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Compañía AQM Copper Inc.

Zafranal Project, Peru

Technical Report on the Pre-Feasibility Study

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Table of Contents

Glossary

Abbreviation, Contraction or Symbol	Full Expression or Meaning
%	Percent
°C	Degrees Celsius
µm	Micrometre
3D	Three dimensional
3LPE	3 Layer Polyethylene
AACE	Association for the Advancement of Cost Engineering
AAS	Atomic Absorption Spectrophotometry
AC	Alternating current
AEP	Annual exceedance probability
Ai	Abrasion index
AMT	Audio-Magnetotelluric (method or technique)
ANA	Autoridad Nacional del Agua (National Water Authority)
ANC	Acid neutralisation capacity
ANFO	Ammonium nitrate - fuel oil
AQM	AQM Copper Inc.
Ar-Ar	Argon-argon
ARD	Acid rock drainage
ASE	Asesoría y Servicios Especializados S.A.
ASTM	American Society for Testing and Materials
Au	Gold
AUD	Australian dollar
Autodema	Autoridad Autónoma de Majes (Majes Autonomous Authority)
Axb	SAG mill comminution test output
BBWi	Bond ball mill work index
BDV	Block dispersion variance
BOE	Basis of Estimate

Abbreviation, Contraction or Symbol	Full Expression or Meaning
BOO	Build, own, operate
BOOT	Build, own, operate, transfer
BRIC	Brazil, Russia, India and China
BS	Base Salary
BVL	Bolsa de Valores de Lima (Lima Stock Exchange)
CAD	Canadian dollar
Capex	Capital Expenditure (capital cost)
CCR	Crusher control room
CCTV	Closed circuit television
CDA	Canadian Dam Association
CFT	Contracts for Tender
CI	Confidence interval
CIF-FO	Cost, Insurance and Freight - Free Out discharge
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIR	Cox-Ingersoll-Ross
cm	Centimetre
CM	Construction Management
CMZ	Compañía Minera Zafranal S.A.C.
CN	Cyanide
COES	Comité de Operación Económica del Sistema Eléctrico Interconectado Nacional del Perú (Committee of Economic Operation of the National Electric Interconnected System of Peru)
COMEX	Commodity Exchange Division of the New York Mercantile Exchange
Cordillera	Mountain range (Spanish)
CSS	Close side setting
Cu	Copper
CuCn	Cyanide soluble copper
CuSS	Sulfuric acid soluble copper
CuT	Total copper
CV	Coefficients of variation

Abbreviation, Contraction or Symbol	Full Expression or Meaning
CWi	Crushing work index
D	Disturbance factor
d	Day
DAM	Delegated Authority Manual
DC	Direct current
DCF	Discounted cash flow
DCS	Distributed Control System
DDH	Diamond drill holes
DHI	DHI Peru S.A.C.
DHSS	Drill hole spacing study
DWI	Drop weight index
E&I	Electrical and instrumentation
ECA	Estándares de Calidad Ambiental (Environmental Quality Standards)
Ecs	Specific energy (comminution)
EDA	Exploratory data analysis
EDGM	Earthquake design ground motion
EDM	Early dark micaceous
EIA	Estudio de Impacto Ambiental (Environmental Impact Assessment)
EIA-d	Estudio de Impacto Ambiental detallado (Environmental Impact Assessment detailed)
EIA-sd	Estudio de Impacto Ambiental semi-detallado (Environmental Impact Assessment semi-detailed)
EMP	Environmental management plan
EPCM	Engineering, procurement and construction management
ERP	Enterprise resource planning
ERT	Electrical resistivity tomography
ET	Evapotranspiration
Etapa	Stage / phase (Spanish)
EUR	Euro (currency)
FBE	Fusion bonded epoxy
FCA	Free carrier (Incoterms)

Abbreviation, Contraction or Symbol	Full Expression or Meaning
FEL	Front-end loader
FO	Fine oxide
FOT	Free on truck
FS	Feasibility study
g	Grams
G&A	General and administration
GA	General Arrangement (drawing)
GAC	Gangue acid consumption
GAC_C	Gangue acid consumption - carbonate
GAC_NC	Gangue acid consumption - non-carbonate
GCL	Geomembrane/geosynthetic clay liner
GDP	Gross domestic product
GPS	Global positioning system
GSI	Geological strength index
GST	Goods and services tax
GT	Grade-tonnage (curve)
h	Hour
ha	Hectare
HAZOP	Hazard and Operability Study
HDPE	High-density polyethylene
HERCO	Amec Foster Wheeler in-house software for Hermitian correction method
HGL	Hydraulic grade line (m).
HSEC	Health, safety, environmental and community
HV	High voltage
ICP	Inductively coupled plasma
ICSG	International Copper Study Group
ID2	Inverse distance squared (estimator)
IDF	Inflow design flood
IFC	International Finance Corporation
IGV	Impuesto General a las Ventas (Peruvian GST or VAT)

Abbreviation, Contraction or Symbol	Full Expression or Meaning
IMO	International Maritime Organization
INEI	Instituto Nacional de Estadística e Informática (National Institute of Statistics and Informatics)
INGEMMET	Instituto Geológico, Minero y Metalúrgico (Metallurgical Mining Geological Institute)
INRENA	Instituto Nacional de Recursos Naturales (National Institute of Natural Resources)
IP	Induced polarization
IRR	Internal rate of return
IUCN	International Union for the Conservation of Nature
JKDWT	JKMRC drop weight tests
JKMRC	Julius Kruttschnitt Mineral Research Centre (at the University of Queensland)
JV	Joint venture
K	Conductivity
kg	Kilogram
km	Kilometre
KNA	Kriging neighbourhood analysis
KP	Knight Piésold
kPa	Kilopascal
kt	Kilotonne
kW	Kilowatt
L	Litre
LAN	Local Area Network
Lb or lb	Pound
LBMA	London Bullion Market Association
LME	London Metal Exchange
LOM	Life of mine
LPP	Labour plus parts (contract)
LV	Low voltage
m	Metre
m^2	Square metre

Abbreviation, Contraction or Symbol	Full Expression or Meaning
m^3	Cubic metre
MA	Mechanical availability
MAOH / MAOP	Maximum allowable operating head (m) / pressure (kPa)
MAQM	Minera AQM Copper Peru S.A.C.
MARC	Maintenance and repair contract
masl	Metres above (mean) sea level (elevation)
mbgl	Metres below ground level
mbgs	Metres below ground surface
MCC	Motor control centre
MCE	Maximum credible earthquake
MDE	Maximum design earthquake
MEF	Ministry of Economy and Finance
mg	Milligrams
MIBC	Methyl isobutyl carbinol
MIDIS	Ministerio de Desarollo e Inclusión Social del Peru Ministry of Development and Social Inclusion of Peru
MINAGRI	Ministerio de Agricultura y Riego del Perú Ministry of Agriculture and Irrigation of Peru
MINCULT	Ministry of Culture
MINEDU	Ministry of Education
MINSA	Ministry of Health
ML	Metal leaching
MLA	Mineral liberation analyses
mm	Millimetres
Mm^3	Millions of cubic metres (used as an exception to SI convention)
mm^3	Cubic millimetres
MMC	Mitsubishi Materials Corporation
MMG	MMG Limited (formerly Minmetals Resources Limited)
Mo	Molybdenum
MPLs	Maximum permissible limits
mpy	Miles per year

Abbreviation, Contraction or Symbol	Full Expression or Meaning
Mt	Million tonnes or Megatonnes
MTO	Material take-off
MV	Medium voltage
mwc	Metres of water column
MX	Mixed (mineral zone)
NBS	National Bureau of Statistics
NCL	NCL Ingenieria y Construccion Limitada
NEA	National Energy Administration
NERs	Neutral earthing resistors
NGA	Next Generation Attenuation
NN	Nearest-neighbour
NPC	Net present cost
NPS	Nominal pipe size.
NPV	Net present value
NRMS	Normalized root mean square
NSR	Net smelter return
NTU	Nephelometric turbidity units
NYMEX	New York Mercantile Exchange
OBE	Operating basis earthquake
OCS	Operator control stations
OD	Outside diameter
OEFA	Organismo de Evaluación y Fiscalización Ambiental (Environmental Auditing Agency)
OGGS	Oficina General de Gestión Social (General Office of Social Management)
OK	Ordinary kriging
OOM	Order of Magnitude
Opex	Operating Expenditure (operating cost)
OSA	On-stream analyser
OX	Oxides
oz	Troy ounce

Abbreviation, Contraction or Symbol	Full Expression or Meaning
P&ID	Piping and Instrumentation Diagram
PAG	Potentially acid generating
Pampa	Plain expanse without tree vegetation
PAX	Potassium amyl xanthate
PCS	Process control system
PEA	Preliminary Economic Assessment
PEAU	Preliminary Economic Assessment Update
PEN	Peruvian Sol
PET	Potential evapotranspiration
PFD	Process flow diagram
PFS	Pre-feasibility Study
PGA	Peak ground acceleration
PLC	Programmable logic controllers
PLS	Pregnant leach solution
PLT	Point load test
PMF	Probable maximum flood
PMI	Purchasing Managers Index
PMP	Probable maximum precipitation
PP	Price participation
PPE	Personal protective equipment
ppm	Parts per million
PS	Pump station
PWM	Pulse width modulation
QA	Quality assurance
QC	Quality control
QP	Qualified person
Quebrada	Narrow and steep pass between mountains, ravine, canyon, watercourse, creek
RAMBO	Reliability, Accessibility, Maintainability, Buildability, Operability
RC	Reverse circulation (drilling)

Abbreviation, Contraction or Symbol	Full Expression or Meaning
RC	Refining charge (US cents per pound of copper)
RFA	Recommendation for award
RMR	Rock mass rating
RO	Reverse osmosis
ROM	Run-of-mine (ore)
RQD	Rock quality designation
RRR	Reserve requirement ratio
RWi	Rod mill work index
S	Siemens (unit of electrical conductivity)
s	Second
SAG	Semi-autogenous grinding
SBK	Single-block kriging
SCG	Social Capital Group
SDF	Spillway design flow
SEIN	Sistema Eléctrico Interconectado Nacional (National Interconnected Electric System)
SENACE	Servicio Nacional de Certificación Ambiental para las Inversiones Sostenibles (National Service for Environmental Certification of Sustainabilities Investments)
SENAMHI	Servicio Nacional de Meteorología e Hidrología del Perú (National Meteorology and Hydrology Service of Peru)
SER	Slip energy recovery
SF	Safety factor
SG	Supergene (mineral zone)
SG	Specific gravity
SHA	Shareholders Agreement
SHFE	Shanghai Futures Exchange
SI	Système International d'Unités (International System Of Units)
SLD	Single line diagram
SMC	SAG mill comminution (test)
SMP	Structural, mechanical and piping
SMU	Selective mining unit

Abbreviation, Contraction or Symbol	Full Expression or Meaning
SMYS	Specified minimum yield stress (of pipeline material)
SOW	Scope of works
SPI	SAG power index
SS	Sulfuric acid soluble
SX-EW	Solvent extraction-electrowinning
t	Tonnes
T&M	Time and materials (contract)
TAC	Total acid consumption
TC	Treatment charge
TCRC	Treatment charge, refining charge (mineral concentrate smelting)
TDC	Total direct cost
TDS	Total dissolved solids
Teck	Teck Resources Limited
Teck Peru	Teck Cominco Peru S.A.
TEM	Time-domain electromagnetic (method or technique)
TES	Tender enquiry summary
TIC	Total indirect cost
TMF	Tailings management facility
TML	Transportable moisture limit
TRS	Throughput rationalization study
TSX	Toronto Stock Exchange
TSX-V	TSX Venture Exchange
UEA	Unidad Económica Administrativa (Administrative Economic Units)
UA	Utilization of availability
UCS	Uniaxial compressive strength
UH	Hydrographic unit
USD	United States Dollar
USGS	United States Geological Survey
UTM	Universal Transverse Mercator
UV	Ultraviolet

Abbreviation, Contraction or Symbol	Full Expression or Meaning
V	Voltage
VAC	Alternating current voltage
VAT	Value-added tax
VDC	Direct current voltage
VES	Vertical electrical soundings
VIP	Value improvement process
VRF	Variance reduction factor
VSD	Variable speed drive
VVVF	Variable voltage variable frequency (VSD)
W	Watt
w/w	Weight to weight
WBS	Work breakdown structure
WC	Working capital
WGS 84	World Geodetic System 1984
WSS	Water supply system
Wt	Wall thickness
WWC	Well water collection (system)
XRD	X-ray diffraction
y	Year

1 Summary

1.1 Introduction

1.1.1 Issuer

AQM Copper Inc. (AQM), through its Peruvian subsidiary, Minera AQM Copper Peru S.A.C. (MAQM), is in a 50:50 joint venture, Compañía Minera Zafranal S.A.C. (CMZ), with Teck Resources Limited (Teck). MAQM is owned 60% by AQM Copper Inc. and 40% by Mitsubishi Materials Corporation (MMC); hence, AQM has a 30% beneficial interest in CMZ and the Zafranal Project (the Project). AQM, through MAQM, has been the operator for the Project since 2009.

1.1.2 Terms of reference

The Project is a greenfield porphyry copper project to develop a mining and processing operation and all associated infrastructure based on the Zafranal Main zone and Victoria zone deposits including:

- Open pit mining of up to 71 million tonnes per year of total material (sulfide ore, oxide and mixed mineral, and waste).
- Processing of sulfide ore through a flotation concentrator from 55 to 64 thousand tonnes per day (after initial ramp-up) depending on the grinding characteristics of the feed.
- Concentrate transport of 379 to 126 thousand tonnes per year (dry basis).

Ausenco Peru S.A.C. (Ausenco) was engaged by CMZ to co-ordinate the preparation of a preliminary feasibility study (PFS) and associated Technical Report for the Zafranal Project, and to prepare substantial elements of both.

1.2 Property Description and Location

1.2.1 Property area and location

The Property is 47,252 hectares in area and is located in southern Peru about 166 km by road (90 km linear distance) northwest of the city of Arequipa, within the Provinces of Castilla and Caylloma. The area to be developed is at elevations from 1,400 to 2,900 masl. The approximate centre of it is located at 16° 02’ 47” south latitude and 72° 15’ 13” west longitude using Universal Transverse Mercator (UTM) coordinates based on the World Geodetic System 1984 (WGS 84), Zone 18 South.

Figure 1-1 – Property location map

1.2.2 Mineral tenure and status

The Zafranal Property comprises 63 concessions and 9 mineral claims. CMZ has secured all legal rights on the mineral concessions comprising the Project; such rights have been recorded at the Public Registry and are enforceable before the Peruvian government and third parties. The proposed project development will involve 14 of these concessions totaling 9,552 ha.

1.2.3 Owner's title and interest

CMZ holds the right of access to the Property to carry out mineral exploration through a renewable lease contract entered into with Autodema, the regional government agency that manages the Majes-Siguas irrigation project. The lease contract term was last renewed on 11 September 2015 and is in force until 27 October 2017. As a result, surface rights for exploration have been secured for the medium term; however, CMZ is required to enter into an agreement with the land owners for the construction and exploitation stages, including surface rights for offsite infrastructure (power line, access roads and water line).

1.2.4 Others interests, royalties and encumbrances

The Zafranal Project property mineral rights are free and clear of recorded liens, encumbrances or agreements. There is a Government royalty of 1% minimum.

1.3 Topography, Elevation and Climate

The Project area sits on the boundary between the Pampa de Majes mid-elevation plain and the Andes Mountains. The topography of the Project area can be separated into three distinct types:

1. Mining area (including primary crusher) and waste dump areas from approximately 1,900 to 2,900 masl - steep and mountainous, dominated by elongated peaks intersected by gullies with slopes in excess of 70% and frequent rock outcroppings
2. Concentrator site from approximately 1,600 to 1,900 masl - the mountainous terrain becomes less dramatic and more undulated
3. Tailings management facility (TMF) from approximately 1,400 to 1,600 masl – a flatter alluvial basin surrounded by mountainous terrain.

The overall site arrangement is shown in Figure 1-2.

The climate in the Project area is temperate desert, based on the precipitation, evaporation and temperature regime, all of which have an altitude dependency. Average annual rainfall is 30 mm at 1,850 masl, Class A pan evaporation rate at the TMF is 3,085 mm/y and temperature for the average altitudes of the open pit, concentrator and TMF locations are respectively 13°C, 16°C and 18°C.

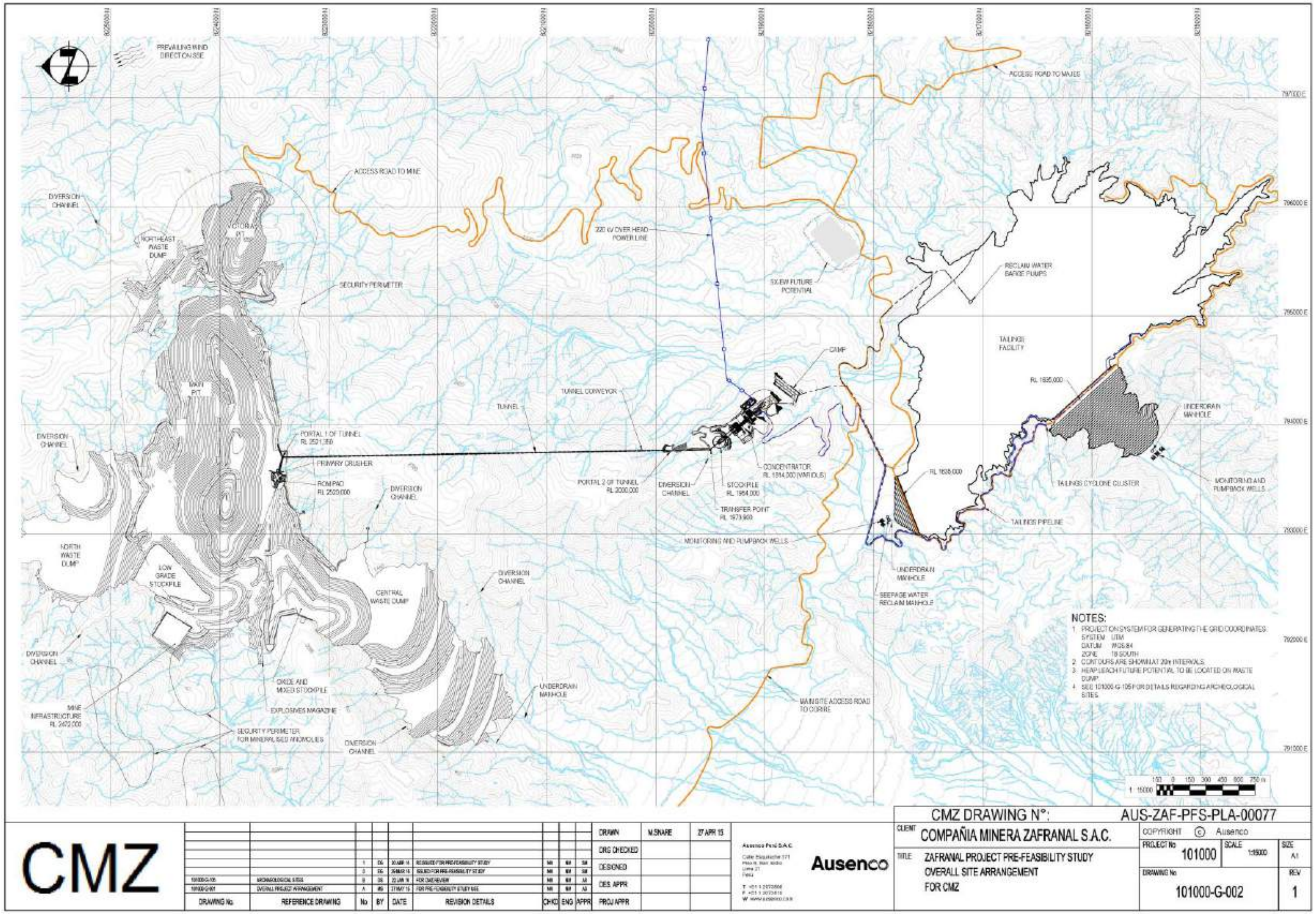

Figure 1-2 –Overall site arrangement

1.3.1 Site geotechnical, hydrogeology and hydrology

1.3.1.1 Site geotechnical

The geotechnical investigation was planned to determine foundation requirements for the design of facilities and to identify construction borrow material sources. There were no fatal flaws identified from the field investigation for site facilities. There are a number of local faults within the project area. Based on surface mapping of the main fault identified, the Incapuquio Fault, there was no evidence that the fault is active.

1.3.1.2 Site hydrogeology

FloSolutions S.A.C (FloSolutions) of Lima, Peru, was contracted by CMZ through Ausenco to complete a series of hydrogeological studies for the Zafranal project.

Within the Zafranal open pit, mining is expected to advance below the water table within 3 years. The Victoria open pit is not expected to be mined below the water table. As a result of dewatering, water elevations could drop by approximately 200 m within the central portion of the Zafranal Main pit. Dewatering impacts are expected to be limited to lowering the water table in the immediate area of the pit and potentially drying up local seeps located within the radius of dewatering. Groundwater inflow is expected to be low. After mine closure, a pit lake is expected to form within the lower 100 m of the pit.

Construction and operation of the TMF is anticipated to create a hydraulic mound within the underlying foundation that exceeds the pre-mining groundwater elevations. To counteract this effect, the TMF design will include measures to protect groundwater and minimize uncaptured seepage. Numerical modeling using the groundwater flow model developed for the Project indicates that uncollected seepage can be limited to less than 1 L/s. Seepage losses of these magnitudes are not expected to significantly affect the water quality of local or regional aquifers, which have no registered users. After closure, seepage will decline as the water levels at the TMF decrease and the tailings dewater. At the cessation of tailing deposition, seepage collected downstream from the TMF will be pumped back to the impoundment and evaporated.

Two potential construction water supply sources have been identified: (1) a deep, confined flowing artesian aquifer within Pampa Ganchos, located 5 km southwest of the TMF; and (2) fractured bedrock, located immediately southwest of the TMF in Quebrada Huacan.

A brackish groundwater resource, located beneath the Majes I irrigation area, located approximately 35 km south of the Zafranal Project, has been identified as a potential operational water supply source. The water supply could be developed through management of the brackish groundwater mound which has built up below the Majes farming areas since irrigation began in 1983.

Pumping from the brackish groundwater mound would have no negative effect on surface activities within the irrigation district and could be beneficial in reducing pore pressure in landslide prone areas. The brackish groundwater quality appears to be unsuitable for irrigation or animal drinking water use. It is, however, suitable for industrial use at the Zafranal Project.

1.3.1.3 Site hydrology

Both rainfall and runoff data were collected for the region and hydrological models were developed which predict the average annual flows are minor and not expected to significantly impact the Project. Extreme event flows were modelled for development of surface water management facilities. In addition, for the TMF the probable maximum flood (PMF) was calculated to be approximately 17 Mm^3.

1.4 Geology and Mineralization

1.4.1 Regional, local and property geology

The Zafranal Main zone deposit, located in the southern Peruvian copper porphyry belt, is a Cretaceous-aged copper-gold porphyry hosted in a window of Jurassic volcanoclastic rocks of the Guaneros and Chocolate formations intruded by Cretaceous-aged dioritic intrusions.

At the Victoria deposit, strongly deformed gneissose, locally mylonitic, rocks of indeterminate age are in contact with altered volcanic rocks. The deformation observed throughout the Main zone and Victoria zone is thought to be linked to strong shearing along district scale faults.

1.4.2 Mineralization

Porphyry copper-gold mineralization occurs within a large, roughly east-west trending hydrothermally altered zone that is more than 7 kilometres in length and as much as 1.7 kilometres in width. Copper occurs within veins and stockworks and as disseminations in both the Main zone and Victoria zone deposits. At the Main zone deposit, primary (hypogene) mineralization is capped by zones of locally oxidized secondary enrichment mineralization (supergene). Based on copper solubility, copper mineralization is categorized into leached, mixed, oxide, supergene and hypogene zones.

The Main zone is bound by two regional east-west trending faults believed to be linked to northwest trending strike-slip faults of the Incapuquio Fault system.

1.4.3 Deposit types description(s)

Zafranal represents a classic example of Andean-style porphyry copper-gold deposit. The emplacement of multiple fertile intrusions at or near structural zones and/or intersections controls mineralization. Mineralizing intrusions range in composition from monzonitic to dioritic and are mainly of Cretaceous age.

Oxidation of primary sulfides led to extensive leaching and re-deposition of copper solutions as enriched copper sulfides at or below the water table. Fluctuating water tables resulted in subsequent oxidation of enrichment blankets.

1.5 Status of Exploration and Development

In 2003, Teck prospected in the area and, following extensive mapping and geochemical sampling programs, completed 11,805 metres of diamond and RC drilling by 2005. AQM optioned the property from Teck in 2009, and initiated drilling in late 2009. To date, the Company has completed 109,531 metres of diamond drilling and 40,603 metres of RC drilling on the Property. Since 2010, AQM has disclosed one mineral resource estimate and two preliminary economic assessments, including mineral resource estimate updates for the Main and Victoria zones.

Having completed the preliminary feasibility study, CMZ will continue discussions with the local communities and authorities to determine the level of support for the Project, including access to the groundwater resource under Majes I as a potential water supply for the operation. Depending on the outcome of these discussions, CMZ will commence the feasibility study (FS) field investigations and engineering required to support the detailed Environmental Impact Assessment (EIA). The EIA elaboration and FS are planned to follow the field investigations and engineering inputs to the EIA.

1.6 Mineral Processing and Metallurgical Testing

1.6.1 Concentrator metallurgy

In general, all of the major rock types, regions and mineralization zones have been sampled and tested; however, not all of the geometallurgical domains have an adequate number of samples. While sample representativity is considered appropriate for the preliminary feasibility study, for the feasibility study more variability testing and more testing of samples from the first five years of production is recommended.

The main outcomes from the metallurgical test work review are as follows:

- The 75th percentile SAG and ball mill specific energy requirements (Ecs) were calculated in SABC circuit configuration as the basis for sizing the required mills (see Table 1-1).
- An optimum primary grind size P_{80} of 150 µm was selected for all ore types.
- Copper recovery of 89% and a final concentrate grade of 28% Cu were used for design based on the results from locked cycle tests
- Three-stage cleaning at elevated pH, in a combination of conventional tank and trough flotation cells with regrinding of rougher, scavenger and cleaner-scavenger concentrates to P_{80} of 40 µm is required for optimum copper recoveries and concentrate copper grades
- Chemical analysis of the copper concentrate indicates low penalty element levels

Comminution testing has shown that all three domain composites are moderately hard and abrasive but very competent; hence, much more power is required for SAG milling than ball milling. The hypogene composite was confirmed to be more competent than the others. As higher proportions of supergene and mixed ores are mined in the early years of the Project, this allows increased throughput.

Table 1-1 – Summary of 75th percentile specific energy requirements

	Mineral zone	Specific energy[a] (kWh/t) for SABC circuit
SAG mill	Supergene	9.2
	Hypogene	11.0
	Mixed	8.7
Ball mill	Supergene	5.1
	Hypogene	4.8
	Mixed	4.4
Total	Supergene	14.3
	Hypogene	15.8
	Mixed	13.1

[a] With RQD effect, assuming an average RQD of 56.3%

1.6.2 Concentrator recovery and concentrate grade estimates

The recovery and concentrate grade models for each of the geometallurgical flotation domains are summarized in Table 1-2. For the overall copper recovery calculation, a cleaner stage copper recovery loss of 2% has been allowed, based on the locked cycle test work.

Table 1-2 – Recovery and grade model algorithms for geomet flotation domains

Domain code	Domain name	Algorithm	Cu conc grade (%)	Cu rec. final (%)	Au rec. final (%)
Lo_S	Low sulfur	S/Fe < 0.2	0	0	0
Lo_CuSS	Low dilute sulfuric acid soluble copper	CuSS/CuT ≤ 0.15	37	89	52
Mid_CuSS	Mid-range dilute sulfuric acid soluble copper	0.15 < CuSS/CuT ≤ 0.3	34	84	55
Hi_CuSS	High dilute sulfuric acid soluble copper	CuSS/CuT > 0.3	32	77	52
Lo_CuCn	Low cyanide soluble copper	CuCn/CuT ≤ 0.3	28	90.5	56

1.7 Mineral Resource Estimate

Geological logging and assay results from 295 core holes totalling 95,618.7 m and 88 reverse circulation (RC) holes totalling 27,041 m completed by CMZ geologists to 16 February 2015 were used as the basis for preparation of three dimensional (3D) wireframe models of geological structures, lithology, alteration, and mineral zonation envelopes for the Zafranal Main and Victoria deposits. The 3D wireframe models were prepared under the direction of Politax S.A. (Politax).

A block model was constructed in Vulcan software with block dimensions of 15 m x 15 m x 12 m high. The main grade variables: copper - total (CuT), Au, copper - acid soluble (CuSS), and copper - cyanide soluble (CuCn) were interpolated into the blocks generally by ordinary kriging (OK) in three passes. An inverse distance squared (ID^2) estimator in two passes was used in the Mixed zone domain and late dykes. Additional grade variables including: Ag, As, Bi, Fe, and, S were interpolated using an inverse distance squared (ID^2) estimator in two passes. Density (specific gravity, SG) was interpolated into density domains using a combination of OK in three estimation passes and ID^2 in two passes.

A separate two-pass ordinary kriging was run to generate the sample counts and distances used for Mineral Resource classification followed by category smoothing. Blocks were classified based on a combination of factors including the number of holes used for each block, the distance to the nearest composites, and CuT mineral zonation.

The Mineral Resource is classified in accordance with the Canadian Institute of Mining, Metallurgy, and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014). In addition to criteria such as sufficient geological continuity, grade continuity, and data integrity, a guideline of drill hole spacing sufficient to predict potential production with reasonable probability of precision over a selected period of time was incorporated into the confidence classification.

To assess reasonable prospects for eventual economic extraction, it was assumed that the Zafranal deposit would be mined utilizing open pit mining methods under a conceptual scenario of 55,000 tonnes per day production using conventional flotation to produce a concentrate grading 28% copper with credit for gold.

Mineral Resources were tabulated within the pit at a cut-off grade of 0.15% CuT. This is above an operating breakeven cut-off grade that covers processing, tailings, and G&A operating costs within the resource pit. Table 1-3 shows the estimated mineral resources reported at a 0.15% CuT cut-off. The effective date of the mineral resource is 14 December 2015.

Table 1-3 – Mineral resource estimate for the Zafranal deposit based on a 0.15% CuT cut-off effective date 14 December 2015, Peter Oshust, P Geo

Classification	Tonnage	Grade		Contained metal	
	(Mt)	CuT (%)	Au (g/t)	Cu (Mlb)	Au (Moz)
Measured					
Mixed	-	-	-	-	-
Supergene	83.3	0.58	0.07	1,056	0.20
Hypogene	120.5	0.28	0.07	744	0.28
Total measured	203.8	0.40	0.07	1,801	0.47
Indicated					
Mixed	23.5	0.28	0.12	146	0.09
Supergene	100.3	0.53	0.07	1,176	0.21
Hypogene	139.7	0.26	0.06	804	0.28
Total indicated	263.5	0.37	0.07	2,126	0.58
Measured and Indicated					
Mixed	23.5	0.28	0.12	146	0.09
Supergene	183.6	0.55	0.07	2,234	0.40
Hypogene	260.2	0.27	0.07	1,543	0.56
Total measured and indicated	467.3	0.38	0.07	3925	1.05
Inferred					
Mixed	7.8	0.22	0.09	37	0.02
Supergene	8.7	0.30	0.04	57	0.01
Hypogene	4.9	0.18	0.03	20	0.00
Total inferred	21.4	0.24	0.06	114	0.04

Notes:

1. Mineral resources are reported inclusive of those mineral resources that have been converted to mineral reserves.
2. Mineral resources are reported within a constraining pit shell developed using Whittle™ software. Assumptions include metal prices of US$3.50/lb for Cu and $1,400/oz for Au; process recoveries of 86% for Cu and 50% for Au in supergene, 86% recoveries for Cu and 50% recoveries for Au in mixed, and 89% for Cu and 50% for Au in hypogene, US$1.58/t of mining at 2,534 masl plus $0.01/bench downward and $0.03/bench upward. US$5.45/tonne for processing, and US$0.38/tonne for G&A.
3. Assumptions include 100% mining recovery.
4. An external dilution factor was not considered during this resource estimation. Internal dilution within a 15 m x 15 m x 12 m SMU was considered.
5. The 1.0% Government royalty was not considered during the preparation of the constraining pit.

   Quantities and grades in a mineral resource estimate are rounded to an appropriate number of significant figures to reflect that they are approximations.

1.8 Mineral Reserve Estimate

Mineral reserves for the Zafranal Project have been estimated by NCL Ingeniería y Construcción S.A. (NCL), using industry best practices and conforming to the CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014). The mineral reserves estimate is summarized in Table 1-4.

The main factors that may affect the mineral reserve estimate are metallurgical recoveries and operating costs (fuel, energy and labour). The base price of copper, even though the most important factor for revenue calculation, does not affect the mineral reserves estimate, as a manual cut-off of 0.15 %Cu is being considered, which is higher than the grade cut-off obtained from the base case economic evaluation. The selected grade cut-off allows for a broad swing in metal prices before they have an effect on the mineral reserves estimate.

Other than the risks identified in this report, NCL is not aware of any other environmental, permitting, legal, title, taxation, socio-economic or political factors that could materially affect the Mineral Reserve estimate.

Table 1-4 – Mineral reserve estimate for the Zafranal Project

Reserve category	Ore		Ore grade		Contained metal	
	Type	Mt	% Cu	G/t Au	Mlbs Cu	Koz Au
Proven mineral reserves						
	Mixed	0.4	0.48	0.11	4	1
	Supergene	97	0.61	0.08	1,308	233
	Hypogene	105	0.28	0.07	660	249
Total proven mineral reserves		*202*	*0.44*	*0.07*	*1,972*	*483*
Probable Mineral Reserves						
	Mixed	2	0.40	0.11	16	6
	Supergene	78	0.50	0.06	861	156
	Hypogene	118	0.27	0.06	694	246
Total probable mineral reserves		*198*	*0.36*	*0.06*	*1,571*	*408*
Total mineral reserves (proven and probable)						
	Mixed	2	0.41	0.11	20	8
	Supergene	175	0.56	0.07	2,169	389
	Hypogene	224	0.27	0.07	1,354	495
Total mineral reserves (proven and probable)		*401*	*0.40*	*0.07*	*3,543*	*891*

Notes to accompany mineral reserves table:

1. The Qualified Person for the estimate is Carlos Guzman, CMC and FAusIMM, an NCL employee. Mineral reserves have an effective date of 31 March 2016.
2. Mineral reserves are reported as constrained within Measured and Indicated mineral resources pit designs, and supported by a mine plan featuring variable cut-off. The pit designs and mine plan were optimized using the following economic and technical parameters: metal prices of US$3.0/lb Cu and US$1,200/oz; recovery to concentrate assumptions according to geometallurgical domains for Cu and Au; copper concentrate treatment charges of US$90/dmt, U$0.09/lb of Cu refining charges and US$4.0/oz of Au refining charges; concentrate charges of US$12/wmt for marketing, US$37.55/wmt for transport, US$20/wmt for port and insurance, US$65/wmt for shipping and 0.3% for transport losses; average payability of 96.9% for Cu and 90% for Au; average mining cost of US$1.84/t, process costs of US$4.47/t for mixed and supergene materials and US$4.75/f for hypogene, and G&A US$1.25/t processed; average pit slope angles that range from 36° to 41°; a 1% Government royalty rate assumption, and an assumption of 100% mining recovery.
3. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.
4. Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces.

1.9 Mining methods

The final pit design was based on the economic shell using the 2015 mineral resource estimate with a 1.0 revenue factor, and variable overall slope angles from 36° to 41°, according to geotechnical domains.

The block model is considered to be a fully diluted resource model; hence, NCL performed pit optimization and mine planning activities without introducing any further mine dilution. NCL also considered a 100% ore mining recovery due to the disseminated characteristics of the ore and the proximity of the economic cut-off to the background copper content of the rock. This assumption requires the mine to adopt strict grade control practices to minimize any misclassification of ore grade material. Mineralization type also has significant impacts on metallurgical recoveries in the flotation plant, further increasing the need for detailed mine planning oversight.

A set of nine mining phases was developed based on the sequence of nested pits obtained from the pit optimization. As a result of this approach, the mine plan is able in most cases to schedule the phase development based on its net smelter return, from highest to lowest over the life of mine.

Pre-stripping and pioneering work is to be performed by a qualified mining contractor. On the basis of normalized bids including Owners' costs, an economic assessment was carried out to determine the optimum duration of the stripping contract and the transition to an Owner-operated mine. The study concluded that a five-year period terminating in production Year 3 was the preferred outcome as it provided an experienced workforce for challenging earthworks, lower initial capital cost and it resulted in an Owner-operated time frame that minimizes replacement of equipment in the latter stages of mine development.

The mine is scheduled to work seven days per week or 365 days per year. Each day will consist of two 12-hour shifts. Four mining crews will cover the operation (two working and two on days off).

The study is based on operating the Zafranal mine with 34 m^3 hydraulic excavators and 220 tonne haul trucks. The selected equipment has the capacity and productivity to achieve a peak annual total material movement of 75 million tonnes (during contractor operation), without compromising mining selectivity for grade and dilution control.

1.10 Recovery methods

1.10.1.1 Process flow diagram

The overall process flow diagram is as described in Section 1.6 and is also presented in Figure 1-3.

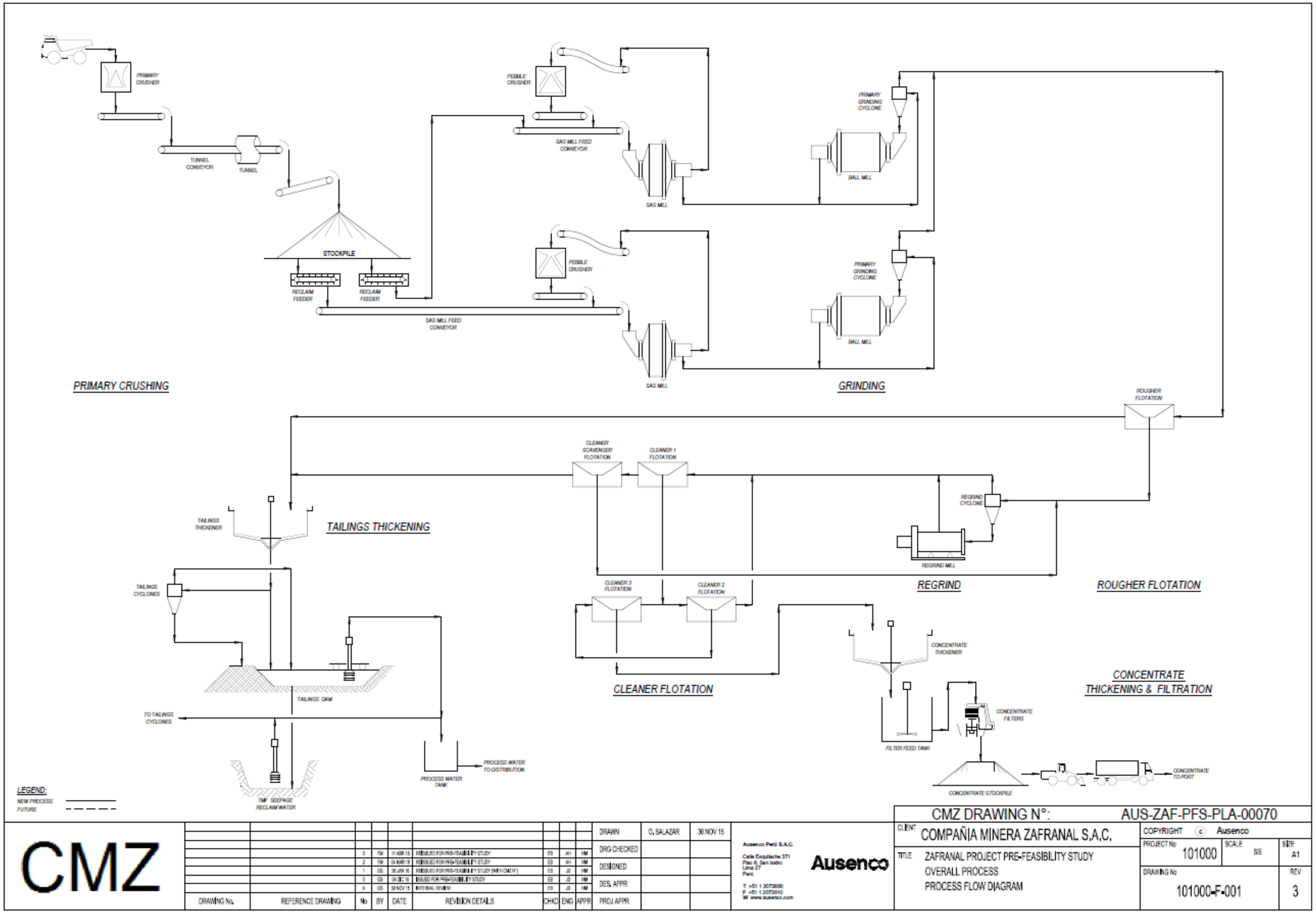

Figure 1-3 – Overall process flow diagram

1.10.1.2 Concentrator design

Concentrator design throughput is up to 64 kt/d depending on the grinding characteristics of the feed. All processing from grinding through to tailings and concentrate thickening will operate on a continuous basis of 24 hours per day, 7 days per week. Allowing for downtime for maintenance and unscheduled stoppages in the operation, the long-term availability is expected to be at least 91.3%. Therefore, the operating time is 8,000 hours.

Run-of-mine or stockpile rehandled ore is received into the dump pocket of a primary gyratory crusher. Primary crusher product is transferred to the crushed ore stockpile by belt conveyors.

Due to the steep topography between the primary crusher and the rest of the concentrator, the transfer conveyor passes through a 3.6 km long tunnel. Most of the tunnel is through rock in which the roof can be supported by bolting but approximately 20% is through mylonite and/or gouge fill requiring steel sets and shotcrete fill for roof support.

An open stockpile with two apron feeders for each grinding line has been selected for crushed ore storage and reclaim. A dual-line grinding circuit was selected to treat design plant throughput.

A single rougher and scavenger flotation circuit is followed by three stages of cleaning and a cleaner scavenger circuit. Final flotation concentrate will be initially dewatered in a high rate thickener prior to filtration. Horizontal plate pressure filters have been selected for the duty. Filtered concentrate will be trucked directly to the Port of Matarani in bulk.

A high rate thickener is suitable for tailings dewatering. Thickened tailings slurry will be gravity fed to the tailings management facility (TMF).

1.10.1.3 Site water balance

The site-wide water balance results indicate that of the total average water requirement of 362 L/s to the site, 84% is expected be supplied from off-site. For design purposes, and due to the uncertainty in this stage of study, it was recommended to use the 75th percentile for the nominal supply of raw water, and the 90th percentile as the maximum. At the 75th percentile, the monthly peak demand of raw water for the mine is 370 L/s, and at the 99th percentile it is 397 L/s (in Years 3 to 6). The water supply system preliminary design is based on these calculations, with an appropriate design allowance.

1.11 Project Infrastructure

1.11.1 Waste dumps and stockpiles

Three waste rock storage areas, located to the north and south of the pits, were designed for the Project according to Piteau geotechnical recommendations.

During the pre-production period, the ROM stockpile will be constructed to the west of the initial pit for later re-handling to the primary crusher. The total ore to be stockpiled during this period amounts to 2.5 Mt.

The low grade ore stockpile will be located to the west of the pit, in the same location of the initial ROM stockpile. The oxide stockpile will be placed to the southwest of the Zafranal Main pit, over a platform built on the central waste dump. The stockpiles are designed to facilitate later re-handling.

A geochemical characterization program is on-going to assess the potential for acid rock drainage and metal leaching (ARD/ML) associated with waste from the open pit. However, median sulfur values are typically between 1% and 3% indicating that open pit waste will be potentially acid generating (PAG) with limited carbonate, which suggests substantial buffering capacity or carbonate mineral dissolution should not be expected.

1.11.2 Tailings storage

The final location of the TMF is 1 km south of the concentrator in a large open valley basin (refer to Figure 1-2 above). It has been sized to contain 396 Mt of tailings, operational water, and the probable maximum flood (PMF), with a capacity that could be expanded in the future. The impoundment basin has a small outlet, Quebrada Huacan, through a range of hills along the southwest side where an initial embankment will be constructed to retain the tailings within the basin.

A 96 m high southwest starter embankment with a 2-year life will be constructed from alluvium from within the impoundment with an upstream liner tied into the bedrock to reduce seepage.

Subsequent raises to contain the tailings through the life-of-mine will utilize underflow sand from a tailings cyclone plant. The embankment height will be 204 metres at the end of the current life of mine (LOM).

A low spot (saddle) in the hills along the northwest side requires a small embankment to contain the proposed tailings production from Year 10. The embankment height will be 40 metres at the end of the current LOM.

Prior to closure, the tailings will be deposited to shape the tailings surface to drain to the northwest embankment, in addition consideration is being given to using a binder to create a surface that is trafficable for construction of the closure cap and surface runoff diversion system. The TMF 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 pass safely through the PMP storm runoff from the facility and watersheds above the facility. In preparation for closure, there will be a spillway developed west of the northwest embankment to discharge the surface flow safely out of the impoundment.

Seepage minimization from the TMF is extensive, including French drains to a collection gallery at the base of the upstream face of the southwest embankment and a network of near-horizontal drill holes under the right abutment of the southwest embankment draining to a drilling chamber and pump system. For the northwest embankment the French drains are tied into a drain installed through its base to a collection pond. Additional seepage collection wells will be located in fault zones down-gradient from each embankment. Water recovered from these systems will be recycled, with reclaim water from the TMF supernatant pond, for cyclone plant feed dilution and return to the concentrator process water system.

1.11.3 Surface water management

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

1.11.4 Product transport

The corridor for concentrate transportation will follow a western route from the site to Camaná and then along the coast on the new Costanera Highway to the Port of Matarani.

1.11.5 Access roads

Site access road designs were developed from both the local towns of Anexo El Pedregal (Corire road) and El Pedregal de Majes (Majes road).

1.11.6 Power supply

The power supply for the Zafranal Project is based on a total installed power requirement of 99 MW and peak demand of 91 MW. The power supply system from the Peruvian National Grid (SEIN) will be via a 95.5 km overhead 220 kV transmission line from the New Socabaya substation.

1.11.7 Water supply

The potential water supply for the Zafranal operation could be from a wellfield of approximately nine wells extracting brackish water from the groundwater resource below the Majes I irrigation area. The wells would be equipped with a submersible pump and supply a transfer tank and pump station. The two transfer pumps would transport water from the transfer tank, via a 32 km pipeline, to a raw water pond adjacent to the concentrator plant at the Project site. Social impacts and community relations risks associated with the water supply system are discussed in Section 1.13.

1.11.8 Site accommodation

The strategy for site accommodation has two camps, one that will be provided by the contract mining company and the second to accommodate the short-term construction phase and long-term operations phase requirements of the Project.

1.11.9 General and administration facilities

The strategy for the Project has been to use modular portable buildings for offices and tent-style (tensioned fabric) structures for warehousing and workshops.

1.12 Product Marketing

1.12.1 Trends for copper prices and forecasts

For industries such as copper that are in structural deficit, that is, with long-term demand significantly in excess of base case production intentions, incentive pricing provides the most appropriate estimate of long-term cycle average prices. The incentive price analysis examines the price required to provide an investor a given rate of return for each project, and calculates the price required in theory to warrant investment in sufficient cumulative capacity to meet potential demand. In this context long-term can be considered the cycle average price over the next cycle, i.e. the decade over which a project would expect to see payback.

Assuming that for the foreseeable future copper demand will be sustained at a growth rate of 2.6% per year roughly 700 kt of new mine copper production would be required each year. Given the environment of high capital costs to build new projects, a higher copper price will be required by mining companies to move ahead with green-field projects. It is estimated that a long-term copper price of US$3.00/lb (US$6,614/tonne) in real US$ terms of 2016 is needed to provide sufficient incentive to ensure that copper does not slip back into a structural deficit and to encourage the development of the next wave of projects required to supply expected global copper demand in the long term.

1.12.2 Copper concentrate treatment and refining charges forecast

The forecast for treatment and refining charges (TCRCs) applied to the sale of copper concentrates point to an average of US$90.0/dmt and 9.0c/lb, CIF main Japanese ports, during the next five years.

For the longer term TCRCs averaging US$90.0/dmt and 9.0c/lb are also estimated, which is equivalent to a charge of 24.1c/lb for a 28% copper grade concentrate, similar to that to be produced by Zafranal.

1.13 Environmental Studies, Permitting and Social or Community Impact

1.13.1 Design considerations

From the evaluation of baseline conditions and early identification of potential risks the following design considerations should be included in further development of the Project.

1.13.1.1 Water controls

One of the key risks is perceptions of the local community around the use of and potential effect on water that could be used for agricultural purposes.

Developing the Majes I brackish groundwater resource as the potential operational water source for the Project minimizes this risk in regard to water use. Studies to date indicate that the brackish groundwater under Majes 1 is not suitable for agriculture or drinking and so would not be a source of competition between the Project and community requirements.

Contact water and seepage from the Project must be contained during operations and adequately managed for closure to avoid the risk of affecting surface water quality in the Majes River or the perception that the surface and/or groundwater that drains to the Majes River is affected by Project mining activities.

Preliminary modeling of the open pit indicates that there is a potential for a pit lake to be formed, reaching steady state conditions at 100 m depth around 100 years post closure. Closure planning must take into account the potential for pit lake formation and seepage collection for perceptions of the Majes River communities regarding potential effects on the river water quality to be addressed.

1.13.1.2 Dust controls

Although the populations surrounding the Project are at a sufficient distance from the Project facilities to not be affected by dust dispersion and the surrounding topography would precipitate fallout of any dust content in the wind before reaching the local communities, this is still considered a concern by the local population; for the potential to affect human health but also potential effects to pasturelands and agricultural crops. The design of the Project must include dust control measures. These will also mitigate potential health risks from dust inhalation for the workforce. Maintaining an adequate level of moisture on the tailings surface through managed deposition will mitigate wind erosion during operations. The provisions for closure are described in Section 1.11.2.

1.13.1.3 Management of sensitive vegetation

There are species of plants of conservation concern that will need to be evaluated further as part of the EIA process. Management of these species will involve collection of seeds, asexual propagation of plants and/or transplanting of individuals. The specific measures will be defined in the EIA.

1.13.1.4 Management of local fauna

There is a local population of guanacos that is of interest to the local residents, as well as being a protected species at the national and international level. During the development of the EIA and the Feasibility Study design, measures to avoid Project impacts to this species and the other species of fauna of ecological interest will be investigated.

1.13.1.5 Archaeology

There are archaeological remains of local interest in the vicinity of the Project. While the location of these remains has been included in the design of the Project at this preliminary feasibility level, with the objective of avoiding them, this criterion must be continued through the remaining design stages of the Zafranal Project.

1.13.2 Stakeholder engagement

All stakeholders in the direct area of influence have been mapped and stakeholder mapping is updated regularly. Different mechanisms and initiatives have been implemented to develop an appropriate level of social engagement with the communities in the direct area of influence, including the establishment of a Working Table. Stakeholder mapping in the indirect area of influence has also been completed.

It is important to maintain and increase the current level of engagement with all stakeholders to avoid disruption of the Project schedule. Keeping the local stakeholders well-informed and meaningfully engaged will reduce the likelihood of opposition groups gaining support in the local environment.

As part of the stakeholder engagement mechanisms, the Project will implement a series of public workshops in support of the EIA. These workshops will include the official public workshops coordinated with the MINEM and SENACE but will also include a series of voluntary information spaces.

1.13.3 Social responsibility

Social responsibility is being managed through a series of programs and plans:

1. Local employment program
2. Local supply and service program
3. Social investment program
4. Communications plan and programs
5. Social commitment monitoring program
6. Program for attention to grievances
7. Program for attention to requests for donations
8. Social climate monitoring and risk prevention program
9. Social crisis management program.

1.14 Capital and Operating Costs

1.14.1 Capital cost

The capital cost of the project over a three-year construction and pre-mining period, and sustaining expenditures over the 19-year mine life, have been estimated. The following basic data pertains to the estimate:

- The base date is 1st quarter 2016
- All costs are expressed in United States dollars (USD or US$)
- The estimate is at pre-feasibility precision ±20-25%
- No allowance has been made in the estimate for escalation from the base date or changes in currency exchange rates.
- All import duties and taxes are excluded from the estimate (not expected to apply).
- Each element of the estimate is developed initially as a base cost only. A growth allowance has then been allocated to each element of the costs to reflect the level of definition in pricing and engineering maturity relating to that element.
- Estimate contingency (provision) is included to address anticipated variances between the specific items contained in the estimate and the final actual project cost assessed as a percentage of direct and indirect costs to arrive at a total project estimate with the required confidence level.
- The following items were not considered in the capital cost estimate:
    - changes in market conditions
    - working capital
    - replacement capital other than in sustaining capital
    - finance charges
    - residual value of temporary equipment and facilities
    - residual value of any redundant equipment
    - cost of any downtime
    - environmental approvals
    - closure costs (included in the economic evaluation)
    - legal costs
    - any further studies
    - force majeure issues
    - future scope changes
    - special incentives (schedule, safety or others)
    - allowance for loss of productivity and/or disruption due to religious, social and/or cultural activities
    - risk management reserve (Owner's contingency), which is covered by positive variance in capital cost in the sensitivity analyses in Section 1.15.

The capital cost estimate summarized in Table 1-5.

1.14.1.1 Definition of costs

The direct costs are those costs that pertain to the permanent equipment, materials and labour associated with the physical construction of the facilities and utilities.

The indirect costs include all costs associated with implementation of the Project and incurred by the owner, engineer or consultants in the engineering design, procurement, construction, project management and commissioning of the project.

Construction contractor's indirect costs are either included explicitly in the direct costs or are in the rates.

1.14.1.2 Estimate Methodology

The estimate was derived based on the methodologies set out in Ausenco Class 4 (equivalent to AACEI Class 4) Pre-Feasibility Study Standards.

Table 1-5 – Overall capital costs

WBS code Level 1	WBS Description Level 1	Total US$M
Initial capital		**1,157**
	Initial direct cost	**703**
1000	Mine	142
2000	Concentrator	430
4000	On-site infrastructure utilities and facilities	33
5000	Off-site infrastructure utilities and facilities	98
	Initial indirect cost	**454**
6000	Project preliminaries	103
7000	Indirect costs (EPCM, vendors, commissioning, spares, etc.)	132
8000	Provisions	151
9000	Owner's costs	68
Sustaining capital		**263**
	Sustaining direct cost	**228**
1000	Mine	213
2000	Concentrator	15
4000	On-site infrastructure utilities and facilities	1
	Sustaining indirect cost	**34**
8000	Provisions	34
Grand Total		1,420

1.14.2 Operating cost

The operating cost of the project has been estimated on the same basis as the capital cost estimate except that there is no growth allowance or estimate contingency included. A summary of the average production period operating costs is shown in Table 1-6.

Table 1-6 – Summary of average production period operating costs

Description	Production period average	
	$/t ore	($/lb Cu produced)
Mining	4.25	0.54
Processing	4.58	0.59
General and administration	1.22	0.16
Concentrate road transport	0.45	0.06
Total Cost	**10.50**	**1.35**

Operating costs vary with time according to total material mined and concentrator throughput. There is also some minor variation related to concentrate output. These are reflected in the economic analysis.

1.15 Economic Analysis

The economic valuation for the Zafranal project has been determined using a discounted cash flow (DCF) analysis, conducted in real term US.

The valuation model fully incorporates the Peruvian tax regime which includes royalties, special mining tax, value added tax (i.e. Impuesto General Ventas or IGV), workers' profit sharing and income tax.

Dividend withholding tax is not considered in the economic evaluation.

The key assumptions employed in the valuation (discount rates, metal prices and foreign exchange) are based on internally developed forecasts with metal prices and treatment and refining charges benchmarked externally (Section 1.12). These forecasts incorporate applicable general economic assumptions as derived from numerous sources including forecasts by Teck Resources Limited and Mitsubishi Materials Corporation, industry studies and financial institutions (i.e. CIBC, Macquarie Bank and the IMF).

The NPV is based on the valuation of CMZ as a Peruvian unleveraged stand-alone asset, on an after-tax basis as of the currently expected decision point of 01 January 2019. A real US$ discount rate of 8.00% has been applied to calculate the NPV. The life-of-mine (LOM) price forecast for copper is US$3.00 per pound and price forecast for gold is US$1,200 per troy ounce. Other forecast variables include long-term oil price of US$85 per barrel and long-term PEN:USD exchange rate of 2.90.

The valuation was undertaken both deterministically and probabilistically (i.e. Monte Carlo simulations). The key variables and assumptions used in the calculation of the cash flows are based on CMZ's mid-range (i.e. expected) estimates. Range analyses were carried out to determine the possible range of values for the key variables.

A summary of the valuation results are contained in Table 1-7.

Table 1-7 – Summary of valuation results[b]

Valuation Results: Mid-range Case Prices	Deterministic	Probabilistic
Total Undiscounted Revenues (US$M real)	9,627	9,569
Total Undiscounted Free Cash Flows (US$M real)	1,649	1,591
Initial Capex with Contingency (US$M real)	1,157	1,161
Mid-range / P50 NPV@ 8% (US$M)	496	460
Mid-range / P50 IRR (%)	15.9%	15.3%
Payback Period - Start of Mill Operations (years)	5.1	5.2
Mid-range / P50 C1 Costs (US$/Cu lb real)	1.59	1.59
Mid-range / P50 C2 Costs (US$/Cu lb real)	2.07	2.09
Mid-range / P50 C3 Costs (US$/Cu lb real)	2.14	2.15
NPV@ 8% (US$M) - Range Min-Max/P10-P90	(706) to 1,425	16 to 932
Probability NPV > $0		91%
Decision/Valuation Date	1/Jan/19	

Table 1-8 shows the copper prices sensitivities on undiscounted cash flows, after tax net present values and internal rate of return. A more detailed sensitivity analysis in presented in Section 22.3.

Table 1-8 – Copper Price Sensitivity

Copper price sensitivity		Copper prices (US$/lb)				
		2.00	2.50	3.00	3.50	4.00
Undiscounted cash flows US$M	Pre-Tax	-311	1,204	**2,711**	4,236	5,753
	After-Tax	-478	648	**1,649**	2,632	3,574
After-tax NPV US$M at discount rates	5%	-515	180	813	1,426	2,029
	8%	-547	-5	**496**	978	1,454
	10%	-567	-101	333	749	1,162
Internal rate of return	IRR %	0.0%	7.9%	**15.9%**	22.1%	27.6%

1.16 Other Relevant Data and Information

1.16.1 Project implementation plan

The implementation strategy is based on an organization with an Owner's project execution team and an EPCM contractor both reporting to a Project Director. The team also includes a Vice President Operations, whose role is to manage pre-production mining activities and prepare for operations, and administrative VPs, whose primary roles are to manage the off-site service and support functions that will support the project execution and facilitate the transition

[b] C1, C2 and C3 are as defined by Wood Mackenzie. Refer to section 22.1.2 for explanation of definitions.

from construction to operations. The Project Director, VP Operations and the administrative VPs all report to the CMZ President, who has ultimate accountability for delivery of the Project.

The key project milestones are summarized in Table 1-9:

Table 1-9 – Key project milestones

Deliverable	Revision 0 End Date
PFS complete	Q2 2016
Gate 2 approval – start FS	Q4 2016
FS complete	Q4 2017
EIA detailed	Q3 2018
Gate 3: approval - FS and EIA	Q4 2018
Board approval project sanction: early funding	Q4 2018
Financing Phases 2 and 3; board approval project sanction: construction	Q3 2019
Beneficiation construction authorization	Q1 2019
Detailed engineering complete	Q1 2019
Construction concentrator complete	Q2 2021
Pre-stripping complete	Q2 2021
First ore	Q4 2021

1.16.2 Operations management plan

CMZ will have three administration centres in Peru, one at the Project site, one in Arequipa and a small office in Lima. The Vice President of Operations will be located at the Project site, reporting to the President, and be responsible for the health, safety and performance of the personnel on site, including mining, concentrator, technical services; health, safety and environment; as well as on-site representatives of human resources, purchasing and warehouse, camp administration, labour relations, security, and cost accounting. The President will be located in Arequipa and be the legal representative for the Company with responsibilities including overall Company performance, external affairs (including community relations), human resources, finance and accounting. The Vice President Finance will be located in Lima, reporting to the President, and will be responsible for information technology, marketing, procurement, treasury and finance activities. Office space will also be provided for personnel traveling to Lima on Company business.

CMZ has an established general and administrative (G&A) and field workforce with several years of service and corporate history. While not all current employees will be retained, the basic structure will remain in place and serve as a solid starting point for building the permanent workforce. Additional CMZ labour will be recruited primarily from Caylloma Province, Castilla Province, Arequipa, and Lima.

1.16.3 Risk management

The Zafranal Project has been subject to proactive risk management from July 2014. The key risks (rated as “high”) to the Project at the end of the PFS phase fall into three categories:

- Community, access and approvals refer to the social and community relations component of the Project, which includes most of the current high risks to the Project. This needs to be actively managed by the Owner with guidance from suitably qualified consultants to manage the risks that have been identified and those that might manifest through the life of the Project.

- Technical risk associated with site conditions refers to the inability to gain timely access to some field sites in the area of the TMF during the PFS phase which has led to the possibility that geotechnical and hydrogeological modeling may change as more information becomes available. These risks are also associated with the community, access and approvals risks above. An archaeological discovery during the land purchase process or during construction could also affect the Project schedule.

- Commercial and budget risk refers to the assumption that the contractor will complete pre-stripping and the initial three years of mining without passing on a new fleet purchase amortization, which is based on the assumption that the contractor will not charge for a new fleet over the six-year life of contract, due to the contractors having large current fleets paid off on other projects.

1.17 Overall Conclusion and Recommendation

The overall conclusion from the PFS is that the Project is suitable to progress to the next phase of investigation and evaluation.

The next phase of project development includes the feasibility study integrated with the elaboration of the Environmental Impact Assessment. Typical expenditure for the FS phase, including EIA, is 2-5% of project capital cost, in this case US$30-70M.

The subsequent phase of project development is the implementation or execution phase, the cost of which is the initial capital cost estimate presented in Section 1.14.1.

