; Society of Economic Geologists, Economic Geology 100th Anniversary Volume, pp 251-298.
- SGS Minerals S.A., 2015. Programa Metalúrgico de Conminución y Flotación en Mineral de Cobre - Oro, Proyecto Josemaría – Fase I y Fase II – 2015.
- Sillitoe, R.H., Devine, F.A.M., Sanguinetti, M.I., and Friedman, R.M., (in preparation): Geology of the Josemaría Porphyry Copper-Gold Deposit, Argentina: Formation, Exhumation, and Burial in Two Million Years; for submittal to Economic Geology.
- Sinclair, W.D. (2007) Porphyry Deposits. In: Goodfellow, W.D., Ed., Mineral Deposits of Canada: A Synthesis of Major Deposit-Types, District Metallogeny, the Evolution of Geological Provinces, and Exploration Methods, Geological Association of Canada, Mineral Deposits Division, Special Publication, Canada, Newfoundland, 223-243.
- Smith, M., Allen, R., & Pereira, L., 1998. Revised Fao Methodology Xa9848174 For Crop-Water Requirements. Management of nutrients and water in rainfed arid and semi-arid areas, 51.
- Thorntwaite, C.W., 1948. An approach toward a rational classification of climate, Geographical Review. (American Geographical Society) 38 (1): 55–94.

28 Date and Signature Page

This technical report was written by the following “Qualified Persons” and contributing authors. The effective date of this technical report is 20 November 2018.

Fionnuala Devine, P.Geo. <i>Merlin Geosciences Inc.</i>	Gino Zandonai, CP (RM CMC) <i>DGCS SA</i>	Bruno Borotraeger, P.Eng. <i>Knight Piésold Ltd.</i>
Robert McCarthy, P.Eng. <i>SRK Consulting (Canada) Inc.</i>	Neil Winkelmann, FAusIMM <i>SRK Consulting (Canada) Inc.</i>	Andy Thomas, P.Eng <i>SRK Consulting (Canada) Inc.</i>
Michael Royle, P.Geo. <i>SRK Consulting (Canada) Inc.</i>	Robin Kalanchey, P.Eng. <i>Ausenco Engineering Canada Inc.</i>	Scott Elfen, P.E. <i>Ausenco Engineering Canada Inc.</i>

All data used as source material plus the text, tables, figures and attachments of this document have been reviewed and prepared in accordance with generally accepted professional engineering and environmental practices.