2 Introduction

2.1 Issuer and Terms of Reference

AQM Copper Inc. (AQM) is a Canadian and Peruvian-listed mineral exploration and development company whose principal purpose is the acquisition and development of base metal deposits in South America. The Company was incorporated in British Columbia on 24 March 2005 and commenced trading on the Toronto Venture Stock (TSX-V) Exchange on 29 November 2006 and the Bolsa de Valores de Lima (BVL) on 7 August 2008.

Through its Peruvian subsidiary, Minera AQM Copper Peru S.A.C. (MAQM), in which Mitsubishi Materials Corporation (MMC) has acquired a 40% interest, AQM is the operator of the Zafranal Project in southern Peru in a 50:50 joint venture with Teck Resources Limited (Teck), Compañía Minera Zafranal S.A.C. (CMZ). Hence, AQM has an indirect interest of 30% in the Zafranal Project. Under the terms of the Joint Venture Agreement, Teck has the option of becoming the operator of Zafranal at the time a construction decision is made by the joint venture stakeholders or when there has been a change of control in AQM.

2.2 Terms of Reference

The Zafranal Project is a greenfield porphyry copper project with this report as the culmination of the Preliminary Feasibility Study (PFS). The proposed Project is to develop a mining and processing operation and all associated infrastructure based on the Zafranal main zone and Victoria deposits including:

- Open pit mining of up to 71 million tonnes per year of total material (sulfide ore, oxide and mixed mineral, and waste).
- Processing of sulfide ore through a flotation concentrator at rates from 55 to 64 thousand tonnes per day (after initial ramp-up) depending on the grinding characteristics of the feed.
- Concentrate transport of 379 to 126 thousand tonnes per year (dry basis).
- Associated on-site and off-site infrastructure.

Ausenco Peru S.A.C. (Ausenco) was engaged by CMZ to co-ordinate the preparation of a preliminary feasibility study and associated Technical Report for the Zafranal Project (the Project), and to prepare substantial elements of both.

Table 2-1 presents a summary of the assignment of scope to the Qualified Persons (QPs) who have reviewed and take responsibility for the respective items of scope of the Technical Report.

Table 2-1 - Summary of scope assignment and Qualified Persons' responsibilities

Qualified Person	Responsible for	
Alvaro Fernandez-Baca	6.2	Exploration and Development by Previous Owners
	6.3	Previous Mineral Resource Estimates
	7	Geological Setting and Mineralization
	8	Deposit Types
	9	Exploration (excluding drilling)
	10.6	Moly Sur Zone Drilling Information
Andy Schissler	17.1.2	Tunnel geomechanics
Anthony Sanford	1	Summary
	2	Introduction
	3	Reliance on Other Experts
	4	Property Description and Location
	5	Accessibility, Climate, Local Resources, Infrastructure and Physiography (except Site Geotechnical, Hydrogeology, Surface Hydrology)
	6.1	Ownership History
	20	Environmental (except Hydrogeology, Surface Hydrology)
	25	Interpretation and Conclusions
	26	Recommendations
	27	References
Carlos Guzman	15	Mineral Reserve Estimates
	16	Mining Methods (except Hydrogeology, Surface Hydrology, and Mining Support Facilities and Utilities)
	18.1	Waste Dumps and Mineral Stockpiles (except surface water management, and foundation geotechnical)
	21.4.1	[Capital Cost] Mining and mining fleet
	21.11	Mining Operating Cost Estimate
Greg Lane	13	Mineral Processing and Metallurgical Testing
	17	Recovery Methods (except Tunnel Geomechanics)
	19	Market Studies and Contracts
	21	Capital and Operating Costs (except Mining)
	22	Economic Analysis
	24	Other Relevant Data and Information
Julio Bruna Novillo	10	Drilling (except 10.6 Moly Sur zone)
	11	Sample Preparation, Analyses and Security
	12	Data Verification

Qualified Person	Responsible for	
Paul Staples	16.9	Mining Support Facilities and Utilities
	18.4	[Project Infrastructure] Product Transport
	18.5	[Project Infrastructure] Access Roads
	18.6	[Project Infrastructure] Power Supply
	18.7	[Project Infrastructure] Water Supply (excluding hydrogeology)
	18.8	[Project Infrastructure] Site Accommodation
	18.9	[Project Infrastructure] General and Administration Facilities
Peter Oshust	14	Mineral Resource Estimate
Ryan Jakubowski	5.9.2	Site hydrogeology
	16.2.1	Hydrogeology considerations
	18.2.13	TMF hydrogeology and seepage management
	18.7.1	Construction (start-up) water supply
	18.7.2	Operational water supply (hydrogeology)
	20.1.2	Hydrogeology
Scott Elfen	5.9.1	Site geotechnical
	5.9.3	Site hydrology
	16.2.2	[Mine] Hydrology considerations
	18.1.2	[Waste Dumps] surface water management (only)
	18.1.3	[Waste Dumps] Geotechnical considerations (foundation conditions)
	18.2	Tailings Storage (except hydrogeology)
	18.3	Surface Water Management
	20.1.1	Hydrology and water balance
Not applicable	23	Adjacent Properties - no relevant adjacent properties
While Anthony Sanford has taken responsibility for ensuring that the sections requiring contributions from all QPs accurately reflect those contributions, the individual QPs remain responsible for their own contributions	1	Summary
	2	Introduction
	3	Reliance on Other Experts
	25	Interpretation and Conclusions
	26	Recommendations
	27	References

2.3 Sources of Information

The main sources of information for the PFS have been previous reports of work by CMZ and its consultants; including those prepared for the technical reports of the updated Preliminary Economic Assessment dated 4 October 2013, the Preliminary Economic Assessment dated 16 January 2013 and the Mineral Resource Estimate dated 7 May 2012; and those prepared for the PFS. A detailed reference list is provided in Section 27.

2.4 Site Visits

- Alvaro Fernandez-Baca from Barracuda Exploraciones, QP for deposit type, exploration history, geology, mineralization, structure and alteration and exploration outside of the main bodies, as shown in Table 2-1, visited the Zafranal Project site on 25-26 September 2015. The field visit included a review of drill core from exploration drilling of the Moly Sur target as well as maps, sections and geochemical results from current and past exploration programs.
- Andrew P. Schissler from Ausenco, QP for geotechnical design of the crushed ore transfer conveyor tunnel visited the Zafranal Project site from 19-22 August 2015. The planned tunnel portals and alignment was traversed on surface. Drill core was inspected at the geotechnical core-logging facility at the site camp. The facility is well-managed with documented procedures and the chain-of-custody is industry standard. Geotechnical drill core from two holes was reviewed. The CMZ and Ausenco geologists demonstrated a high level of knowledge of the relevant geology. The geotechnical drill hole survey is to industry best-practice standards.
- Anthony Sanford from Ausenco, QP for environmental, permitting and social or community impact and other sub-sections, as shown in Table 2-1, visited the Zafranal Project site on 24-25 June 2015. The field visit included an examination of the general area where the proposed mine infrastructure is to be installed, including the area of the proposed tailings management facility and its other siting options.
- Carlos Guzman from NCL, QP for the mineral reserve estimate, and other sub-sections as shown in Table 2-1, visited the Zafranal core-logging facility in Arequipa and the Project site on 28-29 April 2015. Drill core from four holes was reviewed for understanding of the geological model and rock mass quality. At the project site, the location of all future infrastructures was visited.
- Greg Lane from Ausenco, QP for mineral processing, concentrator design, infrastructure, product marketing, capital and operating costs (other than mining), economic analysis, project implementation plan (other than mining), and operations management plan (other than mining), as shown in Table 2-1, visited the Zafranal Project site on 26-27 July 2014. The site visit included the Matarani Port, Camaná Estuary area, potential tailings storage area and concentrator location, the proposed open pit area and the exploration camp. The site visit was conducted with representatives of MMC, Teck, CMZ and AQM.
- Julio Bruna Novillo from Patagonia Geosciences, QP for drilling (except Moly Sur zone); sample preparation, analyses and security; and data verification, as shown in Table 2-1, visited the Zafranal Project site in November 2013, January 2014 and July 2015. He visited the overall site and area of exploration activities. Inspected drill core and drilling sites, reviewed geological, data collection and sample preparation procedures, and carried out independent data verification. In addition, he worked at and inspected the drill core storage area just outside the city of Arequipa, where core was reviewed with the project geologist.

- Paul Staples from Ausenco, QP for mining support facilities and utilities (design) and project infrastructure (product transport, access roads, power supply, water supply [excluding hydrogeology], site accommodation, and general and administration facilities), as shown in Table 2-1, has not visited the Zafranal Project site, due to other commitments precluding such a visit at the appropriate times. As the only one of ten QPs not to visit the site, this is not considered to be of detriment to the PFS.
- Peter Oshust from Amec Foster Wheeler, QP for the mineral resource estimate, and other sub-sections as shown in Table 2-1, visited the Zafranal core-logging facility in Arequipa and the Project site on 17-18 November 2015. The core-logging facility is well-managed with documented procedures and the chain-of-custody is industry standard. Drill core from four holes was reviewed. The CMZ geologists demonstrated a high level of knowledge of the deposit geology. The Zafranal project operation is also well-managed and the entire team showed a strong commitment to safety. The site geologists again demonstrated a firm understanding of the deposit. Drill hole monumenting is to industry best-practice standards and 12 drill hole locations at 8 drill pads were verified correct by hand-held GPS.
- Ryan Jakubowski from FloSolutions, QP for hydrogeology as shown in Table 2-1, visited the Zafranal Project site on 14-15 March 2016. The site visit included the following areas proposed as part of this PFS: tailings management facility; waste dumps; open pit; Pampa Ganchos; and the Majes I irrigation area. Additionally, drill core and cuttings from five holes adjacent to or within the facility footprints were reviewed.
- Scott Elfen from Ausenco, QP for site geotechnical, site hydrology, tailings management facility and other sub-sections as shown in Table 2-1Table 2-1, visited the Zafranal Project site on 24-25 June 2015. The field visit included an examination of the general area where the proposed mine infrastructure is to be installed, including the area of the proposed tailings management facility and its other siting options.

3 Reliance on Other Experts

The Qualified Persons who prepared this report have not reviewed any of the information provided by AQM or CMZ or both, concerning legal, political or tax matters relevant to sections of the PFS Report, including but not limited to:

- Mineral tenure and status
- Owner's title and interest
- Others interests, royalties and encumbrances
- Permitting requirements and status
- Right of access, title and permitting risks
- Surface rights
- Ownership history
- Taxes and royalties.

The Qualified Persons who prepared this report have relied upon information provided by the following experts who are not QPs:

- Lorenzo de la Puente, General Manager of MAQM has been relied on for advice on legal matters relating to the Project.
- Erick Underwood, Chief Financial Officer of AQM has been relied on for advice on tax matters relating to the Project.

The results and opinions expressed in this report are conditional upon the aforementioned information being current, accurate, and complete as of the date of this report, and the understanding that no information has been withheld that would affect the conclusions made herein. Ausenco reserves the right, but will not be obliged, to revise this report and conclusions if additional information becomes known to Ausenco subsequent to the date of this report. Ausenco does not assume responsibility for AQM's actions in distributing this report.

4 Property Description and Location

4.1 Property Area and Location

The Property is 47,252 ha in area and is located in southern Peru about 166 km by road (90 km straight-line distance) northwest of the city of Arequipa, within the Provinces of Castilla and Caylloma. The property is at elevations of 1,400 to 2,900 masl. The approximate center of it is located at 16° 02' 47" south latitude and 72° 15' 13" west longitude using WGS 84 shown in (Figure 4-1).

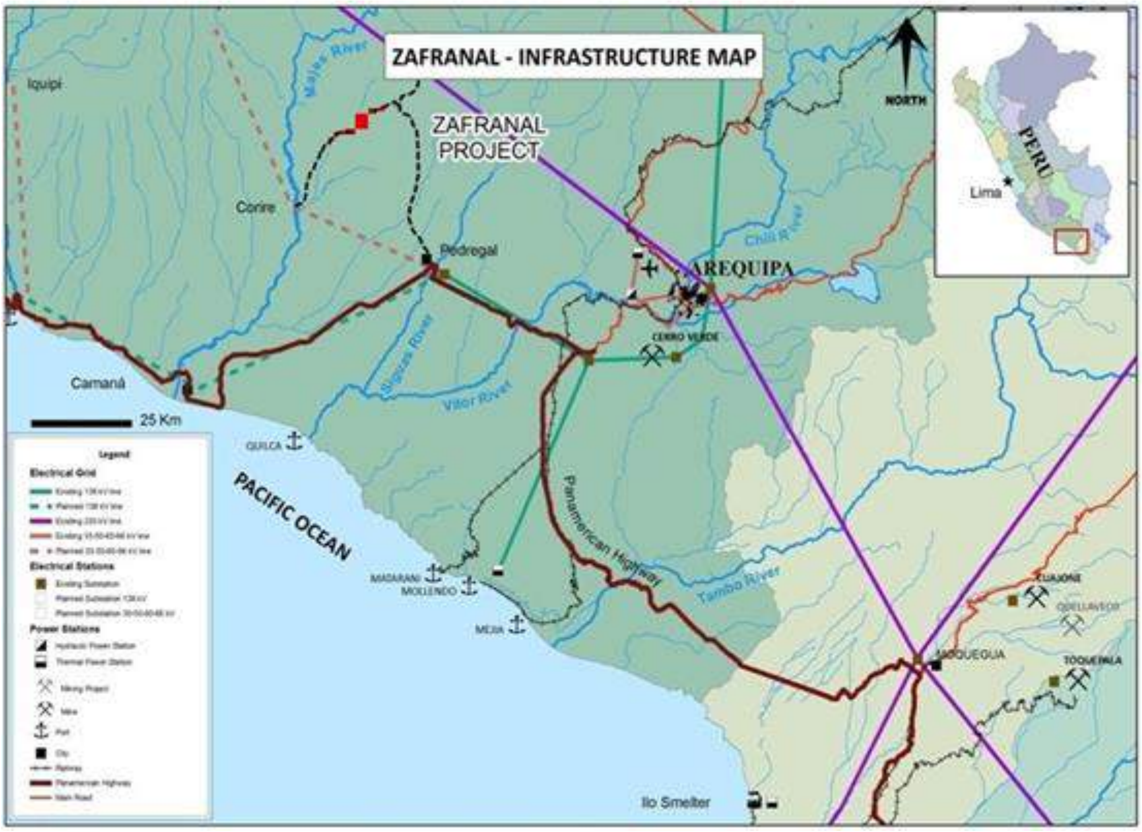

Figure 4-1 – Property location and regional infrastructure

4.2 Mineral Tenure and Status

4.2.1 Peruvian law

Aside from reconnaissance and prospecting activities, all other mining activities in Peru must comply with the Mining Concessions System, governed by the General Mining Law (Single Revised Text approved by Supreme Decree N° 014-92-EM). A mineral concession grants its holder the right to explore and exploit all mineral resources found in the subsoil of the concession area. Such concessions are irrevocable if the concessionaire fulfills the obligations to pay an annual 'Mining Good Standing Fee' and achieve an annual minimum production level, as per the General Mining Act provisions (see Section 4.3).

4.2.2 Zafranal mineral rights

The Zafranal Property comprises 63 concessions and 9 mineral claims, the details of which are presented in Table 4-1. The proposed project development will involve 14 of these concessions totaling 9,552 ha, which have been identified in Table 4-1 and shown in Figure 4-2.

Of the 63 concessions comprising the project, 31 have been included in four administrative economic units ('Unidad Económica Administrativa' – UEA): UEA CMZ 1, UEA CMZ 2, UEA CMZ 3 and UEA CMZ 4. A UEA is a group of mining concessions that are put together for administrative and minimum investment calculation purposes. A UEA allows the titleholder to work only in one or some of such concessions and apply the minimum production level to all the mining concessions comprising the UEA. The details of each UEA are presented in Table 4-2.

CMZ has been engaged in a process to sign an exploitation agreement with a company formed by approximately 100 artisanal miners who are operating within the Property. This process was initiated following the 2012 implementation of government regulations for formalization of artisanal mining. The formalization process requires the signing of an exploitation agreement between the concession owner and the artisanal miners that stipulates the exploitation methods as well as the legal and environmental responsibilities of the artisanal miners. This agreement will not affect areas of the Project infrastructure mine development, and it can be modified should the concession owner's activities conflict with those of the artisanal miners.

Table 4-1 – List of Compañía Minera Zafranal property mineral rights (concessions and mineral claims)

Count	Concession	Code	Area (ha)
1	010172503	AMALIA GUILLERMINA	200
2[1]	010033810	AQM IV	500
3	010209809	AQP I	800
4	010209909	AQP II	500
5	010210009	AQP III	900
6	010145910	AQP V	800
7	010294310	AQP VII	1,000
8[1]	010479310	AQP VIII	900
9	010015111	AQP X	500
10	010166411	AQP XI	700
11	010166311	AQP XII	1,000
12	010420712	AQP XIII	300
13	010420612	AQP XIV	200
14	010372313	AQP XIX	500
15	010218613	AQP XV	100
16	010371913	AQP XVI	700
17	010372413	AQP XVII	400
18	010371613	AQP XVIII	500
19	010371813	AQP XX	700
20	010371713	AQP XXI	600
21	010372113	AQP XXII	900

Count	Concession	Code	Area (ha)
22	10372613	AQP XXIV	300
23	010171415	AQP XXIX	400
24	010372513	AQP XXV	200
25	010372213	AQP XXVI	300
26	010371513	AQP XXVII	600
27	010130914	AQP XXVIII	500
28	010171515	AQP XXX	400
29	010171615	AQP XXXI	400
30	010171715	AQP XXXII	400
31	010278208	CAMPANERO 1	1,000
32	010278108	CAMPANERO 2	400
33	010554007	CHARO 1	1,000
34	010210403	CHICHARRON_11	1,000
35	010209003	CHICHARRON_N_5	1,000
36	010209103	CHICHARRON_N_6	700
37	010209203	CHICHARRON_N_7	1,000
38	010209303	CHICHARRON_N_8	1,000
39[1]	010187809	SANTUARIO 2009	100
40	010248903	SICERA 1	1,000
41	010295003	SICERA 2	500
42	010313703	SICERA 3	900
43	010330303	SICERA 4	1,000
44[1]	010135403	ZAFRANAL 1	1,000
45[1]	010360803	ZAFRANAL 10	600
46[1]	010360903	ZAFRANAL 11	600
47	010260704	ZAFRANAL 12	1,000
48	010260804	ZAFRANAL 13	1,000
49	010260904	ZAFRANAL 14	1,000
50	010261004	ZAFRANAL 15	500
51[1]	010261104	ZAFRANAL 16	1,000
52[1]	010261204	ZAFRANAL 17	1,000
53[1]	010261304	ZAFRANAL 18	1,000
54[1]	010175103	ZAFRANAL 2	27
55[1]	010261604	ZAFRANAL 21	1,000
56[1]	010175303	ZAFRANAL 3	525
57	010262904	ZAFRANAL 34	1,000
58	010263004	ZAFRANAL 35	700

Count	Concession	Code	Area (ha)
59	010263104	ZAFRANAL 36	500
60[1]	010269403	ZAFRANAL 4	800
61	010313803	ZAFRANAL 7	1,000
62[1]	010340003	ZAFRANAL 8	500
63	010357503	ZAFRANAL 9	500

Note (1) Mining concessions included in proposed development plan

	Mineral Claim	Code	Area (ha)
1	AQP XL	010309315	700
2	AQP XXIII	010372013	200
3	AQP XXXIII	010308615	400
4	AQP XXXIV	010308715	500
5	AQP XXXIX	010309215	500
6	AQP XXXV	010308815	600
7	AQP XXXVI	010308915	600
8	AQP XXXVII	010309015	800
9	AQP XXXVIII	010309115	900

Table 4-2 – List of Zafranal administrative economic units ('Unidad Económica Administrativa')

	UEA	Concession	Code	Area (ha)
1	CMZ 1	CAMPANERO 2	010278108	400
2	CMZ 1	CHICHARRON_N_5	010209003	1,000
3	CMZ 1	SICERA 4	010330303	1,000
4	CMZ 1	ZAFRANAL 34	010262904	1,000
5	CMZ 1	ZAFRANAL 36	010263104	500
6	CMZ 2	AMALIA GUILLERMINA	010172503	200
7	CMZ 2	SICERA 1	010248903	1,000
8	CMZ 2	SICERA 2	010295003	500
9	CMZ 2	SICERA 3	010313703	900
10	CMZ 2	ZAFRANAL 10	010360803	600
11	CMZ 2	ZAFRANAL 13	010260804	1,000
12	CMZ 2	ZAFRANAL 15	010261004	500
13	CMZ 2	ZAFRANAL 35	010263004	700
14	CMZ 3	CHICHARRON_11	010210403	1,000
15	CMZ 3	CHICHARRON_N_6	010209103	700
16	CMZ 3	CHICHARRON_N_7	010209203	1,000
17	CMZ 3	CHICHARRON_N_8	010209303	1,000
18	CMZ 3	ZAFRANAL 11	010360903	600
19	CMZ 3	ZAFRANAL 17	010261204	1,000
20	CMZ 3	ZAFRANAL 21	010261604	1,000
21	CMZ 3	ZAFRANAL 7	010313803	1,000
22	CMZ 3	ZAFRANAL 9	010357503	500
23	CMZ 4	ZAFRANAL 1	010135403	1,000
24	CMZ 4	ZAFRANAL 12	010260704	1,000
25	CMZ 4	ZAFRANAL 14	010260904	1,000
26	CMZ 4	ZAFRANAL 16	010261104	1,000
27	CMZ 4	ZAFRANAL 18	010261304	1,000
28	CMZ 4	ZAFRANAL 2	010175103	27
29	CMZ 4	ZAFRANAL 3	010175303	525
30	CMZ 4	ZAFRANAL 4	010269403	800
31	CMZ 4	ZAFRANAL 8	010340003	500

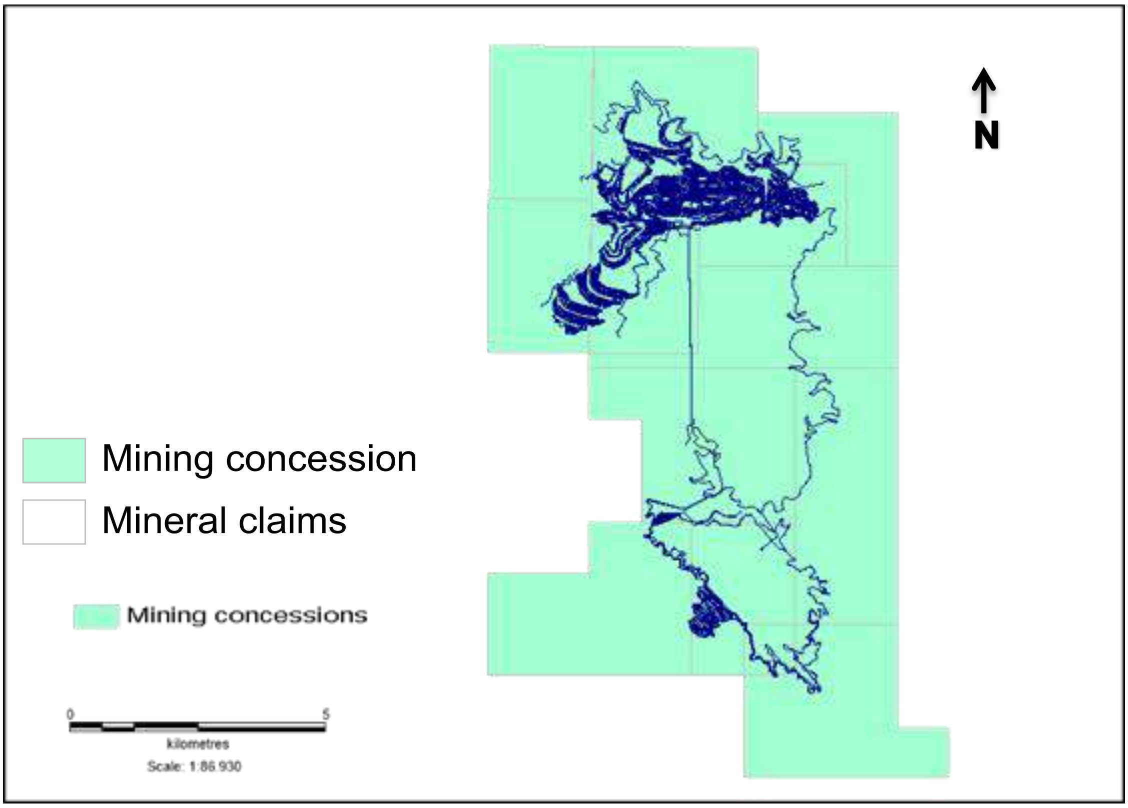

Figure 4-2 – Map of Zafranal Project showing the 14 concessions related to the proposed development

4.3 Owner's Title and Interest

In Peru, surface rights and subsurface mineral rights are legally distinct; therefore, a mineral concessionaire requires the corresponding surface rights to access the land. The Project is located within state owned land held by Autoridad Autónoma de Majes (Autodema), an agency at the Regional Government of Arequipa that manages the Majes Siguas Irrigation Project.

CMZ holds the right of access to the Property to carry out mineral exploration through a renewable lease agreement entered into with Autodema. The lease agreement term was last renewed on 11 September 2015 and is in force until 27 October 2017.

CMZ also reached an agreement with the co-owners of a section of land called 'Fundo Huacan', which partially overlaps Autodema's land. The co-owners filed a suit against Autodema in 2006, claiming a large portion of the land on the grounds that their property right is recorded in the Public Registry. Both the ownership and the size of the 'Predio Huacan' are matters subject to the lawsuit. Rather than risk potential delay while the lawsuit is resolved, CMZ proposes to reach agreement with both parties so that project development activities can proceed regardless of the outcome of the lawsuit.

As noted above, CMZ holds the title to 63 mineral concessions. According to the General Mining Law, mineral concessions are irrevocable as long as the concessionaire fulfills the obligations to pay an annual 'Mining Good Standing Fee' and achieve an annual minimum production level, as per the General Mining Act provisions. The 'Mining Good Standing Fee' is currently equivalent to US$3.00/ha and is payable on an annual basis to maintain the concessions and mineral claims (concession applications) in good standing. Lack of payment for two consecutive years results in the cancellation of the corresponding mineral right.

4.4 Others Interests, Royalties and Encumbrances

The Zafranal Project property mineral rights are free and clear of recorded liens, encumbrances or agreements except for the Chicharron concessions (Table 4-3) that Teck Peru purchased from the Peruvian branch of BHP Billiton World Exploration Inc. The sale of metals from these concessions remains subject to a 1.5% Net Smelter Return (NSR) Royalty capped at US$2.0 million. Once that amount is reached, the NSR Royalty will expire. None of the Chicharron concessions are included in the current development plan for the Project.

Table 4-3 – The 'Chicharron' concessions

	Concession	Code	Area (ha)
1	CHICHARRON_11	010210403	1,000
2	CHICHARRON_N_5	010209003	1000
3	CHICHARRON_N_6	010209103	700
4	CHICHARRON_N_7	010209203	1,000
5	CHICHARRON_N_8	010209303	1,000

There is a Government royalty of 1% minimum.

4.5 Environmental Liabilities

In the area and vicinity of the Zafranal Project there are 490 identified environmental liabilities, consisting of mine adits, chimneys, test pits, and waste dumps, among others. These correspond to informal mining works from the 1990s onward and some previous formal mining exploration activities by other companies.

Minera AQM Copper Perú S.A.C. (MAQM) inventoried and declared these environmental liabilities to the Ministry of Energy and Mines, General Bureau of Mining, during the period when it acted as the concession holder for the Property.

The environmental liabilities within the footprint of the proposed Zafranal Project permanent facilities (open pit, waste rock deposits, tailings management facility) are not being considered for closure and remediation activities, as they will be permanently covered or removed during the development and operation of the Project. Those liabilities that lie within the footprint of the non-permanent Project facilities will be remediated within the scope of the definitive Zafranal Project Closure Plan, which is to be submitted within one year of receiving approval for the Project's Environmental Impact Assessment. Those environmental liabilities outside the footprint of the proposed Project that would have been assigned responsibility to CMZ by the Ministry of Energy and Mines will be addressed in an Environmental Liabilities Management Plan. None of the declared environmental liabilities on the Property are deemed to be significant issues.

4.6 Permitting Requirements and Status

4.6.1 Environmental Impact Assessments

Environmental Impact Assessments are required to be approved by the Ministry of Energy and Mines prior to carrying out mineral exploration and by the Environmental Certification Agency, (SENACE 'Servicio Nacional de Certificación Ambiental para las Inversiones Sostenibles') prior to carrying out mineral exploitation.

Mineral exploration requires an Environmental Impact Declaration (DIA 'Declaración de Impacto Ambiental') or a Semi-detailed Environmental Impact Assessment (EIA-sd 'Estudio de Impacto Ambiental Semi-detallado'). Mineral exploitation requires a Detailed Environmental Impact Assessment (EIA-d 'Estudio de Impacto Ambiental Detallado').

MAQM as operator and CMZ as titleholder have been carrying out exploration work on the Property subject to a Semi-detailed Environmental Impact Assessment (EIA-sd 'Estudio de Impacto Ambiental Semi-detallado') approved in December 2009 through Resolution N° 420-2009-MEM/AAM of the Ministry of Energy and Mines. The EIA-sd was modified in April 2011 and July 2013, respectively through Resolution N° 099-2011-MEM/AAM and Resolution N° 264-2013-MEM/AAM. Technical Reports (ITS 'Informe Técnico Sustentatorio'), covering program modifications within the EIA-sd study envelope, were also approved in January 2014, October 2014 and May 2015, respectively through Resolution N° 049-2014-MEM-DGAAM, Resolution N° 529-2014-MEM-DGAAM and Resolution N° 221-2015-MEM-DGAAM.

4.6.2 Other permits

CMZ's exploration activities are being developed according to the Second Modification to the EIA-sd and the Third Technical Report, ITS 3. These activities also require the following sectoral permits:

Table 4-4 – Other permits

Permit	Agency	Resolution	Validity Term
2015-2016 Exploration water use	AAA	1145-2015-ANA/AAA I C-O	31 December 2016
Potable treatment plant	DIGESA	235-2013/DSB/DIGESA/SA	2 August 2017
Zafranal Camp septic system	DIGESA	224-2013/DSB/DIGESA/SA	Definitive
Ganchos Camp septic system	DIGESA	231-2013/DSB/DIGESA/SA	Definitive
Zafranal Camp fuel storage tank	OSINERGMIN	8662-2013-OS/COR	Definitive
Ganchos Camp fuel storage tank	OSINERGMIN	8662-2013-OS/COR	Definitive

Project construction and operation stages require an EIA-d approved by SENACE and will need additional sectoral permits that will be required throughout the life of the mine.

4.7 Access, Title and Permitting Risks

CMZ has secured all legal rights for the Project's mineral concessions; and these rights have been recorded at the Public Registry. CMZ does not foresee any risk in losing its legal rights for the mineral concessions. Surface rights for exploration activities have also been secured.

CMZ is required to enter into a long-term agreement for land use during the Project's construction and exploitation stages, and is also required to enter into agreements with owners of the land needed for off-site infrastructure.

Other than those identified in this report, there are no other significant factors or risks known that may affect access, title, or the right or ability to perform work on the property.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Topography, Elevation and Vegetation

The Project area sits on the boundary between the Pampa de Majes mid-elevation plain and the Andes Mountains. The topography of the Project area can be separated into three distinct types.

1. Pit area and waste dump areas (approximately 1,900 to 2,900 masl)
2. Plant site (approximately 1,600 to 1,900 masl)
3. Tailings management facility (approximately 1,400 to 1,600 masl).

In the northeast of the property, where the open pit and waste rock dumps would be located, the topography is steep and mountainous, dominated by elongated peaks intersected by gullies with slopes in excess of 70% and frequent rock outcroppings of varied mineralogy. In this area there are few spaces available with moderate to flat slopes; those that are available will be used for the installation of mining support facilities and utilities.

In the surroundings of the concentrator site and the final tailings management facility boundaries, the mountainous terrain becomes less dramatic and more undulated. Although some slopes are steep in some parts of this area, there is a higher frequency of flatter slopes.

To the southwest of the Project area, particularly within the proposed tailings management facility the topography is flatter, dominated by a series of interconnected colluvial-alluvial plains.

The topography and elevation, as well as climatic factors determine the type of vegetation present over the Project area. The higher elevation vegetation consists primarily of diverse species of low-growing grasses and matorral shrubs, interspersed with columnar cacti. In the lower elevations there is a much sparser vegetation cover, although with similar plants.

5.2 Access to Property

The Project can be accessed via 166 km of roadway from the city of Arequipa. About 105 km of this road is paved, including the Pan-American Highway. The last 61 km to the Property is a gravel road of which 12 km is private road. The distance from Arequipa to the Project Site will be reduced to 144 km once a new 8.5 km road extension is constructed from the existing eastern access road.

5.3 Proximity to Population Centres and Nature of Transport

The population centres in proximity to the Zafranal Project are: Huancarqui, Lluta, Anexo El Pedregal (Corire) and El Pedregal de Majes. Transportation between the Project and the population centres is by personal vehicle. The Property is well-removed from agricultural areas and does not contain any communities or settled areas.

5.4 Climate (including any specific effects on operation, e.g. operating schedule)

The climate in the Project area is characterized as being temperate desert, based on the precipitation, evaporation and temperature regime, which has been shown to have an altitude dependency.

The annual average isohyets vary between 30 mm for the TMF area at 1,580 masl and 450 mm on the peaks that dominate the area at 4,500 masl. The annual average precipitation of 152 mm, 112 mm and 50 mm for the valleys containing the open pit, TMF and concentrator, respectively.

During the rainy season, between January and March, 81% of the annual precipitation falls, while during the driest months, between May and November, 12% of the annual precipitation falls.

An analysis of the maximum 24-hour precipitation frequency was developed. The Probable Maximum Precipitation ("PMP") was calculated as 232 mm for the mid-altitude (2,581 m) in the valley above the TMF by the Hershfield method (Reference 27.5.3).

The average multi-annual Class A pan evaporation was calculated to be 3,085 mm/y.

The average annual temperature at the sites of the future Project installations has been calculated for their respective average altitudes as 13°C, 16°C and 18°C for the open pit, concentrator and TMF, respectively. The maximum and minimum temperature-altitude regressions give 19°C, 22°C and 24°C as the maximum temperatures for the open pit, concentrator and TMF, and 7°C, 8°C and 9°C as the respective minimum temperatures for the same facilities.

5.5 Surface Rights (sufficiency for mining operations)

Legal access to the land for mining operations at the mine site requires CMZ to enter into an agreement with Autodema and a co-owner currently contesting land ownership with Autodema. Surface rights required for off-site infrastructure will also need to be negotiated. For purposes of the Transmission Line an easement granted by the Ministry of Energy and Mines may apply.

5.6 Availability of Local, Regional and National Infrastructure, Labour and Contractors

Sufficient electrical power is currently available for a new mining operation with additional power lines and upgrades of existing lines in Arequipa province and in southern Peru under construction. Several hydroelectric projects and gas-fired power stations are also planned to be on line by the start of Zafranal's operations.

Water is a key issue for Zafranal as it is for almost all projects in southern Peru. There are no available permanent surface water courses near the Project site; however, a brackish groundwater resource suitable for operation has been located within 35 km of the concentrator site, a groundwater source suitable for construction water has been identified within 8 km of the concentrator site, and a groundwater source suitable for potable water has been identified within 9 km of the concentrator site. The groundwater source and water supply system suitable for the proposed operation are described later in this report.

Arequipa is a mining district with a number of large operating mines, and the current labour market has availability of mining equipment and concentrator operators or skilled trades technicians, due to the recent market turndown. Existing and new mines have been proactive in skills training in the region. Local people will require up-skill training but will be a good source of base labour.

Peru has a good supply network for mine support fabrication workshops and material and equipment supply companies with wide global support to the region. It also has a large construction company base with long-term experience in mining projects ranging from large to mid-tier organizations operating across the country. These fabricators and constructors are currently experiencing a downturn in the market creating an environment of excess capacity.

There are currently regular bus and freight services between Arequipa, the two major local towns of Corire and El Pedregal de Majes and other minor towns in the region; however personnel and freight transport to the Project site is currently by private vehicles or by specific arrangement. The nearest significant port is Matarani, which is well-suited to support Project construction and operation. The nearest domestic airport is on the outskirts of Arequipa and the nearest international airport is in Lima-Callao.

5.7 Physiography

The physiography of the mining area is characterized by rugged terrain, steep scoured river valleys and generally poor surface rock quality. In this area there are few spaces available with moderate to flat slopes; those that are available will be used for the installation of mining support facilities and utilities and are restricted to higher elevations, which in turn negatively impacts the capital and operating cost of these facilities. These difficulties for infrastructure become advantages for the development of the open pit, in that waste from the mine can be deposited in nearby valleys using relatively short haulage cycles.

The physiography of the Project site has led to the development of the layout of the concentrator, administration buildings, accommodation camp and tailings management facility in three broad locations to obtain sufficient areas and suitable topography for each. The primary crusher requires little area and is located adjacent to the mine with the foundations in rock cut. Due to the steep topography between the primary crusher and the rest of the concentrator, the conveyor that transfers crushed ore to the stockpile feeding the grinding circuit passes through a tunnel 3.6 km long. The rest of the concentrator is terraced down a ridge, with the concentrator buildings, administration buildings and accommodation camp located further downhill on the same ridge. The areas are compact but adequate for the facilities and for construction requirements, including laydown storage. The TMF is located 1 km south southeast from the concentrator in a basin at the foot of the Andes. The basin is surrounded by rugged mountain terrain requiring only an initial embankment across a narrow gorge in the southwest called Quebrada Huacan and a later embankment across a saddle in the northwest.

5.8 Site Geotechnical, Hydrogeology and Hydrology

As part of the pre-feasibility studies for the Project, site-wide geotechnical, hydrogeological and hydrological studies were performed. The following section summarizes these programs. Detailed results from these programs can be found in the appropriate sections of this report, where the information has been used in the development of the Project facilities.

5.8.1 Site geotechnical

The areas investigated for Project facilities were, in general, determined to have suitable foundation conditions at the proposed pad and foundation base levels in the Project design.

The southwest TMF embankment spans Quebrada Huacán. The right abutment is bedrock consisting of carbonate rock at the top and volcanic at the bottom. The base of the quebrada is covered by 50 m of alluvium and below the alluvium is siltstone followed by diorite (upstream footprint) and Chocolate Formation (downstream footprint). The underlying rock formations have a relatively low permeability, however, there are faults that transverse the gully which may have high permeability. The upper layer of bedrock siltstone is fractured to highly fractured, while underlying host rock is less fractured.

Between the southwest embankment and the northwest embankment the hillside contains the Chocolate Formation consisting of carbonate rock. Based on both field mapping and a drilling program the formation consist of marls and marly limestones having low to moderate carbonate content. Small scale dissolution features are present on surface in both the limestones and calcarnites of the Socosani Formation indicating some potential for karstification; however, larger scale dissolution features are not present on surface within the mapped areas. The risk of a large scale release of tailings and pond water from the TMF into karstic features is considered very low considering the lack of karst development in the carbonate rocks within the footprint of the TMF.

Along the crushed ore transfer conveyor tunnel alignment the lithology is mostly andesite, in which the quality Rock Mass Rating (RMR) improves with depth. In addition, mylonites were discovered along the tunnel alignment, which are of poorer quality than the andesite according to RMR. During the preliminary optimization of the crushed ore transfer conveyor tunnel alignment the tunnel was moved to more favorable rock quality lithology, based on surface mapping and a previous geotechnical field investigation.

Only borrow areas for construction materials within the TMF impoundment were investigated because they were included in the field investigation permit for the preliminary feasibility study. The investigation identified an area of approximately 100 ha where appropriate materials exist that can likely be screened to generate the required construction materials.

There are a number of faults within and near the project facilities based on a preliminary literature search and data available from Instituto Geológico, Minero y Metalúrgico (INGEMMET). The Incapuquio fault zone, which includes the Pampacolca Fault, the Lluclla Fault, and other unnamed faults, runs through the Project area.

An active fault is a fault that is expected to be the source of another earthquake in the future. Geologists commonly consider faults to be active if they have moved at least once in the last 10,000 years (USGS 2009). A preliminary literature review of the Incapuquio fault zone in the area of the project indicates it has not been active since late Pleistocene epoch and therefore would be considered inactive. During the geotechnical field investigation, a search for surface expressions on the major Incapuquio faults through the project that might suggest the age of the most recent activity, such as scarps, displacements, and other features did not find any features that would suggest activity within the last 10,000 years. During the feasibility study a more detailed seismic hazard analysis should be performed, along with trenching across identified key faults to confirm that the faults within or near the project are inactive.

5.8.2 Site hydrogeology

The hydrogeology of the Zafranal Project has been divided into highland (mountainous) and lowland (pampa) areas. The highland areas, located north of the TMF and Pampas, includes mine facilities such as the open pit, waste dumps, heap leach (potential future), crushed ore transfer conveyor tunnel, and concentrator; whereas, the lowland areas includes the TMF and groundwater resource targets within Pampa Ganchos and Quebrada Huacan. In addition to the hydrogeological studies carried out at the Zafranal site, a groundwater exploration study to find an operational water supply was completed at the Majes I irrigation area, which is located approximately 25 km south-southeast of Zafranal.

Groundwater movement in the mountainous areas is dominated by faults, whereas groundwater movement in the lowland areas is controlled by faults in addition to permeable strata within alluvium and fractured sedimentary and volcaniclastic rocks. Groundwater movement is generally toward local fault-controlled quebradas where groundwater is shallowest. A regional map with groundwater elevations and flow directions is shown in Figure 5-1.

A numerical groundwater flow model (developed by DHI Peru S.A. using FEFLOW) was constructed to evaluate baseline conditions on a regional and local scale, evaluate potential impacts that may occur as a result of project development, and provide a tool to help guide engineering designs and minimize potential effects. Potential impacts are described in Section 20.1.2.

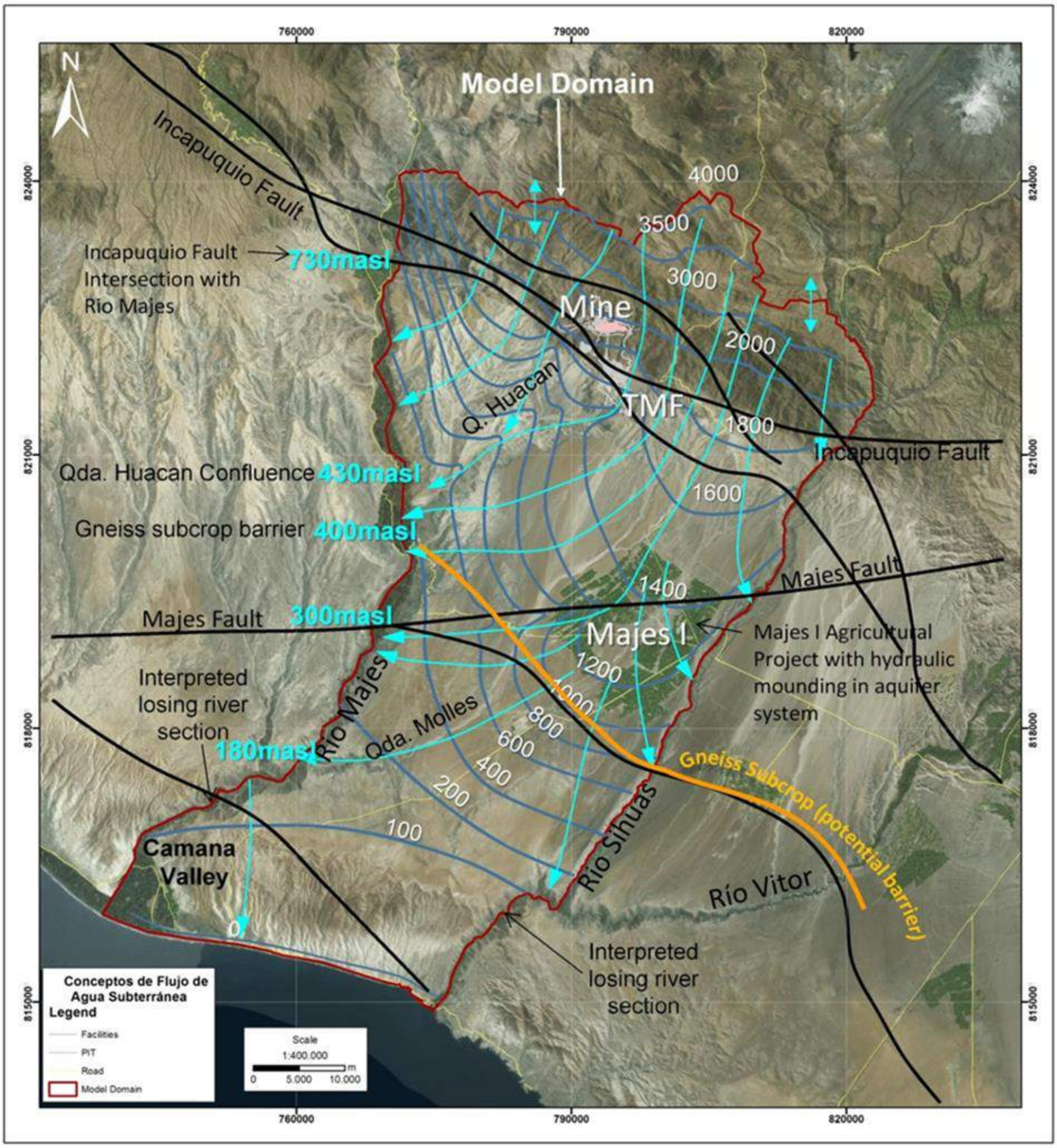

Figure 5-1 – Conceptual regional groundwater flow patterns

5.8.3 Site hydrology

At a Project level, four basins of the Huacán Creek are directly associated with the location of the future facilities for the Zafranal Project. These unnamed basins have been identified for the purposes of the PFS as A, B, C and D, as described below:

- Basin A is the basin where the north waste dump and a portion of the open pit are to be located, down to its intersection with the concentrator-to-mine access road.

- Basin B is the basin where the northeast waste dump and a portion of the open pit are to be located, down to its intersection with the access road.
- Basin C corresponds to the Cachimayo Creek basin down to its confluence with the Lloquelloy creek, and where the concentrator and part of the TMF are located.
- Basin D corresponds to the Lloquelloy creek basin down to its confluence with the Cachimayo creek, and where part of the TMF is located.

Surface runoff into the creeks of the Project area is characterized as ephemeral; it only occurs in times of heavy precipitation that produce effective rain[1]. Ephemeral flows do not intercept groundwater, and therefore have no base flow; instead they feed groundwater by infiltration. Conceptually, for the area of the Project, the groundwater discharges to the Majes River canyon 800 m below the area of the Project; hence, the entire Project area is considered to be an area of recharge. The surface flows were calculated indirectly by applying a rainfall-runoff model since there are no hydrometric stations in channels with features such as those that are present in the Project area.

The average annual flow in the creeks of the area of the Project were obtained from the rainfall-runoff models for situations with and without the Project developed.

The peak watershed flows for different return periods were estimated and the probable maximum precipitation (PMP) for the area of the Project, and specifically for Basins C and D that discharge into the TMF, enabled the estimation of the potential volume for the probable maximum flood (PMF) as 17 Mm^3.

[1] Effective rainfall (or precipitation) is equal to the difference between total rainfall and actual evapotranspiration. SOURCE: SOeS (site : http://www.statistiques.developpement-durable.gouv.fr)

6 History

6.1 Ownership History

The Zafranal Property belonged initially to Teck Cominco Limited, now Teck Resources Limited (Teck) through its wholly-owned Peruvian subsidiary Teck-Cominco Peru S.A. (Teck Peru). In 2009, by means of an Option/Joint Venture (JV) Agreement, AQM Copper Inc. (AQM), through its wholly-owned Peruvian subsidiary Minera AQM Copper Peru S.A.C. (MAQM), acquired from Teck Peru the right to earn a 51% interest in the Property. Teck Peru retained the right to earn back-in up to a 60% interest in the Property. In the event Teck Peru did not elect to exercise the back-in, AQM had the right to increase its interest up to 100%. At that time the Zafranal Property was 24,499.63 ha in area and was comprised of 32 concessions.

In 2010, the option agreement was amended whereby AQM Peru and Teck Peru formed a 50/50 joint venture (JV) to explore and develop the Property. The JV entity, Compañía Minera Zafranal S.A.C. (CMZ), was constituted in January 2011 and the ownership of the Zafranal mining concessions was transferred to it from Teck Peru. Teck Peru's shares in CMZ were then transferred to Teck. MAQM was immediately vested to a 50% interest in the Property. All cash payment obligations held by MAQM and all royalties and back-in rights held by Teck Peru were cancelled in exchange for the granting of 5 million AQM shares to Teck and the commitment by MAQM to fund $10.7 million in exploration expenditures in addition to the $7.5M already spent by MAQM on the Property.

A Shareholders Agreement (SHA) was signed between MAQM and Teck on 1 January 2011, shortly after MAQM completed the total of $18.2M in exploration expenditures. Under the terms of the SHA, Teck has the right to become the Project operator when a production decision has been made and CMZ can choose a new operator if there has been a change of control in AQM. In the meantime, MAQM is the Project operator.

MAQM completed the process of transferring the exploration intangible associated with its $18.2 million expenditure in the Property to the JV Company, CMZ, on 12 September 2011, and received shares in CMZ equivalent to its 50% ownership. In August 2013 Mitsubishi Materials Corporation invested US$22.60M and committed an additional US$15.07M to acquire a 40% interest in MAQM.

In 2013 the JV partners agreed that CMZ would start to directly manage the activities of the Project and the agreement that authorized MAQM to act on behalf of CMZ was terminated. MAQM transferred mining concessions held in its name, all permits and permissions granted by the authorities to conduct exploration on the Property to CMZ, and selected personnel. The termination of the agreement that authorized MAQM to act on behalf of CMZ did not affect MAQM's rights as Project operator.

6.2 Exploration and Development by Previous Owners

Several artisanal or small-scale miners actively mine for gold in the auriferous quartz veins on the periphery of the Zafranal porphyry systems. Although the date when mining started in the district is unknown, artisanal miners have worked on the area continuously over the past decade. In 2003, a group of these miners showed a team of Teck geologists, led by Manuel Montoya, samples from the Zafranal Main zone displaying strong phyllic alteration and quartz veining stockworks. Teck immediately staked the area covering the Zafranal Main zone. In 2009, Teck optioned the Property to AQM Copper after a competitive bidding process. With the exception of gold production by artisanal miners, there has been no commercial production from the Property.

Since the 1990s, due principally to improving security conditions and laws encouraging exploration and mining, Peru has attracted large quantities of exploration investments. Significant copper porphyry discoveries and developments made in Peru over the past decade include Toromocho (Chinalco), Cañariaco (Candente Resources), Haquira (First Quantum) Las Bambas (MMG) and Constancia (Hudbay).

Phelps Dodge had claims in the 1990s over the Zafranal Main zone but did not conduct significant exploration. They had also staked the oxide copper area known as Rosario and drilled four shallow drill holes there in the 1990s as well as some RC holes on the Sicera Sur target. Starting in 2005, Teck consolidated the properties and conducted drilling over the Zafranal Main zone, Campanero, Sicera Sur, Sicera Norte and Ganchos zones.

Western Mining and BHP Billiton held land positions within the current concession area but did not conduct any drilling.

6.2.1 Exploration history – Zafranal Main zone

Teck explored the Zafranal Main zone between 2003 and 2005, starting with surface mapping and rock sampling that confirmed the presence of an extensive area with porphyry-style geology, alteration and copper-gold mineralization. Exploration work also included mapping the Zafranal Main zone deposit at a 1:10,000 scale starting in 2003 with updates made through 2005. Property-scale geological descriptions are available in previous Teck reports by M. Smith and W. Tejada (Smith & Tejada, 2004 - See Reference 27.6.1) and W. Tejada (Tejada, 2005 - Reference 27.6.2). AQM has subsequently re-mapped the area.

Based on the results of its geochemical and surface mapping programs, Teck drilled 22 reverse circulation (RC) holes and four core holes in 2004. The RC drilling identified a 1.2 km long east-west trending supergene enrichment blanket under the leached horizon. Drill holes were oriented in a north-south direction and were generally spaced between 250 to 400 metres apart along strike (east-west direction) and 200 m on section. Discovery hole ZFRC04-008 intersected 110 metres with an average grade of 1.02% copper, followed by similar intervals in holes drilled along an east-west fence through the middle of what is now the Zafranal Main zone resource block.

Core holes were drilled to confirm the validity of the RC drill results. Drill hole ZFDDH04-001 twinned previous drill hole ZFRC04-008, while diamond drill holes ZFDDH04-002, ZFDDH04-003 and ZFDDH04-004 were collared at the same locations of RC drill holes ZFRC04-008, ZFRC04-0010 and ZFRC04-007 respectively, but oriented due north in order to test the possible extension of mineralization to the north of the main mineralized zone. Diamond drill hole DDH04-001 which was a twin of ZFRC04-008, intersected 166 metres with an average grade of 1% copper. All of the twinned diamond drill holes confirmed the existence of a high-grade chalcocite blanket identified by the RC drilling program.

A further ten RC holes were drilled in 2004 to test the southern and northern extensions of the mineralization intersected by the first drill campaign. Most of the holes drilled cut significant copper mineralization, though the drill hole spacing was not systematic and they mostly projected outside of the northern and southern boundaries of the enrichment blanket. Table 6-1 shows the Teck drill campaigns completed over the Zafranal Main zone.

Table 6-1 – Teck drill campaigns on Zafranal Main zone

Period	N. of Drill holes	Total Metres	Drill Type
May-June 2004	12	3,689	RC
September-October 2004	10	3,312	RC
November 2004	4	1,556	Diamond
September 2005	10	3,248	RC
Total	36	11,805	

The September 2005 a reverse circulation drill program consisted of ten drill holes testing the possible western and eastern extensions of the supergene mineralization encountered in previous programs.

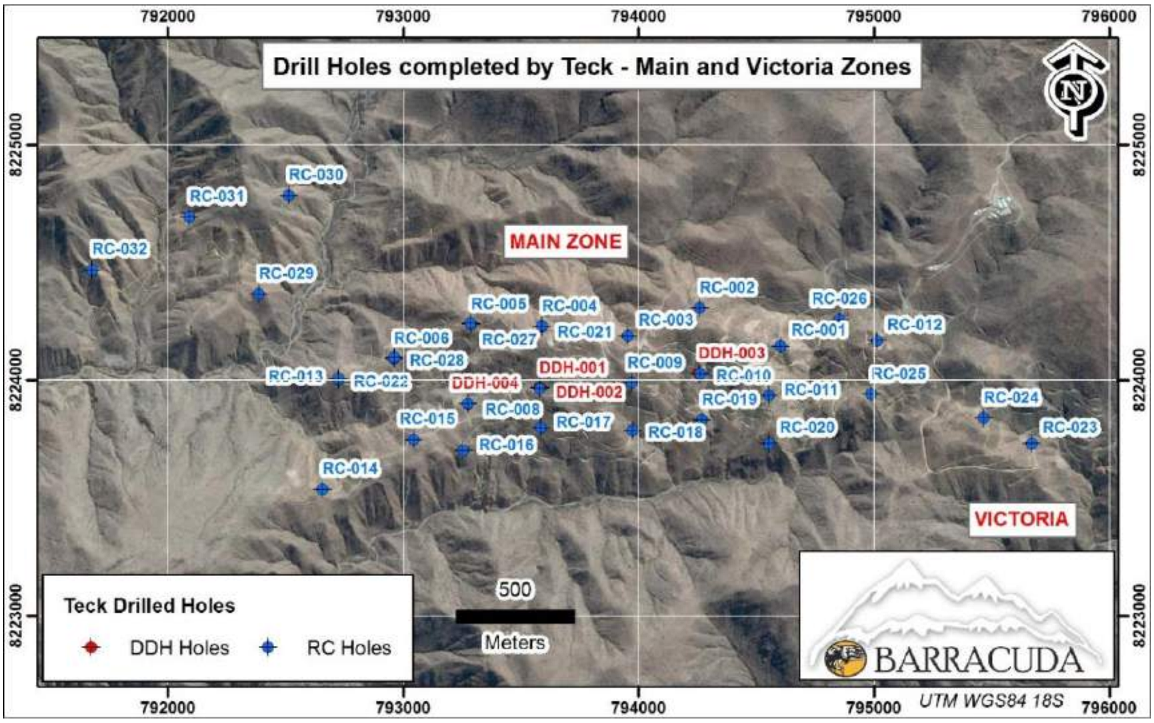

Figure 6-1 – Drill holes completed by Teck between 2003 and 2005

In 2004, following the first phase of drilling, Teck carried out orientation-scale time-domain electromagnetic (TEM) and audio frequency magneto-telluric (AMT) surveys. The AMT survey successfully identified strongly mineralized secondary copper mineralization and represents the best indirect exploration method over the target.

Between 2006 and 2007, Teck conducted scout drilling in peripheral areas of the Property, looking for possible porphyry targets concealed beneath the Tertiary cover and new systems along the regional structural controls. No further work was done on the Zafranal Main zone between 2005 and May 2009. AQM acquired an option on the Property following a thorough onsite review of surface geology and alteration patterns led by Tom Henricksen and José Corzo.

6.2.2 Exploration history - Victoria zone

The Victoria zone is contiguous to the eastern end of the Zafranal Main zone. It was discovered in 2011 as AQM stepped out its drilling eastwards from the Zafranal Main zone. Teck had drilled three RC holes on what is now the southern edge of Victoria, where argillic and phyllic alteration is exposed and where AMT lines had identified weak conductivity anomalies. No significant results were found and the zone remained essentially unexplored until AQM´s systematic drilling campaigns started in 2010.

6.2.3 Exploration history – Other zones on the Property

There is no material exploration history of other zones on the Property: Campanero, Sicera Sur, Sicera Norte and Ganchos. The Moly Sur zone was not previously explored.

6.3 Historical Mineral Resource Estimates and Mineral Reserve Estimates

All previous mineral resource estimates for the Zafranal Project were prepared for and reported by AQM; hence, are not "historical" as defined in NI 43-101, and are superseded by the Mineral Resource Estimate presented in this Technical Report.

There are no previous Mineral Reserve Estimates for the Zafranal Project.

7 Geological Setting and Mineralization

7.1 Regional, Local and Property Geology

7.1.1 Regional geology

The geology of Southern Peru can be subdivided into three main tectonic belts, all trending roughly parallel to the Pacific shoreline. To the east, adjacent to the Archean Brazilian craton, lies the Cordillera Oriental. Along the central spine of the Andes Mountains is the Altiplano belt of highland plateaus and intramontane valleys and to the west lies the Cordillera Occidental mountain range. The Andes mountain range itself, which extends along the entire length of Peru, was created by crustal thickening and subduction of the Nazca plate beneath the South American Plate. This process began during the Jurassic Period and continues to this day.

The Zafranal Property is located within the Cordillera Occidental belt. The southern section of this belt is underlain by Proterozoic metamorphic rocks belonging to the Arequipa massif and younger Paleozoic and Mesozoic sedimentary rocks, all intruded by large intrusive complexes emplaced during the Andean orogeny. This multi-phase intrusion is known locally as the Coastal batholith. In the Cordillera Occidental of Arequipa, it occurred 107 million years ago for the oldest phases and 82-80 million years ago for the youngest phases (Moore, 1983- See Reference 27.7.4). Thick volcanic sequences of intermediate composition that dominate the higher elevations were the result of continued subduction of the oceanic Nazca plate during the late Cretaceous and early Tertiary eras. The belt is host to several porphyry copper deposits and occurrences, all related to intrusions associated with Late-Cretaceous to mid-Tertiary magmatism.

The Cordillera Occidental belt is controlled to the west by a deep-seated fault system known as the Incapuquio Fault. The Incapuquio Fault is comprised of several left-lateral reverse strike slip faults which uplifted the eastern block, dominated by Mesozoic rocks, over the western block which was filled by very thick sequences of Tertiary tuff, ignimbrite and basin-fill sediments (collectively known as the Moquegua formation). Complex fault movement and reactivation, along with dip slip movement, has created dilation and an extensional environment along which imbricate E-W faults developed. In common with several other porphyry deposits in the region, Zafranal is located along one of these E-W fault-bounded corridors spatially linked to the Incapuquio fault.

7.1.2 Local geology

The major rock types within the Zafranal property, from oldest to youngest, are: Jurassic-aged volcanic rocks of the Guaneros and Chocolate Formations, Cretaceous Yura Formation limestone, shale, quartz arenite and marl, a Cretaceous age granodiorite, microdiorite and diorite intrusive suite and Miocene age Moquegua Formation tuff, ignimbrite and sandstone.

Windows of older rocks exposed next to Cretaceous age granodiorite host most of the mineralization in the Property. Exposure of these older rocks occurred due to block movement along two major east-west faults closely linked to the nearby Incapuquio fault. Next to these major faults are gneissic rocks consisting mainly of strongly foliated volcanic and intrusive rocks. This unit was mapped as Precambrian gneiss in the past, but more detailed mapping suggests it as lacking in high-grade metamorphism, and likely to represent highly deformed country rock. Several tectonic events, including major shearing, have given it its strongly deformed (or gneissic) appearance – often with mylonitic textures, which gradually grades into foliated host rock. The geology of the Zafranal Main zone, Victoria and Moly Sur zones is described in more detail below. The 2010 technical report prepared by Amec Minproc (Manfrino, Harbort, &

McCrea, 2010- See Reference 27.7.3) describes the geology of the Sicera Sur, Sicera Norte and Campanero zones, which are not included in the current Preliminary Feasibility Study.

7.1.3 Property geology

7.1.3.1 Zafranal Main zone

The Zafranal Main zone lies within an east-west trending block bounded to the north and south by two large-scale faults. The oldest rocks exposed belong to the Lower Jurassic-age Chocolate Formation and consist of interbedded sedimentary and volcanic rocks. The sedimentary members include siltstone, sandstone and sedimentary breccia (debris flows), while volcanic members include tuff, breccia and andesitic lava flows, together with sub-volcanic units of andesitic composition.

Intrusive rocks in the Zafranal Main zone are classified into four types: Zafranal Diorite, Microdiorite, Quartz-diorite and Post-mineral Diorites. The Zafranal Diorite is the oldest and most abundant of the intrusive rocks, is greenish-grey and porphyritic, and occurs as stocks and dykes. This intrusive body shows sericite-chlorite-biotite alteration (phyllic superimposed on potassic) on surface, together with thin D-type and B-type veinlets, with the latter turning into A-type, at depth.

The Microdiorite is fine-grained and greenish and occurs as stocks at the eastern and western ends of the Zafranal Main zone. It typically has a chlorite-biotite alteration and moderate superimposed sericite. It is potassically altered at depth, with a quartz-secondary biotite-chlorite assemblage accompanied by thin B-type and A-type veins. The Microdiorite cuts the Zafranal Diorite and its emplacement appears to coincide with the main copper-mineralizing event.

The Quartz-diorite is commonly phaneritic, dark grey, and contains disseminated pyrite and magnetite. It cuts both the Zafranal Diorite and Microdiorite and contains low-grade hypogene mineralization.

Post-mineral dikes and minor apophyses are the last-documented intrusive event on the Zafranal Main zone. This phase cuts the preceding three intrusive phases and is unmineralized (except over the supergene enrichment blanket), though in some areas it is propylitically altered. Rhyolitic dykes occur in the north-central part of the Zafranal Main zone. Most post-mineralization dykes may have intruded along faults.

All of the intrusive bodies found on the Zafranal Main zone are believed to be of late Cretaceous age, and are interpreted to represent the final phases of the Coastal batholith. These intrusions are likely to be responsible for the significant hydrothermal alteration found at Zafranal.

A large andesitic breccia occurs at the southwestern edge of the Zafranal Main zone. These rocks are interpreted as Pliocene flows that cover all other rock types and are younger than any alteration or mineralization.

7.1.3.2 Victoria zone

The Victoria zone hosts a variety of strongly deformed and locally metamorphosed northerly dipping extrusive and intrusive rocks of undetermined age. The protolith is principally hornblende diorite, which has been intruded by mafic to ultramafic andesitic dykes and later porphyritic diorite bodies similar in texture to the Zafranal diorite in the Zafranal Main zone. Primary volcanic textures in mafic units suggest that the intrusive rocks were emplaced into a sequence of basaltic andesite volcanic tuffs and/or flows. All of these rock units have been strongly deformed by a regional, dextral shear zone marking the contact with the Coastal Batholith. The strong deformation obscures crosscutting relationships and primary textures. Locally, the deformation is so intense that the rocks are mylonitized.

Unlike in the Zafranal Main zone, the Victoria zone is outside of the fault-bounded block. One Ar-Ar date obtained from primary biotite in a diorite from Victoria yielded a Jurassic age, which suggests that the Victoria zone may represent a different block, which juxtaposes it with the Zafranal Main zone along a regional fault. Figure 7-1 shows the simplified geology of the Zafranal Main and Victoria zones.

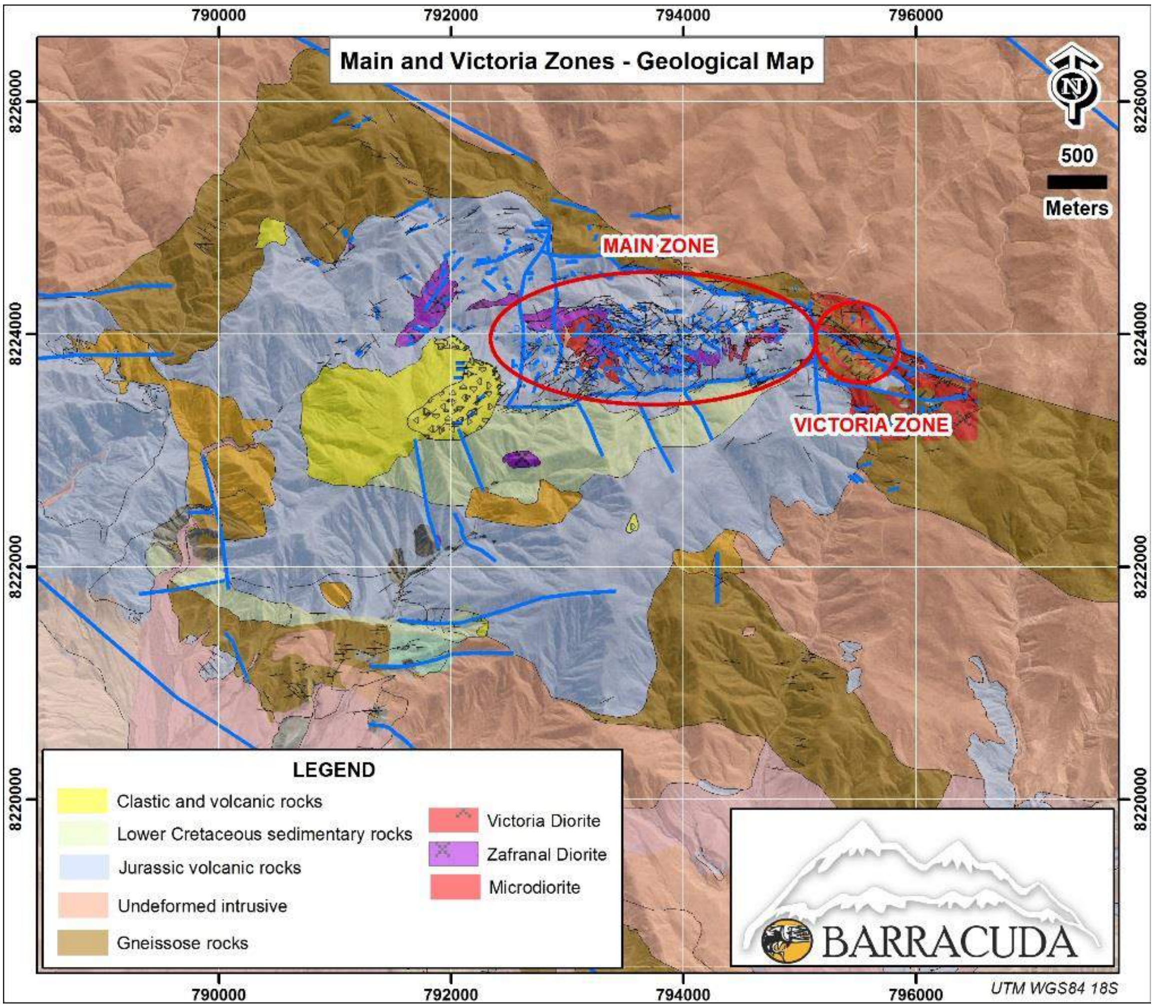

Figure 7-1 – Geological map of the Zafranal Main and Victoria zones

7.1.3.3 Moly Sur zone

Most of the information contained in this section is based on internal AQM geological reports (Callupe, 2015- See reference 27.7.1).

The Moly Sur zone is underlain by andesitic volcanic and a thick sequence of carbonate rocks with fine grained pelitic interbeds, including calcareous mudstone, siltstone and arenite. Based on ammonite fossils, these units are interpreted to belong to the lower Jurassic Chocolate Formation. Conformably above this sequence lie well stratified beds of grey quartz arenite and extensive calcareous arenite. These rock types correlate with the mid Jurassic Socosani Formation found elsewhere in the region. Continental red beds of the Tertiary Moquegua formation cover this Mesozoic sequence.

A series of mafic to intermediate dykes, sills and stocks intrude the sedimentary sequence. Some of these intrusive rocks are strongly deformed along major faults. Extensive outcrops of granodiorite occur in the central portion of Moly Sur, likely belonging to the Coastal Batholith.

The northern side of Moly Sur hosts small outcrops of strongly altered diorite. Drilling on Moly Sur cut several diorite dykes with weak to moderate phyllic alteration and weak pyrrhotite-chalcopyrite mineralization.

7.2 Mineralization, Structures and Alteration

7.2.1 Mineralization

The mineralization descriptions used here are modified from internal AQM reports and from Tetra Tech´s initial PEA report (de Ruijter M. A., et al., 2013- See Reference 27.7.2) and Amec Minproc´s 2011 technical report (Manfrino et al., 2011).

Porphyry copper-gold mineralization occurs within a large roughly east-west trending hydrothermally altered zone that is more than 7 kilometres in length and as much as 1.7 kilometres in width in a north-south direction. Copper occurs within veins and stockworks and as disseminations in both the Zafranal Main zone and Victoria deposits. At the Zafranal Main zone deposit, primary (hypogene) mineralization is overlain by zones of secondary enrichment mineralization. Copper mineralization has been categorized, based on copper solubility, into leached, mixed, oxide, supergene and primary zones.

7.2.1.1 Zafranal Main zone

Drilling on the Zafranal Main zone has demonstrated continuous mineralization over a strike length of at least 2,500 m, widths of between 500 to 800 m, and over a vertical range of up to 1,000 m. The types of copper mineralization are described as follows:

Leached cap and secondary enrichment

The large altered and mineralized area at Zafranal has a subdued colour anomaly. The deposit has been intensely leached with a well-developed leached cap over the top of the supergene-enriched horizon at depth. Very little hypogene sulfide mineralization is present in outcrop, having been strongly leached and/or altered. At surface all forms of sulfides are absent except for occasional pyrite along with chalcopyrite in some more silicified rocks where acid ground waters could not invade the rock and oxidize and leach it.

The best supergene enrichment zone is generally associated with phyllic alteration consisting of a sericite + quartz + chlorite/biotite + clays + pyrite assemblage. The thickness of this zone can reach up to 150 m – averaging 75 m throughout the deposit – with grades up to 7% Cu. Although no age information exists on Zafranal's supergene enrichment, the agreed age of this style of mineralization corresponds to the Upper Eocene to Lower Miocene, similar to other enrichments zones within porphyries in the Paleocene Belt (Quang, Clark, Lee, & Guillen, 2003 - See Reference 27.7.5). Extensional reactivation of the main strike-slip faults (azimuth 130°-140º) is responsible for block faulting of the secondary enrichment layer towards the south.

Copper oxides of chrysocolla, neotocite, malachite and azurite generally occur above the main supergene enriched blanket and below the leached cap. This type of oxide copper mineralization is associated with an intense phyllic alteration (sericite + quartz + clays). Oxide mineralization can be up to 50 m thick. The leached cap itself is 30 m to 200 m thick. The copper-oxide zone defines former supergene enrichment zones (paleoblanket) preserved within the oxidized leach zone. Copper oxides, particularly chrysocolla, chalcantite, neotocite, tenorite and psilomelane, replace chalcocite. Occasionally, copper oxides occur above and below preserved, laterally discontinuous chalcocite blankets that are within the leached capping. Locally, copper oxides and sooty chalcocite of the supergene-enriched blanket is at least partially exposed at surface along the walls of the more deeply incised valleys and gullies where the supergene blanket may have been close to being exposed by rapid erosion. Wherever the enriched blanket is nearest the surface, it has been extensively weakened by surface oxidation and leaching, with splashy copper oxide coating fractures.

Elevated copper values are reported from drilling in the Leached zone, even with no visible copper oxides. Mineralogical studies have confirmed that most of this copper content is included in cupriferous goethite and chlorite.

Some of the most spectacular amounts of copper oxides of chrysocolla, malachite and azurite seen on the property occur within post mineralization dykes that contain enough carbonate or calcite to have precipitated copper in the form of the previously described minerals. Younger, mafic dykes, often contain native copper in the enrichment zone, likely precipitated in that form through lack of sulfur content in the wall rock during fluid circulation.

The enriched copper sulfide blanket occurring below the leached cap is up to 180 m thick and elongated in an east west direction. It is lens-shaped in N-S cross-section and mimics topography. Within the enriched blanket, chalcocite has mainly replaced chalcopyrite, with local coatings on pyrite. The supergene-enriched blanket has progressively been remobilized to deeper levels, as well as laterally, due to erosion. Locally abundant "live limonite", principally hematite, in the leached cap is evidence that the blanket has undergone successive leaching and re-deposition cycles.

Primary sulfide mineralization

Hypogene mineralization occurs as veins, stockworks, and disseminations in the Zafranal diorite and microdiorite units. The Zafranal diorite at surface contains consistent background copper amounts (typically between 1,000 ppm and 1,500 ppm Cu) and typically presents the highest surface geochemical anomalies on the property. At deeper levels higher-grade hypogene mineralization is closely associated to the intrusive contact with the cross cutting microdiorite. The overlying enriched blanket is much smaller than the underlying hypogene mineralization, though with much higher grades.

Primary sulfide mineralization at the Main zone is hosted mainly by the microdiorite and the Zafranal diorite. This primary mineralization is related to a potassic alteration zone with a quartz-secondary biotite-chlorite assemblage. Chalcopyrite is by far the most abundant copper species here and occurs both as disseminations and in veinlets. A-type and B-type veins (quartz-chalcopyrite-pyrite and quartz-chalcopyrite-molybdenite respectively) are abundant at Zafranal and average copper grades range from 0.15% to 0.5% Cu, locally increasing up to 1% Cu.

7.2.1.2 Victoria zone

Mineralization at Victoria is predominantly hypogene and occurs as a tabular body parallel to the main foliation, suggesting that it predates or is contemporaneous with the principal deformation event. Chalcopyrite occurs both as fine-grained disseminations and as patches associated with veins and stockworks. Mineralization within the Victoria zone measures approximately 900 m along strike and up to 500 m width and drilling encountered it down to depths of up to 400 m below surface.

7.2.1.3 Moly Sur zone

No significant porphyry mineralization has been found on Moly Sur to date. Limited drilling in 2015 has identified altered diorite dykes with weak pyrrhotite and chalcopyrite mineralization occurring as disseminations and stockworks in the volcanic and sedimentary wall rock. The lateral extent of this style of mineralization remains unknown.

7.2.2 Alteration

7.2.2.1 Zafranal Main and Victoria zone

Alteration on the Main zone deposit is typical of many Andean porphyry deposits. The main alteration style is phyllic, with pervasive quartz-sericite alteration affecting most lithologies with the exception of the later barren dykes and the gneissose units in Victoria. Here, the texture of the deformed rocks has not been destroyed by alteration and can easily be seen as bands of biotite and or chlorite. Potassic alteration is most commonly seen in the microdiorite intrusive unit and, to a lesser extent, in the Zafranal diorite. No potassic feldspars have been noted to date, but significant hydrothermal biotite occurs throughout the deposit.

A pervasive acid leaching type alteration associated with the leached cap is prevalent over the entire deposit and takes the form of weak argillic alteration. All the rocks that occur within the central part of the deposit are bleached with the plagioclase feldspars altered to a white creamy colour. The plagioclase phenocrysts within the volcanoclastic and diorites exhibit varying degrees replacement by clays.

Along the south side of the deposit is an area of intense argillic alteration. The argillic zone affects all lithologies in the Main zone, including some of the later dykes. The argillically-altered zone is strongly bleached, and is typically whitish in colour with yellowish jarosite and common copper oxides. The diorite intrusive found adjacent to this zone is weakly altered but has the typical chlorite, phlogopite, sericite alteration found elsewhere on Zafranal. Several deep erosion gullies deeply incise the argillic zone, as it is soft and easily eroded.

The general lack of propylitic alteration seen outside of the fault-bounded mineralized block suggests that the faulting occurred post alteration and mineralization. Phyllic alteration within the block appears to terminate abruptly at the fault boundaries across which only weak propylitic alteration occurs. The gneissose rocks to the north of the north-bounding fault are only slightly propylitically altered with epidote and/or chlorite. Similarly the thinner bedded sediments exposed south of the south-bounding fault contain only minor amounts of epidote and chlorite. Peripheral gold and gold-copper veins surround the copper-gold deposit.

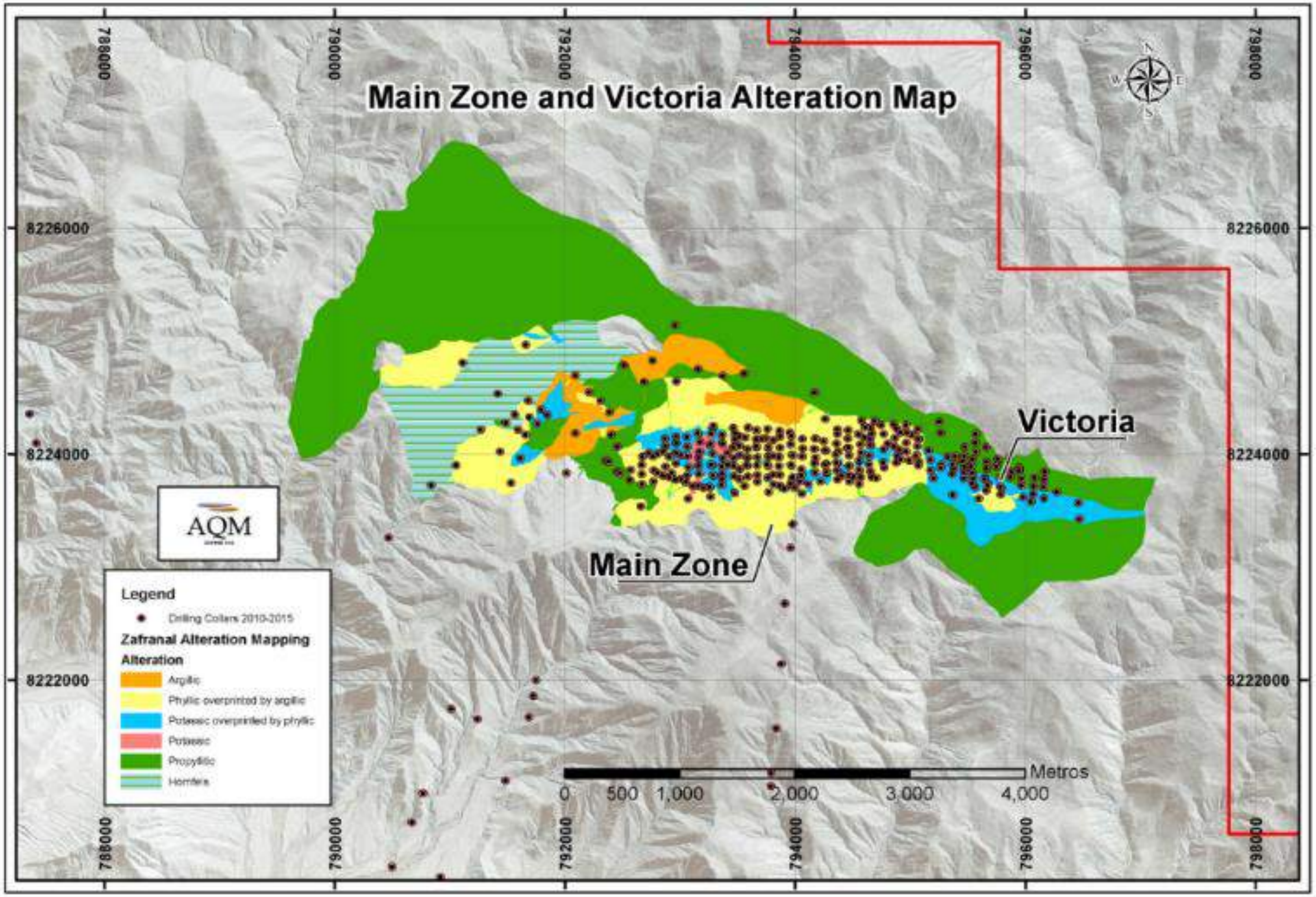

Figure 7-2 – **Main zone and Victoria alteration map**

7.2.2.2 Moly Sur zone

Alteration on Moly Sur is generally weak, with hardly any of the strong phyllic surface alteration that characterizes the Main zone. Strong bleaching, silicification, and some limonite-goethite veining are seen on the sedimentary sequence. Minor zones of pervasive oxidation are mostly limited to calcareous arenite horizons. Leaching, with relict sulfide clasts, occurs over a 0.45 by 0.2 km zone on the northwestern corner of Moly Sur.

A small 10 by 15 metre outcrop of strongly altered intrusive was found on the northern edge of Moly Sur. Here, copper oxides such as malachite and tenorite accompany strong phyllic alteration (quartz-sericite) on fractures.

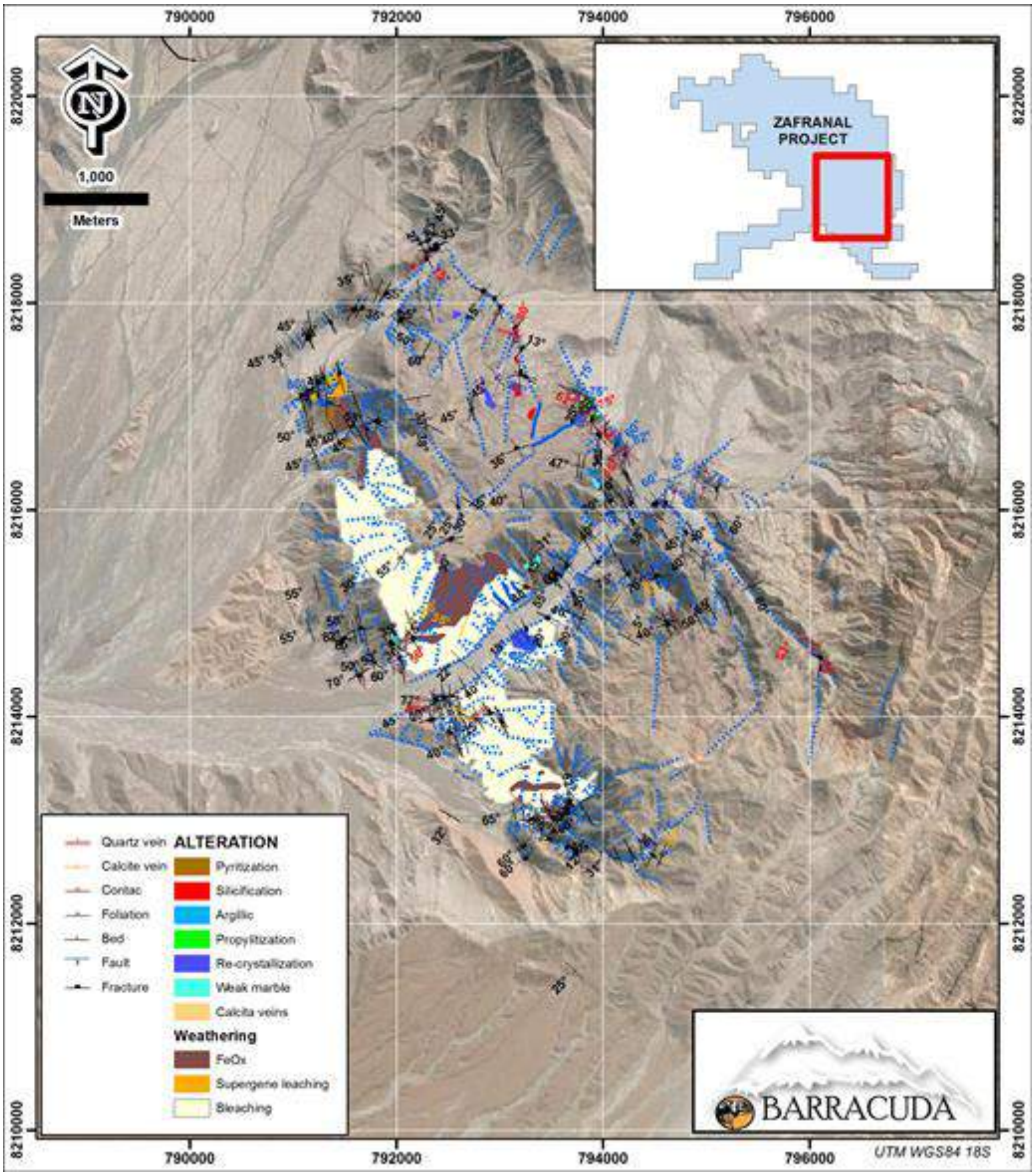

Figure 7-3 – Moly Sur alteration and structural map

7.2.3 Veining

Zafranal mineralization includes multiple stages of veining, all of which are typical of porphyry copper deposits. Observations of the styles of veining identified on Zafranal taken from the 2010 technical report (Manfrino, Harbort & McCrea, 2010- See Reference 27.7.3) are presented in the following subsections.

7.2.3.1 Early veinlets – EDM, A-type and B-type

Early veinlets are those without alteration halo, or with halos composed of rock stable minerals, (quartz, feldspars, biotite or anhydrite). They form in the late magmatic or early hydrothermal (magmatic fluids) stage at relatively high temperatures (+400 °C).

The 'EDM' (early dark micaceous) types are composed of biotite or greenish sericite with or without quartz, andalucite; and are usually the earliest veinlets that carry visible copper sulfides. Some members of this family may be composed only of biotite, of magnetite-quartz, or of amphibole-magnetite-biotite without sulfides.

The A-type are composed of granular quartz, without centreline, with or without K-feldspar halo, with irregular non-matching walls. These veinlets often carry major amounts of chalcopyrite and bornite and are most common in the core of the system. A-type quartz veinlets may amount to 20% or more of the rock volume.

The B-type are usually the major carriers of molybdenum values, and are the first of the brittle fracture domain, have appreciable lateral extent, centreline sutures or banded character and bilateral symmetry, and have either no halo, potassic feldspar, or weak sericite halos.

7.2.3.2 Intermediate veinlets – C-type

Intermediate veinlets often have halos composed of chlorite/sericite or sericite/clay. They form at an early hydrothermal stage and usually carry chalcopyrite, bornite (with or without pyrite, molybdenite); they contribute major amounts of copper to the grade in some deposits. In these early hydrothermal stage veinlets, the sericite in the alteration halo usually has a greenish cast (phengitic) with higher iron and magnesium content.

7.2.3.3 Late veinlets – D, E and F-types

Late veinlets have texturally destructive halos composed of minerals not stable in fresh rock, white sericite or of sericite, carbonate, and clay with gypsum and/or anhydrite. The late veinlets are continuous and through-going and are commonly the result from hydrothermal fluids (150-250 °C) with a large component of meteoric origin.

The earlier D-type carries dominant pyrite (with or without chalcopyrite, molybdenite), with wide halos of greyish sericite sometimes coarse enough to be termed muscovite. When these veinlets have chalcopyrite in the fracture, it is usually subordinate to pyrite, and the halos rarely contain appreciable disseminated chalcopyrite.

The late structurally controlled mineralization (E-type), typical of the Central Andean porphyry copper deposits, is characterized by carbonate (ankerite-dolomite, or rhodochrosite), gypsum-anhydrite-barite, with pyrite, chalcopyrite, tennantite-tetrahedrite, and sphalerite-galena. These veins often have thicknesses from 5 cm to as much as 1 m. The alteration halos consist of sericite/clay and are typically 5 to 10 times vein width. This veining stage is often responsible for almost all arsenic/antimony present as contaminants in porphyry copper deposits.

There are also very late (dying thermal stage) veinlets (F-type), with gypsum, carbonate, chlorite or epidote, some of which have alteration halos of clays (sericite), the clays include both non-swelling types (mapped as kaolinite), and swelling clays (smectite).

7.2.4 Structure

7.2.4.1 Zafranal Main and Victoria zone

Copper-gold porphyry deposits such as Zafranal are open systems formed by the introduction of magmas and hydrothermal solutions into the rock along geological structures caused by the strain applied by regional and localized magmatic forces. The importance of regional-scale strain in the Zafranal deposit is clear, as the observed faults associated to it control the location of the intrusive bodies. The relative importance of structural controls and location of magmatism within the deposit has evolved over time and space as evidenced by the pattern of the different intrusions. Furthermore, structural controls have been very significant during the supergene enrichment process, as faults, joints, foliation and bedding planes have provided permeability for surface waters to percolate to relatively deep levels. Fault zones are frequently more permeable than surrounding rock thus enabling thick supergene enrichment areas to develop and deepen beyond the water table horizon.

The main structural controls defined are chronologically sequenced from oldest to youngest as follows – excerpts taken from (Rivera, Cano, and Huaman, 2010- See reference 27.7.6):

- Northern and Southern Transgressive Faults (Iquipi-Clavelinas System or Incapuquio Fault System): Zafranal is enclosed to the north and south by two east-west faults formed by Andean tectonics. The location of the deposit's intrusive bodies, alteration and mineralization zones fits within these faults which have allowed for the juxtaposition of blocks of Jurassic volcanoclastic and sedimentary rocks with younger diorite intrusions.
- Foliation, E-W Faults, Folds, Lineation: Structural observations made at the outcrop level show the existence of a foliation event affecting the entire deposit. This deformation has affected Zafranal both before and after the intrusive events and hypogene copper alteration-mineralization processes, as evidenced by the foliation of sericite halos of D-type veins. Both on surface and at depth, porphyry intrusions also show crystal lineation, particularly in hornblendes, thus indicating that the shearing event also affected them during their emplacement.
- N-E Fault System (Az. 050°-060°): This fault system occurs on surface as a series of continuous structures mainly along the eastern and western edges of the system. The relative absence of this type of faults in the central part is a result of the prevalence of later dextral strike-slip structures (Az. 130°-140°) that have sliced the system progressively to the northeast.
- Normal Strike-Slip Fault System (Az. 130°-140°): These structures are found in the central part of the deposit and appear as very long continuous structures with well-developed fault breccias up to 2 m wide. These structures run parallel to the Cincha-Lluta (Incapuquio) fault system and are the result of strong regional shearing deformation. Subsequent to the various intrusions, these dextral strike-slip structures show displacements of up to 400 m. Following the secondary enrichment formation, these structures were tectonically reactivated (during the Andean orogeny) as normal faults, progressively down dropping the enrichment blanket to the south. The latest signs of motion (normal faults) can be identified in outcrops, as well as diamond-drill cores, and is much better preserved than earlier strike-slip indicators.

Alteration obscures bedding within the central part of the deposit; however, the overall form of a syncline is interpreted by the outcrop pattern of the Jurassic sedimentary and volcanoclastic rocks. Faulting within the deposit is prolific and the diorite intrusive phases and younger dykes appear to have been intruded along some of these faults. Many of the intrusions, including diorite, quartz diorite or microdiorite bodies, have fault-bounded contacts as well as clearly intrusive contacts with chilled margins.

7.2.4.2 Moly Sur zone

The Moly Sur blocks represent one of the few outcrops of Pre-Cenozoic rocks west of the Incapuquio fault system. Although its structural history remains poorly understood, conjugate movement associated to the principal fault system is interpreted to have caused uplift and tilting, thus exposing the Moly Sur block.

Mapping has identified three main sets of faults at Moly Sur. Early north-south normal faults are cut by later NE-SW and NW-SE trending sinistral and dextral extensional faults respectively.

8 Deposit Types

8.1 Description(s)

Zafranal represents a classic example of an Andean style porphyry copper, gold, and molybdenum deposit. In summary, and using a general description from Panteleev (1995-Reference 27.8.1) such types of deposits display large zones of hydrothermally altered rock with quartz veins and stockworks, sulfide-bearing veinlets, fractures and lesser disseminations in areas often greater than 10 square kilometres in size. These alteration zones are often coincident with hydrothermal or intrusion breccias and dyke swarms.

Potentially economic grades are often controlled by emplacement of fertile intrusions at or near structural zones and/or intersections. The uppermost sections of these intrusions, where strong fracturing has developed because of depressurization and hydrothermal brecciation, as well as at or near the contacts with other rock types, often coincide with the best grades. Andean porphyry systems commonly display multiple intrusion phases and therefore multiple generations of veinlets and stockworks. Mineralizing intrusions range in composition from monzonitic to dioritic and were emplaced during the Andean orogeny, which began during the Cretaceous and continues to this day.

Oxidation of primary sulfides generated in porphyry systems results in circulation of acidic waters above mineralized systems. This later event has a twofold effect on porphyry deposits: it leaches rocks of all or most of the sulfides they contained above the water table; and copper rich solutions re-deposit as enriched copper sulfides at or below the water table. Common sulfides found here are chalcocite, covellite and digenite. Occasionally, native copper will deposit on rocks with insignificant amounts of sulfur, such as young barren dykes. These enrichment zones (or “blankets”) tend to behave as flat zones often parallel to topography. Above the secondary enrichment zone, altered rock often shows no geochemical signature due to intense leaching of all copper-bearing primary sulfides. Thus, typical Andean porphyries have a leached upper zone, an enriched supergene blanket, and a much larger mineralized, albeit at lower grades, primary (or hypogene) zone at depth.

Fluctuating water tables often result in subsequent oxidation of enrichment blankets. Common copper oxide minerals found in these zones are malachite, chrysocolla and brochantite. Occasionally, these copper oxides re-deposit some distance away from the main orebodies to form “exotic” copper deposits.

Alteration in porphyry deposits, from the innermost zones outwards, are: sodic-calcic, potassic, phyllic, and argillic to propylitic. Propylitic or phyllic alteration may overprint early potassic assemblage, and argillic alteration may overprint all other styles.

Other deposit styles associated with porphyry copper deposits (spatially and genetically) include epithermal quartz veins and disseminated precious metal deposits, lead-zinc-silver veins and replacements, and skarns. A schematic model for porphyry deposits with respect to other styles of mineralization is shown in Figure 8-1.

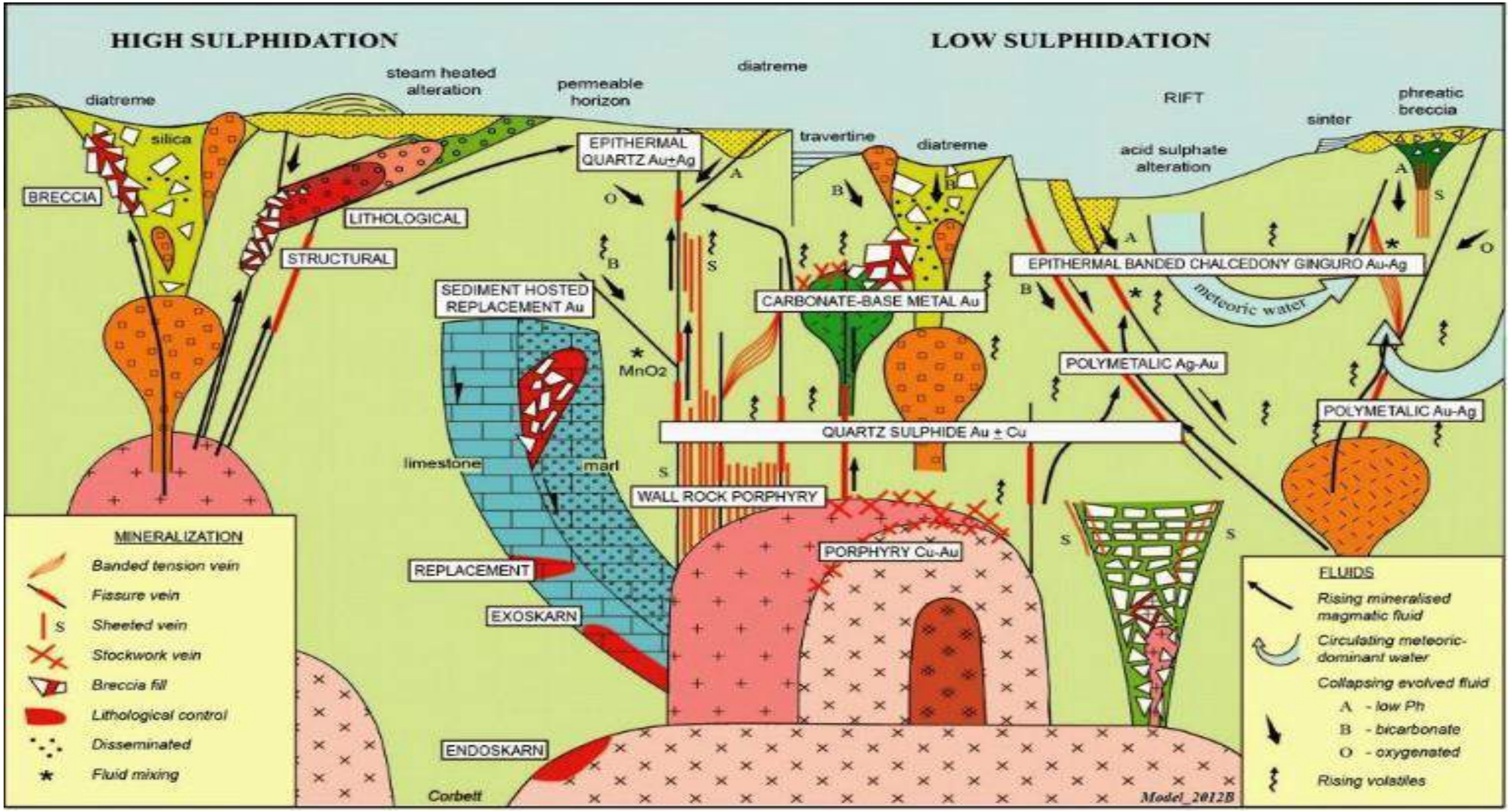

Figure 8-1 – Schematic model for porphyry and associated deposits after (Corbett, 2013*- See Reference 27.8.2)

8.2 Geological Models/Concepts and Basis for Exploration Planning

Exploration at Zafranal has focused on outlining a viable copper-gold porphyry deposit similar to those found elsewhere on the prolific Southern Peru porphyry belt. Initial drilling aimed to determine the existence of a supergene blanket at the Zafranal Main zone. Subsequent drill programs tested the extents of the supergene and the underlying hypogene mineralization. District scale exploration has followed up on structural controls, along which other mineralized intrusive bodies may have been emplaced, as well as on surface alteration, taking into account the typical alteration zonation observed on typical porphyry systems (shown in Figure 8-2).

As with many other mineralized porphyry systems around the world, the geometry of the mineralized shell at Zafranal is determined, at least in part, by the spatial extent of the mineralizing intrusions. All drilling at Zafranal was planned in order to construct a geological model that adequately explained the distribution of the grade and the shape of the mineralized shells, taking into account the vertical zonation inherent to porphyry deposits, the alteration zonation and the nature of the host rocks.

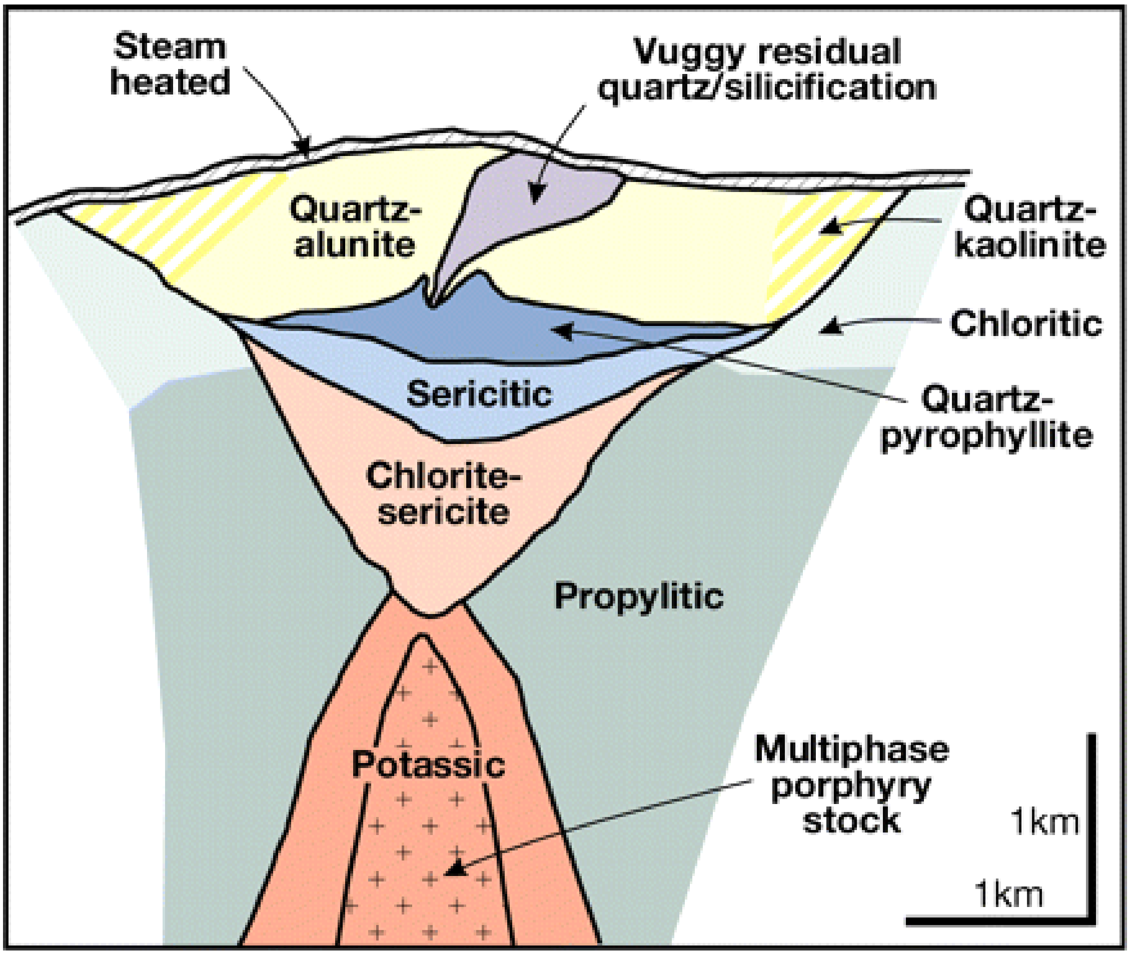

Figure 8-2 – Schematic alteration model for porphyry systems after (Sillitoe, 2010 - See Reference 27.8.3)

9 Exploration (excluding drilling)

9.1 Procedures and Parameters

Soon after acquiring the property in 2009, AQM through CMZ began a systematic geochemical sampling and mapping program over the Main zone. This work confirmed and extended the anomalous zones identified by Teck´s previous work. The geochemical program was based on a 100 x 100 metre grid. Additionally, AQM carried out 5.7 kilometres of magnetotelluric survey over the Main zone, confirming the existence of a 1,000 x 500 metre conductor (or negative resistivity anomaly) roughly coincident with the supergene enrichment blanket over the Main zone. No exploration work other than drilling was carried out over the Victoria zone.

Moly Sur was discovered in 2012 following a regional reconnaissance program and the discovery of oxidation and bleaching of the sedimentary host rocks. Follow-up mapping and further rock sampling extended the anomalous area to cover approximately 4 km by 1.5 km. Test IP lines were carried out over the southern side of Moly Sur, where the strongest molybdenum anomalies occurred. However, the lack of current penetration in the arid terrain hindered the program, which was abandoned before completion. Scout drilling was planned and executed during 2015, aimed at identifying (or ruling out) a buried porphyry system outside of the existing resources areas which may affect any future mine planning.

CMZ has collected 64 rock chip samples from Moly Sur and completed 8 line km of IP geophysical surveys. CMZ completed a ground magnetics survey in late 2015 over the northern part of Moly Sur, which the IP survey did not cover.

Descriptions of past exploration by CMZ on the Sicera Sur, Sicera Norte, Campanero and Ganchos zones are available in the 2010 technical report filed by the Company (Manfrino, Harbort, and McCrea, 2010 See Reference 27.9.1).

9.2 Rock Sampling Methods and Quality

Rock samples were collected using rock hammers to chip away pieces of outcrop every 10-20 cm to cover sample lengths of between 2 and 5 metres. Samples are generally representative of the outcrops from which they were collected and there was no systematic sampling of the zone. Trained technicians, under the supervision of CMZ geologists, carried out the geochemical sampling programs. Samples were located by standard GPS and were collected in double-marked bags and shipped to the preparation laboratory in Arequipa. Pulps were sent to ALS Chemex´s main facility in Lima for gold and multi-element ICP analysis.

9.3 Significant Results and Interpretation

The geochemical anomaly over the Main zone extends beyond the zone of strong phyllic alteration. Geological mapping of the Main zone outlined an east-west trending zone of strong phyllic-alteration over a 4500 m x 500-1500 m area. This altered zone is hosted by a strongly foliated, fine-grained volcano-sedimentary unit of Jurassic age cut by several generations of porphyritic diorite stocks and dykes of dacitic composition of Cretaceous age with weak to moderate phyllic alteration and moderate to strong biotite alteration.

Rock sampling on Moly Sur identified a large, anomalous molybdenum zone 4 km by 1.5 km.

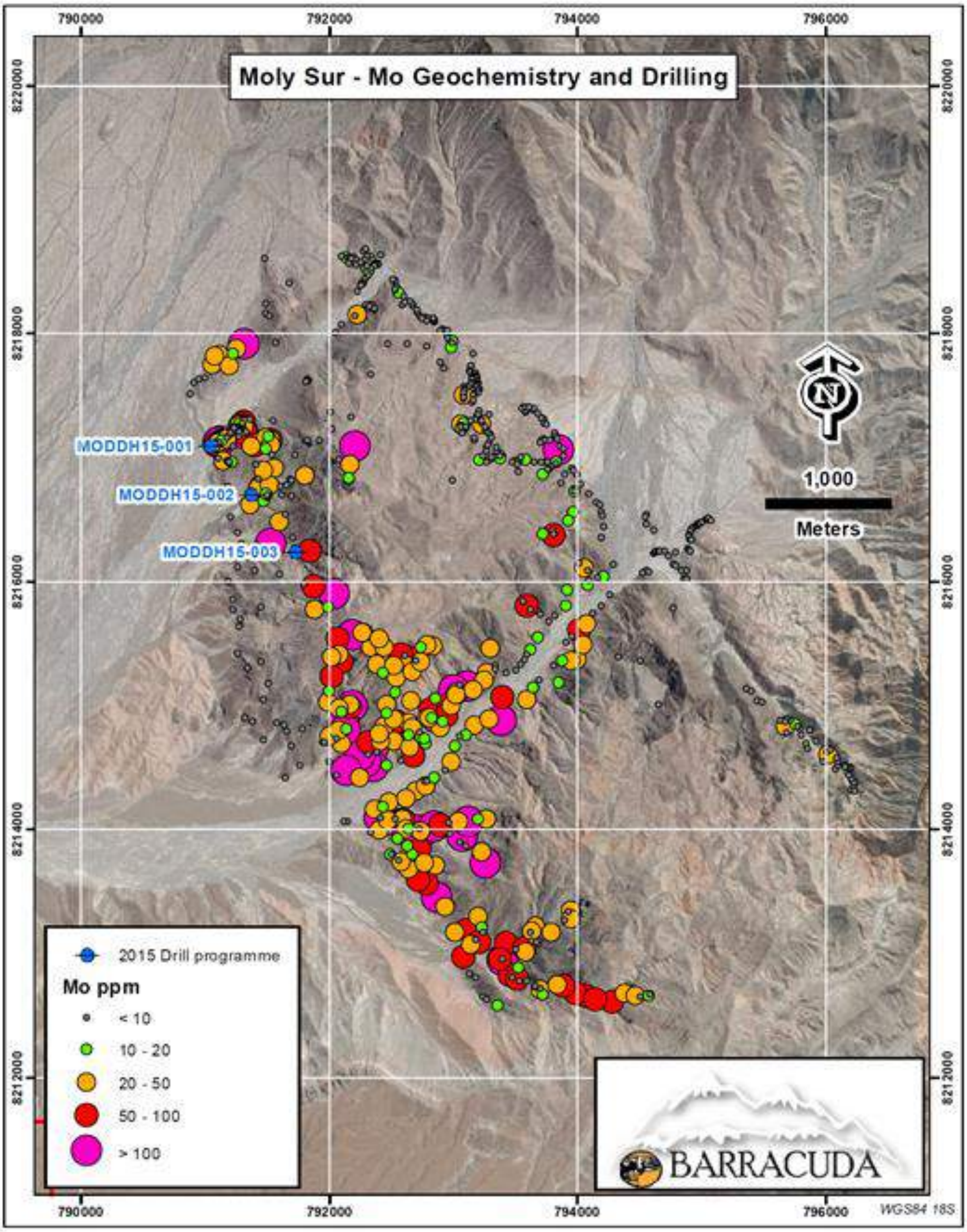

Figure 9-1 – Molybdenum geochemical map and drill location for Moly Sur 2015 program

The 2015 Moly Sur drilling program found no significant base metals results. However, the discovery of porphyry style veining and a pyrrhotite-chalcopyrite-molybdenite assemblage suggests that the alteration observed here may be related to a distal mineralized hydrothermal system.

10 Drilling

10.1 Introduction

A total of 383 exploration drill holes have been completed on the Project (Zafranal Main and Victoria zones), accumulating 122,659.65 m of drilling from 2004 to 2015, by different companies.

Table 10-1 summarizes the details of all drilling campaigns performed.

Table 10-1 – Summary of the drilling campaigns carried out for the Zafranal project.

Company	Year	Purpose	Type	Nº of drill holes	Name of drill holes	Metres drilled
Teck	2004	Exploration	DDH	4	ZFDDH04-001 to ZFDDH04-004	1,556.15
			RC	22	ZFRC04-001 to ZFRC04-022	7,001.00
Teck	2005	Exploration	RC	2	ZFRC05-023 to ZFRC05-024	626.00
			RC	8	ZFRC05-025 to ZFRC05-032	2,622.00
Teck	2009	Exploration	DDH	3	ZFDDH09-005 to ZFDDH09-007	845.5
			RC	3	ZFRC09-033 to ZFRC09-035	636.00
AQM	2010	Delineation	DDH	147	ZFDDH10-008 to ZFDDH10-154	50,759.9
			RC	48	ZFRC10-036 to ZFRC10-084	15,906.00
			RC			
AQM	2011	Delineation	DDH	85	ZFDDH11-155 to ZFDDH11-238	30,937.55
			RC	1	ZFRC11-085A	250.00
AQM	2012	Delineation	DDH	17	ZFDDH12-239 to ZFDDH12-255	5,985.55
CMZ	2014	Delineation	DDH	39	ZFDDH14-256 to ZFDDH14-294	5,534.00
			DDH	295		95,618.65
			RC	88		27,041.00
			TOTAL	**383**		**122,659.65**

10.2 Diamond Core Drilling

The 2014 drilling program was carried out from 17 September 2014 to 7 December 2014. This involved a program of 5,534.00 m of in-fill drilling with 39 inclined HQ diamond drill holes with varying lengths of up to 200 m.

The drilling program was completed by AK Drilling International S.A. (AK Drilling) under the supervision of CMZ's exploration team.

The additional metres of in-fill of diamond drilling for the Zafranal Main zone were intended to better define the limits of the mineralization and increase the measured and indicated component of the Zafranal Main zone mineral resource.

The 2014 drill program was realized in the Zafranal Main zone and is summarized in Table 10-2.

Table 10-2 – Summary of the 2014 drill program in the Zafranal Main zone.

Ref. #	Hole_ID	Easting	Northing	Elevation(m)	Azimuth (º)	Dip (º)	Length (m)	Drill hole type	Date started	Date finished
1	ZFDDH14-256	792,806.00	8,223,999.00	2,590.00	360	-70	160.00	diamond	17-Sep-14	19-Sep-14
2	ZFDDH14-257	792,806.00	8,223,999.00	2,590.00	180	-75	190.00	diamond	20-Sep-14	22-Sep-14
3	ZFDDH14-258	792,805.00	8,223,830.00	2,489.59	360	-65	70.00	diamond	23-Sep-14	24-Sep-14
4	ZFDDH14-259	793,110.00	8,223,717.00	2,610.79	180	-90	150.00	diamond	25-Sep-14	27-Sep-14
5	ZFDDH14-260	793,415.00	8,224,006.00	2,640.56	180	-66	134.00	diamond	28-Sep-14	29-Sep-14
6	ZFDDH14-261	793,515.00	8,223,778.00	2,622.03	180	-61	121.00	diamond	30-Sep-14	02-Oct-14
7	ZFDDH14-262	793,810.00	8,223,980.00	2,759.55	180	-69	124.00	diamond	03-Oct-14	06-Oct-14
8	ZFDDH14-263	793,710.00	8,224,130.00	2,775.24	180	-82	168.00	diamond	06-Oct-14	08-Oct-14
9	ZFDDH14-264	793,810.00	8,223,819.00	2,700.24	180	-89	65.00	diamond	09-Oct-14	10-Oct-14
10	ZFDDH14-265	794,465.00	8,224,047.00	2,810.75	264	-75	191.00	diamond	12-Oct-14	14-Oct-14
11	ZFDDH14-266	794,108.00	8,223,731.00	2,601.00	180	-75	120.00	diamond	15-Oct-14	17-Oct-14
12	ZFDDH14-267	794,505.00	8,223,798.00	2,672.00	180	-70	130.00	diamond	18-Oct-14	19-Oct-14
13	ZFDDH14-268	794,665.00	8,224,177.00	2,708.79	180	-75	83.00	diamond	20-Oct-14	21-Oct-14
14	ZFDDH14-269	794,910.00	8,223,955.00	2,740.15	180	-75	109.00	diamond	21-Oct-14	23-Oct-14
15	ZFDDH14-270	794,900.00	8,224,030.00	2,730.00	360	-70	100.00	diamond	23-Oct-14	24-Oct-14
16	ZFDDH14-271	794,910.00	8,224,031.00	2,734.56	180	-78	94.00	diamond	25-Oct-14	26-Oct-14
17	ZFDDH14-272	794,910.00	8,224,197.00	2,763.20	180	-75	133.00	diamond	27-Oct-14	29-Oct-14
18	ZFDDH14-273	794,665.00	8,223,780.00	2,660.00	180	-75	130.00	diamond	30-Oct-14	31-Oct-14
19	ZFDDH14-274	794,565.00	8,223,792.00	2,707.33	180	-80	124.00	diamond	01-Nov-14	02-Nov-14
20	ZFDDH14-275	794,465.00	8,223,753.00	2,648.63	180	-74	173.00	diamond	03-Nov-14	05-Nov-14
21	ZFDDH14-276	794,365.00	8,223,766.00	2,656.41	180	-60	192.00	diamond	05-Nov-14	08-Nov-14
22	ZFDDH14-277	793,810.00	8,224,119.00	2,785.00	360	-88	165.00	diamond	10-Nov-14	13-Nov-14
23	ZFDDH14-278	794,220.00	8,223,748.00	2,645.37	180	-63	190.00	diamond	09-Nov-14	16-Nov-14
24	ZFDDH14-279	793,515.00	8,224,157.00	2,743.26	180	-78	141.00	diamond	13-Nov-14	15-Nov-14
25	ZFDDH14-280	793,515.00	8,224,070.00	2,710.63	180	-73	172.00	diamond	16-Nov-14	18-Nov-14
26	ZFDDH14-281	794,000.00	8,223,693.00	2,641.20	180	-73	130.00	diamond	16-Nov-14	18-Nov-14
27	ZFDDH14-282	793,515.00	8,223,988.00	2,693.23	180	-80	152.00	diamond	19-Nov-14	21-Nov-14
28	ZFDDH14-283	793,810.00	8,223,725.00	2,656.17	360	-78	187.00	diamond	19-Nov-14	22-Nov-14
29	ZFDDH14-284	793,515.00	8,223,858.00	2,665.19	180	-67	155.00	diamond	22-Nov-14	24-Nov-14
30	ZFDDH14-285	793,905.00	8,223,690.00	2,654.00	180	-75	150.00	diamond	23-Nov-14	26-Nov-14
31	ZFDDH14-286	793,205.00	8,223,697.00	2,595.30	180	-76	144.00	diamond	25-Nov-14	26-Nov-14
32	ZFDDH14-287	794,405.00	8,223,808.00	2,674.00	180	-55	155.00	diamond	26-Nov-14	28-Nov-14
33	ZFDDH14-288	793,005.00	8,223,780.00	2,582.00	180	-75	150.00	diamond	27-Nov-14	29-Nov-14
34	ZFDDH14-289	794,705.00	8,223,780.00	2,634.00	180	-75	95.00	diamond	29-Nov-14	29-Nov-14
35	ZFDDH14-290	792,905.00	8,223,795.00	2,528.00	180	-85	132.00	diamond	30-Nov-14	01-Dec-14
36	ZFDDH14-291	792,905.00	8,223,970.00	2,575.00	180	-65	185.00	diamond	01-Dec-14	04-Dec-14
37	ZFDDH14-292	793,005.00	8,223,987.00	2,559.00	180	-65	140.00	diamond	02-Dec-14	04-Dec-14
38	ZFDDH14-293	793,415.00	8,223,900.00	2,654.00	180	-65	150.00	diamond	04-Dec-14	06-Dec-14
39	ZFDDH14-294	792,905.00	8,223,970.00	2,575.00	360	-65	180.00	diamond	04-Dec-14	07-Dec-14
					Total		**5,534.00**			

10.3 Drilling Procedures

Diamond drilling in the Zafranal Main zone was conducted using two Sandvik DE7104 all-hydraulic, track-mounted rigs, supplied and operated by AK Drilling. The first diamond rig was mobilized on 14 September 2014, with the additional rig being added later in 2014. All drilling equipment at Zafranal was capable of drilling HQ-sized core (63.50 mm diameter) to depths of 700 m.

A total of 39 diamond drill holes were completed on the Zafranal Main zone in 2014. Table 10-3 presents the descriptive statistics for the 2014 assay data set, which includes the principal assays, using a 0.15% Cu cut-off.

Table 10-3 – Descriptive statistics 2014 drill program (0.15% Cu cut-off) in the Zafranal Main zone.

	Length (m)	Cu %	Au g/t	Ag g/t	Mo ppm	Pb ppm	Zn ppm	As ppm	Bi ppm	Fe %	S %
Mean	1.98	0.49	0.10	0.48	22.90	11.09	27.54	3.52	1.47	3.77	2.37
Standard Error	0.01	0.01	0.01	0.02	1.17	4.88	1.03	0.40	0.05	0.05	0.07
Median	2.00	0.33	0.05	0.30	12.00	2.00	21.00	2.00	1.00	3.56	2.19
Mode	2.00	0.16	0.00	0.10	1.00	1.00	16.00	1.00	1.00	3.82	0.01
Standard Deviation	0.19	0.39	0.15	0.60	33.95	142.15	30.04	11.73	1.53	1.53	2.12
Sample Variance	0.04	0.15	0.02	0.36	1,152.32	20,206.19	902.59	137.65	2.33	2.33	4.47
Kurtosis	27.57	3.99	104.95	46.66	41.63	772.24	36.52	369.74	127.01	10.80	0.99
Skewness	- 4.37	1.85	8.18	5.46	4.89	27.26	4.65	17.34	9.67	2.13	1.00
Range	2.50	2.81	2.58	7.00	442.50	4,049.00	394.00	276.00	24.00	15.96	10.00
Minimum	0.50	0.15	0.00	0.10	0.50	1.00	1.00	1.00	1.00	0.89	0.01
Maximum	3.00	2.96	2.58	7.10	443.00	4,050.00	395.00	277.00	25.00	16.85	10.00
Sum	1,675	413	81	404	19,416	9,405	23,358	2,981	1,247	3,195	2,011
Count	848	848	848	848	848	848	848	848	848	848	848

10.4 Drilling, Sampling and Core Recovery Commentary

10.4.1 Drilling methods

The Project has primarily used diamond drill holes (DDH) as the preferred drilling method for the estimation of resources, geometallurgical, geotechnical and hydrogeological studies. Reverse air circulation drilling (RC), was also used with fewer metres for the estimation of resources and hydrogeology.

In all of the diamond drilling campaigns, the drill strings used were HQ (63.50 mm diameter) or NQ (47.60 mm diameter). The drill core was recovered by using the wireline and core tube methods. HQ size core was obtained for metallurgical testing.

10.4.2 Core recovery commentary

For the diamond drill holes, core recovery data were obtained. These recoveries were compared with the length drilled. Average core recovery for diamond drill holes is 98% with 94% of the samples having a recovery greater than 90%. The zones with low recovery are associated with intensely fractured or faulted material.

10.4.3 Geological and geotechnical logging

The Project has developed logging and sampling procedures that have been continuously improved and have been subjected to external audits that have confirmed that the processes implemented are sound and their results have a high level of certainty.

All of the logging was performed on complete drill core using conventional methods (use of geologist's lenses, harness pens, magnets, etc.), contacts, lithologies, alterations, mineralization and structures were identified, and mineralized zones were characterized (Cu oxides, sulfides and secondary sulfides).

In 2014 the Project initiated a re-logging of core from all previous campaigns with the support of an external consultant to unify logging with a written geological logging protocol (Reference 27.10.1 and Reference 27.10.3).

For the geotechnical logging, conventional methods were also used to define the characterization of the rock mass. Traits identified included recovery, frequency, condition and degree of fracturing, RQD (Rock Quality Designation), GSI (Geological Strength Index) and others. A written geotechnical logging protocol was followed (Reference 27.10.2).

10.5 Zafranal Main and Victoria Zones Drill Hole Locations, Orientation and Depth

10.5.1 Grid control

Grid control was established that is approximately 100 m E-W and 80 m N-S on Zafranal. Minesight software was used to plan the drilling program using north-south sections spaced 100 m apart. Individual drill holes were spaced between 50 m and 100 m apart within each section considering all previous programs and the geological model.

10.5.2 Drill hole collars

Drill holes were oriented by Project geologists using Brunton compasses. Drill hole angles were established using inclinometers.

Once a drill hole was completed, a 3 metre long piece of PVC pipe or HQ drill rod was left behind and cemented in for surveying with a total station. The drill hole number, and total depth was marked directly on the cement collar.

The collar location and elevation of all drill holes were surveyed by professional surveyors and locations were referenced to the PSAD56 datum and later converted to WGS84 datum.

10.5.3 Down-hole surveying

During the 2014 in-fill drilling program, the drill hole surveys were measured mainly with Gyro Reflex (non-magnetic tool) to determine deviations of both azimuth and inclination angles. In all cases the surveys were done by Project geologists and Bornav S.A.C. technicians (Reference 27.10.4). The holes were surveyed from the bottom to the top every 5 metres. The surveys were downloaded and validated using Reflex Gyro software and the readings were checked.

The azimuth analysis indicates that 100% of the drill holes have differences between maximum and minimum values of less than 10°. The maximum variation in present value is 6.10º. The dip analysis indicates that 100% of the drill holes have differences between maximum and minimum values, of less than 5°. The maximum variation is 3.60º. These are acceptable values for drill hole deviation.

10.5.4 Surface topography

Topographic maps were prepared by Horizons, a Lima-based airborne photography and mapping company. Flights were completed in late 2009, and 1:2000 and 1:5000 scale maps were generated. Two control points tied in to the Peruvian geodesic network were used to georeference the aerial photos and generate the topographic maps and orthophotos.

All surface mapping and drill planning was done using 1:2000 maps for the Zafranal Main zone. Maps at 1: 5000 were used for outlying areas and satellite targets.

10.6 Moly Sur Zone Drilling Information

10.6.1 Drill hole locations, orientation and depth

CMZ completed 1,502.6 metres of diamond drilling in three scout drill holes on Moly Sur. Drilling aimed to test the existence of porphyry style mineralization under the strongest altered intrusive rocks. Table 10-4 shows a summary of the drilling completed on Moly Sur.

Table 10-4 – Moly Sur 2015 drilling campaign

Drill Hole	PSAD56 East	PSAD56 North	Azimuth	Dip	Depth (m)
MODDH15-001	791045	8217094	085	-65	400.0
MODDH15-002	791375	8216702	025	-65	700.6
MODDH15-003	791730	8216241	070	-75	402.0

10.6.2 Relationship of orientation and sample length with true thickness

Drilling at Moly Sur targeted potential hypogene copper mineralization associated with porphyry intrusions that only outcrop in one location. Although hypogene mineralization is controlled by lithology and by structural breaks, the lack of additional surface expressions meant that there was insufficient information to determine the preferred drilling orientation to intersect mineralized porphyries. The outcropping intrusive rocks appear sub-vertical, and the Moly Sur drilling would have cut these bodies obliquely, with shallower angles giving a better true thickness estimate than steeper ones. Samples were typically 2 metres long unless there were lithological or alteration breaks that required different sample lengths.

10.6.3 Anomalous grade intervals

No anomalous grade intervals were intersected by the 2015 drilling program at Moly Sur.

11 Sample Preparation, Analyses and Security

11.1 Site Sample Preparation Methods and Quality Control for RC Chips

The 2014 drilling program did not include any reverse circulation (RC) drilling. The previous programs that did, followed the below general methodologies:

- RC chips were collected during drilling in 1m intervals, weighed, then put though a sample splitter and a one-third split (approximately 7 kg on average) was placed into a laboratory mesh sack and the remainder of the sample was placed into a rice bag.
- Samples were then transported to camp where blanks of granodiorite (five per 100 samples), duplicates (five per 100 samples) and six standards per 100 samples (two each of three different standards), were added to the sample stream at random intervals.
- Samples were shipped to the independent ALS Peru S.A. (ALS) sample preparation laboratory in Arequipa for crushing and splitting. Shipping took place in entire-hole lots that were transported from the drill site to Arequipa by a CMZ representative.
- Following preparation in Arequipa, pulps were sent by ALS to the independent ALS Chemex analytical laboratory in Lima for analyses.

11.2 Site Sample Preparation Methods and Quality Control for Drill Core

Until 2013, the sampling programs followed the following general methodologies:

- All cores were routinely photographed before logging took place.
- The core was logged by qualified geologists for alteration, mineralization, lithologies, and structures, and assessed by geotechnical technicians for rock quality designation (RQD).
- Once a log was completed, it was digitally entered into the database by CMZ personnel.
- Core mark-up for sampling was undertaken in the sample preparation laboratory.
- Cores were cut in half using gas-powered core saws. Samples were taken on 2 m or smaller intervals, depending on changes in structure, lithology, mineralogy or alteration. The resulting sample was placed in plastic sample bags, weighing approximately 5 to 7 kg per 2 m sample.
- Standards, blanks and duplicates were inserted after the core was cut at the logging facility. Duplicates were cut into quarter core samples (¼ duplicate sample, ¼ original sample, ½ core sample kept in core shack).
- All sample bags were placed in rice bags that accommodated about 35 kg of samples.
- Samples were transported to ALS in Arequipa, for sample preparation, and then submitted to ALS in Lima for analysis.
- The remaining core was returned to the core box and was stored in core warehouses located at the site camp at Zafranal; subsequently a warehouse in Arequipa was used for storage.

Sample preparation, analyses and security protocols have not changed since the effective date of the 2013 resource estimate (Reference 27.11.2, 27.11.3 and 27.11.4), except that geotechnical data collected included RQD, geological strength index (GSI) and point load testing (PLT) for subsequent use in geotechnical studies. In addition, in-situ bulk density determination was carried out every 2 m using the water displacement method.

The 2014 CMZ sampling programs used the following general methodologies:

- All cores were routinely photographed before logging took place.
- Logging by qualified geologists recorded observations and measurements including lithology, alteration, mineralogy, sulfide/oxide mineralogy, sulfide percentages, structural features, veining, and iron oxide characteristics.
- Geotechnical data collected by qualified technicians included RQD, GSI, fracture frequency and PLT in 2 m core intervals.
- Density determination using the water displacement method. Samples were typically measured unwaxed every 2 m; however, waxed determinations were made on selected samples.
- All data collected during logging were loaded online using laptops with GVMapper v.3.6 software (data entry).
- Core was marked up prior to being cut in the sample preparation laboratory.
- Drill core was split using a hydraulic guillotine. In some samples, the presence of fine-grained sooty chalcocite and copper oxides prevented the use of water for core sawing as some mineralization could be washed out and; therefore, lead to sampling bias. Sample preparation was undertaken when the geological and geotechnical mapping was completed.
- Standards, blanks and duplicates were inserted after the core was cut at the logging facility. Duplicates were cut into quarter core samples; (¼ duplicate sample, ¼ original sample, ½ core sample kept in core shack).
- All samples were placed in rice bags that accommodated about 35 kg of sample.
- Samples were transported to the ALS laboratory in Arequipa for processing and subsequently submitted to ALS Lima for analysis.
- The remaining core was stored in core warehouses located at the CMZ camp at Zafranal; final storage is in the CMZ Arequipa warehouse.

11.3 In-situ Bulk Density

A total of 753 samples were collected for bulk density measurements by ALS Lima. Samples were taken at 40 m intervals by project geologists, thus ensuring that all lithologies and alterations present at Zafranal were well represented, whether they contained mineralisation or not. Individual pieces of core measuring over 10 cm in length were collected and put into sealed and padded plastic bags to preserve sample integrity.

Bulk density samples were coated in paraffin and analyzed by ALS using the OA-GRA09 as method, described as follows: *"The rock or core section is weighed and then slowly placed into a bulk density apparatus which is filled with water. The displaced water is collected into a graduated cylinder and measured. From the data, the bulk density is calculated: Density = Weight of sample (g) / Volume of water displaced (cm^3)".*

During the 2014 in-fill program, a total of 1,588 density measurements were taken on site, using the water displacement method. The procedure is regulated and described in the "*Density Protocol_PLT_Zafranal_2014*" from the CMZ exploration geology department.

In line with the procedure, density was calculated on half drill core at 2 m intervals, using a scale. Samples were measured with and without a wax (paraffin) coating.

Table 11-1 presents a summary of the results of the program.

Table 11-1– Comparison of methods for bulk density by lithology.

Lithology	Sample with paraffin-coated		Sample without paraffin-coated	
	Nº Data	Average (SG)	Nº Data	Average (SG)
Migmatite	10	2.67		
Volcaniclastic	397	2.48	920	2.57
Diorite Zafranal	576	2.61	440	2.64
Microdiorite	258	2.59	203	2.80
Late Quartz Diorite	56	2.64	12	2.73
Dry Diorite	5	2.66	13	2.71
Total	**1,302**	**2.61**	**1,588**	**2.69**

Details of the bulk density data are provided in Section 14.

11.4 Sample Security Measures

All sampling was done at the drill site and in the core shack and was performed by CMZ personnel. RC samples were quartered and sealed in cloth bags at the drill site. Samples were then put into large rice bags, which were subsequently sealed and sent to ALS in Arequipa in a closed and padlocked truck.

Half core samples were collected in the core shack and individual sample bags were put into sealed rice bags which were transported to ALS Arequipa using the same security measures as for the RC samples.

Numbered zap-straps supplied by ALS were initially used to seal each individual rice bag, but the practice was discontinued during the 2014 program. This is not considered to be of material detriment to sample security for the 2014 program samples.

The security sample measures for Zafranal are considered adequate to ensure the authenticity of the samples.

11.5 Laboratory Sample Preparation, Assaying and Analytical Procedures

The ALS Peru S.A. analytical laboratory in Lima holds ISO/IEC 17025:2005 and ISO 9001:2008 accreditations for selected analytical methods.

Sample preparation for samples collected prior to 2014 is not recorded in the technical reports prepared at the time. All samples were analyzed for 32 elements by inductively coupled plasma (ICP) (MEICP41). Gold was analyzed by atomic absorption (AA) (AA23) using a 30 g sample. Samples with ICP copper assays greater than 0.2% were subjected to a series of analyses to establish sequential solubility. The values obtained from these analyses were used to model the mineral zones that form the basis of the following resource estimate.

Total copper content was determined by aqua regia digestion followed by an atomic absorption spectrophotometer (AAS) (AA46 for mineralized material-grade copper between 0.01% and 40% copper). Samples were then dissolved by weak sulphuric acid (AA06) to establish the proportion of total copper belonging to the secondary "oxide" phase. Sulphuric acid digestion was followed by cyanide digestion (AA16s) to determine the remaining portion of the total copper belonging to other secondary phases. The cyanide digestion was followed by hydrochloric and nitric acid digestion (AA62S) to determine the residual copper content which is equated to primary or hypogene mineralization.

All the samples collected in 2014 were drill cores of HQ diameter. The procedures used to prepare drill core were: crush to 70% less than 2 mm, riffle split off 250 g, and pulverize the split to better than 85% passing 75 µm (PREP- 31).

All samples were analyzed for 35 elements using method MEICP41. Gold was analyzed by fire assay and AAS (AA23) using a 30 g sample. Assay results were provided in digital format (both spreadsheet and PDF) by the laboratories and were automatically loaded into the database after validation.

Samples with ICP copper assays greater than 0.1% were subjected to a series of analyses to establish sequential solubility. The values obtained from these analyses were used to model the mineral zones that form the basis of the mineral resource estimate.

Total copper content was determined by aqua regia digestion followed by atomic absorption spectrophotometry (AAS) (OG46 for mineralized material-grade copper between 0.01% and 40% copper). Samples were then dissolved by weak sulfuric acid (AA06s) to establish the proportion of total copper belonging to the secondary "oxide" phase. Sulfuric acid digestion was followed by cyanide digestion (AA16s) to determine the remaining portion of the total copper belonging to other secondary phases. The cyanide digestion was followed by hydrochloric and nitric acid digestion (AA62S) to determine the residual copper content which is equated to primary or hypogene mineralization (Table 11-2). The rationale for the assignment of copper to various secondary mineral zones is explained in Section 14.3.

Table 11-2 – ALS main element analytical procedures and detection limits

Analytical Procedures			Detection Limits		
ALS Code	Description	Instrument	Unit	Lower	Upper
Au - AA23	Au 30g fire assay - AAS finish	AAS	ppm	0.005	10
ME - ICP41	35 element aqua regia ICP - AES	ICP-AES	ppm (Cu)	1	10000
Cu-OG46	Ore grade Cu - aqua regia / AAS	VARIABLE	%	0.001	40
Cu-AA06s	Cu sequential - sulfuric leach	AAS	%	0.01	100
Cu-AA16s	Cu sequential - cyanide leach	AAS	%	0.01	100
Cu-AA62s	Cu sequential - residual	AAS	%	0.01	50

11.6 Quality Control and Quality Assurance for the 2009-2010 Drilling Program

A summary of quality assurance and quality control (QA-QC) procedures for the drilling program conducted during the years 2009-2010 (Reference 27.11.2), is presented below.

CMZ followed standard QA-QC procedures with the regular insertion of blanks and certified standards, and collection of field duplicate samples. The quality control data of drilling used in resource estimation was assessed statistically to determine relative precision and accuracy levels between various sets of assay pairs and the variation of relative error over time during the exploration campaigns. A summary of the QA-QC program for 2009-2010 is presented in Table 11-3.

Table 11-3 – Summary of the samples analyzed in drill program 2009-2010 and its control samples

Zone	Samples DDH	Samples QA-QC	% of total samples	Standards		Duplicates		Blanks	
Zafranal Main	34,261	6,106	18%	2,231	7%	1,925	6%	1,950	6%

11.6.1 Standards

A total of 15 certified standards were used over the exploration campaigns, for a range of copper grades. The standards were prepared by the SGS laboratory in Lima (SGS Lima) from coarse drill core sample rejects. They were chosen to cover the grade range likely to occur at Zafranal for total copper (CuTotal) and Au. The samples were prepared in a conventional manner and were subjected to independent round-robin analyses, the results of which were used to determine the expected mean and standard deviation of each standard. The frequency of standards insertion in sample batches was approximately one standard to 15 samples.

Calculated precision and bias are within adequate ranges for copper and gold and only a limited number of isolated results fall outside of the acceptable intervals with the exception of Standard 333 (the highest copper grade standard at 6.87%) which displays a high proportion of erratic results. Within the range of grades in the copper mineralisation encountered at Zafranal, there is no evidence of bias or trend indicating the deterioration of assaying quality over time.

11.6.2 Duplicates

A total of 1,925 field duplicates results were available for analysis. The frequency of field duplicate insertion in sample batches was approximately one duplicate to 20 samples.

When using the hyperbolic method to analyse the adequacy of the results, only 5% and 4% of duplicates respectively are out of the acceptable range for CuTotal and Au which is well within the 10% acceptable range for field duplicates and demonstrates the adequacy of the sampling practices. This is corroborated by coefficients of correlation close to unity for all datasets.

No coarse or pulp duplicates were available for analysis.

11.6.3 Blanks

A total of 1,950 blank results were reviewed with analysis results for both Cu Total and Au. The blank dataset analysed only covers CMZ drill holes. The frequency of blank insertion in sample batches was approximately one blank to 20 samples.

For copper, 23 results exceeded 10 times the detection limit, i.e. approximately 1% of the results, and 2 results exceeded 20 times the detection limit. For gold, no result exceeded 10 times the detection limit.

11.6.4 Conclusions

The review of the analytical database QA-QC for the 2009-2010 drilling program indicates that the sample preparation and assaying conducted by AQM is of reliable and consistent quality, and provides accurate and required information which is suitable for resource estimation and mine planning studies.

11.7 Quality Control and Quality Assurance for the 2011-2012 Drilling Program

A summary of QA-QC for the drilling program conducted during the years 2011-2012 (Reference 27.11.6), is presented below.

CMZ employed standards, duplicates and blank samples in its quality assurance and quality control (QA-QC) program. A total of 21,931 samples were generated, with 18,420 original or primary samples corresponding to HQ diameter diamond drill cores. Table 11-4 shows the number of control samples used and the percentage of the sample population represented by each type of control sample.

Table 11-4 – Summary of the samples analyzed in the 2011-2012 drilling program and its control samples

Zone	Samples DDH	Samples QA-QC	% of total samples	Standards		Duplicates		Blanks	
Zafranal Main	21,931	3,507	16%	1,316	6%	1,093	5%	1,098	5%

Samples that are part of the QA/QC for the grade analysis are:

- Sample Type 1: field duplicate (quarter core)
- Sample Type 4: standards
- Sample Type 6: coarse blanks (rock chips).

Each batch was composed of 100 primary samples and five quality control samples, corresponding to standards, blanks and duplicates.

The responses obtained from the control program are discussed below.

11.7.1 Standards

CMZ used six standards for the 2011-2012 drilling program QA-QC, for a range of copper grades. The standards were also prepared by SGS Lima from coarse drill core sample rejects. Three of the standards were from the 2009-2010 drilling program, with three new copper-only standards added. The samples were prepared in a conventional manner and were subjected to independent round-robin analyses, the results of which were used to determine the expected mean and standard deviation of each standard.

The quality control of analytical accuracy undertaken with the independent reference materials is based upon the difference between the analytical value and the established value. If the difference is >±2 standard deviations (SD) to the original round robin value then the analytical batch is considered to be on “alert”, whereas for differences >±3SD the batch is considered out of control.

The accuracy of the main laboratory is acceptable for Cu for standards (CRMs) 1000, 2000, and 3000; and for Au, for standards 700, 800, and 900. There are failed re-assays for standards 700 and 900. Copper standard 800 had 120 failed re-assays (84% of total); there is strong negative bias (-5%) with respect to the standard’s expected value for Cu. Given these results, it was considered that this standard could not be used as such to control laboratory accuracy, because the sample preparation and analysis of the standard itself is suspect.

11.7.2 Duplicates

A total of 1,093 field duplicates results for CMZ drill holes were available for analysis.

Assay precision from field duplicates is good for Cu and acceptable for Au. Gold has more intrinsic variability, reflected by the standards coefficients of variation (STD 700 and STD 800 with 7%; STD 900 with 5%). No samples were re-assayed on the basis of differences between duplicates.

11.7.3 Blanks

A total of 1,098 samples were inserted and used during the 2011-2012 drilling campaign. These were rock chip samples from barren outcrops of granodiorite, located in areas external to the deposit.

The blank samples from this period do not present significant levels of contamination, which demonstrated that the cross-contamination during the sample preparation process was insignificant.

11.7.4 Conclusions

The review of the analytical QA-QC database for copper and gold indicated that the sample preparation and assaying was of reliable and consistent quality, and provided accurate and precise information which was suitable for resource estimation. To increase the confidence level of quality control, the batches with control sample failures identified should be re-assayed.

11.7.5 Re-assaying program

To increase the confidence level of quality control, a series of recommendations were presented in the QA-QC report for the 2011-2012 drilling campaign at the Zafranal project (Reference 27.11.6). These included re-submission and re-assaying of 10 or 20 samples immediately before and after standards identified as failures during the previous analysis. The decision as to whether 10 or 20 samples were included was dependent on the results of the control samples adjacent to the failed standard.

Of a total of 765 original samples corresponding to original pulp samples of the drill core, 740 pulps were re-assayed for copper and 25 pulps were re-assayed for gold. The conclusions of the re-assaying program (Reference 27.11.5) were:

- The re-assayed data for copper were examined through statistical and graphical tests for a comparison of the failed standard and the re-assayed values. The re-assayed values are acceptable; therefore, the original data analyzed by ALS is considered to be reliable and validated.
- The re-assayed data for gold were examined through statistical and graphical tests for a comparison of the failed standard and the re-assayed values. The re-assayed values are within the acceptance limits (according to the variability of the element); therefore, the original data analyzed by the ALS is considered to be reliable and validated.

11.8 Quality Control and Quality Assurance for the 2014 Drilling Program

CMZ employed standards, duplicates and blank samples in its 2014 QA-QC program. A total of 3,144 samples were generated, with 2,793 original or primary samples corresponding to HQ diameter diamond drill cores. A total of 351 control samples were incorporated in the process, which represents 11% of the total samples (Table 11-5).

Table 11-5 – Control sample for 2014 drill program

Year	Zone	Samples DDH	Samples QA-QC	Total samples %	Standards		Duplicates		Blanks	
2014	Zafranal	3144	351	11%	115	3.7%	120	3.8%	116	3.7%

All the quality control samples were inserted on site. The processes were carried out in line with procedures laid out in CMZ's "*Management Manual QA/QC_V1_2014*".

The samples that are part of the QA-QC for the grades analysis are:

- Sample Type 1: field duplicate (quartercore)
- Sample Type 4: standards
- Sample Type 6: coarse blanks (rock chips).

Each batch was composed of 90 primary samples and 10 quality control samples, corresponding to standards, blanks and duplicates.

Standards, blanks and duplicates are inserted after the cores samples are cut in half using the following systematic sequence every 8 samples: duplicate-blank-standard-standard-blank-duplicate. Most samples were collected at intervals of two metres, stored in bags for a total of approximately 35 kg, numbered and sealed with tape.

The responses obtained from the control program are discussed below.

11.8.1 Standards

CMZ used three standards, for a range of copper grades, prepared by SGS Lima from coarse drill core sample rejects. The samples were prepared in a conventional manner and were subjected to independent round-robin analyses, the results of which were used to determine the expected mean and standard deviation of each standard. Other commercial standards were also used as independent reference materials for quality control of copper and gold analyses, and were purchased from CDN Resource Laboratories Ltd. (Canada).

The quality control of analytical accuracy undertaken with the independent reference materials is based upon the difference between the analytical value and the established value. If the difference is >±2SD to the original round robin value then the analytical batch is considered to be on "alert", whereas for differences >±3SD the batch is considered out of control.

The accuracy of ALS for standards 1000, 2000 and 4000 for Cu in 112 standards is acceptable, with overall negative bias of analytical results of less than -2%; and for Au with positive bias of analytical results of 5%; 100% of the results are within the limit of ±2SD.

11.8.2 Duplicates

A total of 120 field duplicates results for CMZ drill holes were available for analysis.

The duplicate analyses are indicative of a high level of analytical precision for Cu and Au. No samples were re-assayed on the basis of differences between duplicates.

11.8.3 Blanks

A total of 116 samples were inserted and used during the 2014 in-fill drilling campaign. These were rock chip samples from barren outcrops of granodiorite, located in areas external to the deposit.

Of the analyzed data for copper coarse blanks, 100% are below the established limits for the blanks used as a control. Of the analyzed data for gold coarse blanks, 99% are below the established limits for the blanks used as a control.

The blank samples from this period do not present significant levels of contamination, which demonstrated that the cross-contamination during the sample preparation process was insignificant.

11.9 Conclusions

The main conclusions results of QA-QC of the 2014 in-fill drilling and assaying program (Reference 27.11.1) are summarized below:

- The accuracy of the main laboratory is acceptable for Cu and for standards (certified reference materials) 700, 1000, 3000 and 4000; and for Au, for standards 700 and 4000. There were no failed data for either element.
- Assay precision from field duplicates is very good for Cu. Assay precision from field duplicates for Au is acceptable, considering that it has more intrinsic variability, reflected by the standard's coefficient of variations (STD 700 and 4000 with relative standard deviation of 5%). The precision of analyses of the field duplicates for the two elements of interest is according to industry standards.
- There appears to be no contamination, with 100% of the data analyzed for blanks, having lower values than the limit set. Therefore, the results are considered acceptable.
- Cross-laboratory validation with Acme Analytical Laboratories Peru S.A. to evaluate the performance of ALS shows that the data obtained by ALS are reliable. The comparison of the main elements (copper and gold) analyzed by ICP and fire assay of primary samples, is acceptable in all cases.
- Based on the results obtained in the framework of the QA-QC program, the level of confidence of the assays is acceptable and they can be used to support a resource estimate for the project. The results obtained have been confirmed and/or reproduced within reasonable limits by an alternate laboratory.

12 Data Verification

12.1 Procedures

CMZ maintains a procedure for the entry and management of data structured by means of a database of geological/geotechnical and other data related with the drilling process.

The starting point of for the process was the unification of criteria through the implementation of the "*Protocol Geological Logging Zafranal_2014*" (Reference 27.2.1). There was a unified and simplified coding arrangement typical of porphyry copper deposits (Table 12-1).

Table 12-1 – Project abbreviations and coding for geological logging

Rockg Zafranal - Victoria					
Rock_type	**Lito_desc**	**Unidad**	**Ms_code**	**Rockg**	**Colour**
No Sample	NN	NN	0	NA	
Migmatite	MIG	MIGP0	10	10	
Volcaniclastic	VC	VCP0	20	20	
Victoria diorite	DIO VICT	VDP1	30	30	
Victoria microdiorite	MDIO VICT	VMDP2	50	50	
Mafic dyke	DM	DM	55	NA	
Gabrodiorite	GBDIO	GBDIO	65	65	
Granodiorite	GRND	GRND	70	66	
Pegmatite	PGM	PGM	75	NA	
Aplite	APL	APL	80	NA	
Zafranal diorite	ZAF DIO	ZDP1	90	90	
Microdiorite	MDIO	ZMDP2	95	95	
Quartz feldspar porphyry	QFP	QFP	96	96	
Hidrotermal Breccia	BXH	BXH	97	97	
Tectonic breccia	BXT	BXT	98	98	
Late quartz diorite	LT QZDIO	VLQDP3	100	100	
Dry diorite	DRY DIO	DDP4	103	70	
Post diorite	PT DIO	PDP5	105		
Monzodiorite	MZDIO	MZDP5	110		
Late microdiorite	LT MDIO	LMDP4	115		
Porphyritic andesite	AND POR	ANDPY6	120		
Andesite	AND	ANDY6	130		
Post-mineral microdiorite	PT MDIO	PMDP5	135		
Quaternary	QT	OVB	140	140	
Principal fault	FI	FI	600	600	

Altrg Zafranal - Victoria				
Alt_tipo	**Description**	**Ms_code**	**Altrg**	**Colour**
Unalt	Unaltered	1	200	
ARGI	Intermediate argillic	10	210	
K	Potassic	20, 30, 40, 120, 140, 150 and 160	220	
Ep-Cl	Epidote-chlorite	60, 50 and 110	230	
PHY	Phyllic	70 and 130	240	
Ca-Na	Sodic-calcic	80	250	
ARG	Argillic by fault	90	260	
SIL	Silicification	100	270	

Minzog Zafranal - Victoria			
Minzone	**Description**	**Minzog**	**Colour**
LZ	Leached zone	310	
OZ	Copper-oxide zone	320	
MZ	Mixed zone	330	
SZ	Supergene enrichment zone	340	
HZ	Hypogene zone (chalcopyrite)	360	
DYKES	Post-Mineral Dykes	370	
SS	Without Sequential Copper	380	

This protocol is used to set the GVMapper v.3.6 software, using code tables and validation by which the online logging of the 2014 in-fill drilling and assay program and the program of re-logging were performed. All cores have been removed from the Property and are stored in Arequipa. This core storage facility was examined and found satisfactory.

12.2 Limitations and Failures (if any)

There are a total of 65,473 assay intervals, with 56,537 intervals drill prior to 2014. The additional DDH information obtained in 2014 was validated by Politax S.A. in early 2015 (Reference 27.12.3). This is in addition to pre-existing validations by GeoSystems International, Inc. (GSI) of the historical data in 2013 (Reference 27.12.5 and 27.12.7) and AQM in 2013, in reference to collars and changes in base points for survey (Reference 27.12.6, 27.12.8 and 27.12.9).

The excel file called ZAF_Assays_20150224 has a total of 65,473 intervals, of which 140 do not have assays and show as NO SAMPLE. Of these, 80 intervals belong to DDH ZFRC10-083, and were deleted from the database. The additional 60 intervals were coded as -99.

A total of approximately 10% of the database was validated against the original certificates and information. The main conclusions results are summarized below:

- To date over 65,393 samples from the Zafranal Main and Victoria zone have been assayed for Au, ICP_35 elements and CuT-SEQ06_% by a highly reputable commercial laboratory. Of these, 5,725 samples (9%) were compared to the corresponding values on the spreadsheets as well as against the values contained in the CMZ database. The number of discrepancies found in the checking of certificates does not exceed 1% (Reference 27.12.2).
- There were no serious errors in the file ZAF_Assays_20150224, used in geologic modeling and resource estimation. There were some minor discrepancies that were corrected, but that affected less than 1% of total intervals checked. Work proceeded using the corrected information, file ZAF_Assays_JB_13_03_2015.
- The geological logging data was examined for sample overlaps and/or gaps in down hole logging data, with overlaps and duplication identified and removed.
- With respect to the down the hole survey file, the database had been reviewed and corrected in 2013. In the case of the drill holes in 2014, all differences in azimuth and dip on subsequent readings are within acceptable values, with a maximum of 6.10º in azimuth, and 3.60º in dip. No errors in location down the hole were found (Reference 27.12.3, 27.12.4 and 27.12.5).
- With respect to the collar information, for this Resource Model update the database had been corrected in 2014, using a conversion value of -1.4935 m in elevation, as discussed in checks previously documented (Reference 27.12.6 and 27.12.9).

12.3 Adequacy of Data

Based on the preceding analysis, the data are considered to be of adequate quality and acceptable for the purposes of the mineral resource estimate that follows.

13 Mineral Processing and Metallurgical Testing

13.1 Testing and Analysis Procedures and Results

The results of the five phases of metallurgical testwork programs and reports on the earlier phases by other engineering consultants were provided to Ausenco for review.

Various metallurgical testing was done in each phase, without the same testing being done in all phases:

- Phase 1, 2010
  - Comminution - SGS Mineral Services – Chile (SGS)
  - Flotation and leaching - Amdel Laboratories – Australia (Amdel)
- Phase 2, 2011
  - Comminution - SGS
  - Flotation and leaching - Amdel
- Phase 3, 2012
  - Flotation - G&T (ALS Metallurgy) Kamloops (G&T) and C.H. Plenge & Cia (Plenge)
  - Thickening - Pocock Industrial (Pocock)
- Phase 4, 2014-2015
  - Leaching - Plenge
- Phase 5, 2015-2016
  - Comminution, flotation and leaching – Plenge
  - Thickening - Pocock
  - Materials handling - Jenike and Johanson.

The Preliminary Economic Assessment Update (PEAU) completed in October 2013 proposed a concentrator with primary crushing followed by a single grinding circuit consisting of a SAG mill and ball mill with future pebble crushing, flotation circuit with regrind, copper concentrate dewatering and tailings thickening and disposal as the treatment route for the Zafranal hypogene and supergene zones. The proposed treatment routes reflect the results of prior metallurgical testing, and subsequent testing reflects the proposed treatment routes.

13.1.1 Mineralogy

Drilling at Zafranal has shown that mineralization and alteration have a well-zoned pattern consisting of a leached oxide zone, a mixed oxide and supergene sulfide zone, an immature enriched supergene sulfide blanket zone and a hypogene sulfide zone. The oxide zone is dominated by the presence of copper chlorites, with significant amounts of copper goethite and minor amounts of copper biotite. Within the supergene zone chalcocite becomes the dominant copper mineral, with copper chlorite decreasing with depth. Hypogene material is predominantly chalcopyrite, with increasing pyrite at depth.

Gangue material includes iron oxidation minerals (goethite), quartz, chlorite, biotite and epidote. Phase 3 mineralogical tests on three main composites found low content of clays as shown in Table 13-1.

Table 13-1 – Mineralogical analysis of the three main phase 3 composites

Mineral	Mass Percent (%)		
	Hypogene (ZFC-01)	Supergene (ZFC-02)	Mixed (ZFC-03)
Chalcopyrite	1.3	0.6	0.8
Bornite	<0.01	<0.01	<0.01
Chalcocite	<0.01	0.3	0.1
Covellite	0.01	0.1	0.1
Turquoise	<0.01	0.02	0.04
Pyrite	4.5	3.9	4.7
Iron Oxides	0.1	0.7	1.0
Quartz/Feldspars	65.1	63.6	57.2
Micas	22.5	22.4	27.3
Chlorine	4.2	4.4	4.8
“Kaolinite” (Clays)	0.2	1.1	1.4
Epidote	0.6	0.7	0.5
Ti Minerals	0.8	0.6	0.7
Other Gangue	0.7	1.6	1.2

Phase 5 test work results have also indicated that clay content is low, at a maximum of 1.45%.

13.1.2 Physical and chemical characteristics

Copper grades for all flotation samples reported by Plenge in Phase 3 were in the range from 0.2% to 0.99%, iron grades were in the range from 2.3 to 7.1%, silver grades ranged from 0.6 to 3.0 g/t and gold grades were in the range from 0.02 to 0.37 g/t.

Copper grades for all flotation samples in Phase 5 were in the range between 0.12 to 1.08% and gold grades were between 0.05 and 0.76 g/t, whereas iron grades were between 5.6 and 49.7%.

Physical characterization of the Zafranal composites used in flotation tests for hypogene, supergene and mixed ore classifications indicated specific gravities in the ranges respectively from 2.67 to 2.74, 2.70 to 2.78, and 2.67 t/m^3. Average natural pH of the hypogene, supergene and mixed ore types were respectively 7.2, 6.7 and 7.2.

Bulk densities (at 100% -3.35mm) for the hypogene, supergene and mixed ores averaged approximately 1.9 t/m^3.

The nominal copper head grade used in the process design criteria is 0.45% Cu and was based on the average head grade for the mine production schedule from May 2014. Although superseded by more recent revisions, the approach taken to accommodating later revisions was to reference back to this nominal head grade. The maximum copper head grade occurs in the first year of operation at 0.85%, and copper head grades decline after the first year of operation to 0.18% in the last two years of operation.

13.1.3 Comminution

Comminution test work has been undertaken in three phases of testing to date on samples from the Zafranal deposit:

- Phase 1, 2010, SGS Mineral Services – Chile (SGS)
- Phase 2, 2011, SGS Mineral Services – Chile (SGS)
- Phase 5 2015, C.H. Plenge & Cia, Peru (Plenge).

In total 109 composite samples of drill core material have been subjected to comminution testing which consisted of the following tests:

- SAG mill comminution test (Axb)
- Bond ball mill work index (BBWi)
- Abrasion index (Ai)
- JKMRC drop weight tests (JKDWT)
- SAG power index (SPI)
- Rod mill work index (RWi)
- Crusher work index (CWi).

Not all samples were subjected to all tests, however the majority of samples were subjected to SAG mill comminution tests, bond ball mill work index and abrasion index tests.

The results of this program are summarized in Table 13-2.

Table 13-2 – 75th percentile comminution results

Parameter	Unit	Supergene	Hypogene	Mixed
Axb	-	34.6	28.5	36.9
DWi	kWh/m^3	7.5	9.3	7.0
SG	t/m^3	2.60	2.65	2.59
CWi	kWh/t	20.2	24.6	19.0
RWi	kWh/t	15.3	15.6	14.3
BWi	kWh/t	12.8	13.0	11.9
Samples Tested	Number of	55	49	5

Additional comminution tests: JKDWT, SPI, RWi and CWi; were performed on three samples. These tests were performed to calibrate the Axb values which were determined from the SMC test work (a short-format JKDWT style test). The calibration is required to determine the variation in breakage response across a range of size fractions. The smaller size fractions used for the SMC test can result in lower Axb values if uncalibrated. The summarized results from the three samples are shown in Table 13-3.

Table 13-3 – Phase 5 Plenge calibration testwork results summary

Sample	Ore Type	Axb (JKDWT)	Axb (SMC)	BWi (kWh /t)	Ai (g)	ta	RWi (kWh/t)	CWi (kWh/t)	SPI (min)
ZECS-31	Supergene	57.8	53.3	10.4	0.099	0.69	11.7	11.2	38.3
ZECS-33	Supergene	61.6	58.9	10.7	0.056	1.01	11.8	9.54	48.4
ZECS-43	Mixed	64.9	67	13.3	0.063	1.01	11.4	4.79	49.6

The samples selected for the analysis were two supergene samples and one oxide sample but no hypogene samples were tested. As hypogene ore is more competent than both the oxide and supergene ores, hypogene ore is critical to the analysis with low Axb values. The results from the samples tested show that there is increasing bias between the results of the JKDWT and SMC test with reducing Axb. Future testing of hypogene samples is required to assess bias in Axb values below 50.

13.1.4 Flotation

Flotation testwork has been undertaken in four phases of testing to date on samples from the Zafranal deposit:

- Phase 1, 2010, Amdel Laboratories - Australia
- Phase 2, 2011, Amdel Laboratories - Australia
- Phase 3, 2012, G&T (ALS Metallurgy) Kamloops, Canada; and C.H. Plenge & Cia, Peru
- Phase 5, 2015 C.H. Plenge & Cia, Peru.

Metallurgical design for the concentrator was based on the outcomes of the Phase 3 and Phase 5 test work campaigns.

13.1.4.1 Sampling

Samples for the 3 domain composites and 35 variability composites were selected from Main Zafranal and Victoria Zones. Copper and gold head grades for each domain composite and the grade ranges for variability samples are shown in Table 13-4.

Table 13-4 – Phase 3 Composites head grades

Composite	Zone	Cu (%)		Au (g/t)	
		Plenge	G&T	Plenge	G&T
Phase 3					
ZFC-MC01	Hypogene	0.45	0.46	0.15	0.084
ZFC-MC02	Supergene	0.49	0.52	0.13	0.069
ZFC-MC03	Mixed	0.41	0.44	0.13	0.11
Variability	Range	0.20-0.99	0.22-0.88	0.02-0.37	0.03-0.26
Phase 5					
ZEFC-01	Supergene	0.74		0.05	
ZEFC-02	Hypogene	0.25		0.06	
ZEFC-03	Supergene	0.18		0.03	
ZEFC-04	Hypogene	0.49		0.03	
ZEFC-05	Mixed	0.31		0.03	
ZEFC-06	Supergene	0.23		0.01	
Variability	Range	0.12-1.08		0.05 – 0.76	

The head grades results from the two laboratories for the domain composites showed good correlation for copper assays but poor correlation for gold assays.

13.1.4.2 Rougher flotation

13.1.4.2.1 Grind size

Primary grind size was evaluated during the both the Phase 3 and 5 test work campaigns. During Phase 3 testing, primary grind size test work was performed at G&T using three domain composite samples (ZFC-MC01, ZFC-MC02 and ZFC-MC03) which represented supergene, hypogene and mixed mineral zones. Testing was performed at 106 µm, 150 µm, and 212 µm, and the rougher was operated at a pH of 10. PAX additions varied in the rate of 6-11 g/t.

The test work found that there was no change in copper recovery from P_{80} of 212 µm to 150 µm. From P_{80} of 150 µm to 106 µm there was a slight increase in recovery. The lowest gold recovery for hypogene and mixed domain composites is at P_{80} of 150 µm and the highest for both is at P_{80} of 106 µm. For the supergene domain composite, gold recovery decreased with P_{80} from 212 µm to 150 µm to 106 µm. Based on the Phase 3 samples a primary grind size of 150 µm was selected for the PEAU.

The primary grind size was further evaluated during the Phase 5 test work performed at Plenge with tests performed on two composite samples (ZEFC-01 and ZEFC-02) representing supergene and hypogene ore types. The rougher flotation kinetics were evaluated at a P_{80} grind sizes of 75, 106, 125, 150, 212 and 250 µm.

The test work for both supergene and hypogene ore types showed that a laboratory grind size of 125 µm produced good rougher recoveries with selectivity over iron for both the supergene and hypogene ores. For supergene ore a noticeable decrease in kinetics occurs above P_{80} of 212 µm for supergene and P_{80} of 150 µm for hypogene.

A laboratory grind size P_{80} of 125 µm corresponds to a concentrator primary grind of 150 µm once the effect of closed circuit hydrocyclone classification is considered. The optimum primary grind size of 150 µm was selected based on Phase 3 and Phase 5 test work.

13.1.4.2.2 Rougher pH optimization

Rougher pH optimization test work was conducted during Phase 3 at both G&T and Plenge, whilst confirmation work was performed during Phase 5 at Plenge.

The Phase 3 G&T test work assessed the copper and iron recovery at pH values at natural (6-6.5), 9 10 and 11. The work identified that for the supergene (ZFC-MC01), hypogene (ZFC-MC02) and mixed (ZFC-MC03) samples the optimum rougher pH was 10. All samples had similar copper recoveries across the range of pHs tested, however pH 10 provided higher selectivity against iron in all samples except the mixed sample, where iron recovery was slightly higher.

The Phase 3 Plenge test work assessed the copper and iron recovery at pH values of natural (7), 9.5, 10.5. The work identified that based on the same three samples as used for the G&T work, the optimum rougher pH was 10.5. Across the pH ranges tested, copper recovery was similar for all samples, however the highest gold recovery for both supergene and hypogene was observed at pH 10.5 and pyrite rejection was enhanced at higher pH for supergene and mixed samples (Figure 13-1). Copper from iron selectivity was unchanged for the hypogene sample between a pH of 9.5 and 10.5. The analysis concluded that a pH of 10.5 was optimum for all samples tested.

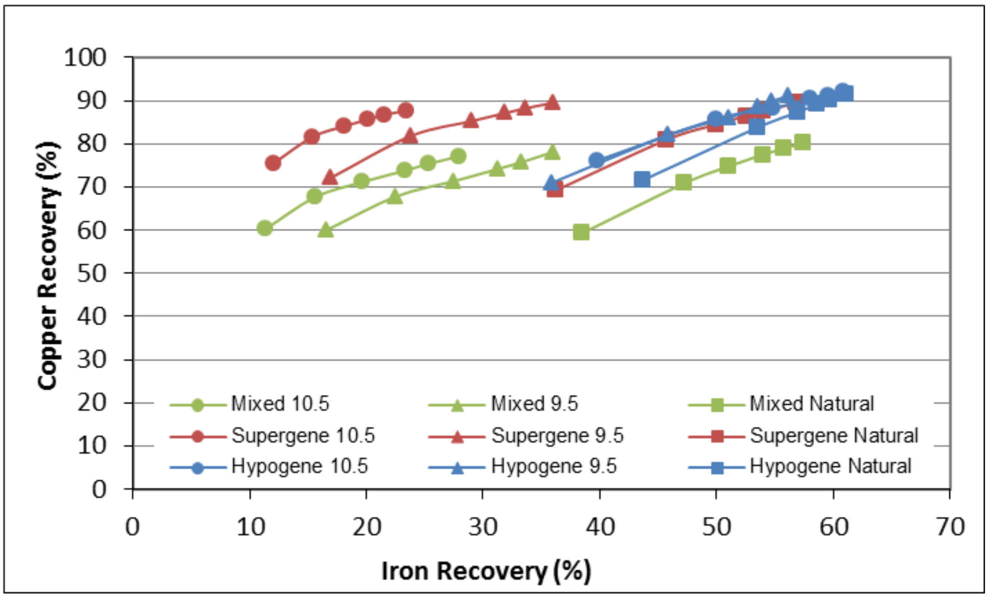

Figure 13-1 – Rougher flotation, pH effect on copper-from-iron selectivity – Plenge

Confirmatory testing was performed during Phase 5 campaign performed at Plenge. The test work considered the difference in the supergene (ZEFC-01) and hypogene (ZEFC-02) samples at pH of 8,9,10. The test work confirmed that a design value for rougher pH of 9-10 was optimum. The test work showed little difference in this range, with the recommendation of the Phase 5 test work that the pH for the rougher circuit be a minimum pH of 9 due to higher selectivity between copper and iron.

The supergene composite tested showed little difference in copper recovery between all pH ranges; however, iron recovery was much higher below pH 9. Iron recovery for the hypogene sample was elevated at a pH of 10. All other pH values were in the same range

During the Phase 5 test work an additional test was performed at a pH of 9 with a split lime addition to the primary grind and conditioner versus lime added only at the rougher conditioning stage. Splitting the lime to the grinding circuit showed improvement in depression of pyrite, with far lower iron recovered in the roughers. This trend was not apparent for the hypogene ore with similar iron recovery in both tests. The option to add lime to the grinding mills was incorporated in to the design.

13.1.4.2.3 Reagent scheme

A number of reagent schemes have been tested throughout the four phases of flotation test work. Parallel tests were conducted during Phase 3 at G&T and Plenge, with 12 different rougher reagent schemes assessed.

Reagent scheme confirmation was performed during the Phase 5 test work over two different campaigns, with different conditions used in the two campaigns. A total of 10 reagent schemes were assessed during the test work. The second campaign used the optimised rougher conditions of a primary grind of 125 µm and pH 9.

The evaluation determined that AP-9950 was the optimum collector for both supergene (ZEFC-01) and hypogene (ZEFC-02) ore composites. This conclusion was based on similar copper kinetic performance to the other collectors but higher selectivity to iron for the supergene. This was a minor change from the optimum for the Phase 3 test work's A3302+A3418 collector scheme.

13.1.4.2.4 Rougher collector dosage

Collector dose optimisation was performed in the rougher circuit during the Phase 5 test work. The test work was performed with the previously identified optimum collector AP-9950, and a primary grind size of 125 µm. The test work showed that a lower collector dose provided similar rougher kinetic performance for supergene (ZEFC-01) whilst reducing the amount of pyrite reporting to rougher concentrate. For hypogene (ZEFC-02), the lower collector dosage rate had lower recoveries at residence times of less than six minutes.

Based on the test work, a collector addition of 30 g/t was selected as the optimum collector dosage rate for both supergene and hypogene ore types. This collector addition rate had a good trade-off between copper and iron recovery for both ore types.

13.1.4.2.5 Rougher dispersant effect

Rougher flotation kinetic tests were performed during Phase 5 to evaluate samples with potentially high content of clays that could produce a viscous pulp. These tests were performed on the same composite samples (ZEFC-01 and ZEFC-02) with the test work showing that there was no benefit in adding sodium silicate as a dispersant to either supergene or hypogene composites. In addition to adding sodium silicate to the sample, the flotation pulp density was decreased from 33%w/w solids to 24%w/w solids. Despite both these changes, no significant benefit in performance was seen for either the supergene or hypogene composite sample.

The samples without a dispersant added and at higher pulp density showed greater selectivity against pyrite with less iron recovered at similar copper recoveries

The test work performed confirmed that there was no benefit of adding a dispersant to the samples and that a rougher feed density of 33%w/w is appropriate for the Zafranal ore. The current PFS design rougher pulp density is 35%w/w solids, which is based on similar operations. The design will be updated in the next phase of test work.

13.1.4.2.6 Rougher kinetics

Phase 3 test work identified that a rougher batch laboratory residence time of 10 minutes was sufficient to achieve high recoveries. Additional test work was undertaken at Plenge as part of the Phase 5 test work. Rougher kinetic tests were performed on five composite samples consisting of two samples of each of supergene and hypogene with one additional sample of mixed.

The results show that for all but one sample the rougher flotation time of 10 minutes is sufficient (Table 13-5). Kinetic results from the rougher variability test work also confirmed that 10 minutes is suitable for the mineral zones tested.

Table 13-5 – Modelled optimum rougher residence time

Sample	Mineral zone	Head grade (% Cu)	Optimum rougher time (min)
ZEFC01	Supergene	0.63	7.9
ZEFC02	Hypogene	0.34	15.5
ZEFC03	Supergene	0.72	7.4
ZEFC04	Hypogene	0.32	7.7
ZEFC05	Mixed	0.25	9.1

13.1.4.2.7 Rougher variability samples

Phase 5 variability rougher flotation was performed on 90 samples. The conditions for the test work were a primary grind size of 125 µm, feed solids of 33%w/w and the optimised rougher regent scheme of 30 g/t AP-9950. Lime was added to a pH target of 9.

The variability test results showing the grade and recovery accumulated after 16 minutes of flotation are specified in Figure 13-2. Rougher recoveries ranged from 11.5 to 96.1% Cu and 39.5 to 94.2% Au, whereas rougher grades ranged from 0.28 to 15.1% Cu and from 0.03 to 2.16 g/t of Au.

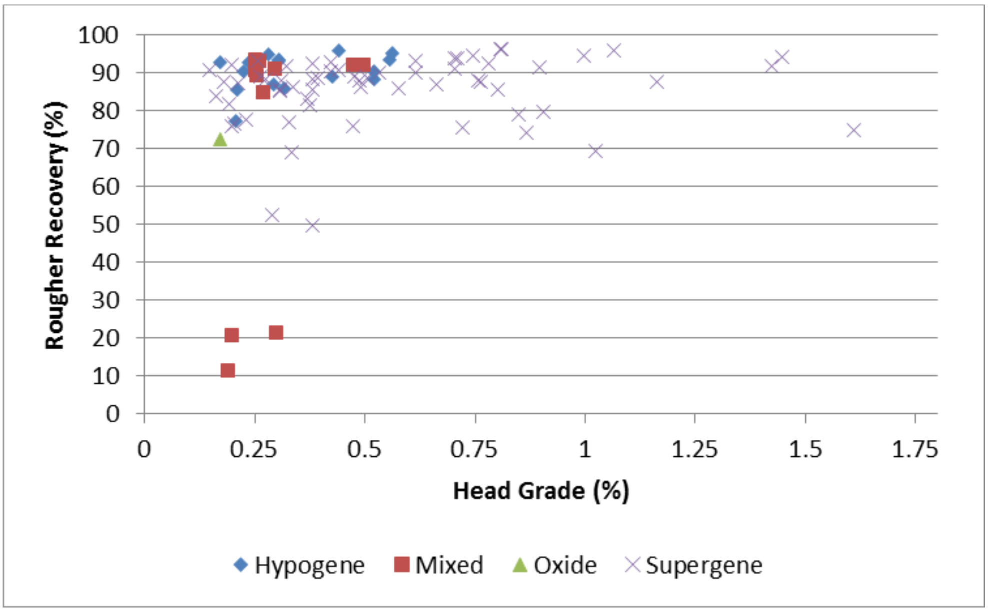

Figure 13-2 – Variability of rougher recovery by head grade

The hypogene recovery showed lowest variability with 75% of copper recoveries above 88%. Mixed recovery was highly variable with three values of 10-20% recovery. Whilst the supergene copper recovery was variable with recovery not related to head grade.

13.1.4.3 Cleaner flotation

13.1.4.3.1 Regrind size

Phase 3 test work performed by G&T identified that the optimum rougher concentrate regrind size for supergene and mixed samples was 40 µm whilst regrinding to 20 µm resulted in a slight increase in copper recovery for the hypogene sample. Gold recovery was found to be maximized at 20 µm for all samples. At a regrind size of 40 µm, final concentrate copper grade is in the range of 25% to 32% copper.

The Phase 3 Plenge regrind analysis test work at tested two regrind sizes of 140 µm and 20 µm. The work showed that copper recovery of the mixed and hypogene samples decreased with regrind size whilst higher recovery was achieved for the supergene ore at 20 µm.

Confirmation test work was performed by Plenge as part of the Phase 5 test work. The regrind tests were performed on two composites representing the supergene (ZEFC-01) and hypogene (ZEFC-02) mineral zones, with open circuit cleaner test work performed. Regrind was evaluated at 90, 60, 40 and 20 µm.

The test work showed that open circuit copper recovery for the supergene was maximized at a regrind P_{80} of 40 µm (Figure 13-3). Decreasing regrind size did not improve copper recovery, perhaps due to insufficient cleaner residence times, insufficient reagent addition or due to sliming of secondary copper sulfides. Gold recovery was maximised at the finest grind and generally decreased as the regrind size increased.

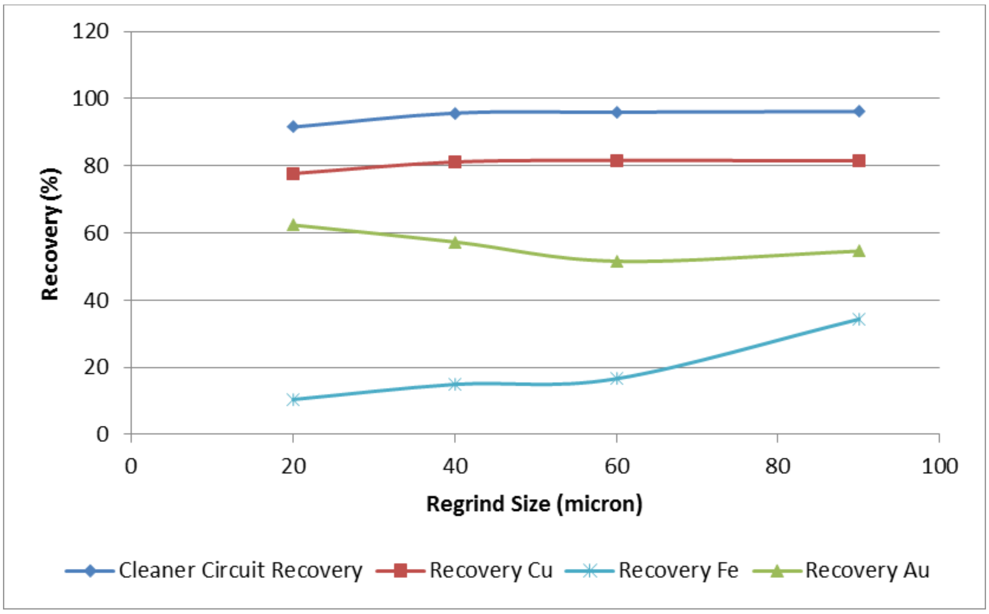

Figure 13-3 – Effect of regrind size on cleaner recovery for supergene sample (ZEFC01)

The hypogene ore showed similar trends for gold and iron recovery; however, copper recovery did not show the same decrease in recovery a regrind size of 20 µm.

Copper and gold concentrate grades were maximized for both samples at a regrind P_{80} of 20 µm due to lower pyrite content in the Cleaner 3 concentrate. Based on the work in Phase 3 and 5 a regrind P_{80} of 40 microns was selected for design.

13.1.4.3.2 Cleaner pH evaluation

The Phase 5 test work also assessed the optimum cleaner pH for the supergene (ZEFC-01) and hypogene (ZEFC-02) composite with four pH values (9.5, 10, 10.5 and 11) investigated.

Copper and gold recoveries were similar for the supergene sample at all pH values tests whilst at a pH of 9.5 iron recoveries increased. Copper grades in the Cleaner 3 concentrate were found to decrease significantly at a pH of less than 10. This corresponded to increased iron grades in the concentrate. Gold grade in the concentrate showed a trend of decreasing with pH.

Similar trends were exhibited by the hypogene sample with copper recovery flat across all pH values tested. For hypogene, both gold and iron recoveries increased at lower pH indicating that they are associated.

Again, at a pH of 9.5 iron grade increased significantly in the Cleaner 3 concentrate. Copper grades decreased with pH, and a large drop occurred below a pH of 10. Gold grade was highest at a pH of 10.

Based on the results of the Phase 5 test work confirming the Phase 3 test work a cleaner pH of 9 to 10 was adopted for design and a cleaner pH of 10 was used in the cleaner tests.

13.1.4.3.3 Cleaner collector dosage

Collector dosage tests were conducted for a supergene (ZEFC-01) and hypogene (ZEFC-02) sample. The tests were conducted at a regrind P_{80} of 40 µm and a cleaner pH of 10. The optimised rougher collector AP-9950 was used for the two tests.

The tests showed that adding no collector to the cleaner circuit produced the lowest iron recovery while maintaining copper recoveries. The gold recovery optimum was at a dosage of 5g/t for supergene.

Cleaner 3 concentrate copper and gold grades increased with lower collector dosages for the supergene due to lower iron recovery.

Similar trends were seen in the hypogene sample with decreased collector causing no decrease in copper recovery. For hypogene, gold recovery also decreased with collector dosage along with iron. This suggests that the gold is associated with pyrite.

As it was for the supergene sample, Cleaner 3 concentrate copper and gold grades increased with decreased collector dosage rates due to decreased recovery of iron.

Based on the test work no collector addition was required for the cleaners. The plant design will however provide allowance for additions to the circuit.

13.1.4.4 Locked cycle test work

Located cycle test work was performed during the Phase 3 test work at both G&T and Plenge. As the optimum conditions determined varied between the laboratories and the test work was conducted in a parallel, the conditions for the locked cycle test work varied between laboratories.

Optimized conditions developed in Phase 3 G&T test work were used to carry out locked cycle tests. The conditions were:

- Rougher pH 10
- Primary grind size P_{80} 150 µm
- Primary collector additions 10 to 15 g/t 3418A
- Rougher flotation time was 8 and 10 minutes
- Cleaner pH 10 for the first series and pH 11 for the second series
- Regrind size P_{80} from 27 to 44 µm
- Cleaner flotation time was 11 to 16 minutes
- Cleaner collector addition was 2 to 9 g/t 3418A.

A pH of 11 gave the best flotation conditions to produce a saleable copper concentrate. Figure 13-4 shows copper grade recovery data for all composites. A concentrate copper grade of 14% Cu was achieved for all composites at a pH 10. Further tests demonstrated that at pH 11 concentrate copper grade was improved to 25% Cu for hypogene and 34% Cu for supergene. Copper recovery for the hypogene was 90% Cu while the supergene produced a slightly lower copper recovery of 85% Cu. The mixed composite produced a concentrate copper grade of 30% Cu with a lower copper recovery of 70% Cu.

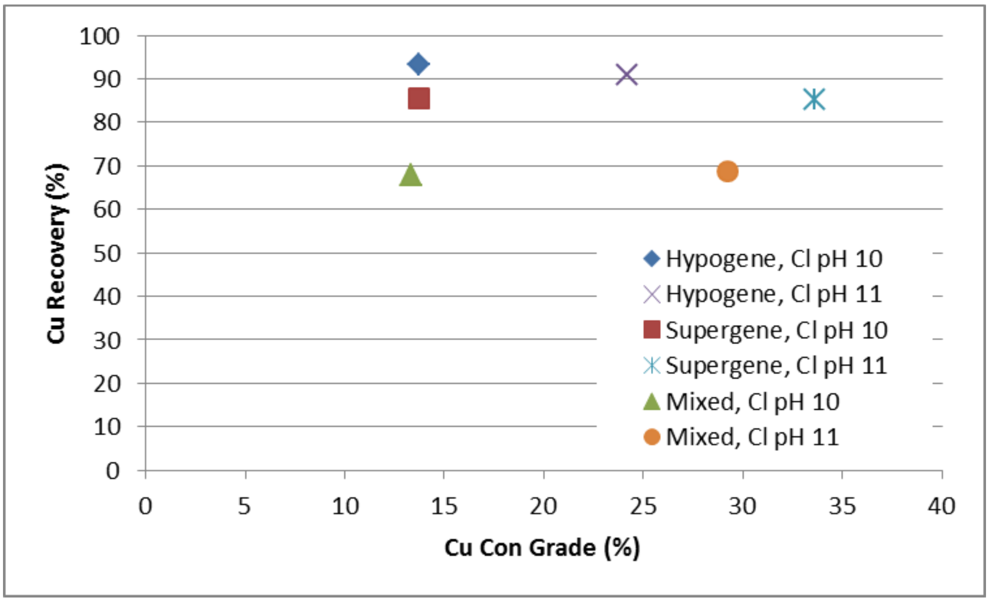

Figure 13-4 – Lock cycle test for domain composites with cleaner pH 10 and 11

Table 13-6 shows elemental analysis of copper concentrate for the hypogene, supergene and mixed composites. Concentrate copper grades were 24% Cu for hypogene, 34% for supergene and 29%Cu for mixed composite. Gold grades were 2.9 g/t for hypogene, 3.8 g/t for supergene and 5.6 g/t for the mixed composite. Mercury, Arsenic, and Fluorine assays are below the penalty levels for all composites. Chlorine assays were not available. It is recommended to assay for chlorides in future tests.

Table 13-6 – Copper concentrate chemical assay for main elements

Elements	Hypogene	Supergene	Mixed
Aluminium (%)	0.66	0.39	0.53
Antimony (%)	0.002	0.001	0.001
Arsenic (g/t)	12	37	10
Bismuth (g/t)	34	19	34
Chlorine (g/t)	not available	not available	not available
Cobalt (g/t)	136	150	130
Copper (%)	24.2	33.6	29.2
Fluorine (g/t)	43	35	41
Gold (g/t)	2.9	3.8	5.6
Lead (%t)	<0.01	<0.01	0.01
Magnesium (%)	0.044	0.012	0.025
Mercury (g/t)	<1	<1	<1
Molybdenum (%)	0.04	0.09	0.05
Silver (g/t)	30	44	32
Zinc (%)	0.01	0.01	0.03

Optimized conditions developed in the Phase 3 Plenge test work were used to carry out locked cycle tests. The conditions are as follows:

- Rougher pH 10
- Primary grind size P_{80} 150 µm
- Primary collector additions 15 g/t 3302 and 15 g/t 3418A added to grinding
- Rougher flotation time was 10 minutes
- Cleaner pH 10
- Regrind size P_{80} not specified, only indicated 2 minutes grinding
- Cleaner flotation time was 15 minutes.

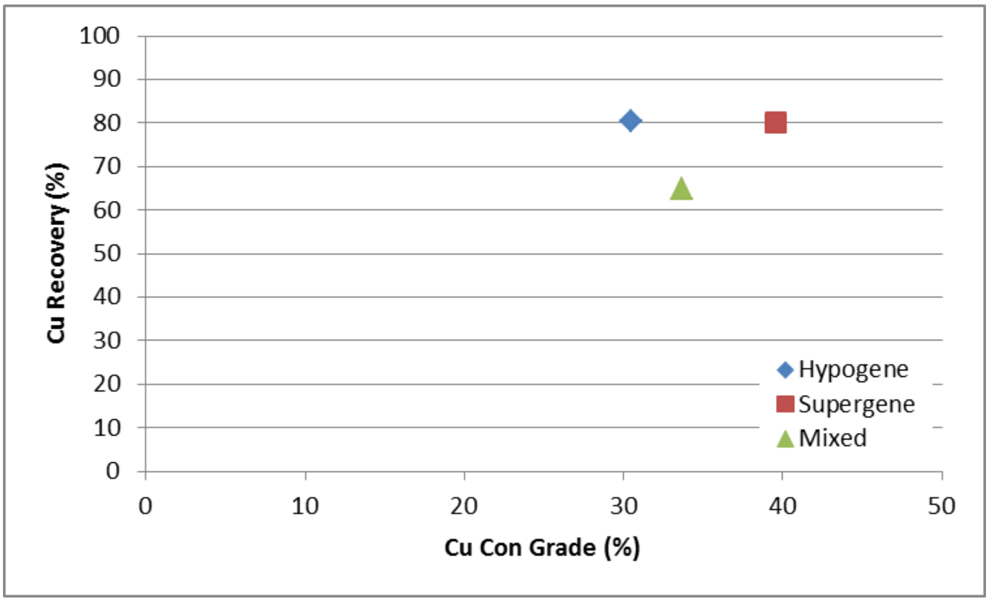

Figure 13-5 – Locked cycle test Cu recovery vs. Cu concentrate head grade

Locked cycle tests results by Plenge as depicted in Figure 13-5 were lower compared to tests conducted by G&T. Metallurgical results from G&T were used for design in process design criteria due to the flotation parameters more closely matching the chosen design.

Table 13-7 shows comparison of the Phase 3 locked cycle test results between G&T and Plenge.

Table 13-7 – Comparison of Phase 3 LCT results for G&T and Plenge

Sample	Cu recovery %	Cu grade %	pH
G&T			
ZEFC-01	93.2	13.9	10
ZEFC-02	85.5	13.8	10
ZEFC-03	67.8	13.3	10
ZEFC-01	91.0	24.2	11
ZEFC-02	85.1	33.8	11

Sample	Cu recovery %	Cu grade %	pH
ZEFC-03	68.7	29.4	11
Plenge			
ZEFC-01	85.6	29.8	10
ZEFC-02	81.3	38.5	10
ZEFC-04	67.3	31.8	10

Locked cycle tests conducted by G&T at pH 10 reported a copper grade in the final concentrate (Cleaner 03 concentrate) below typical commercial grades (<25% Cu) and recoveries higher than 85% Cu for ZFC01 and ZFC02 except ZFC03 with 67.8% Cu as this composite has a high copper oxide (11.4% CuT) content. This LCT results showed that the pyrite was not rejected from the cleaner circuit generating the dilution of copper concentrate. To correct this result an additional locked cycle test was performed at pH 11 in the cleaner flotation stages.

Locked cycle test conducted by G&T at a pH 11 in the cleaners produced a copper grade in the final concentrate (Cleaner 03 concentrate) with higher copper grades (>24% Cu), recovery was 91% Cu for ZFC01, 85.1% Cu for ZFC02 and 68.7% Cu for ZFC03, low copper recovery in ZFC03 was due to high copper oxide (11.4% CuT) in this composite. Pyrite rejection in the cleaner circuit improved at a pH 11. The stability in LCT for ZFC01 composite in the two last cycles (IV and V) was good for all elements and mass. The stability in LCT for ZFC02 and ZFC03 composite in the two last cycles (IV and V) displayed regular and low stability for mass and all elements.

Finally locked cycle tests conducted at Plenge laboratories showed that copper grade in the final concentrate (Cleaner 03 concentrate) reported >25% Cu. Recovery was higher than 80% Cu for ZFC01 and ZFC02 except ZFC03 with 67.3% Cu due to this composite having a high copper oxide (21.6% CuT) content. Pyrite rejection was not a problem as reagent A-3302 is selective against iron sulfide minerals. G&T used A-3418 15 g/t in the roughers and in cleaners between 5-9g/t while Plenge used A-3302 and A-3418 at 15g/t in the rougher flotation stage.

Additional locked cycle tests were performed during the Phase 5 test work to assess flotation results with using the updated design parameters. The test work used four composite samples with two samples each representing the supergene and hypogene mineral zones. The conditions for the locked cycle tests were the optimum conditions determined for rougher and cleaner test work previously described. The conditions are summarised in Table 13-8 below.

Table 13-8 – Conditions and results from three-stage cleaner LCTs

Description	ZEFC-01 (supergene)	ZEFC-02 (hypogene)	ZEFC-03 (supergene)	ZEFC-04 (hypogene)
pH	9	9	9	9
Primary grind P_{80} (µm)	125	125	125	125
Regrind P_{80} (µm)	40	40	40	40
Collector AP-9950 (g/t)	30	30	30	30
Rougher time (min)	10	10	10	10
Cleaner time (min)	15	15	15	15
Cu recovery (%)	81.9	88.9	86.7	87.2
Au recovery (%)	59.4	41.2	56	42.2
Cu concentrate grade (%)	29.5	26.3	30.3	26.8
Cu head grade (%)	0.65	0.37	0.71	0.32

Supergene copper recoveries were 81.9% and 86.7% to a final concentrate grade of approximately 30%. Hypogene copper recoveries were higher with 87.2% and 88.9% to a lower final grade of approximately 26%. The high recoveries for the hypogene were achieved despite low head grades of 0.37 and 0.32% Cu.

Two additional locked cycle tests were conducted to determine whether two stages of cleaning were appropriate for the hypogene and supergene samples. The optimal conditions for rougher, cleaner and cleaner-scavenger stages were used. The locked cycle test was performed with the fixed conditions of collector AP-9950, primary grind P_{80} of 125 µm, regrind P_{80} of 40 µm, and the pH was adjusted by addition of lime.

Copper and gold recoveries for the two-stage cleaner tests were slightly lower than the three-stage tests with recoveries of 80.1% and 61.3% for the supergene and 86% and 38.6%, respectively, for the hypogene. The two-stage cleaner tests resulted in similar concentrate grades to the three-stage tests with 29.5% and 26.3% Cu for both supergene and hypogene. Due to lower recoveries at similar concentrate grades, changing the circuit to two-stage cleaning was not adopted and design remained at three cleaner stages.

13.1.4.5 Sample aging

Sample aging was performed during the Phase 5 test work on three composite samples (ZEFC-01, ZEFC-02 and ZEFC-03) with the samples tested being crushed to -2 mm and spread out on a tray, maximizing the aging effect. Although, due to the test parameters, it is likely that any aging would be slower and with less effect than these tests, the tests provide an indication of the aging potential of the samples.

The aging tests were performed on two samples, which were aged for 45 days and one sample which was aged for 30 days. Rougher flotation was performed for each of the samples before and after aging to determine whether flotation performance was affected.

The test work found that across most samples higher activation of pyrite decreased the concentrate copper grades. The rougher copper recoveries and kinetics were similar for the aged samples and the fresh samples.

13.1.4.6 Flotation water analysis

Water source analysis was performed with five different water sources being assessed during Phase 5 Test work:

1. Tap water ZEWS-00
2. Sea water ZEWS-01
3. Crop water (from Dren El Chorro) ZEWS-02
4. Majes river water (from Huatiapa) ZEWS-03
5. Majes I irrigation aquifer water ZEWS-04.

Rougher test work was performed with six composite samples (ZEFC-01 to 06), and four of the five water types. The test work showed that similar rougher recoveries were achieved from all water samples tested on both supergene and hypogene ores. For the mixed composite, tap water had the highest recovery and both sea water and crop water had similar but lower recoveries.

In the supergene samples, concentrate grades were higher for tap water than for other water types tested. This trend was not apparent for hypogene samples.

For samples ZEFC-02, 04 and 05 crop water produced lower grades than the other samples. Pre-treatment of the crop water was investigated using the following methods:

- Pre-oxidation
- Activated carbon
- Microfiltration
- Lime addition.

The treatment of the crop water did not result in any improvement in flotation performance. In general there was no repeatable problem with the water in any of the tests with the initial poor results unable to be replicated in subsequent test work.

No rougher test work was performed on the Majes aquifer water. Some rougher test work was performed as part of the open circuit cleaner test campaign. The cleaner test work was performed on three composite samples, four different water samples. For this test work Majes river water (from Huatiapa) was not utilised.

The rougher results showed that rougher recovery was highest with seawater by 2% for both supergene and hypogene. Recovery for Majes water was 2% lower than tap water for both zones. Cleaner recovery for sea water was high although the final concentrate grades produced were 18.8% to 20.3%Cu. This was due to high pyrite recovery in the cleaner circuit.

Cleaner copper recoveries were similar for all water types, except for crop water which showed a lower recovery in both tests. The concentrate copper grade for crop water was much higher for hypogene than for the other samples.

Locked cycle test work was performed comparing tap water to Majes Aquifer water on a supergene and hypogene composite. This test work showed that for both composites Majes water produced higher copper concentrate grade than tap water. Copper recovery for the Majes water was lower for both samples. Both sets samples had similar cleaner recoveries, with rougher recovery providing the main difference.

It is recommended that further flotation test work is undertaken in the next phase of development to better assess the impact of the water source chosen for flotation.

13.1.5 Materials handling

Test work was undertaken in the Phase 5 test work by Jenike and Johanson using three head composite samples from the supergene, hypogene and oxide zones. These samples were subjected to material handling test work which included:

- Cohesive strength
- Compressibility
- Wall friction
- Minimum hopper opening
- Chute test
- Angle of repose/ angle of drawdown.

The sample details are shown in Table 13-9.

Table 13-9 – Samples used for materials handling test work by Jenike and Johanson

Sample	Mineral zone	Moisture (%)	Particle density (kg/m^3)	Size (P_{80} mm)
ZEFC-01	Supergene	1.20	2.8	0.81
ZEFC-02	Hypogene	0.90	2.79	0.80
ZEFC-04	Oxide	2.10	2.77	19.0

The test work found that:

- All samples were found to be classified as cohesive material at 5% moisture based on the cohesive strength and consolidating pressure. At 1.5% moisture, the samples were on the limit between cohesive and free-flowing materials.
- Compressibility test work showed that all three samples should be treated as compressible materials in stockpiles, silos or hoppers. The bulk density of the samples varied considerably with the consolidating pressure at the conditions tested.
- The samples were found to have a tendency to form an arch due to their cohesiveness and a strong tendency to form rat-holes when handled in silos, hoppers and transfer chutes at the moisture contents and instantaneous flow conditions tested.

Angle of repose and angle of drawdown tests were performed for the three samples. The tests found that for the samples, the angle of repose varied between approx. 43° and 46° for the moisture content range of 1.5% and 5.0%, respectively. This compares to the concentrator coarse ore stockpile design angle of repose of 35° which was based benchmarking of similar operations.

The angle of drawdown determined from the tests varied between 60° and 75° for the moisture content of 1.5%, and between 70° and 90° for the moisture content of 5.0%. This compares to the concentrator coarse ore stockpile design angle of drawdown of 65%.

All samples were found to fall within the standard design parameters for copper concentrators so no modifications were made to the design conditions based on the materials handling tests.

13.1.6 Thickening, rheology, filtration

Concentrate and tailings thickening test work was conducted for both the copper concentrate and the tailings during the Phase 5 test work. Static thickening test work was performed on the concentrate samples, whilst dynamic thickening test work was performed on the tailings samples. The test work was performed by Pocock on samples produced from LCT of four different composites (Table 13-10).

Table 13-10 – Samples used for settling test work by Pocock

Sample	Sample origin	LCT product	Mineral zone
ZESS-01	ZEFC-01	Final tailings	Supergene
ZESS-02	ZEFC-01+03	Cleaner 3 concentrate	Supergene
ZESS-03	ZEFC-04	Final tailings	Hypogene
ZESS-04	ZEFC-02+04	Cleaner 3 concentrate	Hypogene

The results confirmed that a medium to high molecular weight anionic polyacrylamide flocculant (Hychem AF 304) was suitable for thickening both the tailings and copper concentrate at dosages of between 20 and 30 g/t required for the tailings samples and 20 g/t required for both concentrate samples.

Solids loadings for thickener design were dependent on the thickening technology used. For a conventional thickener the tailings solids loadings were 0.28 t/m^2/h and for a high rate thickener the hydraulic rise rate results ranged from 4.7 to 5.7 m^3/m^2.h. The thickening rates for high rate thickening give similar thickener diameters to the design value of 1 t/m^2/h which was based on the Phase 4 test work.

The tailings thickener underflow densities were 55-59% for supergene and 66-70% for hypogene. This compares to a design value of 70% and nominal underflow density of 60%.

The solids loading determined from test work on the concentrate samples, was 0.3 t/m^2/h for both samples. This compares to the design value of 0.2 t/m^2/h which was based on benchmark operations. The underflow densities for the concentrate ranged from 66 to 71%w/w solids compared to the design and nominal values of 65 and 60% w/w solids.

The tailings thickener underflow densities for the test work were based on the rheology data from the dynamic thickener test work. The work showed no issues with pumping the underflow at design densities.

Overflow clarity was 40-50 nephelometric turbidity units (NTU) for the supergene tailings and 20-40 NTU for the hypogene tailings sample.

Pressure filtration tests were performed for the two concentrate samples. The results confirmed that a cake moisture of 6.8 to 7.8% could be achieved in a standard recess plate type filter with air blow used as the only drying method. The dry cake density varied between the supergene and hypogene samples recording 1,175 and 696 kg/m^3 respectively.

The supergene sample had a higher filtration rate than the hypogene sample with filtration rates of 279.2 kg/m^2.h and 165.9 kg/m^2.h respectively. Both of the filtration rates were lower than the design value of 400 kg/m^2.h which was based on benchmark data. Further work is required to confirm the filter sizing.

13.1.7 Tailings cycloning

Tailings cyclone test work was conducted to determine the amenability of the tailings to produce TMF embankment construction material from cyclone underflow. To be suitable for TMF construction the tailings underflow must have a fines content of <15% -75 um and an underflow solids density of >65%w/w is required.

Preliminary test work was conducted for supergene composite ZEFC-09 which was a sample containing rougher tailing and reground rougher concentrates. No tests produced the required proportion of fines in the underflow.

Additional test work was conducted on two samples which contained rougher and cleaner tailings. The two samples were, supergene tailings composite ZEFC-07 and hypogene tailings composite ZEFC-08.The underflow for the supergene sample contained 10.6% passing 75 µm and the hypogene underflow contained 17.1% passing 75 µm. The yield of sand from each test was 18% and 24% respectively. The underflow density for both samples was approximately 70%.The yield of sand produced in all tests was lower than the design value of 25%. The design value was a conservative value based on both benchmark and supplier modelling. It is important to note that the cyclone selected to produce the sand would include a cyclowash system which minimises fines in the underflow through injecting clean water into the lower cone. This also allows the sand yield to be increased.

It is recommended that additional cyclone tests be conducted on tailings composites utilizing a cyclone with cyclowash to confirm the feed dilution rates, sand yield and quality.

13.1.8 Concentrate characteristics (transportable moisture limit TML, autopyrolysis)

Transportable moisture limit (TML) and autopyrolysis tests have not been conducted to date.

It is recommended to conduct these tests during the Feasibility Study stage to include these parameters in the process design criteria.

These parameters are required to identify any issues with concentrate transportation and to ensure that filter design moisture is below the TML value.

13.1.9 Tailing characteristics (physical, geochemical)

The tailings physical and geochemical characteristics are described in Section 18.2.5.

13.2 Recovery Estimate Basis

13.2.1 Design basis

An overall copper recovery of 89% and a final concentrate grade of 28% Cu were used for design. Copper recovery and grade were based on locked cycle tests results conducted by G&T in the Phase 3 Metallurgical Test work Program which were confirmed for the design conditions by the Phase 5 locked cycle tests.

13.2.2 Geometallurgical model

A geometallurgical analysis of the Phase 3 and Phase 5 variability flotation test work results was conducted by Transmin to determine whether the recovery of copper could be predicted utilizing assay, lithology, alteration or other data. This analysis involved studying a series of correlations to determine whether relationships could be found between sample copper recovery and the following independent variables:

- Lithology
- Alteration
- Spatial location
- Head assays

The analysis found that neither lithology nor alteration play as an important role in rock quality or flotation response as oxidation and enrichment. Consequently many of the variability samples that had unique lithology and alteration types had mixed mineralization types. Therefore the flotation response was erratic within each rock type.

It was also determined that the spatial distribution had no direct effect on recovery. Rather, recovery is affected by geological events that have caused some copper minerals to have higher flotation recovery than others.

The main correlations found during the analysis were with sample head assays. From this analysis, the following groups were found useful for copper recovery prediction:

- Low sulfur
  Lo_S, or low S/Fe ratio, for S/Fe<0.20
- Low dilute sulfuric acid soluble copper
  Lo_CuSS, or low CuSS/CuT ratio, for CuSS/CuT<15%
- Mid-range dilute sulfuric acid soluble copper
  Mid_CuSS, or mid CuSS/CuT ratio, for CuSS/CuT: 15%-30%
- High dilute sulfuric acid soluble copper
  Hi_CuSS, or low CuSS/CuT ratio, for CuSS/CuT>=30%
- Low cyanide soluble copper
  Lo_CuCn, or low CuCn/CuT ratio, for CuCn/CuT<30%

These groups were subsequently labeled “geomet flotation domains”.

13.2.3 Recovery estimate summary

The recovery and concentrate models for each of the geometallurgical flotation domains are summarised below (Table 13-11). For the overall copper recovery calculation, a cleaner stage copper recovery loss of 2% has been assumed based on the locked cycle test work.

Table 13-11 – Recovery and grade model algorithms for geomet flotation domains

Domain name	Algorithm	Cu conc grade (%)	Au conc. grade (g/t)	Cu rec. final (%)	Au rec. final (%)
Lo_S	S/Fe < 0.2	0	0	0	0
Lo_CuCn	CuCn/CuT <= 0.3	28	by calculation	90.5	56
Lo_CuSS	CuSS/CuT <= 0.15	37		89	52
Mid_CuSS	0.15 < CuSS/CuT <= 0.3	34		84	55
Hi_CuSS	Rest	32		77	52

13.3 Sample Representativity

Samples for were selected to maintain representativity across:

- lithology
- alteration
- mineral zone
- grade
- spatial distribution.

To ensure representative sample selection a number of inputs into the selection process were utilised. These included:

- Block model
- Mining plan
- Drilling database

Following selection of the samples, results were processed through the laboratories, and reconciliation of the representativity was performed. This reconciliation found:

- In general, all of the major rock types, regions and mineralization types have been sampled and tested.
- Not all of the domains have an adequate number of samples, so more variability testing is recommended to reduce risk:
  - SG Phase 5 samples were overrepresented in the flotation program, in particular the mid and high CuSS samples.
  - No samples from Victoria deposit were evaluated.
  - For further studies it is recommended to select more flotation test samples from the mixed mineral zone and from Years 1 to 5 of production.
- Further flotation testwork should include sampling based on geomet domains rather than lithology and alteration.
- Further comminution testwork should focus on rock quality, fracturing, depth and weathering within the major mineral zones rather than lithology and alteration.

13.4 Significant Processing Factors (affecting economic extraction)

13.4.1 Comminution

Table 13-12 shows the preliminary assessment of comminution data (Phases 1 and 2) by Ausenco. During development of the preliminary design criteria, some issues were identified:

- No rod mill data are currently available and the Bond rod mill work index (BRWi) was assumed to be 20% higher than the Bond ball mill work index (BBWi).

- The crushing work index (CWi) data were considered by Phillips LLC to be biased low due to the core pieces provided for the tests being smaller than generally desired. Previous design criteria used the 75th percentile of all the available crushing work index data, including oxide, without any correction factor being applied.
- The average SG was used, since the use of a 75th percentile SG value combined with a 75th percentile DWi can bias the SAG mill power calculation to a higher value.

Table 13-12 – Ausenco preliminary comminution design criteria

Parameter	Unit	Ausenco Preliminary Assessment
SMC Drop Weight Index (DWi)	kWh/m^3	8.7
JK Axb	-	30.5
Specific Gravity (SG)	t/m^3	2.59
Crusher Work Index (CWi)	kWh/t	23
Rod mill Work Index (RWi)	kWh/t	14.6
Ball mill Work Index (BWi)	kWh/t	12.2
Abrasion Index (Ai)	-	0.32
Cyclone Overflow Grind Size (P_{80})	µm	150

The data were statistically analysed to identify the significant drivers of comminution performance. The analysis of the comminution database (Phases 1, 2 and 5) identified the following:

- There was a significant difference between hypogene and supergene DWi
- There was no significant difference between hypogene and supergene BWi
- There was an increase of DWi with depth, however the BWi differences with depth are still negligible:
  - The hypogene DWi increases below 250 m
  - The overall DWi increases with depth, most likley because the proportion of hypogene also increases with depth
- No significant differences between lithologies and/or alterations were determined

tAusenco has used its in-house comminution model (Ausgrind) to calculate the SAG and ball mill specific energy requirements (Ecs) in SABC circuit configuration for every sample in the comminution database (Phases 1,2 and 5). The 75th percentile of the resulting Ecs values for SAG and ball mill respectively, were determined to define the base case design criteria and the design comminution parameters (DWi and BWi) to be used as the basis for sizing the required mills.

This approach was adopted to mitigate the risks associated with taking the 75th percentiles of individual parameters when there is a combination of low competency (DWi) with moderate BWi, which could result in underestimating the ball mill power requirements (or *vice versa*). The resulting design criteria ore properties are presented in Table 13-13.

Table 13-13 – PFS comminution design criteria

Parameter	Unit	Ausenco Design Criteria	P_{75}	Average
SMC Drop Weight Index (DWi)	kWh/m^3	8.4	8.6	6.7
JK Axb	-	30.6	30.1	38.6
Specific Gravity (SG)[1]	t/m^3	2.6	2.7	2.6
Crusher Work Index (CWi)[2]	kWh/t	22.9	22.9	18.1
Rod mill Work Index (RWi)[3]	kWh/t	15.6	14.6	13.2
Ball mill Work Index (BWi)	kWh/t	13.0	12.2	11.0
Abrasion Index (Ai)	-	0.32	0.32	0.22
Primary Grind Size	µm	150	N/A	N/A

The design criteria ore properties were used to calculate the specific energy requirements for both SABC and SACB circuit configuration.

The circuit specific energy requirements (Ecs), calculated using Ausgrind for a final product of 150 microns, are presented in Table 13-14.

Table 13-14 – Calculated specific energy

Circuit	Specific Energy, Ecs (kWh/t)		
	SAG mill	Ball mill	Total
SABC	9.7	5.6	15.3

13.4.2 Flotation

The flotation circuit design was based on a circuit capacity of 20.1 Mt/y. The copper design head grade of 0.85% copper was based on maximum monthly average feed grade in Year 1.

A copper recovery of 89% and a final concentrate grade of 28% Cu were used for design. Copper recovery and grade were based on locked cycle tests results conducted in Phase 3 Metallurgical Testwork Program.

The design rougher concentrate mass pull of 11.4% was based on variability on rougher flotation testwork completed in Phase 3 which was confirmed by Phase 5 test work.

The selection of the flotation cells for the modified and new flotation circuit duties was based on the design parameters shown in Table 13-15.

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1 The average SG was used, since the use of a 75th percentile SG value combined with a 75th DWi can bias the SAG mill power calculation to a higher value.

2 As discussed above, the CWi data were considered by Phillips LLC to be biased low. Ausenco has used an inferred CWi.

3 No rod mill data are currently available and the rod mill work index (RWi) was assumed to be 20% higher than the BWi.

Table 13-15 – Carrying rate and froth lip loading parameters for flotation cell design

Parameter	Units	Value
Froth lip loading (design)		
Rougher	t/h/m	1.5
Cleaner 1	t/h/m	1.0
Cleaner 2	t/h/m	1.0
Cleaner 3	t/h/m	2.0
Cleaner Scavenger	t/h/m	1.0
Froth carrying rate (design)		
Rougher	$t/h/m^2$	1.5
Cleaner 1	$t/h/m^2$	1.5
Cleaner 2	$t/h/m^2$	1.5
Cleaner 3	$t/h/m^2$	2.3
Cleaner scavenger	$t/h/m^2$	1.0

Table 13-16 summarises laboratory and design residence time for all flotation stages. Laboratory residence times are based on locked cycle tests conducted by Plenge in Phase 3 and confirmed during Phase 5 test work. A factor of 2.5 was used to scale up flotation residence time from the laboratory to the plant scale. Note that the residence times used in G&T's locked cycle tests were similar to Plenge except Cleaner 2 and Cleaner Scavenger, were slightly different (i.e. 4 and 2 min respectively).

Table 13-16 – Laboratory flotation residence time

Flotation Stage	Laboratory (min)	Plant (min)
Rougher	10	25
Cleaner 1	5	12.5
Cleaner 2	3	7.5
Cleaner 3	3	7.5
Cleaner Scavenger	4	10

13.4.2.1 Concentrate filtration

A specific filtration rate of 0.4 $t.m^2/h$ was used for the design of the concentrate filter. The design value is based on benchmarking of similar operations treating sulfide copper minerals using a horizontal plate pressure filter. Confirmatory test work is required as filtration results from Phase 5 were not in line with benchmark operations. It is anticipated that the filter cake moisture will be 9.5% w/w which is similar to the benchmarked operations.

13.4.2.2 Variable throughput

Following design for the concentrator being completed, analysis of the Phase 5 comminution test work identified that there were significant differences in the competency of the supergene and hypogene zones. This allowed the concentrator to process increased tonnage early in the mine life by maximising utilisation of the installed mill power for the less competent supergene ore. To exploit the differences in competency, the mining schedule was updated with the different throughput estimates for supergene and hypogene zones.

Following update of the mining schedule, a significant increase in throughput was realised in the first years of operation. Peak through put was scheduled in Year 3 at 63.7 kt/d (Figure 13-6). Due to high copper head grades during ramp-up, maximum copper production was realised in the 2nd half of year one. The copper production rate in this semester was 17 t/h of copper due to the high head grade of 0.96%Cu.

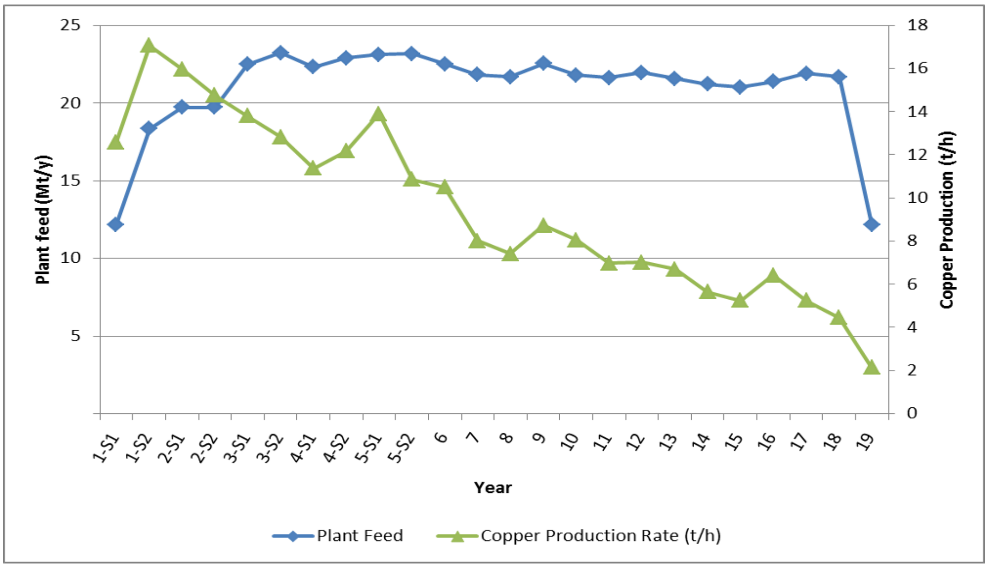


Figure 13-6 – Variable throughput production schedule

As the concentrator had been designed at a throughput of 55 kt/d at a maximum head grade of 0.85%Cu which corresponds to a copper production rate of 17.9 t/h, a check was performed to determine whether there was sufficient design factor to run at the maximum rates presented by the variable throughput production schedule. For the assessment, an additional 10% was added to the average head grade to allow for variability of the feed to the plant. This increased the design copper production rate for the variable throughput case to 20.5 t/h.

The maximum capacity of the major equipment was assessed to determine whether they presented any bottlenecks which prevented achievement of the maximum throughput.

The analysis found that due to design factors and the granularity of equipment selection, the maximum rates presented by the variable throughput production schedule could be achieved through the crushing, milling and flotation circuits with changes to the following equipment:

- Additional length added to the ball mills to allow full power draw from the selected motors
- One additional Cleaner 3 cell
- Additional concentrate filter and increased size of existing filter
- TMF starter embankment and fill curve require more assessment due to the higher rate of rise.

Other major equipment was deemed fit for the maximum throughput by either utilisation of excess capacity, minor modification to parameters (ie speeding up the drive or change of impellor) or relaxation of conservative design parameters.

Following the analysis, the design for the concentrator was updated to include the additional equipment required to achieve the variable throughput production schedule.

13.5 Product Analysis

Table 13-17 shows chemical characterization of the final concentrate of the locked cycle test performed by G&T. Mercury, arsenic, antimony, cadmium, lead, zinc and fluorine assays are below the penalty levels for all composites.

Table 13-17 – Concentrate analysis for main elements

Elements	Hypogene	Supergene	Mixed	Penalty limits
Aluminium (%)	0	0	0	3
Antimony (g/t)	7	25	18	500
Arsenic (g/t)	8	178	13	250
Bismuth (g/t)	212	245	232	
Cadmium (g/t)	12.2	12.6	11.7	200
Chlorine (g/t)	<10	<10	<10	
Cobalt (g/t)	54	71	82	
Copper (%)	30.5	39.6	33.7	
Fluorine (g/t)	43	35	41	400
Gold (g/t)	4.70	5.21	2.95	
Lead (%)	0.01	0.02	0.02	1
Magnesium (%)	0.13	0.10	0.09	
Mercury (g/t)	0.36	0.47	0.55	30
Molybdenum (g/t)	650	1962	1063	
Nickel (g/t)	31	51	43	
Silver (g/t)	24	37	24	
Zinc (%)	0.04	0.05	0.07	3

13.6 Interpretation and Conclusions

The following conclusions arise from the information provided in the previous sections:

- SAG and ball mill specific energy requirements (Ecs) were calculated in SABC circuit configuration for every sample in the comminution database based on Ausenco's grinding model. The 75th percentile of the resulting Ecs values for SAG and ball mill respectively were determined to define the design comminution parameters (DWi and BWi) to be used as the basis for sizing the required mills. This approach was adopted to mitigate the risks associated with taking the 75th percentiles of individual parameters when there is a combination of low competency (DWi) with moderate BWi, which could result in underestimating the ball mill power requirements (or *vice versa*)

- Comminution testing has shown that all three domain composites are moderately hard and abrasive but very competent leading to much higher specific energy; hence, much more power required for SAG milling than ball milling. All composites are of similar hardness and abrasiveness with similar Bond ball mill work indices (BWis) but the hypogene composite is more competent than the others with a significantly higher drop weight index (DWi). As higher proportions of supergene and mixed ores are mined in the early years of the Project, this presents an opportunity to increase throughput in those early years
- Table 13-18 summarises 75th percentile specific energy for the Hypogene, Supergene and Mixed ore types.

Table 13-18 – Summary of 75th percentile specific energy requirements

	Mineral Zone	Specific Energy (kWh/t)
SAG mill	Supergene	9.2
	Hypogene	11.0
	Mixed	8.7
Ball mill	Supergene	5.1
	Hypogene	4.8
	Mixed	4.4
Total	Supergene	14.3
	Hypogene	15.8
	Mixed	13.1

- The comminution parameters derived from specific energy requirements and from other comminution tests for each mineral zone and the LOM blend are presented in Table 13-19.

Table 13-19 – Summary of rock type properties

Parameter	Unit	Supergene	Hypogene	Mixed	LOM
Axb	-	34.6	28.5	36.9	31.2
DWi	kWh/m^3	7.5	9.3	7.0	8.4
SG	t/m^3	2.60	2.65	2.59	2.62
CWi	kWh/t	20.2	24.6	19.0	22.4
BRWi	kWh/t	15.3	15.6	14.3	14.9
BBWi	kWh/t	12.8	13.0	11.9	12.4

- Additional analysis of the comminution database identified the following:
  - A significant difference between hypogene and supergene DWi
  - No significant difference between hypogene and supergene BWi was detected
  - There is an increase of DWi with depth, however the BWi differences with depth are still negligible

- There are no significant differences between lithologies and/or alterations

- An optimum primary grind size P_{80} of 150 µm was selected for all ore types
- Copper and gold recoveries and concentrate grades from locked cycle tests are shown in Table 13-20:

Table 13-20 – Summary locked cycle test results

Parameter	Units	Supergene	Hypogene	Mixed
Copper Recovery	%	85	91	69
Copper Concentrate Grade	% Cu	34	24	29
Gold Recovery	%	46	51	36

- Flotation recovery of copper in the mixed composites is lower than hypogene or supergene due to higher copper oxide content (11.4% of total copper present), which also led to lower recovery of gold, due to the mineralogical association of gold with copper. The lower concentrate copper grade in the hypogene composite is because the dominant copper mineral is chalcopyrite, whereas there is more chalcocite in mixed and supergene composites. Regrinding of rougher, scavenger and cleaner-scavenger concentrates is required to achieve optimum copper recoveries with sufficient pyrite rejection to achieve concentrate copper grades acceptable for marketing
- A copper recovery of 89% and a final concentrate grade of 28% Cu were used for design. Copper recovery and grade were based on locked cycle tests results conducted in Phase 3 Metallurgical Testwork Program. Design margin for higher recoveries was allowed through designing the concentrator for a maximum head grade of 1.01% Cu
- Design gold recovery for the circuit was approximately 50% and dependent on gold head grades
- A moderate mass recovery (11% w/w) in the rougher flotation circuit under typical processing conditions was achieved for both main ore types
- The grind size rougher flotation testwork indicates an optimum primary grind size of 150 µm
- Regrinding to a nominal size of 40 µm represents the optimum regrind size based on the available data
- Three-stage cleaning at elevated pH, in conventional flotation cells is expected to be adequate to achieve target concentrate grades and recoveries
- Elemental analysis of the copper concentrate indicates low penalty element levels. The copper concentrate contains gold concentrations and should therefore be relatively easy to market.

13.7 Recommendations

The following recommendations arise from the information provided in the previous sections:

- Conduct further comminution testwork to confirm comminution characterization of ore types and determine potential gains in plant throughput and assess potential operating and capital cost savings
- Determine if pebble crushing is required in the initial years of the project and whether it can be deferred to later years of the project. Deferral of the pebble crushing circuit is a function of comminution parameters and planned ore supply to the mill

- Analyse likely feed size distribution based on drill and blast modelling used by the mine
- Further laboratory flotation testwork including locked cycle tests and variability tests using Majes 1 aquifer water is recommended to confirm copper recoveries and concentrate grades in the rougher and cleaner flotation circuits and overall flotation circuit
- Additional tests are required during the next study phase to confirm filtration rates using pressure filter and moisture levels at a solids feed density between 60-65% w/w
- Conduct TML and auto-ignition testwork of copper concentrate to confirm benchmarking data used in process design criteria
- A concentrator comprising a single primary crusher feeding a dual-line grinding circuit, each line consisting of a SAG mill and ball mill circuit with pebble crushing; flotation circuit with regrind of rougher, scavenger and cleaner-scavenger concentrates, copper concentrate dewatering and tailings thickening and disposal, is the appropriate treatment route for the Zafranal hypogene, supergene and mixed ores at the proposed throughputs.

14 Mineral Resource Estimates

14.1 Introduction

Geological logging and assay results from 295 core holes totalling 95,618.7 m and 88 reverse circulation (RC) holes totalling 27,041 m completed by CMZ geologists to 16 February 2015 were used as the basis for preparation of three dimensional (3D) wireframe models of geological structures, lithology, alteration, and mineral zonation envelopes. The 3D wireframe models were prepared by Politax S.A. (Politax) in Leapfrog Geo software.

A block model was constructed in Vulcan software with block dimensions of 15 m × 15 m x 12 m high. The main grade variables: total copper (CuT), Au, sulfuric acid soluble copper (CuSS), and cyanide soluble copper (CuCn) were interpolated into the blocks generally by ordinary kriging (OK) in three passes. An inverse distance squared (ID^2) estimator in two passes was used in the Mixed Zone domain and late dykes. Additional grade variables including: Ag, As, Bi, Fe, and, S were interpolated using an inverse distance squared (ID^2) estimator in two passes. Density (SG) was interpolated into density domains using a combination OK in three estimation passes and ID^2 in two passes.

A separate two-pass ordinary kriging (OK) interpolation was run to generate the sample counts and distances used for Mineral Resource classification followed by category smoothing. Blocks were classified based on a combination of factors including the number of holes used for each block, the distance to the nearest composites, and CuT mineral zonation.

14.2 Database

In 2014, CMZ acquired GVMapper (v.3.6) software for online logging which is designed to collect and manage information which is stored in a centralized database resident on a server. This software is a configurable tool for creating, editing, and viewing maps, and drill hole logging data columns. Any of the GVMapper database tables can be exported to any database via ASCII files or spreadsheets and all data on other databases can be imported via ASCII files. The drill hole database provided by CMZ was a database dump from a master database in comma-separated variables (CSV) format.

14.3 Data Entry

All data was provided by the geological team of CMZ, and was reviewed and validated by Politax. The following records were received in hard copy format and were input in a digital format (Microsoft Excel) using a double entry system to minimize possible errors:

- Geological logging record
- Geotechnical logging record
- Density measurement record
- Assay record.

The information on the locations of the drill hole collars, the drill hole deviation surveys, and drill core photographs were received in a digital format.

14.4 Data Reviews

14.4.1 Amec Minproc, 2010

As part of preparation of the Technical Report for the December 2010 Resource Estimate dated 25 February 2011, Amec Minproc, a predecessor company to Amec Foster Wheeler, completed a review of the database and assay sampling QA/QC. It was concluded at the time, that the database provided was thoroughly checked and suitable for use in resource estimation. The review of the analytical QA/QC database indicated that the sample preparation and assaying conducted by AQM was of reliable and consistent quality, and provided accurate and precise information which was suitable for resource estimation and mine planning studies.

14.4.2 Site Visit 2015

During the site visit, the following were observed:

- Geological mapping and interpretation was undertaken using industry-standard methods.
- Core sampling was good. Core logging and re-logging was reasonable. Core storage was well implemented.
- The Zafranal geological data are currently not stored in a relational database. The database for logging is managed using GVMapper. The assay data are compiled in a master MS-Excel spreadsheet, as well as the downhole survey, collars and density results.

Appropriate measures should be implemented to ensure data integrity and should include use of proper database management software to ensure the integrity of the database. It is understood that the selection process for a suitable data management solution is underway.

14.4.3 2014 Drilling Program QA/QC

A total of 3,144 samples were analyzed by ALS Chemex del Perú (2,793 original samples and 351 QC samples) from 39 drill holes. The Zafranal QA/QC program samples comprised 12% of the submitted samples, and included 4% coarse blanks, 4% field duplicates and 4% standard reference materials (SRMs). It was concluded that additional QA/QC measures should be implemented for future drilling programs, but the existing data were acceptable to support confidence classifications of Indicated and Inferred mineral resources.

14.4.4 Geological Interpretation

The geological interpretation is reliable in the area drilled on an 80 m spacing. At depth the spacing is wider and the interpretation is more tentative.

The dyke modelling should be reviewed. During validation, it was observed that the volume of the dykes could be underestimated. The late dykes have been interpreted differently in the Leapfrog wireframing to that of the original interpretation. The late dykes and late quartz diorite are conservative in the Leapfrog wireframes relative to the original interpretation. Additionally, there appears to be smoothing in Leapfrog wireframes, as the thicker intervals in the drill holes are not honoured by the wireframes, thereby reducing the volume of the dykes. The possible underestimation of the dyke volumes (waste) inside the copper mineralized zone may affect the estimate as the internal grade dilution could be smoothed by the lower copper grades in the dykes.

It is recommended that vein density, as another logging requirement, is included for the Victoria zone.

There is a good correlation between the mineral zones that have been defined on the basis of sequential copper ratio and the Leapfrog wireframes for the mineral zone model for all of the mineral zones except the oxides zone. The oxides zone interpretation does not correlate well with respect to the closest drill hole information. It is recommended that for future models, no interpretation for any zone should be extended beyond 120 m from the closest drill hole.

14.5 Re-Logging Program

From August 2014 until February 2015, as part of the process of re-logging and review of the 3D geological models; a process of unification of geological criteria and re-coding was implemented. The re-logging considered the 2013 model and the conceptual models of lithology, alteration, mineralization, and structures in accordance with observations made during the previous mapping of holes, historical geological knowledge of the deposit, and existing reports.

14.6 Geological Modelling

The geological modelling of the Zafranal Main and Victoria zones was updated from the previous model developed by CMZ based on historical drill hole data and new information from the drill holes in the 2014 campaign.

The model describes all of the major primary units within the deposit area, and was developed from drill core logging and geological interpretation in areas not covered by those datasets. The drilling and logging datasets used were collected from 2004 to 2015. North-south cross sections with 50 m spacings were used for control purposes. The models include a simplified surface geological map prepared by CMZ for appropriate surface connection.

Atticus Consulting SAC (Atticus) and Politax defined the criteria and procedures to be used to obtain robust models for the updating of the 2015 model. The information used to interpret, define the contacts and build the solids of the lithology model are, in order of importance: the drill hole intercepts; the surface outcrop map; the interpreted geological map; and the drawn north-south sectional interpretations at 100 m spacing. The drill hole data were composited to 2 m for geological wireframe modelling.

14.6.1 Structural model

The structural model was built using information extracted from the geological map and intercepts in the diamond drill holes. Fault surfaces were recognized and built only when there was more than one point of evidence and displacement could be seen across the fault.

The deposit was divided into 10 structural blocks, six of which contain mineralization. The Zafranal zone was divided into five blocks. The Victoria zone is considered as one block. Displacement has been mapped on all of the faults used in the fault blocks. However, the displacement across the faults that subdivide the Zafranal zone lithologies is difficult to quantify as the contacts are sub-vertical. The distribution of the structural blocks is illustrated in Figure 14-1.

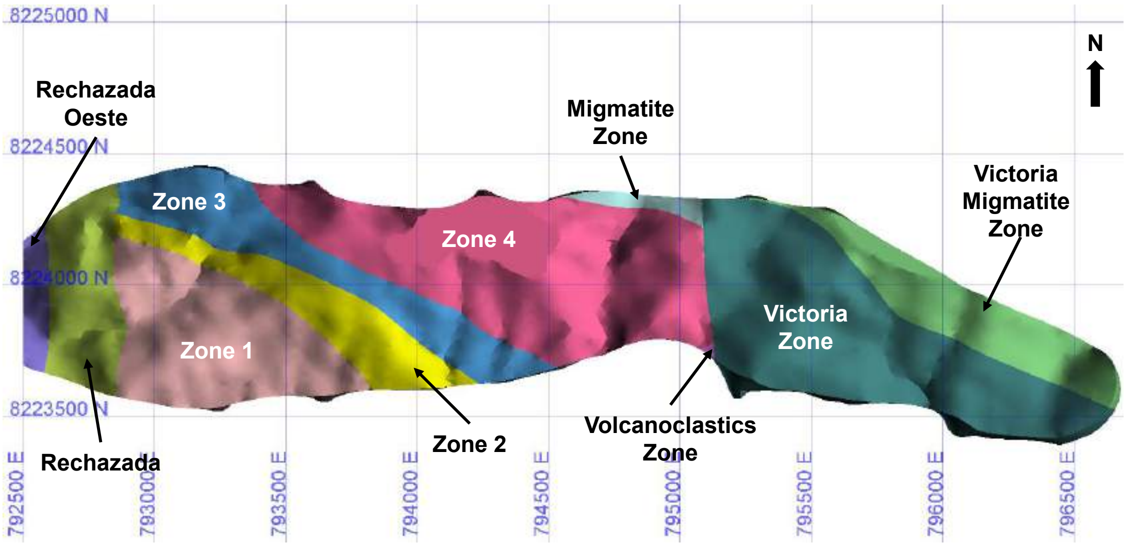


Figure 14-1 –Distribution of the structural blocks (by AFW, 2016)

14.6.2 Lithological wireframe models

Wireframe models of the main lithological units at the Zafranal project were constructed in Leapfrog Geo software from, in order of importance, the drill hole intercepts, the surface outcrop map, the interpreted geological map, and, sectional interpretations based on 100 m spaced vertical sections orientated north to south. All data were provided by the CMZ geological team.

The lithology model was constructed using the event modelling concept; building each contact surface following the geological chronology and cutting each lithological unit out of the model boundary in sequence. A 3D cut-away view of the lithological model is provided in Figure 14-2. The extent of the modelled lithologies encompasses the limits of the known mineralization and lies fully within the range of the block model.

The dykes were modelled separately for each block, as the displacement cannot be accurately depicted from the current density of data, and the margin of error in defining the location of the dykes appears to be less than the predicted displacement.

The intrusive contacts were modelled using a filter set with a minimum interval width of 7.5 m and accepting an internal dilution of 7.5 m. The unmineralized late dykes that were emplaced during a post-mineralization event were modelled to a greater degree of accuracy.

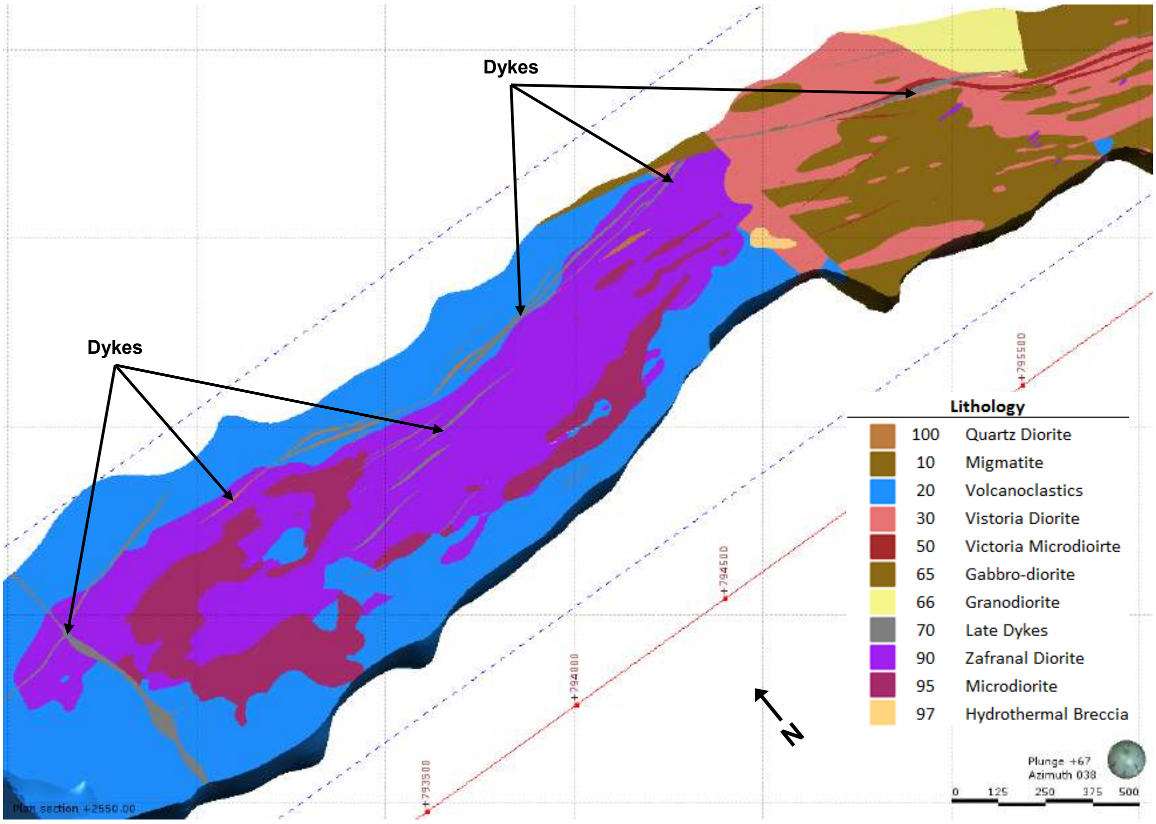


Figure 14-2 – A 3D section of the lithological model at 2500 masl looking northeast (by AFW, 2016)

14.6.3 Mineral zone model

The copper mineral zonation (minzone) model was constructed in Leapfrog Geo primarily from drill hole data. The sectional interpretations provided by CMZ were used only as a guide, as they were based upon different sequential copper thresholds. No outcrop map was provided for the mineral zone model. The definition of the copper mineral zonation is determined from the sequential copper assay ratios. Therefore, only the drill holes that were analyzed for sequential copper were used to construct the model.

A script was applied to the 4 m composite table containing the sequential assay copper results in order to generate the additional data column 'MZ' from which the model was created. Table 14-1 details the definition of the parameters used in the script.

The criteria for defining the mineral zones are based upon work carried out by Transmin Metallurgical Consultants and reviewed by Ausenco and are detailed in Table 14-1. Transitional material, occurring as a sporadic thin zone between the Supergene and the Hypogene, was found to have recoveries similar to hypogene material and hence has not been separated from, but modelled with the hypogene. The presence of material classified as hypogene in the near-surface environment has defined a zone of non-leachable oxide that was not recognized in earlier models.

Table 14-1 –Definition of copper mineral zones

Code	Description	Definition
310	Leached	Sterile material occurring above the top of Supergene or Hypogene
320	Oxide	CuSS/CuT ≥ 55%
325	Non Leachable Oxide	Material defined as 'Hypogene' occurring above the top Supergene
340	Supergene	CuCn/CuT ≥ 30% and CuSS/CuT <55%
330	Mixed	CuSS/CuT < 55% and CuCn/CuT <30% and (CuSS+CuCn)/CuT ≥ 30%
360	Hypogene	(CuSS+CuCn)/CuT < 30%

14.6.4 Alteration model

The data used to define the contacts and build the alteration model solids, in order of importance, are: the drill-hole intercepts and the drawn 100 m spaced sectional interpretations provided by CMZ.

The alteration model was constructed assuming each alteration assemblage as a separate input. Only the major faults were used to define the fault blocks for this model: the Rechazada, Zafranal, and Victoria blocks. No displacement in alteration assemblages could be determined across the minor faults.

14.6.5 Wireframe model volume validation

The wireframe models for minzones and lithology units, and the volumes of the CuT and Au estimation domains, were validated against unconstrained nearest-neighbour (NN) interpolations of back-tagged composites. The NN interpolation of back-tagged composite data provides an objective method of checking for bias in the volume of the models. A target of ±5% difference between the modelled and NN volumes in well-informed blocks is sought. A block selection of Measured and Indicated blocks was used.

The volume differences of the estimation domains summarize the volume differences of the minzone and lithology models. The modelled volume of the dykes is understated as indicated by higher NN volumes of dyke blocks. The validation check on the CuT and Au estimation domains indicates that the modelled volumes in measured blocks are somewhat optimistic relative to the NN interpolation. However, the modelled volumes are within ±5% difference for Measured plus Indicated blocks, indicating that the biases in the modelled volumes are within acceptable ranges.

14.7 Assay Compositing

Drill core assay intervals for CuT, CuSS, CuCn, and Au were composited down-hole in Vulcan to a fixed length of 2.0 m from the top of the drill-holes. Composite intervals were broken at minzone boundaries. Short-length composite intervals <0.5 m were appended to the previous composite interval in the same geology using the 'merge' function in the Vulcan composting tool. Drill core samples for SG were used without compositing.

Prior to 2014, only CuT grade samples with assay values > 0.2 Cu% were assayed for soluble Cu. Since 2014, CuT grade samples with assay values > 0.1 Cu% have been assayed for soluble Cu. Composite intervals for unsampled or unassayed intervals were ignored.

The composite intervals were back-tagged from the lithology, Minzone, and alteration wireframe models to assign the LITMOD, MINMOD, and ALT codes. The Cu and Au estimation domain codes (ED_CU1 and AU_ED1) were assigned by database manipulation script from the grouped LITMOD and MINMOD values. The back-tagged codes were used for exploratory data analysis and block grade estimation. The composite intervals were also back-tagged from the alteration wireframes, but the alteration codes were used only for exploratory data analysis. Checks were made to ensure back-tagging worked as expected.

14.8 Exploratory Data Analysis

Exploratory data analysis (EDA), including preparation of box plots, histograms and probability plots, contact plots, swath plots, outlier analysis, and variography was undertaken on 2 m composites for the main grade elements CuT, Au, CuSS, CuCn, and the supplementary grade elements Ag, As, Bi, Fe, and S, and SG to develop an understanding of the grade distributions leading to an estimation plan for block grade estimation. The analysis of the grade distributions was undertaken on declustered data. NN interpolation of the 2 m grade composites was used to decluster the data. The final iteration of data analysis and validation review was conducted on 2 m grade composites in the Measured and Indicated blocks.

The results of EDA for CuT, as the most economically important grade element, are provided as examples.

14.8.1 Box plots

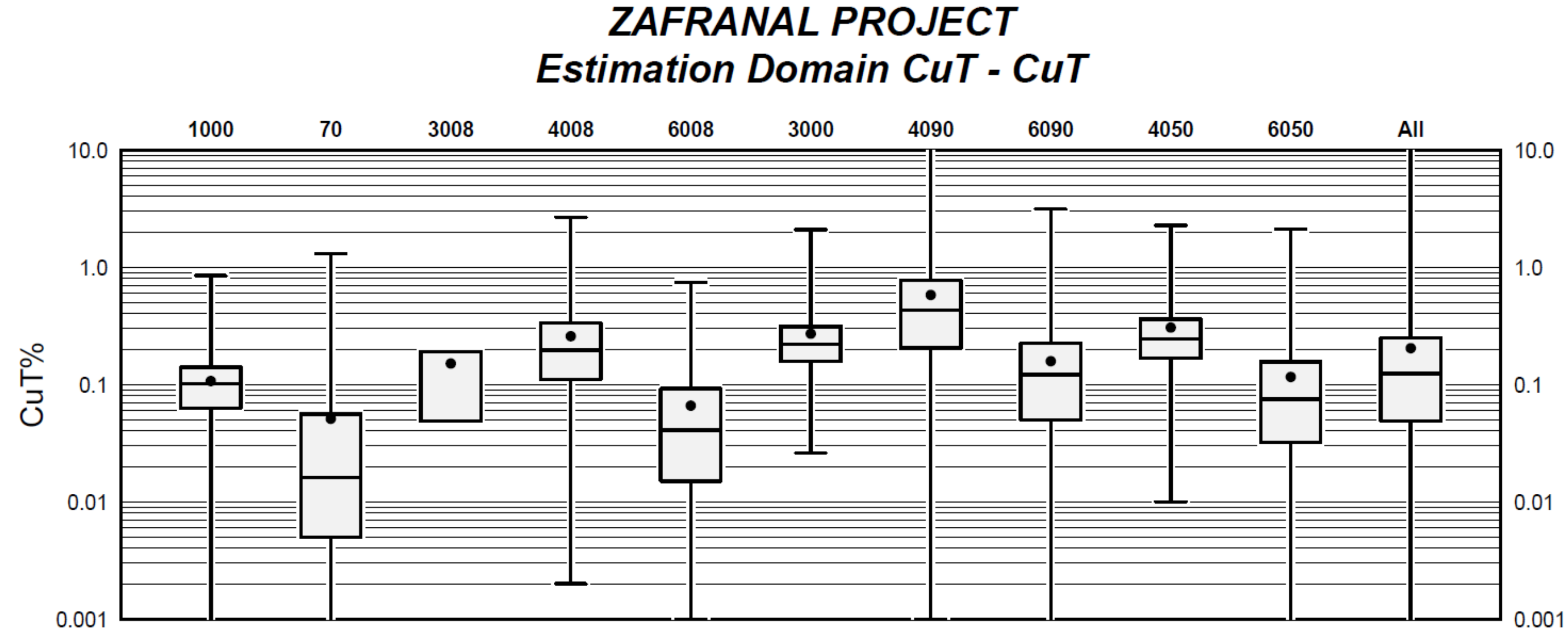
Box plots were prepared from the uncapped declustered grade data grouped by back-tagged lithology, minzone, alteration, and estimation domain.

The highest grades in composites back-tagged by lithology are seen in the Zafranal diorite and microdiorite. The lowest grades occur in the volcanoclastics, Victoria diorite, and late quartz diorite. Some reduction in the coefficients of variation (CV) is observed in the division of the grade distribution by lithology.

The analysis of box plots divided by alteration shows that the coefficients of variation (CV) increase indicating that heterogeneity is being introduced by the division of the grade distribution by alteration zone.

Analysis of the box plots indicates that dividing by minzone with subdivision by lithology will produce the lowest CVs and most similar means. Subdivision by alteration delivers the least reduction in CVs.

The analysis supports the division and groupings of the CuT estimation domains. The box plot for uncapped CuT in the Zafranal zone divided by Cu estimation domain is shown in Figure 14-3. The CuT estimation domain codes are provided in Table 14-2.



	1000	70	3008	4008	6008	3000	4090	6090	4050	6050	All	
No. of data	7893	28501	39	16080	58244	37411	140694	610103	21441	196594	1117000	No. of data
Mean	0.107	0.051	0.151	0.258	0.066	0.271	0.580	0.158	0.306	0.116	0.204	Mean
Maximum	0.847	1.310	0.191	2.670	0.741	2.100	21.655	3.130	2.280	2.120	21.655	Maximum
U.Quartile	0.140	0.056	0.191	0.333	0.092	0.312	0.776	0.224	0.361	0.156	0.250	U.Quartile
Median	0.101	0.016	0.191	0.196	0.041	0.220	0.432	0.121	0.245	0.075	0.124	Median
L.Quartile	0.063	0.005	0.049	0.111	0.015	0.157	0.204	0.050	0.167	0.032	0.049	L.Quartile
Minimum	0.000	0.000	0.049	0.002	0.001	0.026	0.001	0.000	0.010	0.000	0.000	Minimum
Variance	0.004	0.011	0.004	0.054	0.005	0.037	0.383	0.024	0.056	0.017	0.090	Variance
CV	0.624	2.018	0.423	0.902	1.074	0.707	1.068	0.986	0.770	1.121	1.473	CV
Skewness	2.205	5.948	-0.969	2.994	2.145	2.762	10.029	2.871	2.780	3.120	13.484	Skewness

Figure 14-3 – Box plots and summary statistics for uncapped CuT by Cu estimation domain

Table 14-2 – CuT block grade estimation domain codes and description

Code	Description
70	All dykes
1000	Leached zone
3000	Mixed and oxides in the Zafranal and Victoria zones
3008	Mixed and oxides in the west Zafranal zone
4008	West Zafranal supergene
4050	Zafranal supergene
4090	Victoria supergene
6008	West Zafranal hypogene
6050	Zafranal hypogene
6090	Victoria hypogene

14.8.2 Swath plots

Swath plots were prepared from the uncapped declustered grade data grouped by back-tagged lithology, Minzone, alteration, and estimation domain.

Swath plots were prepared in three dimensions for CuT, CuSS, CuCn, and Au composites. No significant grade trends were observed that would negatively impact variography and block grade estimation.

14.8.3 Contact analysis

Contact plots were prepared from the uncapped declustered grade sample composite data for CuT, CuSS, CuCn, and Au between block grade estimation domains. The same plots were repeated for block grade data and compared to the grade composite plots on the same sheet as part of the block grade estimate validation process. A bin width of 15.0 m was used.

The boundaries between grade estimation domains were all treated as hard; no grade sample composite data sharing across domain boundaries was permitted. The contact analysis confirms decisions made on the grouping of minzone and lithology into estimation domains. The contact analysis indicates that, in most cases, this approach is warranted. In some cases limited sharing or soft boundaries would be acceptable.

14.9 Outlier Analysis

Outlier analysis was completed on the CuT, Au, CuSS, and CuCn grade sample composites. Grade sample composite values above 99.7% of the samples in the distributions were considered to be high grade outliers. The analysis for CuT and Au in the Zafranal and Victoria supergene and hypogene also included inspection of histograms, probability plots, cutting statistics, metal-at-risk, and decile analysis. The summary of the CuT outlier analysis is provided in Table 14-3. The high-grade outliers were not capped but, rather, were treated specially with reduced search distances during block grade estimation as described in Section 14.10. This strategy is appropriate as composite outliers are somewhat clustered and are generally bracketed by composites in the next lowest grade bin.

Table 14-3 – Summary of the outlier analysis for CuT (%)

Copper Estimation Domain		Top-cut Estimation Domain	High Yield Limit CuT%	Metal-at-Risk CuT%	Histogram CuT%	Cutstat CuT%
4008	West superergene	Supergene M+I	1.1	2.2	5	not warranted
4050	Victoria supergene		1.6			
4090	Zafranal superergene		3.8			
6008	West hypogene	Hypogene M+I	0.6	not warranted	1.5	not warranted
6050	Victoria hypogene		0.9			
6090	Zafranal hypogene		1.2			
1000	Leached		0.3	N/A		
3008	West oxides+mixed		0.3			
3000	Victoria+Zafranal oxides+mixed		1.4			
70	Dykes		0.6			

14.10 Variography

Variogram maps, down-the-hole and traditional experimental semi-variograms (variograms) were calculated for the CuT, Au, CuSS, and CuCn grade sample composites in Snowden Supervisor™ software. The experimental variograms were computed from data within the grade element estimation domains. The variograms were modelled with two or three spherical structures plus a nugget effect. The nugget effects were modelled from the down-the-hole variograms. A summary of the variogram model parameters and rotations for CuT is provided in Table 14-4.

Table 14-4 – A Summary of CuT variogram model parameters

						Rotation (°)			Range 1st Structure (m)			Range 2nd Structure (m)			Range 3rd Structure (m)		
Cu_Ed Variogram	Nugget C0	C1	C2	C3	Type	Bearing	Plunge	Dip	Major	Semi-Major	Minor	Major	Semi-Major	Minor	Major	Semi-Major	Minor
						Z	X	Y	Y	X	Z	Y	X	Z	Y	X	Z
1000	0.075	0.270	0.280	0.375	Sph	292	0	-45	65	30	30	280	100	150	400	350	250
3000	0.105	0.270	0.250	0.375	Sph	299	15	-43	20	20	10	50	30	20	120	50	50
4008	0.070	0.300	0.360	0.270	Sph	119	-15	-137	10	10	10	60	50	60	80	80	80
4050	0.195	0.250	0.250	0.305	Sph	52	15	-43	20	20	20	30	30	35	60	70	90
4090	0.070	0.275	0.350	0.305	Sph	74	15	-43	28	20	20	160	50	50	190	160	140
6050	0.070	0.265	0.305	0.360	Sph	68	22	-90	100	160	25	200	235	40	280	255	140
6090	0.085	0.250	0.250	0.415	Sph	99	20	-114	40	20	20	150	180	100	600	400	140

Note: LH Rotation about the Z axis
RH Rotation about the X' axis
LH Rotation about the Y' axis
Rotations for the 1st and 2nd structures are the same

14.11 Block Model Assignments

14.11.1 Lithology and Minzone

Blocks were assigned lithology and minzone codes using the wireframes prepared in Leapfrog. Block lithology and minzone assignment was based on majority rule; yet the proportion of lithology in each block was stored as well. The proportional models were used for block grade estimate weighting.

14.11.2 Estimation domains

Block grade estimation domains were derived from the minzones and lithology grouped by structural domains. The allocation by minzone and lithology within structural domains is supported by the results of the EDA. The categorisation of the CuT estimation domains is provided in Figure 14-4 and Table 14-5.

The decision to develop estimation domains from minzone and lithology is appropriate. The highest impact on the reduction of CVs occurs by using minzone-based domains. The seeking of domains with mutually low CVs is advantageous because it improves the accuracy of estimation (does not mix composites with dissimilar grades). Low to moderate differences in means and CVs still exists between lithologies within estimation domains. During future resource updates instituting lithological control within minzones in the grade estimation should be evaluated.

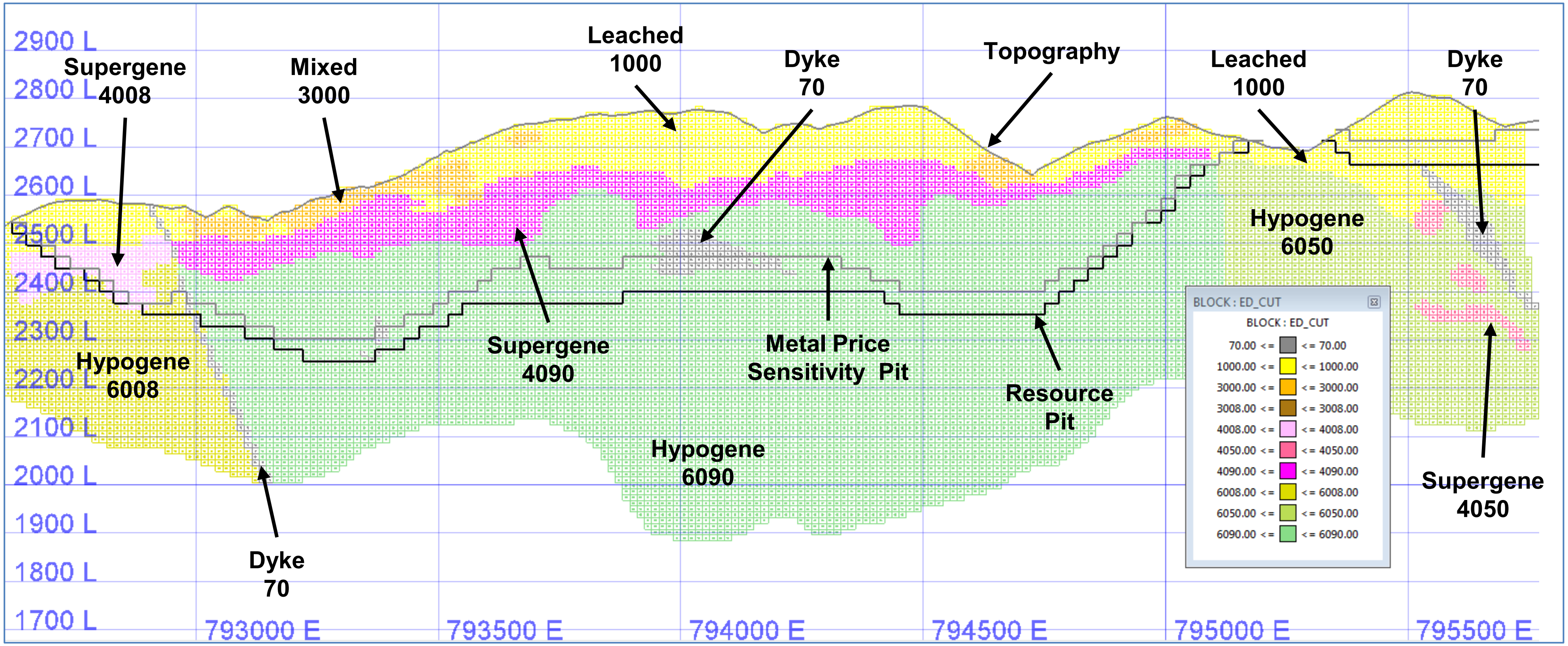


Figure 14-4 – The CuT estimation domains in an east-west longitudinal section at 8,224,000 mN (by AFW, 2016)

Table 14-5 – The categorisation of the CuT estimation domains

Structural Domain	Mineral Zone	Lithology	CuT Estimation Domain
All Structural Domains	Leached	All Lithologies	1000
	Dykes		70
	Mixed		3000
West Sector (1,8)	Mixed	Microdiorite	3008
		Zafranal Diorite	
		Volcanoclasitc	
	Supergene	Microdiorite	4008
		Zafranal Diorite	
		Volcanoclasitc	
	Hypogene	Microdiorite	6008
		Zafranal Diorite	
		Volcanoclasitc	
Zafranal Sector (2, 3, 4, 5)	Supergene	Microdiorite	4090
		Zafranal Diorite	
	Hypogene	Breccia	6090
		Microdiorite	
		Zafranal Diorite	
		Volcanoclasitc	
Victoria Sector (6, 7)	Supergene	Victoria Diorites*	4050
		Migmatite	
	Hypogene	Victoria Diorites*	6050
		Migmatite	

*Gabbrodiorite, Granodiorite, and Victoria Microdiorite

14.12 Block Model Grade Estimate

The CuT, Au, CuSS, CuCn, and SG block values were interpolated using OK and ID² estimators. An OK estimator was used for the minor grade elements: Ag, As, Bi, Fe, and S. A two-pass or three-pass estimation approach was used with each successive pass having greater search distances and less restrictive sample selection requirements. Kriging neighbourhood analysis (KNA) was used as a guide to evaluate the impact of the number of samples and of the variograms parameters on the block grade estimates.

A hard boundary approach was employed for all block grade estimates. The rotation angles of the search ellipses were based on the variography, if available, or on an isotropic search. A block discretization of 4 m x 4 m x 3 m was uniformly used for all block grade estimates. Outlier search restrictions were applied to high grade outliers identified during EDA. A summary of the high-yield search restrictions based on outlier analysis is provided in Table 14-6.

The block grade models can be considered diluted in that proportional block models were constructed by domains and then used to weight the grades at the 15 m x 15 m x 12 m block scale.

Table 14-6 –Summary of the high yield outlier search restriction for Cu

Estimation Domain	Estimator	High Yield Limit			
			Radii (m)		
		Threshold CuT %	Major Axis	Semi-Major Axis	Minor Axis
1000	OK	0.3	15	15	12
3000	OK	1.4	15	15	12
3008	ID²	0.3	15	15	12
4008	OK	1.1	15	15	12
4050	OK	1.6	15	15	12
4090	OK	3.8	15	15	12
6008	ID²	0.6	15	15	12
6050	OK	0.9	15	15	12
6090	OK	1.2	15	15	12
70	ID²	0.6	15	15	12

The soluble copper models, CuSS and CuCn, are less well developed than the CuT model as a result of limited sampling and assaying for soluble copper. These models are reasonable in the Supergene zone. However, the block grades may be high-biased in other lower grade domains because of a lack of soluble copper analyses for samples running less than 0.1 to 0.2 CuT%. For future resource updates, it is recommended that additional assays are made available and that the soluble copper model is developed from the composite sample (CuSS+CuCn)/CuT ratio and applied to the full CuT model as the preferred methodology.

14.13 Block Model Validation

The block model grade estimates were validated by visual inspection comparing composite grades to block grades, statistical checks, and selectivity checks.

14.13.1 Visual validation

Visual validation comprised inspection of composited grade samples and block grades on vertical sections and plan views. The CuT grade composites, block grades, and the CMZ 2015 resource pit shell are shown in plan view at 2,500 masl elevation in Figure 14-5, on vertical section looking north at 8,224,000 mN in Figure 14-6, and on section looking west at 793,150 mE in Figure 14-7. The block model grades generally correlate with the composite data well, and grade extrapolation is controlled where sufficient data exist.

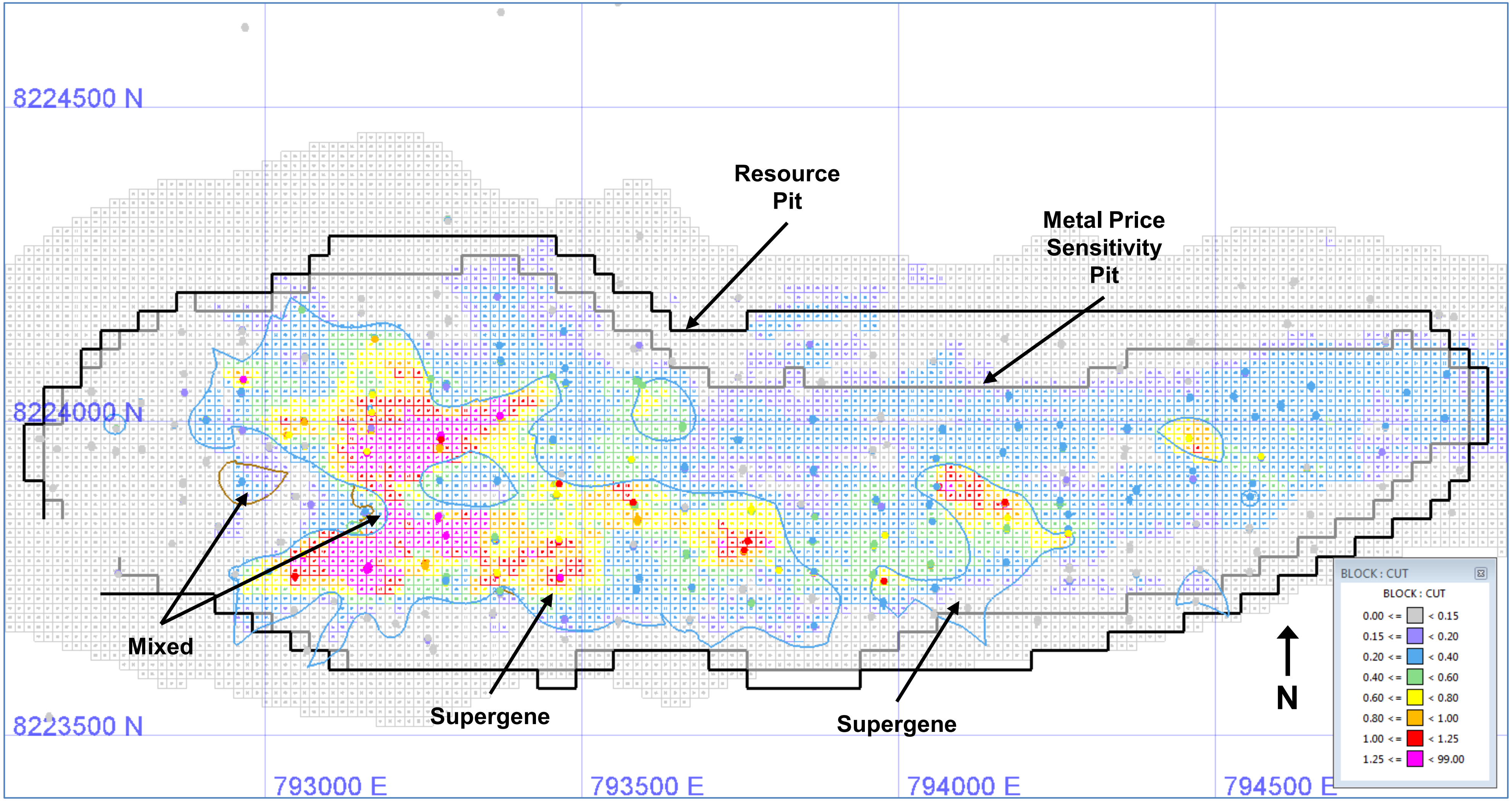


Figure 14-5 – CuT composites and block grades in plan view at 2,500 masl (by AFW, 2016)

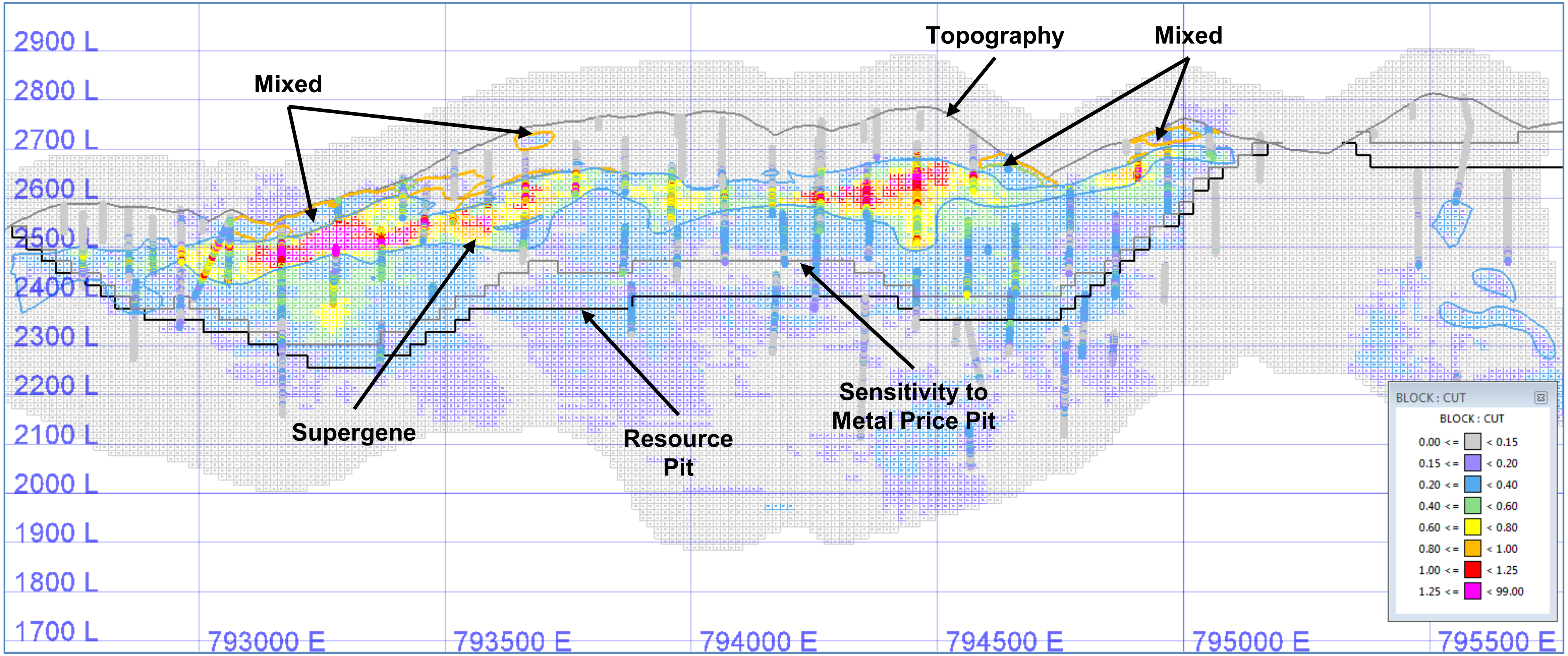


Figure 14-6 – CuT composites and block grades on section at 8,224,000 mN looking north (by AFW, 2016)

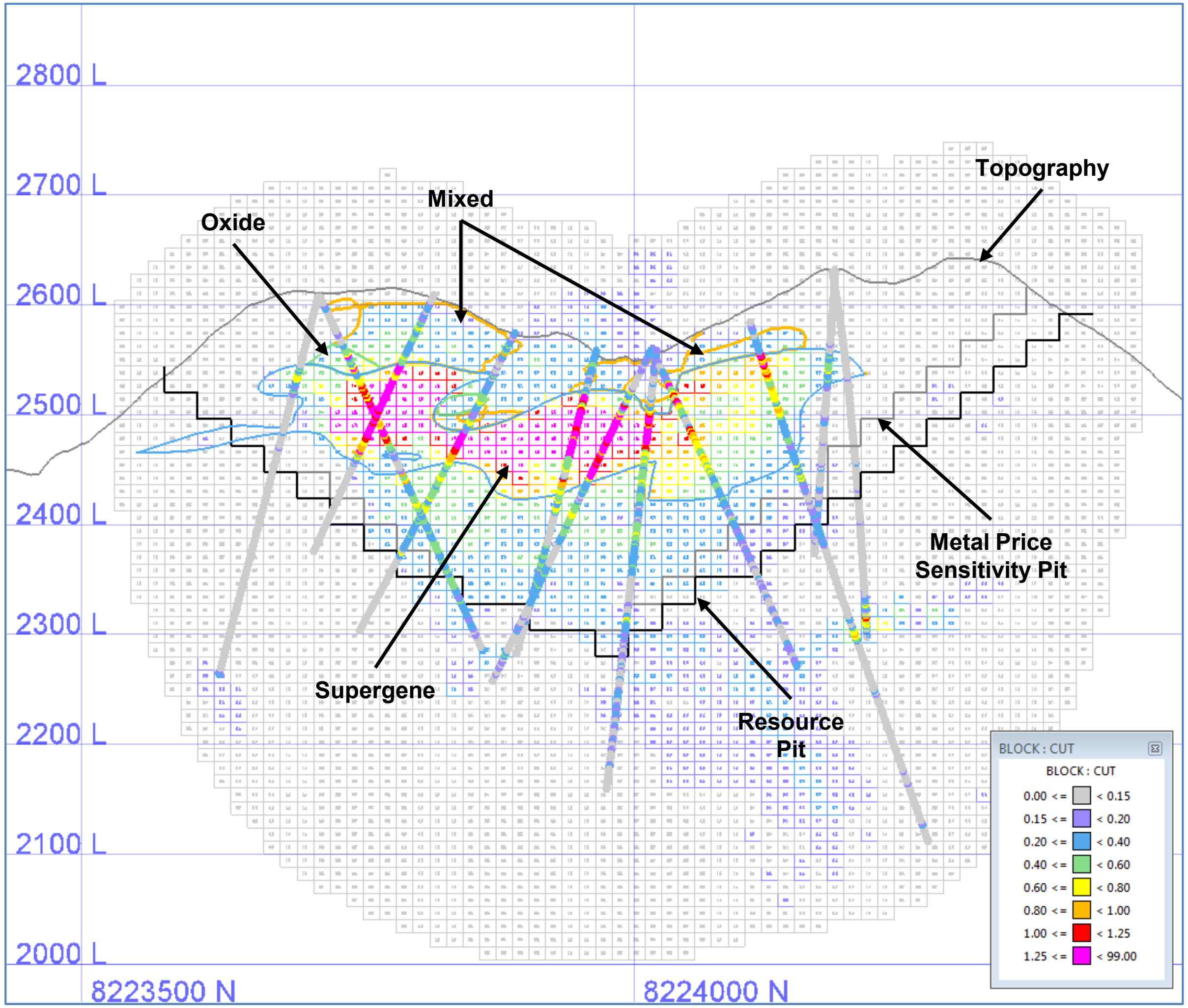


Figure 14-7 – CuT composites and block grades on Section 793,150 mE looking west (by AFW, 2016)

14.13.2 Global grade bias check

Block grade estimates were checked for global bias by comparing the average grades (with no cut-off) with those obtained from the NN interpolated model estimates in Measured and Indicated blocks. The NN grade model is a declustered composite grade distribution and provides a globally unbiased estimate of the average value when no cut-off grade is imposed. The results show that relative differences in mean grades between the OK and NN model are generally well within the target of ±5%. High biases in leached, oxides and mixed, and dyke materials do not pose a project risk during the current level of study; however, these biases should be addressed for the next level of study.

14.13.3 Predicted metal loss

A strategy of outlier search restriction was used to mitigate the influence of high-grade sample outliers during block grade estimation, as opposed to direct capping of the grade sample composites, as described in Section 14.8. The impact of outlier restriction has resulted in generally 2% or less reduction in predicted metal. The greatest impact of outlier restriction is seen in the West Sector Supergene blocks where the predicted metal loss is 4%. The predicted metal reduction in the West Sector Supergene affects a relatively small number of blocks and does not have a material impact on the overall Mineral Resources.

14.13.4 Local grade bias check

Swath plots were prepared to compare grade profiles for the OK and NN block estimates, and grade sample composites in east-west, north-south, and vertical swaths or increments. Swath intervals are 60 m in the easterly, 30 m in the northerly directions, and 12 m vertically. The comparison was limited to measured and Indicated blocks, which are reasonably well informed during estimation.

Swath plots for CuT in the Zafranal Supergene estimation domain Measured and Indicated blocks are provided in Figure 14-8. The grade profiles are generally in very good agreement. As expected, the OK and NN grade profiles diverge slightly where block counts fall. The swath plots indicate that no systematic local bias has been introduced in the block grade estimate.

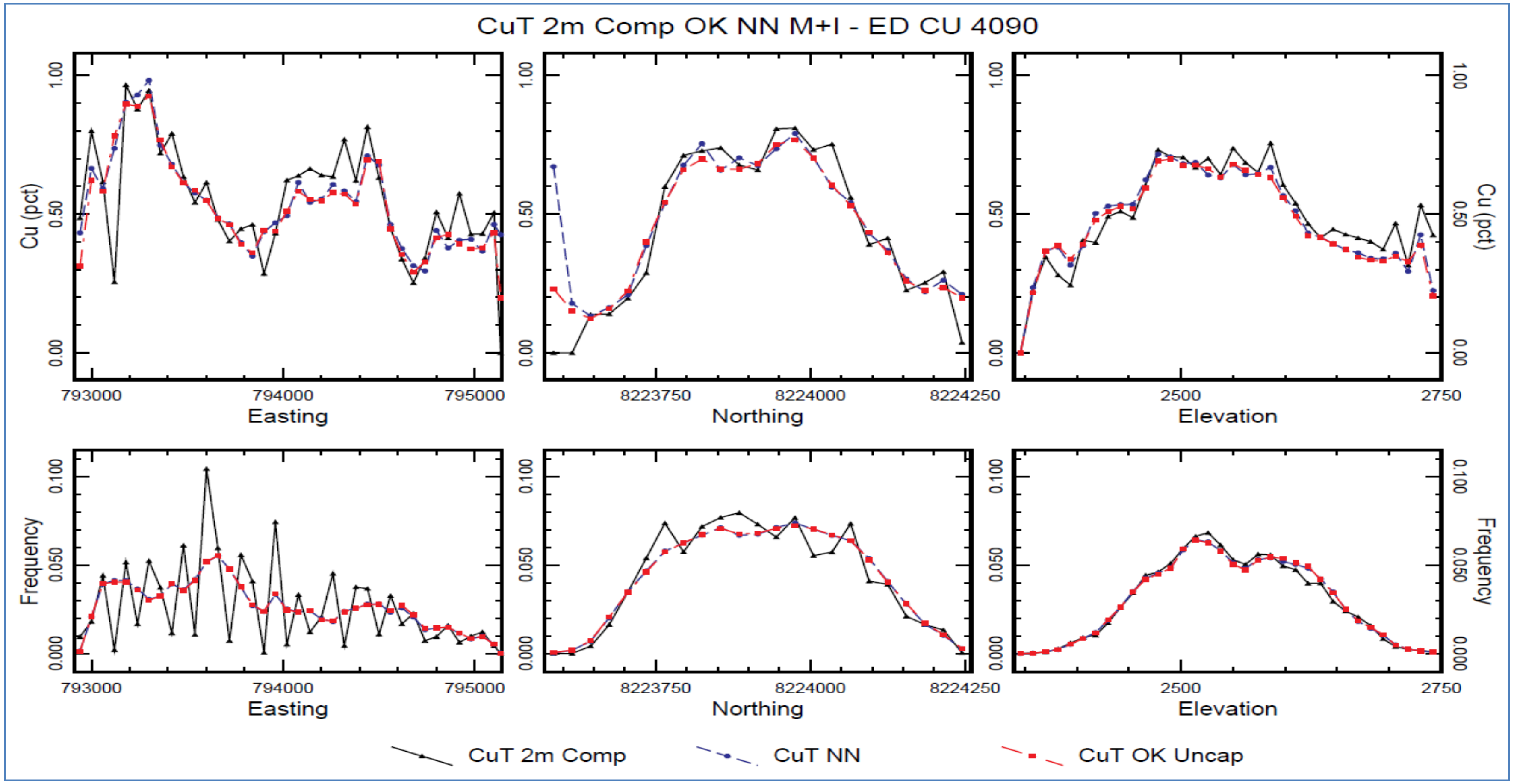


Figure 14-8 – CuT swath plots in the Zafranal supergene estimation domain in measured and indicated blocks (by AFW, 2016)

14.13.5 Contact analysis

Contact plots were prepared from the uncapped declustered grade sample composite data and block grade estimates for CuT, CuSS, CuCn, and Au between block grade estimation domains as described in Section 14.12.

The results of the contact analysis indicate that, in general, the strategy of using hard boundaries between estimation domains is appropriate.

14.13.6 Selectivity check

Selectivity analysis for CuT was completed using the Discrete Gaussian Model for change of support from composite size to a selective mining unit (SMU) size. This was done by a Hermitian correction method using Amec Foster Wheeler's in-house software (HERCO). The aim of this analysis was to assess whether the estimated resource reasonably represents the recoverable resources (represented by grade and tonnage curves) relative to the proposed mining method.

The results of the HERCO analysis are generally discussed in terms of smoothness. An over-smoothed model may over-estimate the tonnes and under-estimate the grade. The model with an appropriate amount of smoothing will follow the HERCO grade and tonnage (GT) curves for values corresponding to different economic, or grade cut-offs. The target amount of smoothing is 5% or less relative difference between the estimated block tonnes and grade and the corrected HERCO tonnes and grade. As with swath plots, the HERCO analysis was limited to Measured and Indicated blocks which are reasonably well informed during estimation.

The selectivity analysis assumed a 15 m x 15 m x 12 m block, the resource block size, as the SMU size. Measured and indicated blocks in the Zafranal supergene domain category (dom_cat block variable) which encompasses leached, mixed, and dykes blocks were selected for the analysis. The grade cut-off of 0.15 CuT% was used as the economic cut-off of interest.

The variogram that was modelled for the block grade confidence classification was used as an input in the Amec Foster Wheeler proprietary Single-Block Kriger (SBK) application to calculate the variance adjustment factor for the blocks.

The analysis shows that the CuT grade estimate is over-smooth at the resource model block scale: change-of-support grade-tonnage curves indicate that the model over-predicts tonnage by 6% and under-predicts grade by 5% relative to an ideal distribution at a grade cut-off of 0.15 CuT%. These values are at or near the target level of smoothing considered as the industry standard. Nevertheless, there is an opportunity to increase NPV and cash flows in the future by mining a lesser tonnage at a higher average grade above cut-off and keeping the contained copper content essentially constant. Amec Foster Wheeler recommends, that for future resource updates, the modellers optimize the kriging parameters, introduce lithological control, or use local uniform conditioning to develop a recoverable mineral resource model.

14.14 Mineral Resource Classification

The Mineral Resource is classified in accordance with the Canadian Institute of Mining, Metallurgy, and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014).

A separate two-pass OK interpolation was run to generate the sample counts and distances used for Mineral Resource classification, followed by category smoothing. Blocks were classified based on a combination of factors including the number of drill holes used for each block, the distance to the nearest composites, and CuT mineral zonation.

In addition to criteria such as sufficient geological continuity, grade continuity, and data integrity, a guideline of drill hole spacing sufficient to predict potential production with reasonable probability of precision over a selected period of time was incorporated into the confidence classification. A drill hole spacing study (DHSS) was conducted taking into account both grade continuity and resource tonnage / volume uncertainty. The DHSS seeks to determine the drill hole spacing required for a 90% grade confidence interval (CI) ±15% based on quarterly production volume for Measured resources and annual production volume for Indicated resources. These CIs are widely accepted in industry for grade confidence levels for the declaration of Mineral Resources.

The following criteria for classification of Mineral Resources at the Zafranal deposit were established from the drill hole spacing study:

- Measured Mineral Resources:
  - Three drill holes with a nominal spacing of 75 m in Supergene
  - Three drill holes with a nominal spacing of 85 m in Hypogene
- Indicated Mineral Resources:
  - Two drill holes with a nominal spacing of 110 m
- Inferred Mineral Resources:
  - All remaining blocks within the estimation search criteria.

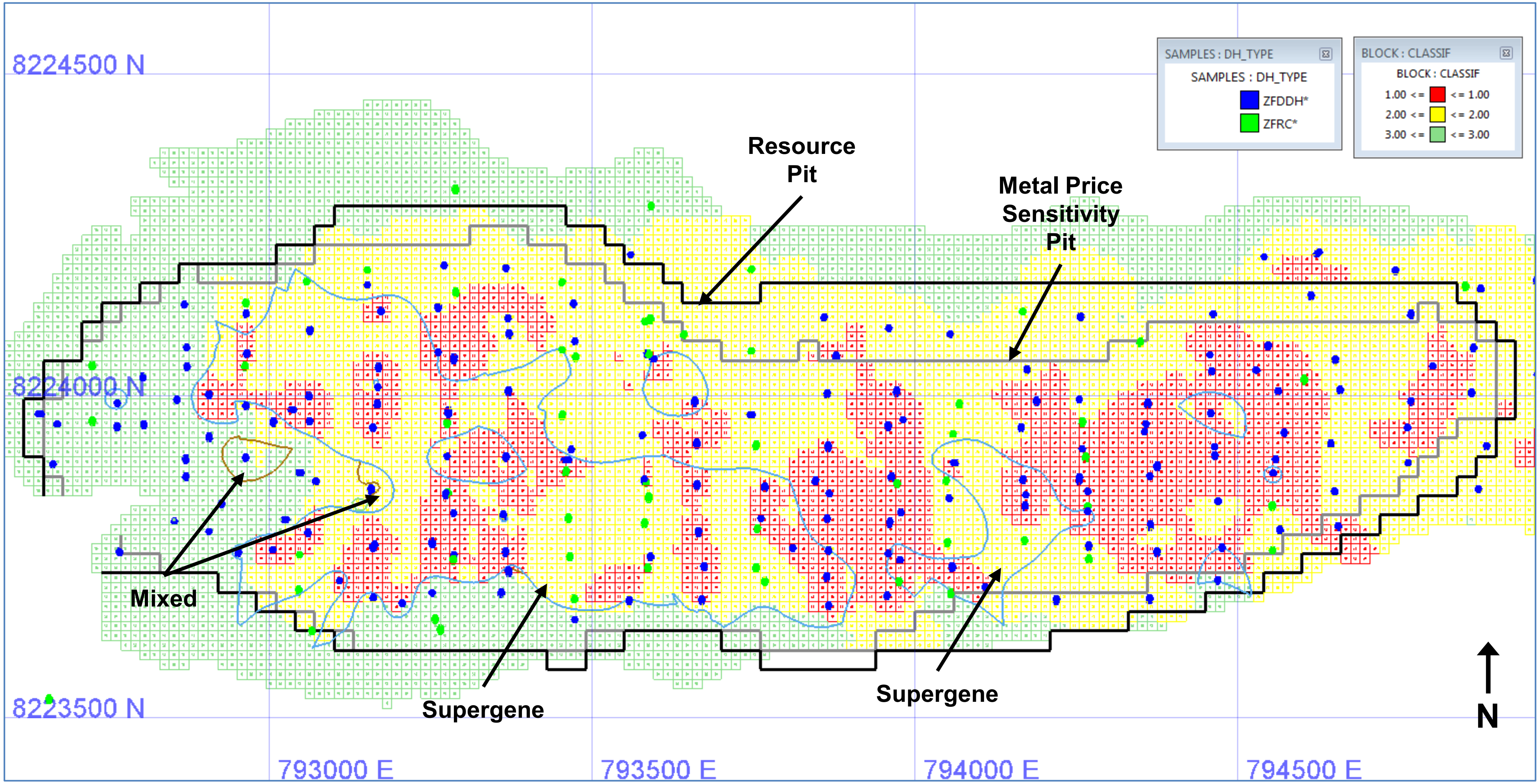
The resulting preliminary classification was reviewed visually on vertical sections and plan views. A small number of 'spots' or discontinuous groups of blocks were observed in the preliminary classification. A smoothing algorithm was used to reduce the number of spots to a reasonable level. The algorithm calculated the number of blocks of each category within a moving window on a block-by-block basis, assigning the final category to the majority classification. A window size of 30 m x 30 m x 25 m; one surrounding block on each block face, was empirically selected after several iterations, as producing an acceptable degree of smoothing.

Several modifying factors based on deposit history, data confidence, and metallurgy were also applied in arriving at the final mineral resource classification:

- Reverse circulation drill holes were not used for Measured mineral resources
- Oxide and mixed material is, at best, classified as Indicated mineral resources
- Leached material is, at best, classified as Inferred mineral resources.

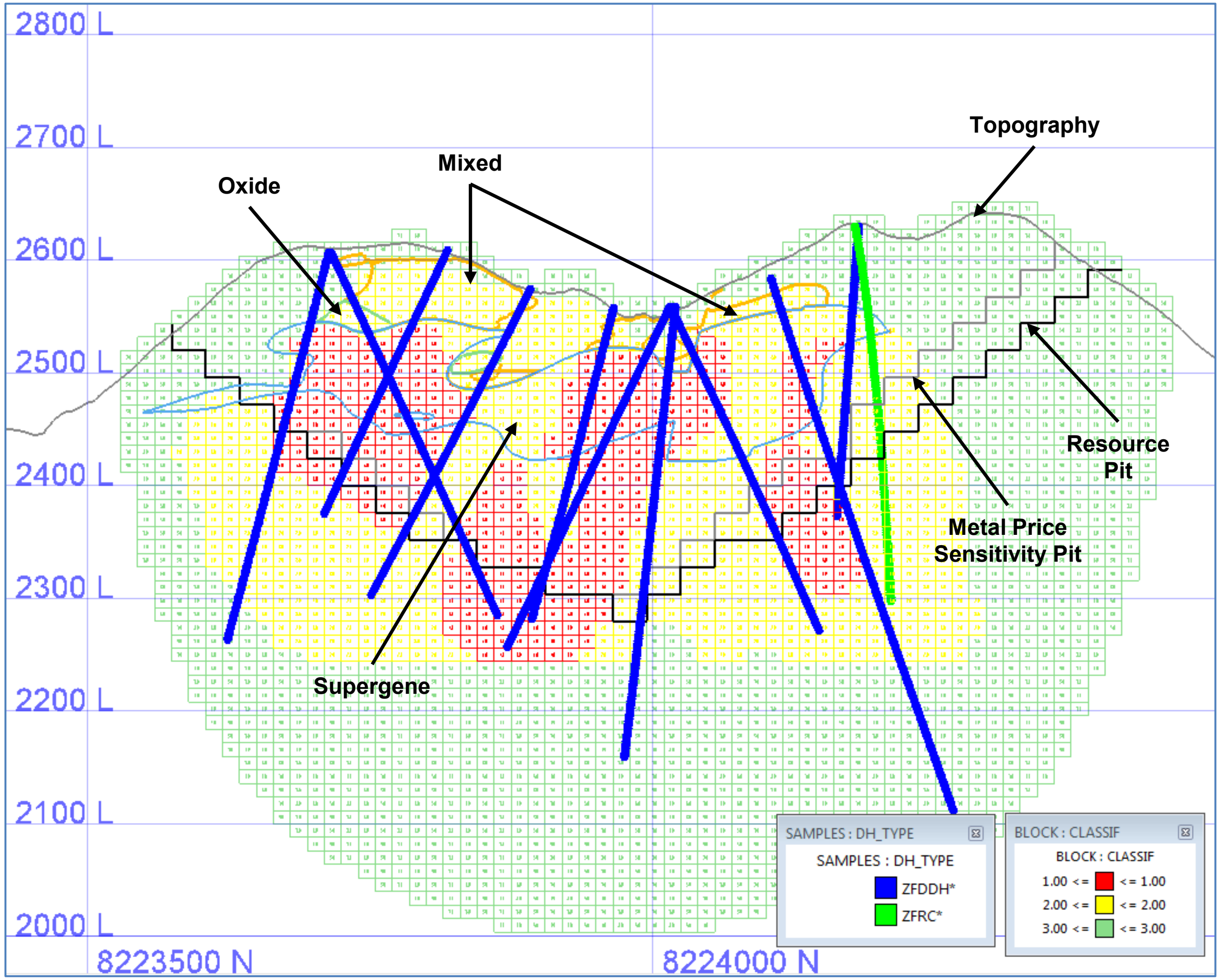
There are only Inferred Mineral Resources below 2,232 masl elevation.

The Mineral Resource classification map in plan view at 2,500 masl elevation is presented in Figure 14-9 and at section 793,150 mE in Figure 14-10.



Note: Block classification: 1.0 = Measured, 2.0 = Indicated, and 3.0 = Inferred

Figure 14-9 – Mineral Resource classification in plan view at 2,500 masl (by AFW, 2016)



Note: Block classification: 1.0 = Measured, 2.0 = Indicated, and 3.0 = Inferred

Figure 14-10 – Mineral resource classification at Section 793,150 mE looking west (by AFW, 2016)

14.15 Reasonable Prospects for Eventual Economic Extraction

To assess reasonable prospects for eventual economic extraction, it is assumed that the Zafranal deposit would be mined utilizing open pit mining methods under a conceptual scenario of 55,000 tonnes per day production using conventional flotation to produce a concentrate grading 28% copper with credit for gold.

Whittle™ pit optimiser software was utilized to prepare a conceptual pit designed as a constraining pit shell, with inputs on parameters and costs for mining, processing, general and administrative (G&A), transportation, and smelting and refining. Preparation of the conceptual pit was based on economic and technical assumptions listed in Table 14-7. Some of these assumptions differ from the final parameters and costs presented in the PFS as they were developed at an earlier stage. The price assumptions used to develop the constraining pit shell are more optimistic than those used in the economic analysis of the PFS.

Table 14-7 – Economic and technical assumptions used to prepare a constraining pit

Description	Units	Cu	Au
Metal price - copper	US$/lb	3.50	-
Metal price - gold	US$/oz	-	1,400
Mill recovery			
Supergene	%	86	50
Mixed	%	86	50
Hypogene	%	89	50
Transit losses	%	0.3	-
Concentrate moisture	%	9.5	-
Concentrate grade	Cu%	28	-
Transportation	US$/t wet	65	-
Smelting and refining		-	-
Smelter charges	US$/t dry	90	-
Refining charges	US$/lb	0.09	-
	US$/oz	-	4
Smelter recovery	%	96.4	90
Pit slope	°	45 - 50	-
Mining			
At 2,534 masl	US$/t	1.58	-
Adjustment per bench downward	US$/t	0.01	-
Adjustment per bench upward	US$/t	0.03	-
Processing	US$/t	5.45	-
G&A	US$/t	0.38	-

The conceptual pit was constrained within property boundaries.

14.16 Mineral Resource Estimate

Mineral resources were tabulated within the pit at a cut-off grade of 0.15% CuT. This is above an operating breakeven cut-off grade that covers processing, tailings, and G&A operating costs within the resource pit.

Table 14-8 shows the estimated mineral resources reported at a 0.15% CuT cut-off. The effective date of the Mineral Resource Estimate is 14 December 2015.

Table 14-8 – Mineral Resource Estimate for the Zafranal Deposit based on a 0.15% CuT Cut-off, Effective Date 14 December 2015, Peter Oshust, P Geo

Classification	Tonnage	Grade		Contained metal	
	(Mt)	CuT (%)	Au (g/t)	Cu (Mlb)	Au (Moz)
Measured					
Mixed	-	-	-	-	-
Supergene	83.3	0.58	0.07	1,056	0.20
Hypogene	120.5	0.28	0.07	744	0.28
Total Measured	203.8	0.40	0.07	1,801	0.47
Indicated					
Mixed	23.5	0.28	0.12	146	0.09
Supergene	100.3	0.53	0.07	1,176	0.21
Hypogene	139.7	0.26	0.06	804	0.28
Total Indicated	263.5	0.37	0.07	2,126	0.58
Measured and indicated					
Mixed	23.5	0.28	0.12	146	0.09
Supergene	183.6	0.55	0.07	2,234	0.40
Hypogene	260.2	0.27	0.07	1,543	0.56
Total Measured and Indicated	467.3	0.38	0.07	3925	1.05
Inferred					
Mixed	7.8	0.22	0.09	37	0.02
Supergene	8.7	0.30	0.04	57	0.01
Hypogene	4.9	0.18	0.03	20	0.00
Total Inferred	21.4	0.24	0.06	114	0.04

Notes:

1. Mineral resources are reported inclusive of those Mineral Resources that have been converted to Mineral Reserves.
2. Mineral resources are reported within a constraining pit shell developed using Whittle™ software. Assumptions include metal prices of US$3.50/lb for Cu and US$1,400/oz for Au; process recoveries of 86% for Cu and 50% for Au in Supergene, 86% recoveries for Cu and 50% recoveries for Au in mixed, and 89% for Cu and 50% for Au in Hypogene, US$1.58/t of mining at 2,534 m plus US$0.01/bench downward and US$0.03/bench upward. US$5.45/tonne for processing, and US$0.38/tonne for G&A.
3. Assumptions include 100% mining recovery.

4. An external dilution factor was not considered during this resource estimation. Internal dilution within a 15 m x 15 m x 12 m SMU was considered.
5. The 1.0% Government royalty was not considered during the preparation of the constraining pit.
6. Quantities and grades in a mineral resource estimate are rounded to an appropriate number of significant figures to reflect that they are approximations.

14.17 Reporting Sensitivity

Table 14-9 shows the sensitivity of the Zafranal Mineral Resource Estimate to changes in CuT cut-off grade. The base case CuT cut-off is highlighted in bold text.

Table 14-9 – Sensitivity of the mineral resource to changes in CuT cut-off grade (base case cut-off)

Measured Mineral Resources

Cut-off	Tonnage	Grade		Contained metal	
(CuT%)	(Mt)	Cu (%)	Au (g/t)	Cu (Mlb)	Au (Moz)
0.125	216.1	0.39	0.07	1,839	0.49
0.150	**203.8**	**0.40**	**0.07**	**1,801**	**0.47**
0.200	172.6	0.44	0.08	1,678	0.43
0.250	137.0	0.50	0.08	1,501	0.36
0.300	103.4	0.57	0.09	1,299	0.29
0.350	79.2	0.65	0.09	1,126	0.24

Indicated Mineral Resources

Cut-off	Tonnage	Grade		Contained metal	
(CuT%)	(Mt)	Cu (%)	Au (g/t)	Cu (Mlb)	Au (Moz)
0.125	289.1	0.35	0.07	2,206	0.61
0.150	**263.5**	**0.37**	**0.07**	**2,126**	**0.58**
0.200	207.1	0.42	0.08	1,908	0.50
0.250	154.8	0.48	0.08	1,649	0.40
0.300	116.1	0.55	0.09	1,415	0.32
0.350	90.2	0.62	0.09	1,231	0.26

Measured and Indicated Mineral Resources

Cut-off	Tonnage	Grade		Contained metal	
(CuT%)	(Mt)	Cu (%)	Au (g/t)	Cu (Mlb)	Au (Moz)
0.125	505.2	0.36	0.07	4,044	1.10
0.150	**467.3**	**0.38**	**0.07**	**3,928**	**1.06**
0.200	379.7	0.43	0.08	3,586	0.93
0.250	291.8	0.49	0.08	3,149	0.76
0.300	219.5	0.56	0.09	2,715	0.61
0.350	169.4	0.63	0.09	2,357	0.50

Inferred Mineral Resources

Cut-off	Tonnage	Grade		Contained metal	
(CuT%)	(Mt)	Cu (%)	Au (g/t)	Cu (Mlb)	Au (Moz)
0.125	26.7	0.22	0.05	130	0.04
0.150	**21.4**	**0.24**	**0.06**	**114**	**0.04**
0.200	11.6	0.30	0.06	76	0.02
0.250	6.0	0.37	0.06	49	0.01
0.300	3.6	0.43	0.06	34	0.01
0.350	2.3	0.49	0.06	25	0.00

Table 14-10 shows the sensitivity of the Zafranal Mineral Resource Estimate to changes in metal price assumptions used to generate the resource pit shell. In this sensitivity case, the prices used are US$3.00/lb for Cu and US$1200/oz for Au. The base case CuT grade cut-off is highlighted in bold text.

Table 14-10 – Sensitivity of the mineral resource to changes in metal price assumptions (US$3.00/lb Cu and US$1,200/oz Au)

Measured Mineral Resources

Cut-off	Tonnage	Grade		Contained metal	
(CuT%)	(Mt)	Cu (%)	Au (g/t)	Cu (Mlb)	Au (Moz)
0.125	186.4	0.41	0.07	1,685	0.44
0.150	**177.8**	**0.42**	**0.07**	**1,658**	**0.42**
0.200	154.6	0.46	0.08	1,571	0.39
0.250	126.6	0.51	0.08	1,432	0.34
0.300	98.8	0.58	0.09	1,264	0.28
0.350	77.4	0.65	0.09	1,110	0.23

Indicated Mineral Resources

Cut-off	Tonnage	Grade		Contained metal	
(CuT%)	(Mt)	Cu (%)	Au (g/t)	Cu (Mlb)	Au (Moz)
0.125	233.5	0.38	0.07	1,945	0.53
0.150	**218.1**	**0.40**	**0.07**	**1,899**	**0.51**
0.200	180.7	0.44	0.08	1,753	0.45
0.250	141.9	0.50	0.08	1,561	0.37
0.300	110.6	0.56	0.09	1,373	0.31
0.350	88.0	0.62	0.09	1,211	0.25

Measured and Indicated Mineral Resources

Cut-off	Tonnage	Grade		Contained metal	
(CuT%)	(Mt)	Cu (%)	Au (g/t)	Cu (Mlb)	Au (Moz)
0.125	419.9	0.39	0.07	3,631	0.97
0.150	**395.9**	**0.41**	**0.07**	**3,557**	**0.93**
0.200	335.3	0.45	0.08	3,324	0.85
0.250	268.5	0.51	0.08	2,993	0.72
0.300	209.5	0.57	0.09	2,637	0.59
0.350	165.4	0.64	0.09	2,321	0.49

Inferred Mineral Resources

Cut-off	Tonnage	Grade		Contained metal	
(CuT%)	(Mt)	Cu (%)	Au (g/t)	Cu (Mlb)	Au (Moz)
0.125	20.7	0.23	0.06	107	0.04
0.150	**17.3**	**0.25**	**0.06**	**96**	**0.04**
0.200	10.5	0.30	0.06	70	0.02
0.250	5.7	0.37	0.06	46	0.01
0.300	3.5	0.43	0.06	33	0.01
0.350	2.2	0.49	0.06	24	0.00

14.18 General Consideration of Other Factors

Other than the risks identified in this report, Amec Foster Wheeler is not aware of any other environmental, permitting, legal, title, taxation, socio-economic or political factors that could materially affect the Mineral Resource estimate.

14.19 Conclusions and Recommendations

Amec Foster Wheeler concludes the following:

- The construction of the 2015 CMZ resource model has followed 2014 CIM Definition Standards for Mineral Resources and 2003 CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines.
- The modeling and grade estimation process used is appropriate for a porphyry-style copper-gold deposit and the resource model is reasonable and is appropriate for the declaration of Mineral Resources to support mine planning at a PFS level for a large-scale open pit mine.

Amec Foster Wheeler makes the following recommendations:

- The CuT grade estimate is over-smooth at the resource model block scale; HERCO change-of-support grade-tonnage curves indicate that the model over-predicts tonnage by 6% and under-predicts grade by 5% relative to an ideal distribution at a grade cut-off of 0.15 CuT%. There may be an opportunity to increase net present value (NPV) and cash flows in the future by mining a lesser tonnage at a higher average grade above cut-off keeping the contained copper content essentially constant. For future resource updates, the modellers should optimize the kriging parameters, introduce lithological control, or use local uniform conditioning to develop a recoverable mineral resource model.
- The soluble copper, CuSS and CuCn models are less well developed than the CuT model as a result of limited sampling and assaying for soluble copper. These models are reasonable in the supergene zone. For future resource updates, additional assays should be made available and the coluble copper model developed from the composite sample (CuSS+CuCn)/CuT ratio and applied to the full CuT model as the preferred methodology.
- The decision to develop estimation domains from minzones and lithology is appropriate. The highest impact on the reduction of CV occurs by using minzone-based domains. There still exist low to moderate differences in means and CVs between lithologies within estimation domains and, for future resource updates, study and consideration should be given to instituting lithological control within minzones in the grade estimation.

- Examination of the cross sections shows that the spacing at which continuity of grade is confirmed in the Supergene domain can vary from 50 to 100 m. For the next level of study, conditional simulation is recommended to establish the recommended drill spacing to declare Measured mineral resources on a local basis. In-fill drilling should be planned to increase confidence, so that at least 80% of the mineralization to be mined in the first five years of production, could support a Measured classification.
- Amec Foster Wheeler considers that appropriate measures should be implemented to ensure data integrity that should include use of proper database management software to ensure the integrity of the database. Amec Foster Wheeler is aware that the selection process for a suitable data management solution is underway.

15 Mineral Reserve Estimates

15.1 Introduction

NCL Ingeniería y Construcción S.A. (NCL) was commissioned by Compañía Minera Zafranal S.A.C. (CMZ) to provide mine planning services for the Zafranal Project in Peru. This work was done in support of a preliminary feasibility study (PFS) prepared by Ausenco.

NCL's Scope of Work is summarized as follows:

1. Develop a mine plan and mine production schedule for the life of mine (LOM), including the design of waste rock storage areas.
2. Determine the mine equipment and labour requirements for the LOM.
3. Estimate the mine capital and operating costs for the LOM.
4. Estimate the mineral reserves of the LOM.

15.2 Source and Types of Materials

15.2.1 Resource model

NCL was provided with the CMZ 2015 Resource model that was developed by CMZ and reviewed by AMEC Foster Wheeler (AFW) and which included the 2014 drilling program results. The block model included Mineral Resources that were classified as measured, indicated or inferred. Pit optimization, mine design and mine planning were carried out using resources classified as Measured or Indicated. Inferred Mineral Resources were treated as waste.

A block size of 15 m E x 15 m N x 12 m EL was selected for the block model. The selected block size was based on the geometry of the domain interpretation and the data configuration.

The following corresponds to the list of variables contained in the received block model data:

1. Mineral zone (minzone) code: Oxides, mixed, supergene and hypogene
2. Bulk density (t/m^3)
3. Copper grade in percent (%Cu)
4. Gold grade in grams per tonne (Au g/t)
5. Secondary elements:
    a) Sulfuric-soluble copper in percent (%CuSS)
    b) Cyanide-soluble copper in percent (%CuCn)
    c) Iron in percent (%Fe)
    d) Sulfur in percent (%S)
    e) Silver in grams per tonne (Ag g/t)
    f) Arsenic in parts per million (As ppm)
    g) Bismuth in parts per million (Bi ppm).
6. A class code to distinguish measured, indicated, and inferred resource blocks.

15.2.2 Geometallurgical domains

The method of predicting the metallurgical performance of the concentrator was provided by Transmin Metallurgical Consultants (Transmin) and is based on metallurgical testwork.

The material contained separately within each of the mixed, supergene and hypogene mineral zones was re-classified in five geometallurgical domains according to the sulfur, iron, sulfuric acid-soluble copper, cyanide-soluble copper, and total copper (CuT) content, per Table 15-1.

Table 15-1 – Geometallurgical domains

Domain	Algorithm	Concentrate grade% Cu	Copper recovery(%)	Gold recovery(%)
Low S	S/Fe < 0.2	-	0.0	0.0
Low CuCn	CuCn/CuT <= 0.3	28	90.5	56.0
Low CuSS	CuSS/CuT <= 0.15	37	89.0	52.0
Mid CuSS	0.15 < CuSS/CuT <= 0.3	34	84.0	55.0
High CuSS	Rest	32	77.0	52.0

15.3 Assumptions and Parameters

15.3.1 Dilution and mine losses

Using the provided CMZ 2015 Resource Model, NCL performed a pit optimization update without introducing any additional factors to account for dilution; the block model is considered to be a fully diluted resource model. NCL used 100% mining recovery because of the disseminated characteristics of the ore.

It is becoming common in the industry to develop resource models which essentially take into account potential dilution within the blocks, or adopt selective mining units (SMU) as part of the resource modelling process. The Zafranal mineral resource model has been developed to account for potential dilution within the blocks and is considered to be a 'diluted' model.

The relationship between the recommended mining fleet and the block size in the block model allows for selective mining when proper mining practices are followed and thus full mining recovery is expected.

15.3.2 Input parameters

Table 15-2 summarizes the base case economic parameters used for Lerchs-Grossman economic pit shells analysis.

The mining cost estimate for the pit optimization process is based on studies developed by NCL during 2014 as part of the conceptual study phase. The estimated average project mining cost was separated into various components such as fuel, explosives, tires, parts, salaries and wages based on similar mines operating in Peru. Each component was updated for 2nd quarter 2015 prices and for the exchange rate from Peruvian New Soles to US dollars. This resulted in a mining cost estimate of an average of US$1.84/t shown in Table 15-2. A nominal mine sustaining capital cost of US$0.08/t is included in this value. The metal prices, processing costs, refining costs, and processing recoveries were provided to NCL by CMZ. These were reviewed by NCL and considered reasonable for pit optimization purposes.

Table 15-2 – Pit optimization parameters

Parameter	Unit	Value
Capacity assumptions		
Concentrator	Mt/y	20
Mine	Mt/y	65.00
Model and parameters		
Resource model (blocks)	m	15 x 15 x 12
Classifications considered	-	Measured/Indicated
Mining dilution	%	0%
Mining recovery	%	100%
Prices		
Copper price	$/lb	3.00
Gold price	$/oz	1,200
Recoveries and concentrate grade		
Flotation supergene Cu	%	per geometallurgical domain
Flotation supergene Au	%	per geometallurgical domain
Flotation hypogene Cu	%	per geometallurgical domain
Flotation hypogene Au	%	per geometallurgical domain
Flotation mixed Cu	%	per geometallurgical domain
Flotation mixed Au	%	per geometallurgical domain
Concentrate grade	%Cu	per geometallurgical domain
Smelter terms		
Minimum deduction	%Cu	1.00%
Payable copper	%	(Concentrate grade – 1) / Concentrate grade Average = 96.9%
Treatment charges - concentrate (dry)	$/t concentrate	90
Refining charges - payable Cu	$/lb Cu	0.09
Payable gold	%	90.00%
Refining charges - payable Au	$/oz Au	4
Other concentrate selling costs		
Concentrate moisture content	%	9.50%
Marketing costs	$/wmt Cu concentrate	12.00
Concentrate transport (third party)	$/wmt Cu concentrate	35.77
Port and insurance charges	$/wmt Cu concentrate	20.00
Shipping costs to China	$/wmt Cu concentrate	65.00
Transport losses (concentrator to loading at port)	%/wmt Cu concentrate	0.30%
Royalties		
Net smelter royalty (NSR)	% sales income	1.00%
Operating costs		
Mining		
Base cost at 2534m elevation	$/t mined	1.58
Additional cost/bench down	$/t.bench	0.01
Additional cost/bench up	$/t.bench	0.03
Sustaining cost	$/t mined	0.08
Average mining cost	$/t mined	1.86
Processing		
Flotation – mixed	$/t feed - concentrator	4.47
Flotation - supergene	$/t feed - concentrator	4.47
Flotation - hypogene	$/t feed - concentrator	4.75
G&A		
General & Administration	M$/y	25.00

15.3.3 Cut-off grade

The pit optimization process was started using the estimated metal prices, process recoveries, refining/transport costs and royalties above. A number of calculations were performed in the model to determine the net smelter return (NSR) of each individual block, to account for the value of copper and gold. The parameters in Table 15-2 are initial assumptions based on previous project studies and estimates based on a process capacity of 55 kt/d; these are not the final economic parameters used for the PFS cash-flow analysis.

Preliminary results indicate a low quantity of oxide ore available for leaching, therefore no oxide treatment is considered in this pit optimization study.

The internal (or mill) cut-off of US$5.72/t milled for mixed and supergene materials and US$6.00/t milled for hypogene material incorporates all operating costs except mining. This internal cut-off is applied to material contained within an economic pit shell where the decision to mine a given block was determined by the pit optimization.

Figure 15-1 shows the steps perform to the block model for the NSR calculation.

```
REM NSR calculation for Transmin 19-04 Geomet Zones
REM Blocks Mixed, Spg and Hyp (MIN>=330)
cu=BlockModel.Model( "Standard","CU", Column, Row, Level )
au=BlockModel.Model( "Standard","AU", Column, Row, Level )
cus=BlockModel.Model( "Standard","CUS", Column, Row, Level )
cucn=BlockModel.Model( "Standard","CUCN", Column, Row, Level )
s=BlockModel.Model( "Standard","S", Column, Row, Level )
fe=BlockModel.Model( "Standard","Fe", Column, Row, Level )

REM prices
prcu=3
prau=1200

REM     lc= concentrate grade
REM    rcu= copper recovery
REM    rau= gold recovery
If fe>0 then
 r=s/fe
Else
 r=0
End If

If r<0.2 then
 lc=0
 recu=0
 reau=0
Else
 If (cucn/cu)<0.3 then
  lc=28
  recu=0.905
  reau=0.56
 Elseif (cus/cu)<0.15 then
  lc=37
  recu=0.89
  reau=0.52
 Elseif (cus/cu)<0.3 then
  lc=34
  recu=0.84
  reau=0.55
 Else
  lc=32
  recu=0.77
  reau=0.52
 End If
End If

REM dmtcpl= dry metric tonnes of concentrate at the plant
REM pay=(lc-1)/lc*100 payability

If lc>0 then
 dmtcpl= (cu*rcu)/lc
 pay=(lc-1)/lc
Else
 dmtcpl=0
 pay=0
End If
```

```
REM wmtcpl= wet metric tonnes of concentrate at the plant (9.5% humidity)
REM dmtcpu= dry metric tonnes of concentrate at the port (- transport losses)
REM wmtcpu= wet metric tonnes of concentrate at the port

wmtcpl= dmtcpl * (1+9.5/100)
dmtcpu=dmtcpl * (1- 0.3/100)
wmtcpu=dmtcpu*(1+ 9.5/100)

REM lbpag = pounds of payable copper
REM aucon = gold grade (g/t) of the concentrate
REM ozpag = ounces of payable gold (90% payability)

lbpag=dmtcpu*(lc/100)*(pay/100)*2204.62

if dmtcpl>0 then
 auconc=(au*rau)/dmtcpl
else
 auconc=0
end if

ozpag=dmtcpu*(auconc/31.103)*(90/100)

REM off-site costs = mkt con + transp con + port & insurance + shipping con + TC/RC

cmark = 12*wmtcpl
ctran= 37.55*wmtcpl
cport= 20*wmtcpu
cship= 65*wmtcpu
csmel= 90*dmtcpu
cref = 0.09*lbpag + 4*ozpag
c= cmark+ctran+cport+cship+csmel+cref

REM rev= revenue

rev = prcu*lbpag + prau*ozpag

REM royalty= 1% over revenue less off-site costs

if rev>c then
 roy=0.01*(rev-c)
else
 roy=0
end if

if rev>(c+roy) then
 nsr= rev - c - roy
else
 nsr=0
end if

BlockModel.Model( "Standard","NSR (Transmin 18-02) (19-04)", Column, Row, Level )=nsr
```

Figure 15-1 – NSR calculation (US$/t)

15.3.4 Slope angles

Overall slopes angles previously used for the pit optimization process were based on the configuration of a pit design prepared in May 2014, with associated geotechnical zones defined in a slope design study prepared by Knight Piésold (see Reference 27.15.1). The geotechnical recommendations were adjusted to incorporate recommendations from reviews performed by Piteau Associates in 2014 (see Reference 27.15.2) and 2015 (see Reference 27.15.3) on updated pit configurations. These recommendations have been reviewed and accepted by NCL. The revised overall slope angles are shown in Figure 15-2.

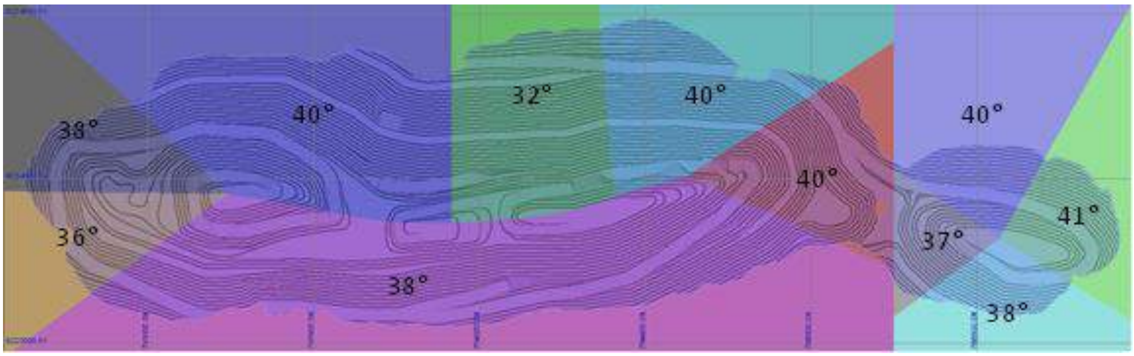


Figure 15-2 – Overall slope angles for pit optimization

15.4 Mine Optimization

15.4.1 Pit optimization results

The Base Case pit optimization was considering the following assumptions:

1. 15 m E x 15 m N x 12 m EL model
2. Undiluted
3. Resources classified as Measured and Indicated (Inferred as waste)
4. Supergene, Hypogene and Mixed to Flotation (Cu and Au)
5. No oxide treatment.

Table 15-3 shows the results of the Base Case optimization run for Zafranal. Nested pit shells were generated for several revenue factors applied to the base case values. Only Measured and Indicated Mineral Resources were used. Figure 15-3, Figure 15-4 and Figure 15-5 show those results graphically.

Whittle pit shell #41 is the revenue factor 1.0 case, chosen as the basis for the detailed ultimate pit design. This shell contains 396 Mt of ore to flotation feed, with an average copper grade of 0.40% Cu. The total tonnage in the shell is 938 Mt for a strip ratio of 1.37 to 1.

15.4.2 Sensitivity analysis

Pit optimization sensitivity analysis was carried out on main variables:

- Metal prices: ±10%, ±20%
- Mining cost: -10%, +10%, +20%, +30%. The mining cost is not expected to decrease by more than 10% or increase by more than 30%, which can be considered comparable with a contract mining option.

- Processing Cost + G&A: ±10%, ±20%
- Slope angles: ±2°, ±5°
- Measured, Indicated and Inferred resources, compared with Measured and Indicated only.

In spite of the variation in economic results for the selected sensitivities, the minimum cutoff grade was not allowed to fall below 0.15%Cu. The basis for this decision was the low reliability of projected metallurgical recoveries for material below the marginal cut-off grade.

As expected, the major impacts reflect variations on the metal prices, with an impact of -25%/+1% on ore tonnes and -15%/+9% on recovered copper.

The rest of the variables have smaller but similar impacts in these results, with a greater impact on ore tonnes than on recovered copper. This is because the decrease/increase of the cut-off grade adds/subtracts significant amounts of marginal tonnes, which have less impact on overall contained metal.

Variability in pit slope anges yielded the smallest impact on the ore tonnes and contained copper, representing -6%/+9% and -4%/+6% respectively for a ±5° variation.

The 30% increase in mining cost to US$2.41/t can be viewed as comparable to a contract mining option. This option for LOM would have an effect of -13% on ore tonnes, but only -8% on contained copper.

The location of mine infrastructure, notably the primary crusher, was tested with an expanded pit shell using a 20% increase in copper and gold prices, and inferred mineral resources. This sensitivity confirmed that any realistic potential pit expansion would not affect mine related infrastructure as shown in Figure 15-8.

Table 15-3 – Pit optimization results – base case

Pit	Revenue factor	Total material Mt	Strip ratio	Flotation: supergene + hypogene + mixed								NPV (US$M)		Cost					
				Mt	%Cu	Au g/t	kt Cu rec	koz rec	Mlb Cu pay	koz pay	Mlb CuEq pay	Best case	Worst case	Mine (US$/t)	Flot+ G&A (US$/t)	Off-site (US$/lb)	Royalty (US$/lb)	Total (US$/lb)	Increm (US$/lb)
1	0.20	31	2.45	8.9	1.21	0.12	94	18	200	16	206.9	412	412	1.79	5.72	0.39	0.03	0.93	0.93
5	0.28	69	1.77	25	0.97	0.11	213	47	454	42	471.0	859	858	1.80	5.74	0.39	0.03	0.99	1.14
6	0.30	122	1.79	44	0.87	0.11	332	81	710	73	738.9	1,255	1,234	1.81	5.74	0.39	0.03	1.05	1.16
10	0.38	151	1.49	60	0.77	0.10	407	105	869	95	907.5	1,461	1,418	1.82	5.76	0.39	0.03	1.10	1.40
11	0.40	208	1.77	75	0.74	0.10	488	129	1,042	116	1,088.4	1,658	1,588	1.83	5.76	0.39	0.03	1.16	1.45
12	0.42	347	1.76	126	0.65	0.09	719	201	1,535	181	1,607.5	2,112	1,918	1.83	5.76	0.39	0.03	1.26	1.47
15	0.48	419	1.51	167	0.59	0.09	860	257	1,838	232	1,930.8	2,324	2,024	1.84	5.79	0.39	0.03	1.32	1.68
20	0.58	489	1.37	206	0.54	0.08	979	301	2,091	271	2,199.6	2,450	2,030	1.84	5.81	0.40	0.03	1.38	1.93
25	0.68	560	1.26	247	0.50	0.08	1,082	343	2,312	308	2,434.8	2,522	1,969	1.84	5.83	0.40	0.03	1.45	2.17
30	0.78	653	1.23	293	0.46	0.08	1,187	387	2,537	348	2,675.9	2,565	1,870	1.85	5.84	0.41	0.03	1.52	2.42
35	0.88	772	1.28	339	0.43	0.07	1,292	428	2,761	385	2,914.7	2,586	1,720	1.85	5.86	0.41	0.03	1.61	2.66
38	0.94	854	1.32	368	0.42	0.07	1,356	459	2,896	413	3,061.6	2,592	1,649	1.86	5.87	0.41	0.03	1.66	2.82
39	0.96	900	1.34	384	0.41	0.07	1,389	472	2,967	425	3,137.3	2,594	1,605	1.86	5.87	0.42	0.03	1.69	2.87
41	1.00	938	1.37	396	0.40	0.07	1,415	485	3,022	436	3,196.8	2,594	1,571	1.86	5.87	0.42	0.03	1.72	2.98

Notes:
kt Cu rec: Thousand tonnes of recovered copper
koz rec: Thousand troy ounces of recovered gold
Mlb Cu pay: Million pounds of payable copper
koz pay: Thousand troy ounces of payable gold
Flot+G&A: Flotation processing cost plus general and administrative cost
Increm: Incremental cost

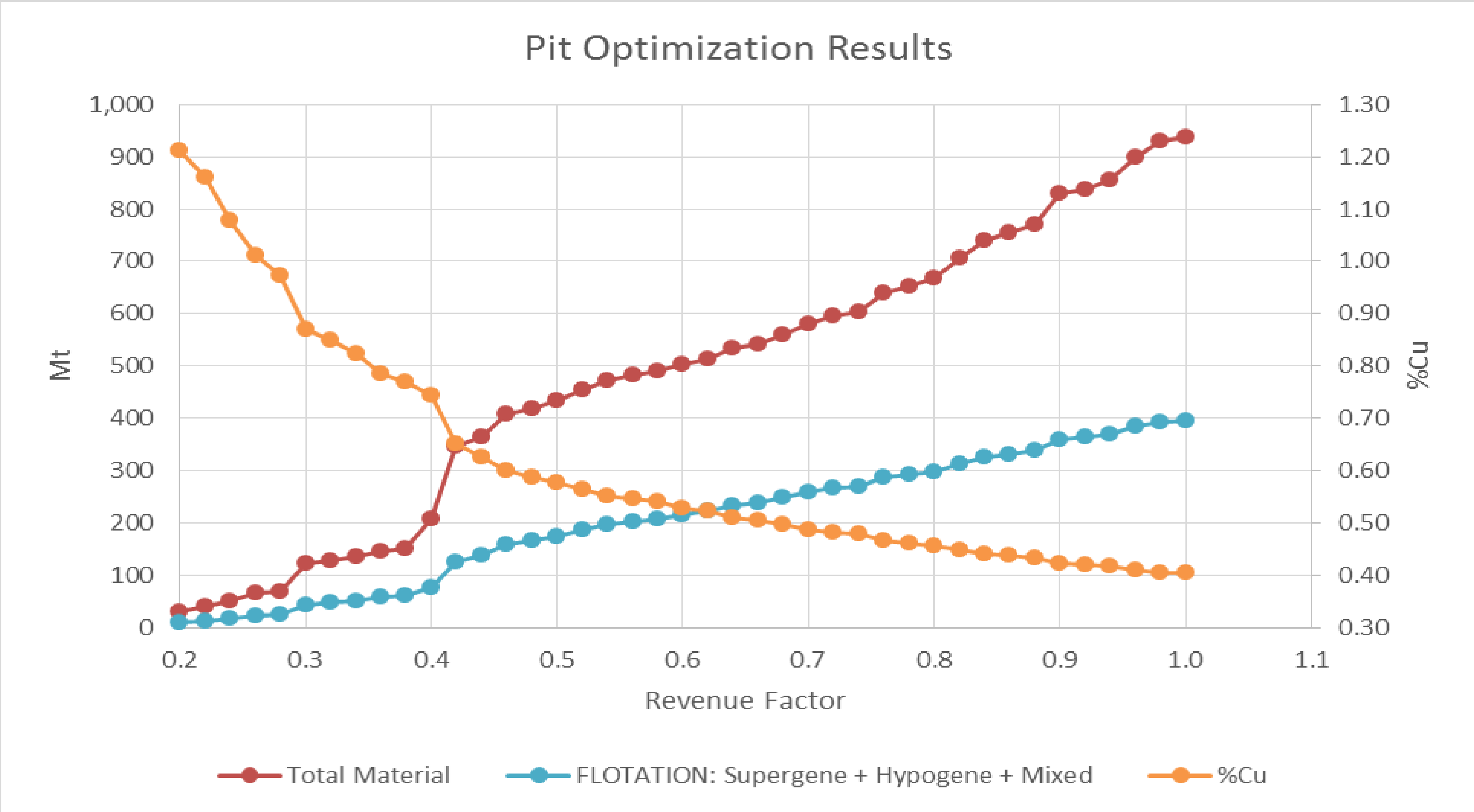


Figure 15-3 – Pit optimization results

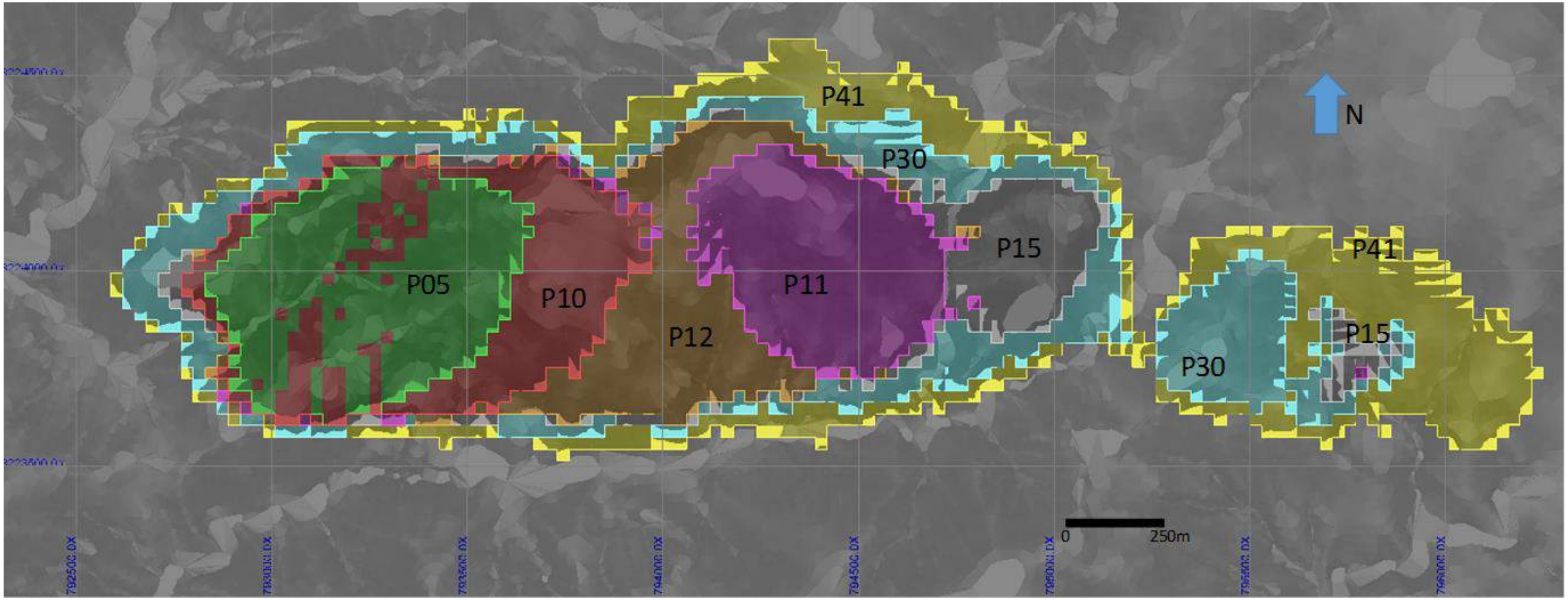


Figure 15-4 – Nested pit shells – plan view

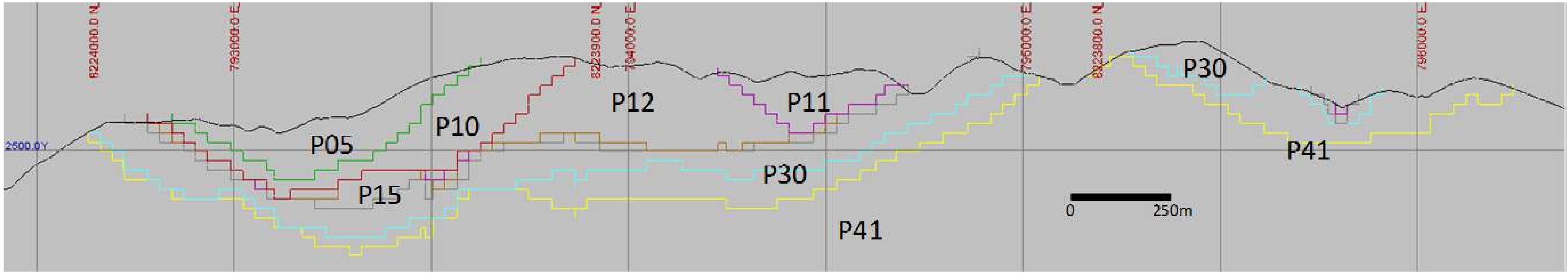


Figure 15-5 – Nested pit shells – cross section

Table 15-4 – Pit optimization sensitivity analysis

Option	Total mined Mt	Strip ratio	Flotation: supergene + hypogene + mixed					Comments
			Concentrator feed			Recovered metal		
			Mt	%Cu	Au g/t	kt Cu	koz	
Base Case	938	1.37	396	0.404	0.07	1,415	485	Revenue factor 1
Sensitivity 1.1	828	1.31	359	0.422	0.07	1,337	449	Prices -10%
Sensitivity 1.2	668	1.24	298	0.457	0.08	1,200	392	Prices -20%
Sensitivity 1.3	1,067	1.46	434	0.389	0.07	1,493	518	Prices +10%
Sensitivity 1.4	1,152	1.53	455	0.382	0.07	1,537	536	Prices +20%
Sensitivity 2.1	1,026	1.43	422	0.394	0.07	1,469	509	Mining Cost -10%
Sensitivity 2.2	898	1.34	384	0.410	0.07	1,388	472	Mining Cost +10%
Sensitivity 2.3	835	1.30	362	0.420	0.07	1,342	452	Mining Cost +20%
Sensitivity 2.4	780	1.28	343	0.430	0.07	1,300	431	Mining Cost +30%
Sensitivity 3.1	1,002	1.41	416	0.396	0.07	1,455	502	Processing Cost + G&A -10%
Sensitivity 3.2	1,069	1.46	435	0.389	0.07	1,495	519	Processing Cost + G&A -20%
Sensitivity 3.3	895	1.34	382	0.411	0.07	1,385	471	Processing Cost + G&A +10%
Sensitivity 3.4	828	1.31	359	0.422	0.07	1,336	449	Processing Cost + G&A +20%
Sensitivity 4.1	990	1.49	398	0.403	0.07	1,417	487	Slope angles -2°
Sensitivity 4.2	960	1.58	372	0.413	0.07	1,359	459	Slope angles -5°
Sensitivity 4.3	961	1.24	428	0.392	0.07	1,486	516	Slope angles +2°
Sensitivity 4.4	973	1.25	433	0.390	0.07	1,495	519	Slope angles +5°
Sensitivity 5.1	948	1.31	410	0.399	0.07	1,446	494	MI+Inferred

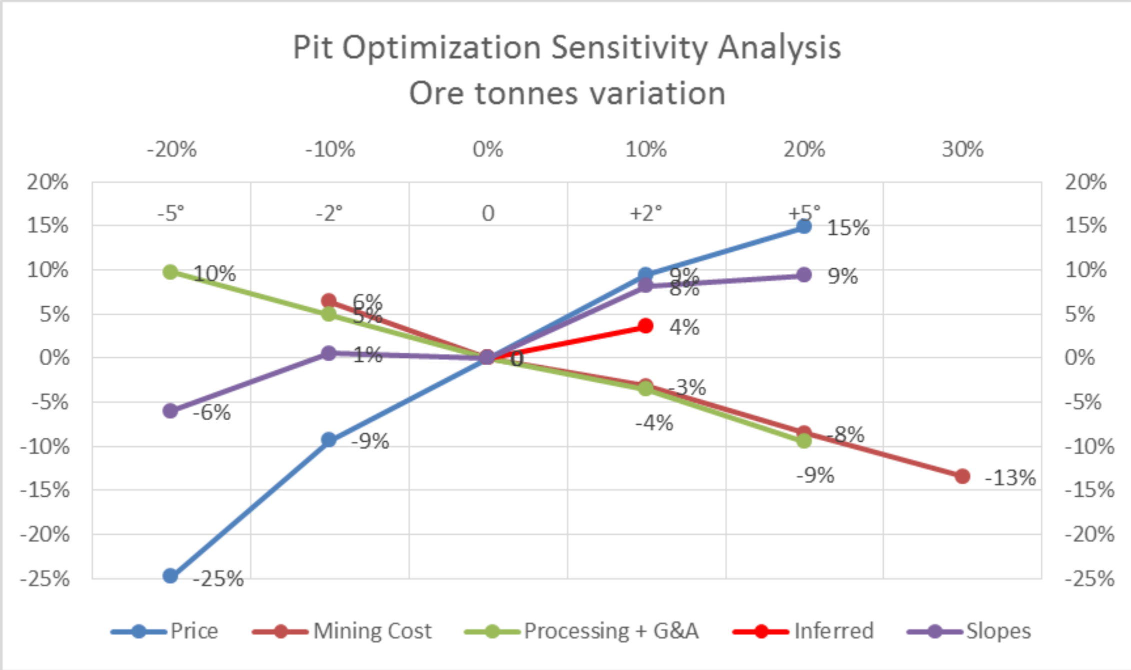


Figure 15-6 – Sensitivity analysis - ore tonnes variations

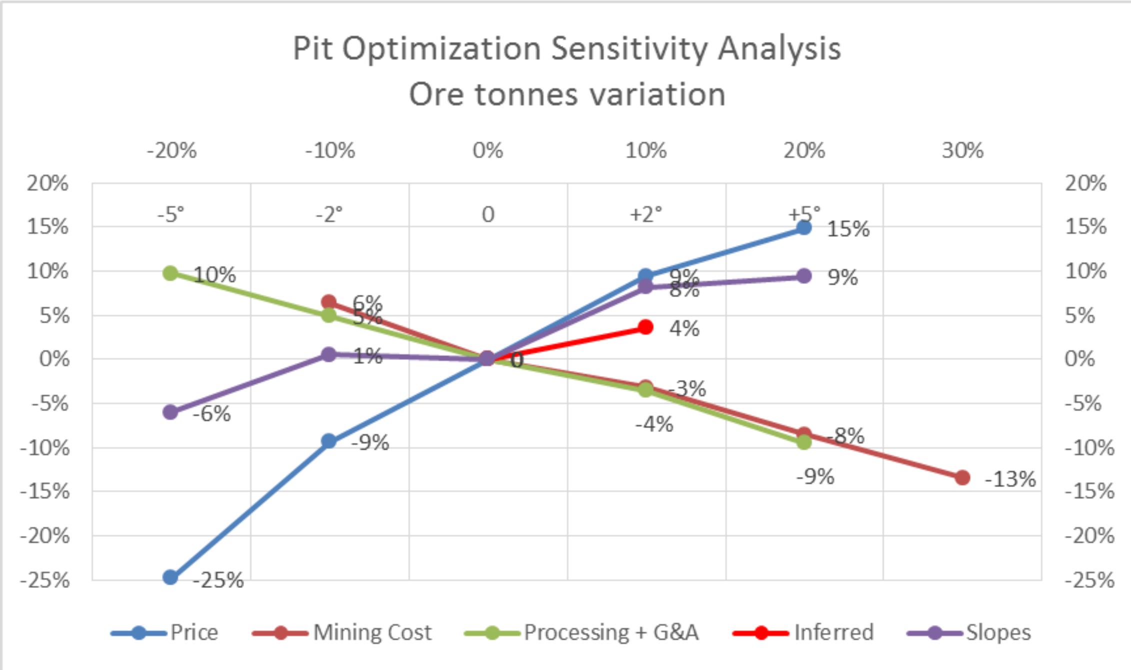


Figure 15-7 – Sensitivity analysis – recovered copper variations

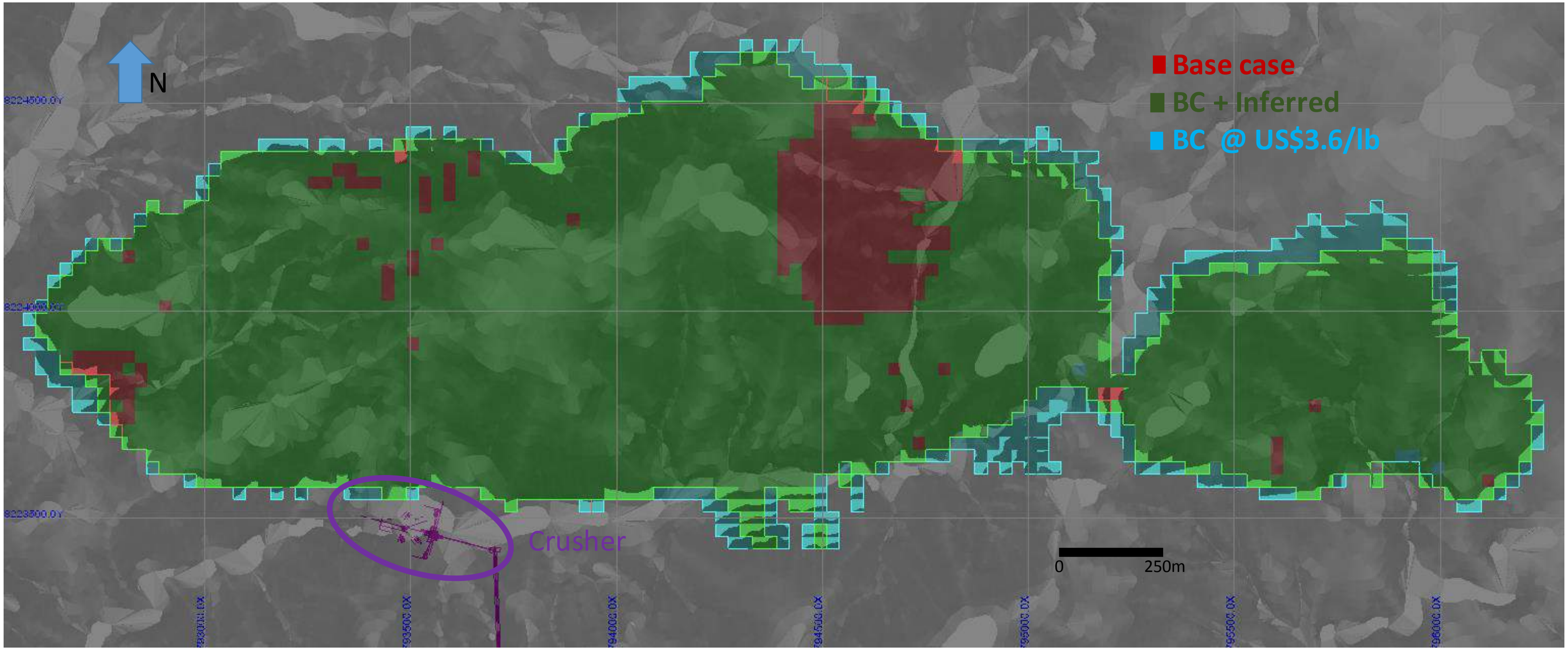


Figure 15-8 – Pit shells showing crusher location

15.5 Mineral Reserve Statement

The Mineral Reserves estimate for the Zafranal Project has been prepared using industry best practices and conforms to the requirements of Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves - Definitions and Guidelines (2014).

The Mineral Reserves estimate is summarized in Table 15-5 and has an effective date of 31 March 2016. The Qualified Person for the estimate is Carlos Guzman, CMC and FAusIMM, an NCL employee.

Table 15-5 – Zafranal mineral reserves statement

Reserve category	Ore		Ore grade		Contained metal	
	Type	Mt	% Cu	g/t Au	Mlbs Cu	koz Au
Proven Mineral Reserves						
	Mixed	0.4	0.48	0.11	4	1
	Supergene	97	0.61	0.08	1,308	233
	Hypogene	105	0.28	0.07	660	249
Total Proven Mineral Reserves		*202*	*0.44*	*0.07*	*1,972*	*483*
Probable Mineral Reserves						
	Mixed	2	0.40	0.11	16	6
	Supergene	78	0.50	0.06	861	156
	Hypogene	118	0.27	0.06	694	246
Total Probable Mineral Reserves		*198*	*0.36*	*0.06*	*1,571*	*408*
Total Mineral Reserves (proven and probable)						
	Mixed	2	0.41	0.11	20	8
	Supergene	175	0.56	0.07	2,169	389
	Hypogene	224	0.27	0.07	1,354	495
Total Mineral Reserves (proven and probable)		*401*	*0.40*	*0.07*	*3,543*	*891*

Notes to accompany mineral reserves table

1. The Qualified Person for the estimate is Carlos Guzman, CMC and FAusIMM, an NCL employee. Mineral reserves have an effective date of 31 March 2016.
2. Mineral reserves are reported as constrained within measured and Indicated pit designs, and supported by a mine plan featuring variable cut-off. The pit designs and mine plan were optimized using the following economic and technical parameters: metal prices of US$3.0/lb Cu and US$1,200/oz; recovery to concentrate assumptions according to geometallurgical domains for Cu and Au; copper concentrate treatment charges of US$90/dmt, U$0.09/lb of Cu refining charges and US$4.0/oz of Au refining charges; concentrate charges of US$12/wmt for marketing, US$37.55/wmt for road transport, US$20/wmt for port and insurance, US$65/wmt for shipping and 0.3% for transport losses; average payability of 96.9% for Cu and 90% for Au; average mining cost of US$1.86/t, process costs of US$4.47/t for mixed and supergene materials and US$4.75/f for hypogene, and G&A US$1.25/t processed; average pit slope angles that range from 36° to 41°; a 1% Government royalty rate assumption, and an assumption of 100% mining recovery.
3. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.
4. Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces.

15.6 General Consideration of Other Factors

In the opinion of the QP, the main factors that may affect the Mineral Reserve estimates are metallurgical recoveries and operating costs (fuel, energy and labour). The base price of copper, even though it is the most important factor for revenue calculation, does not affect the Mineral Reserves estimate as a manual cut-off of 0.15%Cu is being considered, which is higher than the grade cut-off obtained from the base case economic evaluation. The selected grade cut-off allows for a broad swing in metal prices before they have an effect on the Mineral Reserves estimate.

Other than the risks identified in this report, NCL is not aware of any other environmental, permitting, legal, title, taxation, socio-economic or political factors that could materially affect the Mineral Reserve estimate.

16 Mining Methods

16.1 Mine Geotechnical and Geochemical Considerations

16.1.1 Mine geotechnical considerations

Minera AQM Copper Perú S.A.C. (MAQM) retained Knight Piésold Consultants S.A. (Knight Piésold) to conduct the prefeasibility level geotechnical study for the Zafranal and Victoria pits of the Zafranal Project. The main objective was to determine the angles of the bench slopes, inter-ramp slopes, and overall slopes, for the corresponding open pit design.

The proposed Zafranal mine will develop two open pits (Zafranal Main and Victoria) with a combined area of 2.47 km^2. The final pits will be 2.6 km long and 1.0 km wide and shall reach a maximum depth of 554 m.

Knight Piésold completed a geotechnical site investigation program between May and October of 2012. This program included:

- Structural geological mapping
- 2017 m of drilling over 6 oriented drill holes
- Geomechanical mapping using the cell method in 30 stations
- 350 strength tests with a Schmidt Hammer
- 339 point load tests (PLTs) of select rock samples from the drilling and mapping programs.

The strength of the intact rock was determined by PLT and laboratory uniaxial/unconfined compressive strength (UCS) testing. The test results indicate that the rocks of the Zafranal and Victoria pit areas range from very poor to very high, dependent on depth, weathering degree, cut areas and hydrothermal alteration of the rock. The average strength of the volcanoclastics and the Zafranal diorite is intermediate (48 and 44 MPa, respectively) and of the intrusive rocks is intermediate to high (variable between 69 and 133 MPa). The strength of the volcanoclastic rocks and of the mylonitized intrusive (fault-influenced zones) rock, is at an average of 6 MPa in certain cases.

The quality of the rock mass was assessed using the rock mass rating (RMR) classification system (Bieniawski, 1989). The RMR ranges widely within the pits, from poor to good quality rock. Seven geotechnical units (GUs) have been defined for the purpose of completing the pit slope design, labeled GU-I to GU-VII. GU-I is formed by quaternary deposits, GU-II is formed by fault areas, shear (mylonite) zones and fragmented rock areas, and GU-III to GU-VII are formed by rock material with different levels of rock quality, from poor (RMR less than 35) to good (RMR over 60). Table 16-1 shows the distribution of the rock mass ratings for the Zafranal and Victoria pit.

Table 16-1 – Distribution of the RMR for the Zafranal and Victoria Pit

Geotechnical Unit	Quality according to RMR	Zafranal (%)	Victoria (%)
I	Soil	<1	0
II	Very bad (RMR<=35)	2	2
III	Bad (RMR: 36-40)	10	6
IV	Regular (RMR: 41-45)	35	25
V	Regular (RMR: 46-50)	20	20
VI	Regular (RMR: 51-60)	30	45
VII	Good (RMR>=61)	3	2

Seven design sectors have been defined for the Zafranal Main pit and four for the Victoria pit. Each one of these sectors groups the areas with similar geotechnical characteristics. Kinematic analyses have been performed to identify potential failure modes within the pit walls in each design sector. Additionally, inter-ramp and overall slope stability was assessed using limit equilibrium methods.

The kinematic analyses of bench and inter-ramp slopes assumed a friction angle of 33° for minor structures (joints) and 30° for major structures. The results of the kinematic analyses indicate that variable inter-ramp slopes between 45° and 48° can be achieved. The exposure of fault zones along the upper pit walls would result in flatter slope angles.

The limit equilibrium rock mass stability analyses targeted a factor of safety of 1.3 for static conditions and 1.0 for pseudo-static (earthquake) conditions. The results of the limit equilibrium analyses indicate that all design sectors for the Zafranal Main and Victoria pits are stable (assuming a blasting disturbance factor (D) of 0.85), except for two sectors, where the calculated factor of safety is below the acceptable minimum.

These two sectors were analyzed considering slope depressurization and simulating controlled blasting techniques to reduce the blasting disturbance factor (D). The results of this analysis indicate that the targeted FS could be achieved when a lower disturbance factor (D) of 0.7 is applied; therefore, it is recommended to use good controlled blasting practices and implement an instrumentation installation and slope monitoring program.

AQM requested Piteau Associates Perú S.A.C (Piteau) to conduct a review of the geotechnical study prepared by Knight Piésold. The main objectives of the review were to conduct a brief review of the design processes and design criteria summarized in Knight Piésold's 2013 PFS study report and prepare recommendations for any supplementary work required to bring the project to a PFS level. Two phases of reviews were carried out by Piteau: June 2014 and May 2015. These reviews resulted in reduced overall slope angles compared with the original Knight Piésold (KP) study as a result of the incorporation of geotechnical berms 25 m wide incorporated for every 144 m of wall height for a single bench configuration of 12 m height with constant berm width of 7.5 m and face bench angle of 65°, These revised criteria were reviewed by NCL and considered reasonable for mine design purposes.

16.1.2 Mine geochemical considerations

Waste rock material from both the Zafranal and Victoria pits was evaluated for geochemical characteristics (acid generating potential and metals leaching). A wide distribution of sulfur, predominantly as chalcopyrite and pyrite, was reported between the various rock types at the two deposits. Median sulfur values typically ranged from 1% to 3%. These values are consistent with porphyry deposits at which acid rock drainage (ARD) or metal leaching (ML) or both are important waste management issues. Limited carbonate within the Zafranal rocks suggests that substantial buffering capacity or carbonate mineral dissolution from the waste rock is not expected. ABA results suggest that the majority of waste rock associated with both deposits should be considered potentially acid generating (PAG) with roughly 75% to 85% of samples tested and classified as PAG. Samples classified as clearly non-acid generating account for between approximately 15% to 25% and are largely sourced from the leached cap zones of both deposits and potentially from a portion of the later-stage diorite intrusives. All waste material is assumed to be PAG and the design criteria for waste dumps consider this, as discussed in Section 18.1.

Greater early-stage metal leaching potential likely exists in rocks from the supergene and mixed zones of the deposits where soluble secondary copper mineral forms occur. If contact water remains buffered or slightly alkaline, metals and metalloids such as Mo, As, Sb and Se may be mobile. If contact water becomes acidic, increased concentrations of other parameters such as Al, Cu, Cd, Fe, Ni, and Zn, etc. would be expected. Management of mine contact water is discussed in Section 20.

16.2 Hydrogeology and Hydrology Considerations

16.2.1 Mine hydrogeology considerations

General aspects related to the open pit hydrogeology as it relates to mine dewatering are discussed here. Additional details and potential effects from open pit development are discussed as part of Section 20.1.2. At the Zafranal Main open pit, groundwater occurs at depths of approximately 120 to 180 metres below ground level (mbgl) along the north fault and 50 to 100 mbgl along the south fault. Groundwater in higher elevations of the open pit is commonly found between 230 to 250 mbgl. The north and south faults bound the mineral deposit and are believed to be principal hydrogeological features that control groundwater movement. Several cross-cutting fault zones have been mapped within the open pit area and are likely to be in hydraulic connection with both the north and south faults. Pre-mining groundwater movement follows principal faults, from east to west as shown in Figure 16-1, towards Quebrada Ganchos and to a lesser extent Quebrada Huacan located near the TMF.

Ore extraction will occur from the development of two open pits: Zafranal Main and Victoria. Mining of the Zafranal Main open pit will occur beneath the water table within 2 to 3 years and require dewatering infrastructure. Ex-pit and in-pit dewatering wells will be installed during operations beginning at Year 1; additionally, horizontal drains will be also be installed beginning at Year 2 along with in-pit sumps and ditches. The main components of the dewatering system are shown in Figure 16-1. The Victoria pit is expected to be mined under dry conditions. At the end of mining, the Zafranal Main pit will be excavated approximately 200 m below groundwater within the central portion of the pit. Groundwater inflow is expected to be low (of the order of 5 to 15 L/s throughout operations) due to limited saturated thickness, low-permeability rock mass, and low recharge.

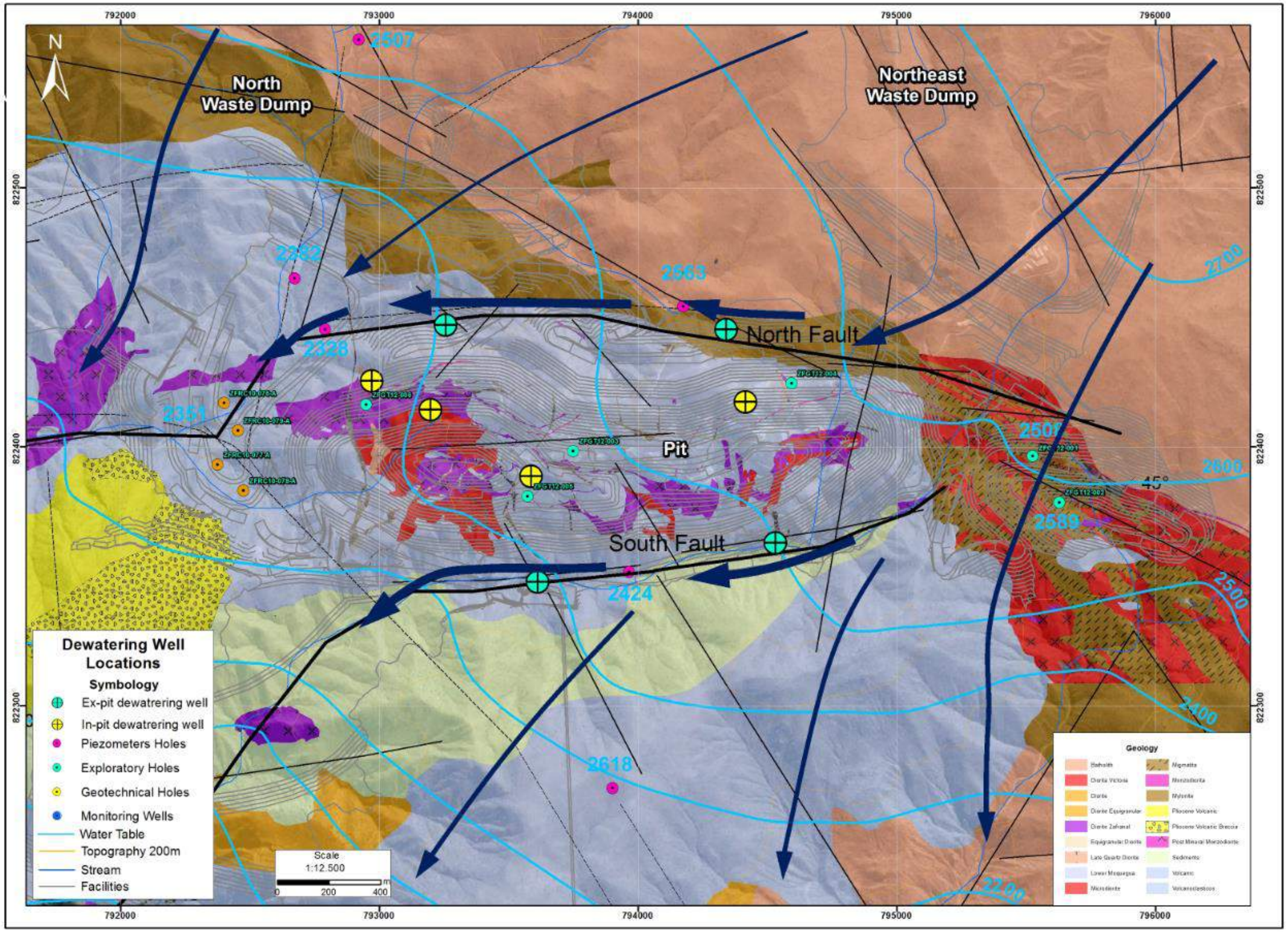


Figure 16-1 – Pre-mining groundwater flow patterns and proposed dewatering well locations (preliminary)

16.2.2 Mine hydrology considerations

In the area of the pit and waste dumps the average annual flow, average annual volume and annual yield for runoff from each watershed is summarized in Table 16-2 as well as peak flows, catchment area and peak flow to area ratio for streams inside the catchments.

Table 16-2 – Flow rates and yield for the catchments surrounding pit and waste dumps

Installation	Annual volume	Average flow	Peak flow - 100 year ($Q_{max\text{-}100\,y}$) of internal catchments	Areas (A) of internal catchments	$Q_{max\text{-}100\,y}$ / A in internal catchments
	Mm^3	(L/s)	(m^3/s)	(km^2)	($m^3/s/km^2$)
Pit and north waste dump	0.093	2.95	0.38 - 12.43	0.039 - 3.725	3.34 – 9.69
Pit and northeast waste dump	0.029	0.91	0.06 - 5.02	0.007 - 1.000	5.02 – 8.73
Central waste dump	0.004	0.14	0.11 - 1.39	0.010 - 0.164	8.60 – 10.46

Management of surface water run-off outside the pit and around the waste dumps is described in Section 18.3.

16.3 Mine Design

The final pit design was based on the economic shell using the 2015 Mineral Resource estimate with a 1.0 revenue factor, and variable overall slope angles between from 32° to 41°, according to geotechnical domains.

The block model is considered to be a fully diluted resource model; hence, NCL performed pit optimization and mine planning activities without introducing any further mine dilution. NCL also considered a 100% ore mining recovery due to the disseminated characteristics of the ore and the proximity of the economic cut-off to the background copper content of the rock. This assumption requires the mine to adopt strict grade control practices to minimize any misclassification of ore grade material. Mineralization type also has significant impacts on metallurgical recoveries in the flotation plant, further increasing the need for detailed mine planning oversight.

A set of nine mining phases was developed based on the sequence of nested pits obtained from the pit optimization. As a result of this approach, the mine plan is able in most cases to schedule the phase development based on its Net Smelter Return, from highest to lowest over the life of mine.

16.3.1 Final pit design

The final pit design reflects the decision to employ a stripping contractor for the pre-mine and early production years (to Year 4), in that the bench height and block size (SMU) were tailored for the use of medium-sized diesel powered equipment. The selected SMU also had the added benefit of producing the least amount of grade dilution within the Block Model.

The key open pit design parameters are shown in Table 16-3.

Table 16-3 – Mine design parameters

Parameter		Unit	Value
Haul road width		m	38 - 26
Maximum haul road grade		%	10
Bench height		m	12
Nominal minimum mining phase width		m	100
Continuous inter-ramp vertical height		m	144
Safety berm		m	25
Geotechnical parameter	Batter height (m)	Batter angle (°)	Berm width (m)
All pit walls	12	65	7.5

Slopes angles used for the pit design process were in accordance with the geotechnical recommendations of reviews performed by Piteau in 2014[1] and 2015[2]. These recommendations resulted in reduced overall slope angles compared with the original KP study[3].

The road width of 38 m will accommodate the selected 220 t trucks for two-lane traffic. NCL used a maximum 10% road gradient, which is common in the industry for this type of truck. A traffic analysis was carried out on the pit design and for those segments of ramp in the bottom benches of a phase with fewer than 12 trucks per hour the design changed to single-lane traffic and the width was reduced to 26 m. The single-lane design was limited to a maximum of 6 benches from the pit bottom on the final pit and each of the mining phases.

The current mine plan and operating costs were developed for 12 m benches in all sectors, ore and waste. Additional 25 m wide safety berms were included in the design when the slope exceeds 144 m of vertical height, accordingly to the geotechnical recommendations. Figure 16-2 shows the final pit design.

[1] Piteau Associates, Memorandum, Zafranal Project – Review of Knight Piésold Prefeasibility Study; June 26, 2014.

[2] Piteau Associates, Memorandum, Review of Prefeasibility-Level (PFS) Requirements for the Zafranal Project; May 1, 2015.

[3] Knight Piésold; Zafranal and Victoria Open Pit Slope Design; Prefeasibility Study; Final Report; May 27, 2013

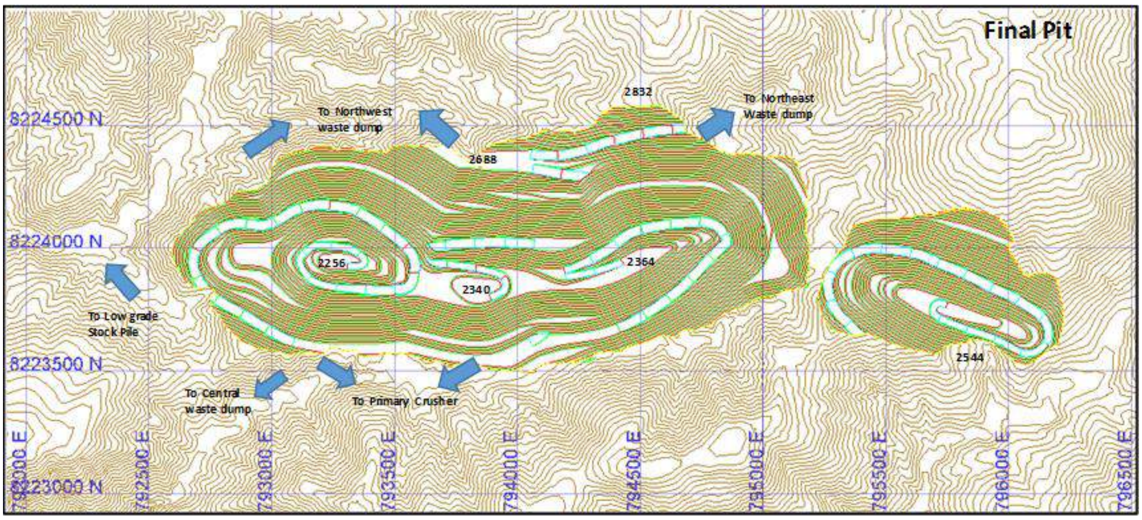


Figure 16-2 – Final pit design

The Zafranal Main pit has two exits on the south side and two on the north side to provide access to the, primary crusher, stockpiles and waste dumps. The main pit is 2,500 m long in the east-west direction and 1,000 m wide in the north-south direction. The pit bottom is at 2,280 m elevation, at the western end. The highest wall is about 456 m on the northeast side. The total area disturbed by the main pit is about 250 hectares.

To the east of the Main Pit, a small Victoria Pit was designed 950 m long and 500m wide, with the pit bottom at 2,532 m elevation, the highest wall of 180 m to the north and with 47 hectares of disturbed area.

Table 16-4 shows the contained resources by mineral zone, resource category and cut-off. The table includes only Measured and Indicated Resources using the updated CMZ 2015 block model. Inferred Resources are considered to be waste material.

Table 16-4 – Final pit by mineral zone, resource category and cut-off

Final pit	Measured											
Cut-off	Mixed to concentrator			Supergene			Hypogene			Total to concentrator		
%Cu	Mt	%Cu	Au g/t	Mt	%Cu	Au g/t	Mt	%Cu	Au g/t	Mt	%Cu	Au g/t
0.50	0.2	0.65	0.13	51	0.87	0.10	4	0.57	0.13	55	0.85	0.10
0.40	0.2	0.59	0.12	63	0.79	0.09	13	0.48	0.12	76	0.74	0.10
0.30	0.3	0.51	0.11	76	0.71	0.09	38	0.39	0.10	115	0.61	0.09
0.25	0.4	0.49	0.11	84	0.67	0.08	60	0.35	0.09	145	0.54	0.08
0.20	0.4	0.49	0.11	92	0.64	0.08	84	0.31	0.08	175	0.48	0.08
0.15	0.5	0.48	0.11	97	0.61	0.08	105	0.28	0.07	202	0.44	0.07
Final pit	Indicated											
Cut-off	Mixed to concentrator			Supergene			Hypogene			Total to concentrator		
%Cu	Mt	%Cu	Au g/t	Mt	%Cu	Au g/t	Mt	%Cu	Au g/t	Mt	%Cu	Au g/t
0.50	0.5	0.66	0.13	30	0.80	0.09	4	0.57	0.13	35	0.78	0.10
0.40	1	0.58	0.13	41	0.71	0.08	12	0.48	0.12	54	0.66	0.09
0.30	1	0.50	0.12	54	0.62	0.08	34	0.39	0.10	90	0.53	0.08
0.25	1	0.44	0.11	62	0.58	0.07	54	0.35	0.08	117	0.47	0.08
0.20	2	0.42	0.11	70	0.54	0.07	83	0.31	0.07	155	0.41	0.07
0.15	2	0.40	0.11	78	0.50	0.06	118	0.27	0.07	198	0.36	0.06
Final pit	Measured + Indicated											
Cut-off	Mixed to concentrator			Supergene			Hypogene			Total to concentrator		
%Cu	Mt	%Cu	Au g/t	Mt	%Cu	Au g/t	Mt	%Cu	Au g/t	Mt	%Cu	Au g/t
0.50	1	0.66	0.13	82	0.85	0.10	8	0.57	0.13	90	0.82	0.10
0.40	1	0.59	0.12	104	0.76	0.09	25	0.48	0.12	130	0.70	0.09
0.30	1	0.50	0.12	131	0.68	0.08	72	0.39	0.10	204	0.57	0.09
0.25	2	0.45	0.11	146	0.63	0.08	114	0.35	0.09	262	0.51	0.08
0.20	2	0.43	0.11	162	0.59	0.07	166	0.31	0.08	330	0.45	0.07
0.15	2	0.41	0.11	175	0.56	0.07	224	0.27	0.07	401	0.40	0.07
Final pit	Mixed and Oxide to stockpile											
	18	Mt	(above 0.15%CuSS+CuCn)									
	0.25	%CuSS+CuCN										
Final Pit	Waste and TOTAL material											
	566	Mt	Waste									
	984	Mt	TOTAL									

Notes:
%CuSS+CuCn: The sum of sulfuric acid soluble copper and cyanide soluble copper

16.3.2 Mining phases design

As described earlier, the mining phases were designed on the basis of their relative NSR value, which generally reflected ore grade and stripping ratio. The series of nested pit shells obtained from the pit optimization process was used as the basis of the phase designs.

Phase 1 targets the ore with the highest grade and lowest strip ratio in the west area, down to 2,424 masl. Phase 2 is an expansion to the east, down to 2,376 masl. Phase 3a is an independent pit to the west, targeting a high grade area, down to 2,568 masl; and Phase 3b joins the east with the west, down to 2,472 masl.

Phase 4 is an expansion of the central area, down to 2,364 masl. Phase 5 corresponds to the last expansion of the west side, down to 2,280 masl. Phases 6 and 7 are successive expansions to the east, down to 2,460 masl and 2,388 masl, respectively.

Phase 8 is an independent pit, to the east of the main pit, named Victoria, down to 2,532 masl.

Figure 16-3 shows the phase outlines on the levels 2568 and 2670 masl.

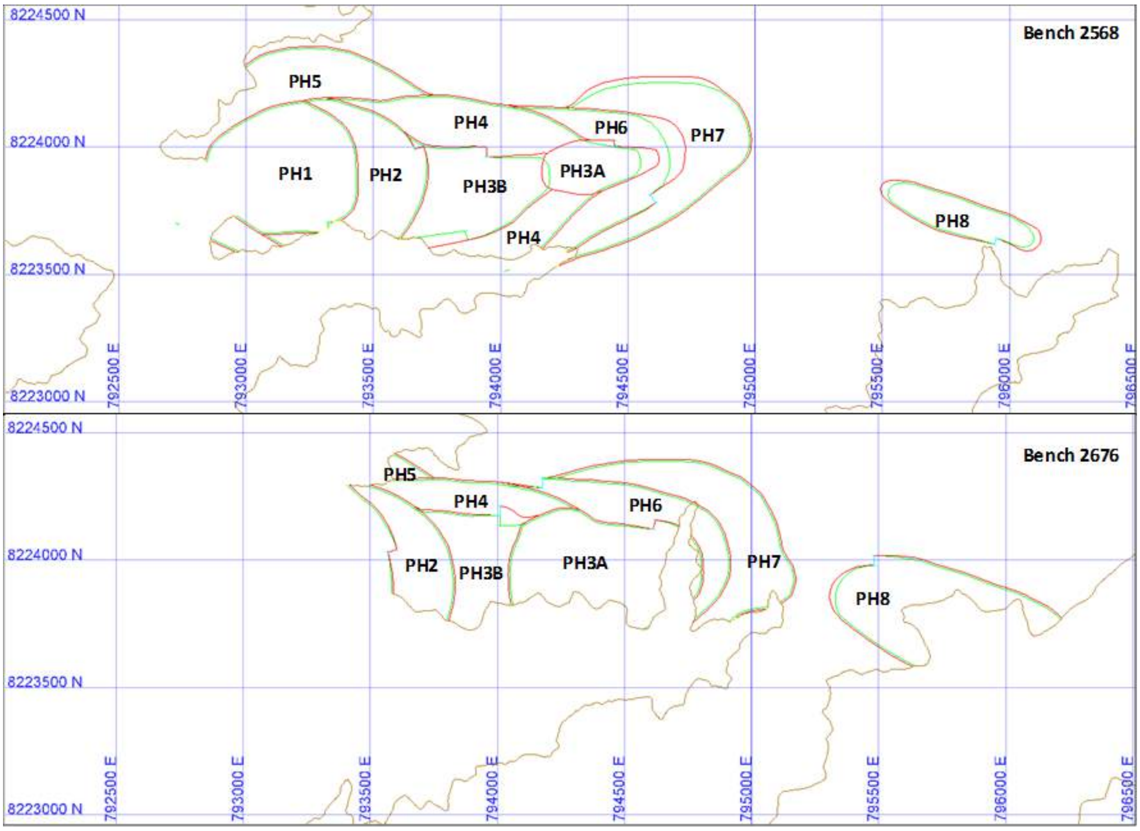


Figure 16-3 – Mining phases outlines

Figure 16-4 shows the development ratio for the mining phases. This ratio is defined as the total tonnes rock of the phase divided by the tonnes of its contained copper. Phases 1 and 2 have a ratio lower than 400, Phase 8 has values higher than 1,200 and the total pit has an average of 613.

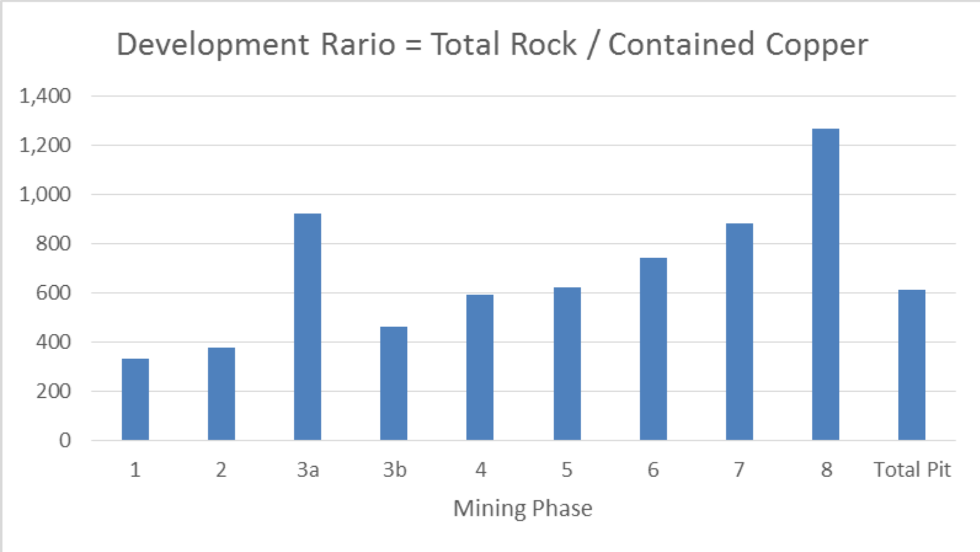


Figure 16-4 – Development ratio per mining phase

16.4 Mining Operations

Pre-stripping and pioneering work is to be performed by a qualified mining contractor. Two such contractors were identified in Peru and both participated in providing budget quotes for the work. On the basis of normalized bids including Owners´ costs, an economic assessment was carried out to determine the optimum duration of the stripping contract and the transition to an Owner-operated mine. The study concluded that a five-year period terminating in Production Year 3 was the preferred outcome as it provided an experienced workforce for challenging earthworks, lower initial capital cost and it resulted in an Owner-operated time frame that minimize replacement of equipment in the latter stages of mine development.

A mine plan was developed for the Zafranal Project to process a variable throughput in the range of 55 to 64 thousand tonnes per day, with a peak total material movement rate of 71 M tonnes per year. The mine is scheduled to work seven days per week or 365 days per year. Each day will consist of two 12-hour shifts. Four mining crews will cover the operation (two working and two on days off).

The study is based on operating the Zafranal mine with 34 m^3 hydraulic excavators and 220 tonne haul trucks. The selected equipment has the capacity and productivity to achieve an annual total material movement of 75 million tonnes during pre-production, without compromising mining selectivity for grade and dilution control.

A modestly over-sized drilling fleet will ensure continuity of blasted material with diesel-powered rigs capable of drilling 251 mm (9 ⅞”) diameter blast holes for ore and waste. Auxiliary equipment includes track dozers, wheel dozers, motor graders and water trucks.

The mine fleet also includes the necessary equipment to re-handle ore from the ROM crusher pad area and low-grade stockpile to feed the primary crusher. This operation will be carried out with a 20 m^3 front-end-loader using 220 tonne haul trucks

16.5 Mine Production Schedule

A mine production schedule was developed to show the ore tonnes, metal grades, waste material and total material by year, throughout the life of the mine (Table 16-7) [4] .

The distribution of ore and waste contained in each of the mining phases was used to develop the schedule, ensuring conformity with criteria such as continuous ore exposure, mining accessibility, and consistent annual material movements.

NCL used an in-house system to evaluate several potential mine production schedules. The required annual ore tonnes and user-specified annual total material movements were provided to the algorithm, which then calculated the alternative mine schedules.

Several runs at various proposed total material movement rates were done to determine an optimum production schedule strategy. While this program is not a simulation package, it is a very useful tool for developing a mine schedule and haulage profiles for a given set of phases and constraints as set by the user.

The schedule is based on process plant variable throughput of 55,000 t/d to 63,600 t/d. The production period mined material movement peaks at 71 Mt/y during Years 1 to 4. The production is limited by the number of benches that it is possible to mine in a single phase in a year, or the amount of vertical development per phase.

As recommended by Ausenco, the concentrator throughput considers three stages, as follows:

- ***Stage1***: First 12 month ramp-up to achieve the design capacity of 55,000 t/d, as per Table 16-5, for a total of 16.0 Mt for the period.
- ***Stage 2***: Second year at design capacity of 55,000 t/d, corresponding to 20.1 Mt.
- ***Stage 3****:* Variable throughput from the third year of commercial production, as a function of the mineral zone, with different vales for mixed, supergene and hypogene, as per Table 16-6, from a minimum of 57,370 t/d if 100% of the plant feed is hypogene to a maximum of 75,536 t/d if 100% of the plant feed is mixed material.

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[4] All tonnages show in this section correspond to dry metric tonnes

Table 16-5 – First year concentrator ramp-up

Month	Throughput (%)	kt/d	kt/month
1	36	19.8	614
2	57	31.4	878
3	63	34.7	1,074
4	70	38.5	1,155
5	80	44.0	1,364
6	83	45.7	1,370
First half year average	**65**	**35.7**	**6,454**
7	87	47.9	1,483
8	92	50.6	1,569
9	96	52.8	1,584
10	96	52.8	1,637
11	97	53.4	1,601
12	100	55.0	1,705
Second half year average	**95**	**52.1**	**9,578**
Annual average	**80**	**43.9**	**16,033**

Table 16-6 – Variable throughput by mineral zone

Mineral zone	Throughput (t/d)
Hypogene	57,400
Supergene	65,300
Mixed	72,500

The total waste mined will be sent to three valley waste dumps in proximity to the open pit using the wrap-around dumping on existing topographic contours to achieve flatter slopes similar to a bottom-up dump without the long initial haulage cycles. French under-drains will be constructed in the base of the dumps before dumping commences in these areas. (Refer to Section 18).

Most mined ore will be hauled to the primary crusher for direct tipping. Low grade ore will be mined and hauled to a stockpile located to the west of the main pit until Year 10. This material will be re-handled and will become part of the concentrator feed in the later years. From Year 11 on, the low grade ore will be sent directly to the crusher. The total stockpiled low grade ore amounts to 38.7 Mt.

The oxide material is treated as waste in the mine plan. No economic process has been defined to treat this material; however, a stockpile area for the approximately 17.6 Mt of oxide material with copper content greater than 0.15% CuSS+CuCn was set aside so this material could be stockpiled for possible future processing.

The pre-stripping and the first five years of commercial production were detailed on a half year basis; and from Year 6 to the end of the life of mine on a yearly basis.

The pre-production period requires the mining of 45 Mt of total material to expose sufficient ore to start commercial production in Year 1. The pre-production period will be approximately 18 months. The ore mined during pre-production will be stockpiled in the low grade area and will make up part of Year 1 ore production. The total stockpiled ore amounts to 2.6 Mt.

The production plan showing material sent to the concentrator and to stockpiles is included in Table 16-8.

Table 16-7 – Mine production schedule summary

Period	Cut-off %Cu	Mined ore			Mined low grade ore (to stockpile)			Oxide and Mixed (to stockpile)		Waste	Total
		kt	Cut (%)	Au (g/t)	kt	Cut (%)	Au (g/t)	kt	%CuSS+CuCn	kt	kt
Y-2H2										5,000	5,000
Y-1H1	**0.25**	0.1	0.31	0.12	6	0.17	0.15	605	0.20	14,389	15,000
Y-1H2	**0.30**	2,491	0.68	0.11	146	0.21	0.06	2,879	0.23	19,484	25,000
Y01H1	**0.20**	5,865	1.06	0.12	113	0.23	0.10	3,151	0.26	26,296	35,425
Y01H2	**0.20**	7,677	0.98	0.12	457	0.25	0.10	1,429	0.24	25,861	35,425
Y02H1	**0.20**	9,955	0.80	0.10	512	0.18	0.06	577	0.21	24,381	35,425
Y02H2	**0.20**	10,120	0.73	0.10	437	0.18	0.05	902	0.25	23,966	35,425
Y03H1	**0.20**	11,570	0.61	0.09	485	0.18	0.03	572	0.32	22,798	35,425
Y03H2	**0.20**	11,629	0.55	0.10	867	0.18	0.05	1,390	0.29	21,539	35,425
Y04H1	**0.20**	11,263	0.51	0.10	1,222	0.17	0.04	1,615	0.25	21,325	35,425
Y04H2	**0.20**	11,601	0.53	0.09	960	0.17	0.03	2,140	0.25	20,724	35,425
Y05H1	**0.20**	11,590	0.60	0.10	1,140	0.17	0.03	788	0.27	13,733	27,250
Y05H2	**0.20**	11,600	0.47	0.08	1,843	0.18	0.05	151	0.26	13,657	27,250
Y06	**0.20**	22,513	0.46	0.08	3,560	0.18	0.05	109	0.24	28,318	54,500
Y07	**0.20**	21,826	0.36	0.07	6,878	0.17	0.04			25,796	54,500
Y08	**0.20**	21,683	0.34	0.07	7,395	0.17	0.05			25,422	54,500
Y09	**0.20**	22,542	0.39	0.05	7,029	0.17	0.04	12	0.23	24,917	54,500
Y10	**0.20**	21,787	0.37	0.06	5,649	0.18	0.04			22,364	49,800
Y11	**0.15**	21,626	0.32	0.06				62	0.31	28,112	49,800
Y12	**0.15**	21,984	0.32	0.06				218	0.28	27,598	49,800
Y13	**0.15**	21,606	0.31	0.06						28,194	49,800
Y14	**0.15**	21,214	0.26	0.06						28,586	49,800
Y15	**0.15**	21,011	0.24	0.07						28,789	49,800
Y16	**0.15**	21,371	0.30	0.07				41	0.16	27,388	48,800
Y17	**0.15**	11,754	0.30	0.04				982	0.19	16,041	28,777
Y18	**0.15**	5,593	0.30	0.03				13	0.22	1,369	6,976
Y19	**0.15**										
TOTAL		**361,871**	**0.43**	**0.07**	**38,698**	**0.18**	**0.04**	**17,638**	**0.25**	**566,047**	**984,253**

Table 16-8 – Plant Feed Production Schedule Summary

Period	Total plant feed							To stockpile	Re-handling
	Mixed	Supergene	Hypogene	Total					
	kt	kt	kt	kt	Cut (%)	Au (g/t)	Cu rec (%)	kt	kt
Y-2H2									
Y-1H1								6	
Y-1H2								2,637	
Y01H1	364	6,086	4	6,454	1.02	0.12	86.08	113	589
Y01H2	414	8,784	381	9,579	0.92	0.12	87.28	457	1,902
Y02H1	102	8,783	1,069	9,955	0.80	0.10	88.25	512	
Y02H2	261	8,376	1,484	10,120	0.73	0.10	88.23	437	
Y03H1	328	9,196	2,046	11,570	0.61	0.09	87.93	485	
Y03H2	16	8,801	2,812	11,629	0.55	0.10	87.95	867	
Y04H1	107	7,051	4,106	11,263	0.51	0.10	88.37	1,222	
Y04H2	155	8,332	3,114	11,601	0.53	0.09	87.99	960	
Y05H1	38	9,858	1,694	11,590	0.60	0.10	88.03	1,140	
Y05H2	21	8,551	3,028	11,600	0.47	0.08	87.87	1,843	
Y06		12,944	9,569	22,513	0.46	0.08	88.73	3,560	
Y07		7,296	14,531	21,826	0.36	0.07	89.08	6,878	
Y08		6,114	15,569	21,683	0.34	0.07	88.96	7,395	
Y09	12	13,165	9,365	22,542	0.39	0.05	87.11	7,029	
Y10		6,973	14,814	21,787	0.37	0.06	88.66	5,649	
Y11	8	5,630	15,988	21,626	0.32	0.06	89.32		
Y12	25	8,548	13,411	21,984	0.32	0.06	88.72		
Y13	26	5,432	16,147	21,606	0.31	0.06	88.96		
Y14		2,256	18,958	21,214	0.26	0.06	90.05		
Y15		589	20,423	21,011	0.24	0.07	90.35		
Y16		3,546	17,825	21,371	0.30	0.07	89.28		
Y17	130	8,641	13,246	22,017	0.24	0.04	85.95		10,263
Y18	103	6,759	14,921	21,783	0.21	0.04	84.57		16,190
Y19	75	2,849	9,320	12,244	0.18	0.04	83.69		12,244
TOTAL	**2,183**	**174,561**	**223,825**	**400,569**	**0.40**	**0.07**	**88.14**	**41,189**	**41,189**

16.6 Blasting and Explosives

The drilling equipment will consist of diesel units capable of drilling 251 mm (9⅞") diameter holes for ore and waste. Additionally, support units capable of drilling 152 mm (6") diameter holes for pre-splitting are included in the fleet. Three units will be required for the pre-production period. During commercial production four units will be required from Year 1 through Year 4 and three again from Year 5 through to Year 16. The number will then drop to the end of mine life as less material is mined during Year 16 and Year 17.

A general design for the drilling and blasting patterns has been carried out. According to the drill pattern specified, blasting powder factors of 189 g/t for waste and 346 g/t for ore were estimated. Both estimated values are common for fresh rock material.

16.7 Mining Equipment and Fleet

Contractor´s and Owner´s mine equipment requirements were calculated based on the annual mine production schedule, the mine work schedule, and estimates of equipment annual production capacities. The equipment will need to perform the following work:

- Construct roads to the initial mining areas as well as to the crusher, waste storage areas and stockpiles. A qualified mining contractor will perform this task. Construct additional roads as needed to support mining activity.
- The pre-production development required to expose ore for initial production. A qualified mining contractor will perform this task.
- Mine and transport ore to the primary crusher.
- Mine and transport waste from the pit to the waste storage areas.
- Maintain all the mine work areas, in-pit haul roads and external haul roads; and maintain the waste storage areas.
- Re-handle the ore and marginal ore (load, transport and auxiliary equipment) from the stockpiles to feed the primary crusher.
- Mine contractor and Owner may select different equipment types and brands. Equipment fleet requirements are shown for comparative purposes only.

The mine major equipment was selected based on the mine production schedule, 18 months of pre-production and approximately 19 years of commercial mining operations. The pre-production period will include preparing roads, preparing bench openings and pre-production stripping. The total material mined during pre-production is 45 M tonnes. Re-handling of ore will be required in Year 1 for material mined during pre-production to complete the plant feed requirement and at the end of mine life for feeding the plant with the low grade ore stockpiled during the 10 first years of production.

The Owner´s major equipment was selected based on the mine production schedule from Year 4, which represents approximately 15 years of commercial mining operations. The Contractor´s major equipment will be based on the requirements of the pre-production period that includes preparing roads and preparing bench openings as well as the requirements of the production period for three years. The total material mined during pre-production will be 45 M tonnes. Re-handling of ore will be required in Year 1 for material mined during pre-production to complete the plant feed requirement and at the end of mine life for feeding the plant with the low grade ore stockpiled during the 10 first years of production.

An average dry bank density of 2.56 t/m^3 was used for ore, 2.45 t/m^3 for oxide and mixed material sent to stockpile and 2.47 t/m^3 for waste.

The density values are based on the resource block model values for the various materials tabulated from the mine production schedule. The material handling swell was estimated at 30%. NCL assumed moisture content of 2%, which represents the weight percent of water in the wet weight of the material. The density of wet, loose material was used to calculate allowable truck payload limits.

A job efficiency factor of 86% was used to correct for operational losses, which in turn were used to estimate the size of loading units and their productivities. This efficiency corresponds to 51.6 minutes of effective work per operating hour. A job efficiency of 89% was used for the haul trucks.

The peak equipment requirements for the pre-production and mine life are included in Table 16-9. Fleet requirements by period are included in Table 16-10.

Table 16-9 – Peak fleet requirements for pre-production and commercial production

Mobile equipment	Contract mining		Owner mining
	Pre-production	Peak requirement	Peak requirement
FEL 994H EHL (19 m^3)	1	1	1
Hydraulic shovel 6060FS FO	2	4	3
Haul truck Cat793F	9	20	17
Support drill (6") MD6290 FO	1	1	1
Ore-waste drill (9 ⅞") MD6420 FO	2	4	3
Bulldozer 1 D10	1	2	2
Bulldozer 2 D11	3	4	3
Wheel dozer 1 834K	1	2	1
Wheel dozer 2 854K	2	3	2
Motor grader 1 16M	1	1	1
Motor grader 2 24M	2	2	2
Water truck 785D WT	2	3	2
Backhoe 326DL	1	1	1
Fuel truck 85 m^3	1	1	1
Mobile crane AC200	1	1	1
Lowboy truck	1	1	1
FEL 966 - 5 yd^3	1	1	1
Tyre handler 988K TH	1	1	1
Lighting tower RL4060D-4MH (4x1000w)	10	12	10

During pre-production two hydraulic shovels will be required. Four operating shovels will be required during the contract mining period, from Year 1 through Year 4, and three during the owner mining period, from Year 5 through Year 16, after which the number will then drop as less material is mined.

The number of front-end loaders required is less than one for all of the mine life. The front-end loader will also be used as back-up for production loading activities.

The number of truck units required was obtained by dividing the annual of truck transport capacity for each combination and period by the corresponding tonnage according to the defined assignment per loading unit. Truck operating hours were calculated per period, type of material and loading unit dividing the tonnage that has to be transported by the hourly productivity of each combination.

The total haulage distance varies from a minimum of 2.2 km to a maximum of 3.8 km (Table 16-6).

Truck speeds were determined using typical values obtained from supplier information and similar operations. The truck cycle assignments included fixed times for loading, dumping and queuing. Two minutes have been added to every cycle for dumping and queuing.

Operational indices considered for the trucks were:

- Mechanical Availability (MA): Variable profile according to vendor and fleet life, from 87% to 81%
- Use of availability (UA): 84.6%
- Operational losses: 89% (accounting for operator factor, inspection, training).

The number of trucks required during pre-production is 14. Because of the characteristic of the mine and the location of the waste dumps, the hauling profile is relatively constant and with a significant proportion of downhill hauling (52%, see Figure 16-5); therefore, the number of required trucks varies from 20 to 17 units throughout the mine life. From Year 18 the requirement decreases as less material is mined and re-handling of the low grade ore is the major activity during Years 18 and 19.

The primary duties that will be assigned to the auxiliary equipment are as follows:

- Mine development including access roads, drop cuts, temporary service ramps, safety berms
- Waste rock storage area control; this includes maintaining access to the dumping areas and maintaining the travel surfaces
- Ore stockpile area control; this includes maintaining access to the stockpile areas and maintaining the travel surfaces
- Maintenance and clean-up in the mine and waste storage areas
- Maintenance and clean-up of water diversion channels around the dumps and open pit
- Drilling for pre-split blasting.
- Equipment types included in the auxiliary mine fleet are as follows or as equivalent brands:
- Caterpillar D10 track dozer (600 HP)
- Caterpillar D11 track dozer (850 HP)
- Caterpillar 834K wheel dozer (496 HP)

- Caterpillar 854K wheel dozer (814 HP)
- Caterpillar 16M motor grader (297 HP)
- Caterpillar 24M motor grader (533 HP)
- Caterpillar water truck 785D WT (85 m^3)
- Caterpillar MD6290 support drill (6").

In general, six track dozers, five wheel dozers, three motor-graders and three water trucks will be required.

Table 16-10 – Fleet requirements by period

	Contractor mining period									Transition		Owner mining period															
	Y-2	Y-1	Y-1	Y1	Y1	Y2	Y2	Y3	Y3	Y4	Y4	Y5	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19
	H2	H1	H2	H1	H2	H1	H2	H1	H2	H1	H2	H1	H2														
FEL 994H EHL (19 m^3)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Hydraulic shovel 6060FS FO	1	2	3	4	4	4	4	4	4	4	4	3	3	3	3	3	3	3	3	3	3	3	3	3	2	1	1
Haul truck Cat793F	3	9	14	20	20	20	19	20	20	20	20	15	15	16	14	16	17	17	17	17	16	17	17	17	17	7	5
Support drill (6") MD6290 FO	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Ore-waste drill (9 7/8") MD6420 FO	1	2	3	4	4	4	4	4	4	4	4	3	3	3	3	3	3	3	3	3	3	3	3	3	2	1	0
Bulldozer 1 D10	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Bulldozer 2 D11	2	2	3	4	4	4	4	4	4	4	4	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2
Wheel dozer 1 834K	1	1	1	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Wheel dozer 2 854K	1	2	2	3	3	3	3	3	3	3	3	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Motor grader 1 16M	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Motor grader 2 24M	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Water truck 785D WT	1	2	2	2	3	3	3	3	3	3	3	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Backhoe 326DL	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel truck 85 m^3	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mobile crane AC200	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lowboy truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
FEL 966 - 5 yd^3	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tyre handler 988K TH	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lighting tower RL4060D-4MH (4x1000w)	5	8	10	12	12	12	12	12	12	12	12	10	10	10	10	10	10	10	10	10	10	10	10	10	8	6	4

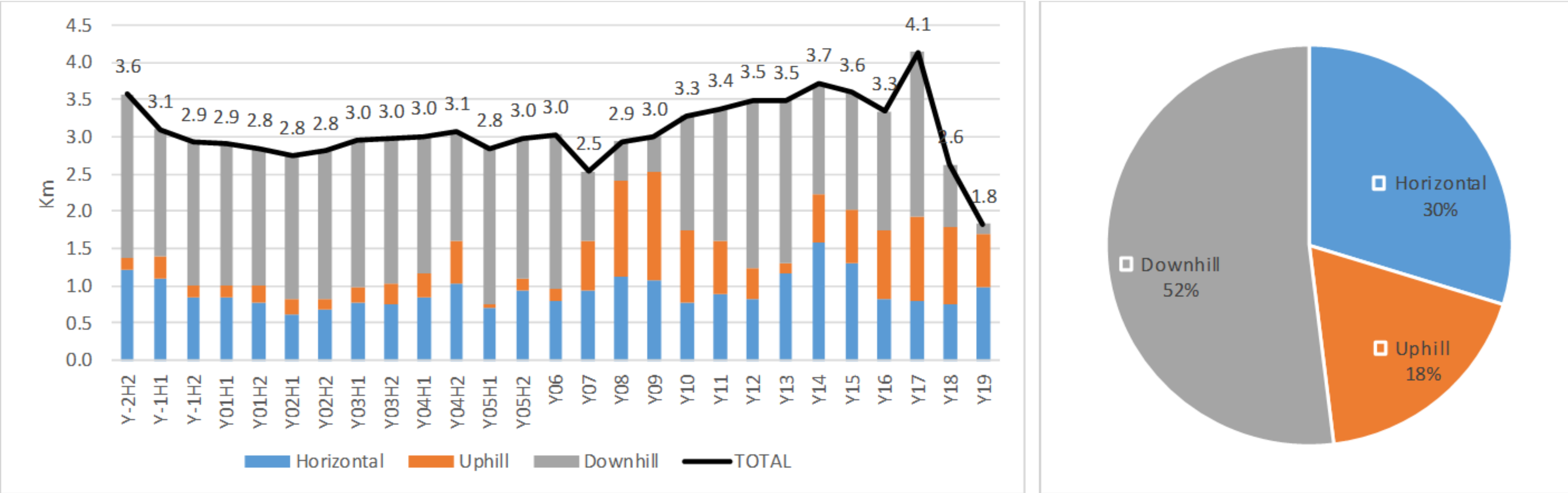


Figure 16-5 — Total Hauling Distances

17 Recovery Methods

17.1 Process Flow Diagram (PFD) and Concentrator Description

The concentrator associated support facilities and utilities will process run-of-mine (ROM) ore delivered to the primary crusher and produce a copper-gold concentrate for sale. The process encompasses crushing of the ROM ore, grinding, flotation including regrind, and concentrate thickening and filtration to produce copper-gold concentrate that is then transported by road for subsequent shipping to customers. The flotation tailings will be thickened prior to discharge to the tailings management facility (TMF).

The process design criteria were based on review of completed test work to date as well as benchmarked data from similar operations. Where gaps remained in the test work, benchmark data was used until confirmatory test work is performed in the next phase.

The key criteria selected for comminution, flotation, intermediate concentrate regrind and final concentrate and tailings thickening are summarised below:

- Treatment of an average 55,000 dry metric tonnes per day (t/d) for 365 days per year.
- Sufficient plant design flexibility for individual treatment of either of the main ore types tested (whole resource is supergene 47% and hypogene 53%), as tested in the metallurgical test work program, at design throughput.
- Capacity to treat a maximum throughput of 2,915 t/h (63.7 kt/d) and a copper production rate of 20.5 t/h at 91.3% (8000 h/y) availability, based on a maximum head grade of 1.01% Cu.

The concentrator includes the following unit processes and facilities:

- Primary crusher and transfer conveyors
- Crushed ore stockpile and reclaim
- SABC grinding
- Flotation comprising rougher flotation, concentrate regrind, three-stage cleaning and cleaner scavenger flotation
- Concentrate thickening and filtration
- Concentrate storage and load-out
- Tailings thickening and water recycle
- Reagents mixing and distribution (including lime slaking, flotation reagents and flocculants)
- Grinding media storage and addition
- Water services (including raw water, fire water, gland water, cooling water and process water)
- Potable water treatment and distribution
- Air services (including high pressure air and low pressure process air).

The overall process plant flowsheet is shown in Figure 17-1.

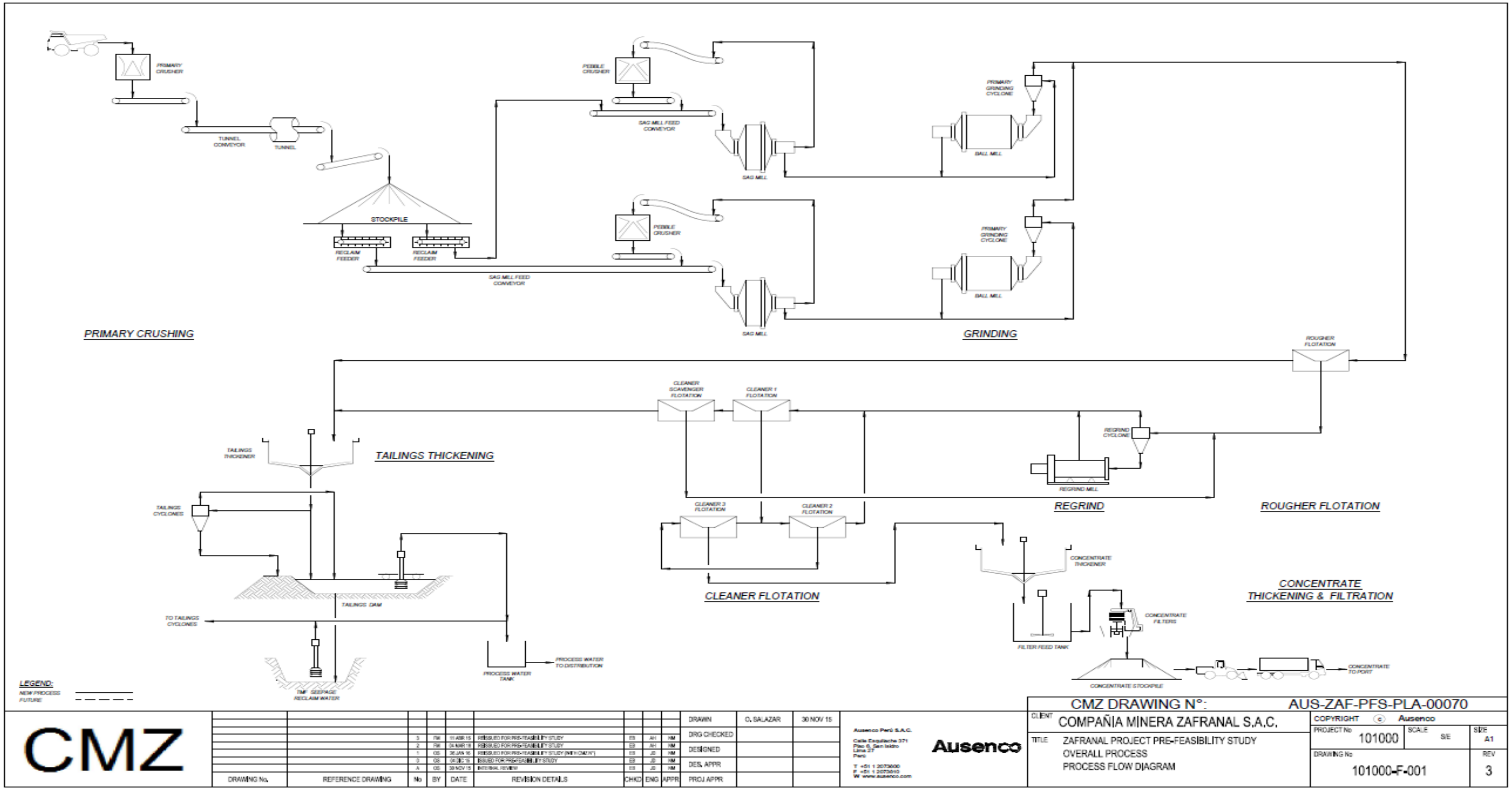


Figure 17-1 – Overall Process Flowsheet

17.1.1 Primary crushing and crushed ore transfer

The run-of-mine (ROM) pad and the primary crusher are located in close proximity to the mine pit exits and in an area where the topography allows the crusher structure to be founded in cut. The ROM pad elevation of 2,520 masl also matches the mine pit exit elevation.

Based on the design throughput, run-of-mine (ROM) ore maximum lump size and ore comminution characteristics, a single gyratory crusher 1,600 mm by 2,400 mm (63" x 95") or equivalent has been selected for primary crushing run of mine ore at a rate of 3,274 t/h based on availability of 70% for 6,100 hours operating per year.

The primary crushed product size distribution is based on a gyratory crusher operating close side setting (CSS) of 120 mm to produce a product top size of 280 mm and a crusher product size P_{80} of 91 mm when treating ROM ore with a feed top size of 1,000 mm and an F_{80} of 450 mm.

Auxiliary crusher equipment includes a rock breaker; the overhead crane described above, and dust suppression system. Water sprays are used to minimise dust at the crusher dump pocket and conveyor transfer points.

The primary crusher discharge conveyor is fitted with a weightometer, belt monitoring systems, and a metal detector to detect tramp metal.

From the sacrificial discharge conveyor, the primary crushed ore reports to the crushed ore stockpile. Due to the steep topography between the primary crusher and the rest of the concentrator, the 3.9 km long conveyor that transfers crushed ore from the crusher discharge conveyor to the crushed ore stockpile feed conveyor passes through a 3.6 km tunnel.

The crushed ore stockpile has 16 hours of live capacity to ensure stable feed to the SAG mill. The crushed ore stockpile will also provide buffer to allow continuous SAG mill feed during periods of crusher maintenance. Additional dead capacity on the stockpile can be dozed into the reclaim feeders during longer shutdowns.

For primary crushed ore, provided adequate water is used for dust suppression, an open stockpile is acceptable. Two apron feeders per grinding line were selected for coarse ore reclaim. Two reclaim feeders for each line allows for continuity of grinding operation in case of problems with blockages or mechanical failure. Each apron feeder has been sized to reclaim 110% of nominal capacity.

The climate at the site is not extreme with regards to heat, cold or rain, so both crushing and grinding structures are to be open, with general maintenance access by mobile crane. Due to the distance from the rest of the concentrator and concentrator workshop, the primary crusher will be serviced by a permanent bridge crane. Davit hoists are provided for routine cyclone cluster maintenance.

17.1.2 Tunnel

The crushed ore transfer conveyor tunnel will be developed by conventional drill and blast techniques, with an arch roof of top radius 2.25 m and finished cross-section 4.5 m wide at the base and 4.2 m high at the top of the arch. The dimensions are sufficient for the conveyor and a roadway, suitable for a small truck or light vehicle for inspection and maintenance, with adequate clearance either side of both and between them. A drainage channel in the floor will be on the conveyor side. Ventilation will be provided by four fans at one end of the tunnel. Personnel passing bays will be provided at intervals, some of which will have been the loading stations used during construction.

The total tunnel length, not including immediate portal structures is 3,515 m; 2,265 m for the first segment/traverse and 1250 m for the tunnel extension. The first segment/traverse was the target of the main geotechnical investigation, including drilling, based on the tunnel alignment and length proposed in the PEAU. Later optimization of the site layout and concentrator layout resulted in the addition of the tunnel extension, which was investigated by surface mapping, but was included in the PFS design too late to be investigated by drilling.

A total of six geotechnical drill holes were completed to characterize the tunnel geology, four with full core logging to compile rock mass rating and collection of samples for physical strength testing. A total of 23 tests on rock specimens were done including unconfined compression, triaxial compression, Brazilian tension, and direct shear. Major rock types occurring in the traverse are: andesite, mylonite, granodiorite, and gouge. The structure, rock joint dip and strike were compiled. Faults and structure were surface mapped and correlated to drill hole intercepts. The tunnel is projected to cross up to ten faults.

Of the tunnel's 3,515 m length, 688 m or 20% of the length occurs in mylonite and/or gouge fill requiring steel sets and shotcrete fill for roof support. The remaining 2,827 m or 80% occurs in rock projected to be supported by bolting. The keystone method, based on rock joint dip and strike, was employed using water pressure-expandable rock bolts and welded wire mesh. Fault crossings are also projected to require steel sets and shotcrete fill.

The tunnel will require more geotechnical drilling, sampling, and testing for the feasibility study. The first 2,265 m of tunnel geotechnical investigation is at PFS level, but the tunnel extension is not, due to the lack of drilling and sampling. However, in the context of the overall Project capital cost estimate it is not a material impact and can be mitigated by an appropriate cost allowance.No ground control threat or fatal flaw was identified using the information available but more drilling is needed to define the tunnel, particularly in the extension.

An incident risk is represented by potential vehicle damage to the transfer conveyor. This can be mitigated by restricting access to the tunnel to maintenance and inspection only.

17.1.3 Concentrator design

The concentrator wet plant design incorporates an open-air layout that minimizes crane requirements by maximizing accessibility for mobile cranes for maintenance. The plant will use a conventional processing flow sheet and industry standard equipment.

Figure 17-2 shows the concentrator plant plan and elevation.

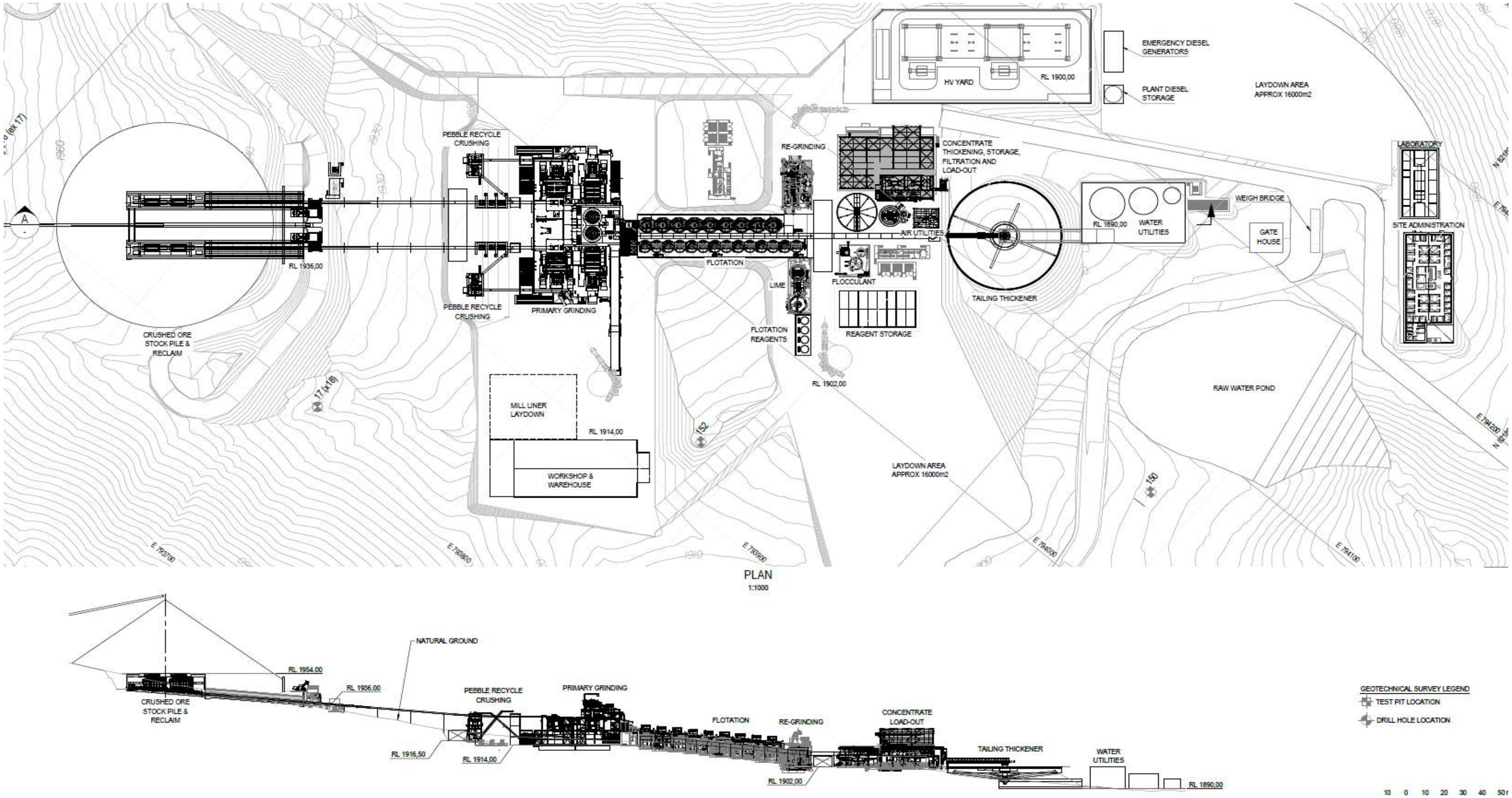


Figure 17-2 – Concentrator wet plant plan and elevation

17.1.3.1 Head grade

The plant is designed to treat sulfide ore at a maximum copper head grade of 1.01%, which allows for a design margin on peak annual average copper head grade of 0.92% in Year 1. The life of mine average copper head grade of 0.40 %

17.1.3.2 Operating hours and throughputs

All processing from crushed ore reclaim through to tailings and concentrate thickening will operate on a continuous basis of 24 hours per day, 7 days per week. Allowing for downtime for maintenance and unscheduled stoppages in the operation, the long-term availability is expected to be at least 91.3%. Therefore, the operating time is 8,000 hours, equivalent to an average operating throughput of 2,509 t/h over the life of mine. The maximum plant throughput of 2,915 t/h occurs in year 3.

The throughput of the concentrate filtration circuit is dictated by the availability of the pressure filter. Typical design availability for pressure filters is 70% which allows additional capacity for surge catchup. This corresponds to an operating time of 6,100 hours, equivalent to concentrate production rate of 36 t/h for an average annual of 218 kt/y. However, significant variations in head grade in Year 1 are expected and design throughput of 71 t/h has been specified which corresponds to a peak concentrate production rate of 430 kt/y.

17.1.3.3 Grinding

Sizing of the mills was based on analysis of the comminution data completed during the Phase 1-5 metallurgical test work. Different mill configurations were assessed in a trade-off study to determine the most cost effective configuration which would meet an average daily throughput of 55 kt/d.

The outcome of the trade-off study was selection of a dual-line SABC circuit with twin-pinion drives on the SAG mills, based on the throughput, ore characteristics, and flexibility to increase throughput, without changing the fundamental design, for similar capital cost to a single-line circuit with a gearless motor drive SAG mill that would be the largest installed (i.e. any increase in design throughput would require change to a dual-line circuit).

The parameters used to establish the power drawn are summarized in Table 17-1.

Table 17-1 – Summary of SAG and ball mill design parameters

Parameter	Units	Value
Throughput	t/h	2,510
Number of grinding lines		2
Mill type		SAG
Number of mills, per grinding line		1
Installed power	kW	16,000
Mill type		Ball mill
Number of mills, per grinding line		1
Grind size, P80	µm	150
Installed power	kW	8,000

The installed motor power for the SAG mill incorporates the allowances for drive train losses to determine the motor power from the pinion power, as well as contingency to allow adjustment in the mill operating conditions to handle ore variability. Installed ball mill power of 8,000 kW incorporates similar allowances.

The competency for the supergene rock type was then determined to be lower than that for the hypogene rock type. An assessment found that whilst the mills selected would provide sufficient power to achieve the design throughput of 55 kt/d and achieve the target cyclone overflow grind size of 80% passing 150 μm, there was an opportunity to increase throughput early in the mine life when the less competent supergene ore was being processed.

The 75th percentile Ecs values for each rock type are presented in Table 13-20.

The grinding circuit consists of two parallel lines of grinding mills, each treating 50% of the concentrator throughput. Each grinding line consists of a single semi-autogenous grinding (SAG) mill with pebble crushing system, followed by a single ball mill operating in closed circuit with a cyclone cluster. The area will be open with all maintenance activities completed using mobile equipment, with the exception of a cyclone cluster jib crane.

The SAG mill feed conveyor transfers ore (along with crushed pebble and grinding media) to the SAG mill feed chute where it is combined with mill feed dilution water and lime slurry to increase slurry pH to approximately 9-10. Each SAG mill is fitted with a twin pinion drive arrangement and is driven by a variable-speed drive comprised of a slip energy recovery (SER) hyper-synchronous drive. Both SAG mills are fitted with discharge grates with pebble ports to retain grinding media and larger particles while allowing smaller particles to discharge from the mill.

Each SAG mill is fitted with a trommel screen to separate slurry and pebbles. Trommel oversize from each SAG mill reports to a surge bin and pebble crusher via transfer conveyors. Each pebble transfer conveyor is fitted with self-cleaning magnet and each pebble crusher feed conveyor has a metal detector. Tramp metal from each magnet is diverted to a bunker, and remaining metal detected on the pebble conveyor is by-passed to the SAG feed conveyor belt. Pebble crusher product is conveyed back to the SAG mill feed conveyor. 'High angle' pebble conveyors have been used to minimize earthworks and have been deemed suitable based on recent industry feedback.

SAG mill trommel undersize gravitates to the primary cyclone feed hopper where it is combined with the discharge from the ball mill.

Each grinding line includes a single variable-speed cyclone feed pump sending slurry to a dedicated cyclone cluster. A rotable spare bare-shaft cyclone feed pump (wet end) is stored on site to allow quick change-out.

Process water for dilution is added to the cyclone feed hoppers before the slurry is pumped to the respective cyclone cluster for classification. Flotation primary collector is also added to the cyclone feed hoppers.

The cyclone underflow streams report to the respective ball mill feed chute via a boil box. Cyclone underflow is ground in the ball mills to further reduce particle size. Both ball mills are driven by wound-rotor motors through reduction gearboxes and ring gear and pinion drives.

The cyclone overflow streams gravitate separately through cross-stream samples and are combined in a single boil box before entering the copper flotation circuit.

A single ball storage bunker is provided for both ball mills. Balls are transferred by magnet and hoist to a ball feeding bin. Separate ball feeders transfer the grinding media to each ball mill.

Two feed chute transporters are used to remove the feed chutes from the four mills for inspection and reline.

Two separate liner handlers are provided, one for servicing the SAG mills and one for servicing the ball mills. This enables both the SAG mill and ball mill (from the same line) to simultaneously undergo liner maintenance and minimize shutdown durations. Likewise, for each liner handler, two liner bolt removal tools are provided for relining.

17.1.3.4 Flotation

The design flow sheet selected consists of a combined rougher-scavenger circuit followed by three stages of cleaning and a cleaner scavenger, all in tank flotation cells except the third cleaner stage which is in trough flotation cells. The cleaners are to be run in open circuit with cleaner scavenger tailings reporting to final tail, as open-circuit cleaners generally provide better operability and control than cleaners closed to rougher feed.

Forced air, mechanically agitated flotation cells have been selected for all duties. There is nothing in the mineralogy or testwork to suggest that there would be any advantage gained by employing in-grinding flotation (flash or unit cell flotation) or column flotation cells.

Table 17-2 shows a summary of the cell requirements for the flotation circuit based on:

- Satisfying the design criteria
- Using cell sizes proven in operation
- Minimising the number of different cell sizes to rationalise spares holdings
- Having appropriate numbers of cells in each stage to minimise short-circuiting.

Table 17-2 – Flotation cell capacity requirements

Flotation stage	Cell size, m^3	N°
Rougher	300	9
Cleaner 1	100	5
Cleaner 2	100	2
Cleaner 3	38	3
Cleaner scavenger	100	7

The regrind circuit regrinds all rougher concentrate and cleaner scavenger concentrate from an F_{80} of 125 microns to a P_{80} of 40 microns. Regrind mill pinion power requirements calculations considers only the rougher concentrate throughput as most of the cleaner scavenger concentrate is below 40 microns.

The regrind size of 40 microns was selected based on Phase 3 and 5 metallurgical test work. Table 17- 3 shows pinion power requirements to regrind the rougher concentrate from 125 microns to 40 microns at the average copper head grade of 0.45%Cu.

Table 17- 3 – Regrind mill requirements

Parameter	Units	Value
Regrind circuit feed (rougher concentrate)	t/h	285
Specific comminution energy (design)	kWh/t	11

Parameter	Units	Value
Feed size, F80	µm	125
Product size, P80	µm	40
Mill motor size	kW	3,700
Installed power available	kW	3,330
Total pinion power required	kW	3,135
Specific comminution energy (design)	kWh/t	11
Design slurry density	%v/v	20

For this power requirement, a single IsaMill™ with a 3,700 kW motor has been selected for this application. The IsaMill™ was selected as it can be run in open circuit with a preclassification cyclone.

As the climate at the site is not extreme with regards to heat and cold or rain, and there are relatively few cells; (relatively few mechanisms to service) the flotation structures can be open, with maintenance access by mobile cranes. There is no evidence from the testwork of the concentrate froths being fragile, so additional protection of the active surfaces of the flotation cells from wind and rain was not considered necessary.

17.1.3.5 Concentrate thickening, filtration, storage and load-out

The concentrate thickener consists of a 20 m diameter high rate thickener to thicken copper concentrate to approximately 60-65% w/w solids. The concentrate thickener overflow reports to the process water tank. A solids loading of 0.20 t/h/m^2 was used for the design of the concentrate thickening facility. This design value was based on benchmarking of similar operations treating sulfide copper minerals.

Thickened concentrate slurry is pumped to an agitated storage tank to provide 24 h surge capacity prior to filtration allowing filter maintenance to be conducted without impacting on overall concentrator utilisation.

Two horizontal plate pressure filters have been selected, which provides a fit-for-purpose design that allows operational flexibility to treat the large differences in concentrate production due to head grade fluctuations over the life of mine. A filter cake of approximately 9.5% w/w moisture is discharged by gravity to a stockpile and stacked in the concentrate shed for on-site storage by front-end loader (FEL).

Various options for handling and transporting the concentrate filter cake were considered with the option of concentrate trucked directly to the port in bulk selected. An axle weighbridge will be used to measure the mass of each load of concentrate. Concentrate trucks will pass through a wheel wash to remove any adhered materials as an environmental control.

As the climate at the site is not extreme, the concentrate thickener and storage tanks are to be open, with maintenance access by mobile cranes. Pressure filtration equipment requires protection from the elements due to the large number of moving parts with tight alignments and fine tolerances, and the sensitivity of some filter cloths to sunlight, so this area will be enclosed. The concentrate filter cake storage will also be enclosed to keep the concentrate dry and to contain dust.

17.1.3.6 Tailings thickening, disposal, and cycloning

The tailings thickener consists of a 62 m diameter high rate thickener to thicken rougher and cleaner scavenger flotation tailings to approximately 60% w/w solids and recover process water prior to discharge to the tailings management facility. Tailings slurry will be gravity fed to the tailings management facility where it discharges directly into the impoundment or reports to the tailings cycloning station.

A portion of the underflow from the tailings thickener gravitates to the dilution box for the tailings cyclone station where it is diluted to 35% solids with water reclaimed from the TMF. Excess tailings not required for construction of the TMF bypass the cyclone dilution box and are discharged to the TMF impoundment via discharge spigots.

The diluted tailings are fed to the cyclone station, which separates the tailings into a 70% underflow with <15% fines less than 75 um. To minimise the fines content in the underflow, the cyclones use a cyclowash system which utilizes clean water injection into the lower cone of the cyclone to displace fines from the underflow. The design for the cyclone station is based on a 25% split of cyclone feed to underflow. This split is based on benchmark data and information from the Phase 5 test work.

The underflow is used for construction of the TMF embankment, and the overflow from the cyclones is discharged via a spigot system to the TMF impoundment. Water is recovered from the TMF by pontoon-mounted pumps and a return water pumping station. Water from the pumping station is pumped to the plant process water tank for re-use in the plant.

Where required, the reclaim water can also flow under gravity from the tailings return water tank to the tailings cyclone water tank where it is used for dilution at the tailings cycloning station and as cyclowash injection water. The main source of water for the tailings cyclone water tank is the seepage water from the TMF impoundment alluvium. The seepage is collected by a series of well pumps located at the northern end of the main tailings embankment and pumped to the tailings cyclone water tank.

17.1.3.7 Concentrate storage and load-out

Concentrate filter cake is discharged by gravity to a covered stockpile. A front-end loader (FEL) is used to optimize concentrate storage within the covered building. The covered building provides storage capacity for up to 7 days at average production rates.

Concentrate is loaded by FEL into road trucks for transport from the mine site to the storage terminal at the port facility.

An automated truck wheel wash washes concentrate from the road trucks as they leave the concentrate storage building. Wash water is recycled and solids are recovered in a sump and pumped to the concentrate thickener.

Road trucks are weighed on a weighbridge located at the main security gate prior to leaving the site.

17.1.4 Reagents

To ensure consistency of reagent strength, separate mixing and storage tanks will be provided for all reagents that require mixing.

Reagent selection is per the metallurgical test work conducted, including lime as the pH modifier, copper and gold flotation collectors and frother. Dosing rates have been estimated for each of the reagents based on the metallurgical test work consumption which are provided in Table 17-4.

Table 17-4 – Reagent consumption rates

Reagent	Unit	Value
Quicklime	kg/t feed	1.1
Primary collector	g/t feed	15
Secondary Collector	g/t feed	15
Frother - MIBC	g/t feed	30
Flocculant Concentrate thickening	g/t concentrate	15
Flocculant – Tailings thickening	g/t tailing	30

Table 17-5 provides grinding media consumption rates selected for design.

Table 17-5 – Grinding media consumption rates

Grinding media	Type	Unit	Value
SAG mill	100-150 mm steel balls	kg/t plant feed	0.73
Ball mill	50 – 75 steel balls	kg/t plant feed	0.43
Regrind mill	Keramax MT-X Ceramic	kg/t regrind feed	0.046
Lime slaker mill	12 mm steel balls	kg/t slaker feed	0.26

17.1.4.1 Lime

Lime is used to increase slurry pH and subsequently depress pyrite in copper flotation.

The quicklime slaking system is a proprietary slaking system comprising storage silo, feeders and vertical slaking mill.

Quicklime is delivered to site in 30 tonne bulk road tankers and unloaded pneumatically to a 540 tonne storage silo at the lime slaking plant. Quicklime is transferred from the silo at a controlled rate via a screw feeder and fed to a vertical stirred mill. The quicklime is slaked in the mill with process water to produce a milk of lime slurry containing 20% w/w solids.

The lime slurry overflows from the mill to a cyclone feed hopper where the lime slurry is pumped to cyclones for further classification. The cyclone overflow gravitates to an agitated storage tank.

Lime circulating pumps and a pressurised ring main deliver lime slurry to dosing points in the grinding and flotation circuits. Pinch valves are used to control lime addition at each dosing point.

17.1.4.2 Primary collector (xanthate allyl ester)

Primary collector for the flotation circuit is delivered as a solution in 20 t tanker trucks. The xanthate allyl ester solution is transferred to a storage tank and pumped to addition points via dedicated dosing pumps.

17.1.4.3 Secondary collector (dialkyl dithiophosphinate)

Secondary collector for the flotation circuit is delivered as a solution in 20 t tanker trucks. The dialkyl dithiophosphinate solution is transferred to a storage tank and pumped to addition points via dedicated dosing pumps.

17.1.4.4 Frother (methyl isobutyl carbinol)

Frother to provide a stable froth in the flotation circuit is delivered as a liquid in 20 t tanker trucks. Methyl isobutyl carbinol (MIBC) is transferred to a storage tank, and is pumped to addition points via dedicated dosing pumps. As MIBC is a combustible liquid, flame-proof motors and other appropriate provisions are made in this area.

17.1.4.5 Flocculant (Magnafloc 338)

Flocculant is used as a settling aid in the concentrate and flotation tailings thickeners. The flocculant mixing system consists of a storage bin, screw feeder, air blower, auto jet-wet mixer, mixing tank, storage tank, and dosing pumps. Flocculant is delivered as a dry powder in 700 kg bags. An electric hoist is used to load bags at the flocculant system.

Dry powder is transferred to a heated hopper and blown into the jet-wet mixer wetting head to produce a 0.25% w/w flocculant solution. Flocculant is further mixed in an agitated tank and transferred to a storage tank.

Dedicated dosing pumps deliver flocculant from the storage tank to the respective thickeners. A standby dosing pump is provided for each flocculant dosing point.

17.1.4.6 Spare reagent facility

An additional facility is provided to store an alternative collector or depressant for dosing to the flotation circuit. The spare facility consists of a storage tank to connect with other system.

17.1.4.7 Miscellaneous reagents

Additional reagents are required in the concentrator; however, these are expected to be used in relatively small quantities. These include chemicals for the potable water treatment plant and cooling water systems. Antiscalant and biocide are dosed to the cooling water system.

17.1.5 Tailings water reclaim and recovery

See Section 18.2.14.

17.2 Process Plant Support Facilities and Utilities

17.2.1 Raw and fire water distribution

17.2.1.1 Raw water

Raw water is assumed to be sourced from the Majes 1 well field. Raw water is pumped from Majes 1 well field, via a pumping station, to the raw water storage pond which provides capacity of 24 hours at average consumption. Raw water storage pond pumps transfer raw water from the storage pond to the concentrator raw water tank and process water tank.

The concentrator raw water tank serves as a combined raw water and fire water tank with the lower section dedicated for fire water service and the remainder available for general use.

Raw water is used to supply the following services:

- Potable water treatment plant
- Gland water system
- Crusher spray water
- Mine raw water and fire water tank
- Concentrator equipment as required
- Cooling water system.

Raw water to the process water tank is used as make-up water for the process water system.

17.2.1.2 Fire water

The raw water tank contains a dedicated firewater reserve with a minimum capacity of 320 m^3. The fire water pumping system comprising three fire water pumps (electric, jockey and diesel) provides fire water to fire hydrants, hose reels, and sprinkler systems throughout the concentrator wet plant, tunnel and primary crusher via a dedicated fire water ring main and pipe system.

17.2.1.3 Gland water

Water for the gland water system is sourced from raw water tank. Gland water is distributed to the plant by two gland water pumps.

17.2.1.4 Cooling water

Cooling water is used in the SAG mills and ball mills to cool oil lubrication systems.

Cooling water is supplied from a closed-loop evaporative cooling system. Water from the cooling water tank is pumped through the evaporative coolers to the heat exchangers in the grinding circuits. Warm water is returned to the cooling water tank and recirculated.

Inhibitor and biocide solutions are dosed to the evaporative cooling towers by dedicated dosing pumps.

17.2.1.5 Process water

Process water is comprised of tailings thickener overflow, concentrate thickener overflow; TMF supernatant pond and seepage reclaim water, and raw water as process water make-up.

Process water pumps distribute process water to the grinding mill; flotation and regrind circuits; concentrate thickener, concentrate filter and lime system.

17.2.2 Potable water treatment and distribution

Raw water (transferred from the plant raw water tank) is treated in a vendor-packaged water treatment facility to produce potable water. The water treatment will be by reverse osmosis, with chlorination and ultraviolet light sterilisation.

A potable water tank provides storage capacity of 48 hours at average consumption. Potable water is reticulated for general use in the concentrator and supplies the safety shower system. The potable water pumps also delivers potable water to the camp, general administration facilities, mine workshop, and mine facilities.

Accumulators are used to supply potable water to the safety showers and eye-wash stations in the event of power failure.

17.2.3 Power distribution

Power will be distributed to the concentrator area substations through 22.9 kV feeders to step-down transformers, medium voltage (MV) and low voltage (LV) switchboards each supplying one or more areas. The primary crushing substation (2110-SS-001) will be fed from the tunnel conveyor and crushed ore stockpile substation (2120-SS-010) via a cable through the tunnel. All other remote area substations will be fed via single circuit overhead-line feeders.

17.2.4 Air services

Low pressure air for the flotation circuit is supplied by two blowers (operating as one duty and one standby). The rougher cells and cleaner cells operate at different air pressures. Air pressure to the cleaner cells is reduced to the required pressure via an in-line pressure control valve.

Two separate air compressors (one duty and one standby) provide high pressure air for instruments and general service points. Compressed air is dried and filtered to instrument air quality prior to storage in plant air receivers and subsequent distribution.

The primary crusher is serviced by its own dedicated air compressor. The system is equipped with a receiver, an air dryer and filter system.

17.2.5 Buildings

The PFS focused on optimizing the size of buildings with the aim of reducing capital cost. The concentrator support buildings, workshop-warehouse, laboratory and reagents store, are located within the concentrator area

17.2.6 Process control

A process control strategy document was developed as part of the PFS. The process control strategy provides an overview of the process control system (PCS) requirements and to form the basis of the process control philosophy, which will be developed during the Feasibility Study phase.

The general strategy adopted for the Zafranal Project is as follows:

- Integrated control via the process control system (PCS) for areas where vendor programmable logic controllers (PLC) are not available and equipment requires remote start/shutdown, sequencing, and process interlocking.
- Monitoring of all required operating conditions on the PCS and recording of selected information for data logging and/or trending.
- Control loops using the PCS except where vendor PLCs directly controls vendor packages.

There are two purpose built control rooms:

1. The concentrator main control room (MCR) is located near the grinding circuit to maximize access to critical plant areas.
2. The crusher control room (CCR) is located close to the primary crusher.

17.2.7 Mobile equipment

The mobile fleet requirements were developed based on similar projects developed by Ausenco as a basis, then refined to be suitable for the Zafranal Project.

17.2.8 Sample preparation, analytical laboratory, metallurgical laboratory

The metallurgical accounting is based on a combination of weightometer readings and process sampling. This process information is used to monitor shift and daily plant performance.

Mass flow measurement of dry material on conveyors is achieved by belt weightometers. The belt weightometers have a local display of instantaneous belt loading and integrated wet tonnage flowrate. The moisture content of the ore is determined in the site laboratory from samples provided from belt cuts and manually entered into the PCS to allow the dry tonnage flowrate and totalised dry tonnes to be calculated on a shift, daily and continuous tonnage basis.

In conjunction with the relevant ore feed weightometer readings there are three essential stream samples required for metallurgical accounting. These are:

- Rougher feed (cyclone overflow)
- Final concentrate (third cleaner concentrate)
- Final tailings (combined rougher and cleaner scavenger tailings).

Multi-staged samplers are used for the high volume rougher feed and final tailings streams to improve the quality of these samples. Metallurgical samples are prepared and analyzed in the site metallurgical laboratory.

In the flotation circuit, process streams are sampled via an on-stream analyzer (OSA) to provide information for control and to define the real-time mass balance for specific evaluation of rougher and cleaner flotation performance. These are:

- Rougher feed
- Rougher concentrate
- Rougher tailings
- Cleaner concentrate
- Cleaner scavenger tailings.

Pressure pipe samplers are used to sample concentrate streams and cleaner scavenger tailings. The pressure pipe samples are delivered to the OSA system using peristaltic pumps. The rougher tailings stream is sampled directly from the tailings box of the last rougher flotation cell and pumped to the OSA system using a peristaltic pump. The OSA is calibrated on a routine basis; calibration samples are prepared and analyzed in the metallurgical laboratory.

Metallurgical samples from metallurgical testwork conducted in the metallurgical laboratory and monthly composite samples are assayed in the site metallurgical laboratory.

17.3 Mass, Water and Process Materials

17.3.1 Concentrator mass and water balance

The concentrator mass and water balance was developed to enable sizing of major equipment. The key inputs were documented in a process design criteria.

The mass and water balance for the concentrator are based on the process flow diagrams (PFDs).

17.3.2 Site water balance

The site water balance was prepared for the original base case of 55 kt/d concentrator throughput and a period of operation of 18.3 years with the TMF embankments raised using cycloned tailings. The site water balance was developed using Goldsim® modeling to assess the impact of uncertainty in the parameters and evaluate 'what-if' scenarios.

Due to parallel development of the site water balance and water supply engineering, the design capacity of the water supply exceeded the final water demand modelled for the original base case for the 99th percentile of the life-of-mine average by a sufficient margin to accommodate the increase in water demand pro-rated from the increased throughputs of the variable throughput case. On that basis it was considered unnecessary to update the site water balance for the variable throughput case for the preliminary feasibility study.

Later comparisons for the peak demand year (Year 3) showed that while the pro-rated average flow is accommodated within the water supply design capacity, the 75th and higher percentiles are not. However, the exceedance, even for the 99th percentile, is within the precision of PFS capital cost estimates, and the main component of the increase in operating cost, power consumption, is incorporated in the operating cost.

For the next phases of the Project, revision of the site water balance model for the variable throughput case should be a high priority.

Water will be used and in some cases will circulate through the following facilities:

- Tailings management facility (TMF)
- Concentrator
- Open pit and waste dumps
- Camps and offices
- Roads, for dust control.

The water balance model was developed on a monthly basis and unfolds in various modules, one related to the balance of water in the TMF, one for the concentrator, and one that develops the comprehensive balance of the site, incorporating the TMF, concentrator, open pit and the hydrological model for creeks upstream. Concentrator raw water demand (make-up water) is the main result of the water balance.

The conclusions from the site water balance modelling are as follows:

- The site wide water balance results indicated that of the total averaged water requirement of 362 L/s to the mine, 84 % is expected be supplied from off-site (abstracted from the Majes I groundwater wellfield), 14% is expected to be supplied from on-site water capture both groundwater and runoff, and 2% come as humidity in ore.
- The averaged water demand to plant process of 417 L/s is supplied 68% by make-up water, 29% by reclaim water and 2% by moisture in ore.
- From the water balance of the concentrator, make-up water requirement is 0.45 m^3/tonne of ore.

- Based on sensitivity analysis water balance calculation for 99% non-exceedence probability, a peak net make up water demand of 397 L/s occurs in the third year of operation which is maintained for next 3 years, dropping steadily over the rest of the life of mine to 337 L/s in the last year of the project.
- The demand for raw water is slightly higher in earlier years. A slight peak in the third year is due to more water loss in pores tailings (less settled density) and alluvial field capacity.
- For design purposes, and due to the parameters uncertainty in this stage of the study, it is suggested to use the 75th percentile for the nominal supply of raw water, and the 90th percentile as the maximum. At the 75th percentile, the monthly peak demand of raw water for the mine is 370 L/s, and at the 99th percentile it is 397 L/s.
- Of the water in the water budget only 4% is derived from direct precipitation and runoff, so precipitation and assumed runoff coefficients are relatively minor components of the overall water balance
- The main water sinks are pore retention and evaporation from wet beach (active beach) areas.
- In summary, uncertainty analysis on permeability show that make-up water is slightly modified (less than ±15 L/s; but it affects pumping energy consumption due to changes in the predominant source of reclaim water (pond, alluvial drainage, or downstream seepage capture).

17.4 Recommendations

Recommendations for additional works include the following:

- Based on the revised comminution parameters for the hypogene and supergene ore types, conduct a trade-off study prior to commencement of the Feasibility Study to assess the optimum grinding circuit configuration and layout.
- Assess the potential to defer the pebble crushing circuit until Year 6 during the next phase of the study, based on the revised mine plan and the revised comminution parameters for the hypogene and supergene ore types.
- Review sizing of lime slaking plant based on updated lime consumptions from Phase 5 metallurgical test work.
- Consider increasing the size of the cleaner and cleaner scavenger cells from 100 to 150 m^3 to reduce the number of cells in the banks. Compare the costs and benefits with minimizing the number of cell sizes to minimize spares holdings.
- Tunnel:
    - Perform an overcoring test to measure horizontal stress in two of the first 2,265 m of tunnel and 2 in the bulk mylonite zone in the extension. High horizontal stress cannot be ruled out until this occurs.
    - Drill a horizontal geotechnical core hole through the mountain at a bearing and dip in the center of the tunnel and tunnel extension. This will give specific information for feasibility level and detailed engineering-level design. The hole will also give specific information on fault intercepts.
    - Drill 4 geotechnical holes to delineate the mylonite and granodiorite zones in the tunnel extension for characteristic and strength variation vertically and horizontally.
    - Perform physical strength testing on the major rock types of the tunnel extension.

18 Project Infrastructure

18.1 Waste Dumps and Mineral Stockpiles

18.1.1 Design

Three waste dump areas will be located to the north and south of the pits. Three mineral stockpiles will be located to the west and southwest of the main Zafranal pit. The final configuration is shown in the Site Layout (Figure 1-2).

The pre-stripping activities will generate approximately 33.1 Mt of waste rock that will be transported by trucks to the waste dumps.

During the pre-production period, the run-of-mine (ROM) stockpile area will be constructed to the west of the initial pit for stockpiling of ore for later re-handling to the primary crusher. The total ore to be stockpiled during this period amounts to 2.5 Mt.

After the pre-production period, a low grade ore stockpile will be located to the west of the pit, in the same location as the initial ROM stockpile area. An oxide stockpile will be placed to the southwest of the main pit, on a platform built on the initial lift of the central waste dump. These stockpiles are designed with 25 m lifts and minimum 17 m set-backs to facilitate later re-handling.

The construction sequence of the mine waste dumps is from bottom to top. The waste dumps and ore and mineral stockpiles were divided into modules, with the horizontal extension of the full areas and the capacity of each section calculated every lift. This wrap-around method follows existing topographic contours to avoid dump slopes that exceed 100 m in height. The general strategy applied was to reduce long horizontal and uphill hauling distances within the waste storage areas when mining occurs at greater depths in the pit. The destination assignment to the different mine waste dumps is based on minimizing cycle time.

Table 18-1 shows the waste dump and mineral stockpile design and used capacity by destination.

Table 18-1 – Dumping sequence by destination by year of operation

Destination	Crusher	Oxide	LG Plat	Stock PP	Stock LG	Central dump	Northeast dump	North dump
Design Capacity (kt)		17,638	17,462	2,491	40,150	497,943	12,837	230,827
Used Capacity (kt)	400,569	17,638	17,462	2,491	38,698	384,409	12,837	151,339

Notes:
LG Plat: low grade stockpile platform
Stock PP: high grade stockpile during pre-striping
Stock LG: low grade stockpile

18.1.2 Geochemical considerations and surface water management

A geochemical characterization program, described in Section 20.1.4, is on-going to assess the potential for acid rock drainage and metal leaching associated with waste rock from the open pit. However, median sulfur grades are typically between 1% and 3%, and acid-base accounting and kinetic testing on the waste rock materials indicate that the majority of waste rock will be potentially acid generating (PAG). If contact water remains buffered or is only slightly alkaline, metals and metalloids such as Mo, As, Sb and Se may be mobile. If contact water becomes acidic, increased concentrations of parameters such as Al, Cu, Cd, Fe, Ni, Zn and others would also be expected. Limited carbonate in the Zafranal rocks suggests that no substantial acid neutralizing capacity or buffering capacity from carbonate mineral dissolution from the waste rock is to be expected.

For acid rock drainage to be generated from sulfide minerals requires exposure of such minerals to air and water, so the fundamental principles for managing PAG materials to avoid acid rock drainage are to avoid or minimize contact with air or water or both. The limited availability of carbonate in waste rock at Zafranal indicates that avoiding or minimizing contact of PAG materials with air and water by encapsulation is not practical except for a very limited quantity of the highest potential acid-generating material. Management of PAG materials has; therefore, focused on minimizing contact with water by means other than encapsulation, and on capture of any contact water.

As described in Section 5.4, the mine and associated waste dumps are located in an arid area with low rainfall (average 152 mm/y) and high evaporation (average 3,085 mm/y). Intense rainfall events of short duration occur resulting in a few days of runoff with high peak flows; however, there may be years without any runoff at all.

The surface water management system is designed to minimize contact water and to prevent contact water from entering the environment. This is achieved by routing clean, non-contact surface water runoff around disturbed areas and minimizing contact water and sediment discharge from the site. Diversion channels are incorporated in the design for this purpose around the north, northeast, and central waste dump areas, the mine workshop, stockpile areas and the open pit area. These drainage structures and their characteristics are described in Section 18.3.

During the feasibility study further characterization of all waste materials at Zafranal will be performed and any opportunity for preferential dumping of potentially acid generating materials in the north waste dump and construction of the central dump in a manner by which PAG materials can be isolated from water and air during mine operation and at closure will be evaluated to establish a dumping plan that minimizes the potential for acid rock drainage. As noted above, at this stage the potential for management by encapsulation appears low, so no provisions have been made for it.

On this basis, diverting non-contact water flows away from disturbed areas will be of primary importance. Any contact water entering disturbed areas during operations will be directed to the open pit where it will either be used for dust suppression or naturally evaporated.

Dumps will be maintained with compacted surfaces to minimize any rain water or dust suppression water infiltrating into the dump. In addition, dump surfaces will be graded to shed rain water to shallow, lined evaporation ponds; water from these ponds will be used for dust control on the waste dumps or evaporated by natural methods. Collection ponds will be maintained downstream of the central waste dump and water quality will be monitored through laboratory testing. Water that is unsuitable for release to the environment will be used for on-site purposes during operations or evaporated. The volumes of contact water from this source are not expected to be high and are expected to be naturally evaporated so no provision for treatment has been allowed. Water that meets the Peruvian maximum permissible limits (MPLs) for discharge will be discharged to the dry quebrada downstream of the central dump. It is thought that this may serve as a recharge to local groundwater.

On closure, all waste dumps and immediate surrounding topography will be contoured such that the separation of non-contact water from the dump structure is maintained. Any through-dump drainage or runoff from the north and northeast waste dumps will be directed to the pit lake. A detailed hydrogeological study of the open pit area is planned for the feasibility study to determine the characteristics of the pit lake and what, if any, remedial work may be needed at closure. At this stage, no remedial work is expected to be required in regard to the pit lake.

After closure, the contact water collection pond below the central waste dump will be monitored for compliance with the Peruvian MPLs and water released if appropriate. The volumes that can be expected from the central waste dump where water quality does not meet the MPLs, are not expected to be high and are expected to be naturally evaporated so no provision for treatment has been allowed. Studies will be undertaken during the FS to further evaluate the likely volumes.

18.1.3 Geotechnical considerations

18.1.3.1 Slope stability

In accordance with Piteau's geotechnical recommendations, which NCL has reviewed and accepted, the facilities were designed in lifts of a maximum 100 m height. Each lift will be constructed at an approximate angle of repose of 37°. A set-back between each lift will maintain the overall slope at 2:1 to facilitate reclamation and long term stability. A constant 2.0 t/m^3 loose density (after natural compaction) was assumed in the design.

18.1.3.2 Foundation conditions

The waste dumps are located around the limits of the open pit. The general landscape within the footprint of the waste dumps is steep terrain, characterized by series of hills with rounded peaks and deep canyon valleys carved into intrusive, volcanic and metamorphic rocks.

Ausenco's site geotechnical investigation identified that the lithology of the area is characterized by the presence of relatively thin residual and colluvial soils overlying intrusive, volcanic and metamorphic host rocks which are moderately fractured to very fractured.

The residual soils identified in the area of the waste dumps are predominantly granular with a thickness of a few to several metres, which is dependent on the local weathering profile, with a higher fines content in areas of volcanic and metamorphic rocks. In general, these deposits are classified as medium dense to dense, sub-rounded to angular and of variable size. On the flanks and hilltops, colluvial soils consisting of brown, silty sands, low plasticity with gravels, cobbles and boulders. At the bottom of the valleys there are streambeds consisting of shallow alluvial deposits that are classified as loose gravels with a sand and silt matrix along with the presence of cobbles and boulders from past historical high flow events.

The geo-hazards identified for the waste dumps would be due to runoff erosion, debris flows, ancient landslides, and local faults. No ancient landslides were discovered in the toe regions of the waste dumps that would affect their global stability.

The structural features are formed by principal faults with a northwest to southeast to north-northwest to south-southeast trend, and minor faults with an east to-west trend. The faults near the waste dumps, especially the north waste dump, appear to be inactive based on lack of surface features from the Holocene period to today to affect the location of the waste dumps for the PFS. During the feasibility study these faults should be analyzed, i.e. trenching, to determine if they are active or inactive.

Based on the geotechnical investigation it is recommended that superficial soil along slopes (less than 0.5 m) and loss colluvial-alluvial deposits (less than 1.0 m) be removed prior to placing waste rock. In addition, it is recommended that the design of the waste dumps have control features for the eventual occurrence of debris flow above these facilities.

18.2 Tailings Storage

18.2.1 Introduction

The anticipated tailings stream from the concentrator will be at a typical solids throughput of approximately 55 kt/d (after ramp-up) ranging up to a maximum of 63 kt/d over 19 years. The tailings will be discharged at a thickened slurry solids content of about 60 percent by weight. The tailings stream to the TMF will be directly discharged to the impoundment or directed to a cyclone station. The sand fraction (cyclone underflow) will be recovered for embankment raising. The cyclone overflow (fine tailings) will report to the impoundment for storage and to provide a low-permeability seal on the alluvium base and the natural walls of the impoundment.

The tailings management facility (TMF) is located 1 km southeast of the concentrator (refer to Figure 1-2). Due to the favourable topography in the TMF area providing a natural basin for the impoundment, only one embankment is initially required in the southwest of the impoundment with a small embankment required later in the northwest. It has been sized to provide sufficient capacity to store approximately 396 Mt of tailings based on the mine and concentrator production schedules (including sand cycloned from the whole tailings used to raise the two embankments).

The ultimate facility occupies an area of approximately 5.4 Mm^2, including the tailings embankments. The TMF has additional capacity to expand to an estimated maximum storage capacity of approximately 770 Mt, with appropriate increases in embankment heights.

18.2.2 Site characteristics

18.2.2.1 Topography and surface hydrology

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

18.2.2.2 Rainfall and Evaporation

The information and analyses provided in Section 5 were used in the design of the TMF.

18.2.2.3 Geology

The TMF impoundment and embankments foundations consist of unconsolidated and medium-consolidated alluvial deposits and rock outcrop. The alluvial thickness ranges from 0 to 60 m within the impoundment and 40 to 50 m within the canyon leading to Quebrada Huacan (the location of the southwest tailings embankment). Approximately 90% of the alluvial layer is underlain by low-permeability siltstones of the Lower Moquegua Formation and the rest is underlain by volcanoclastic-carbonate sequence and quartz diorite. Minimal evidence of karstification on surface and below surface was observed during the mapping and drilling program within the TMF. While potential karstification adjacent to the TMF cannot be ruled out conclusively, the existing data set does not suggest a significant presence.

18.2.2.4 Site seismic assessment

A preliminary review of the regional seismicity has been carried out to enable selection of an appropriate design earthquake event for the TMF by Knight Piésold (2011), which was utilized in the PFS along with data available from other projects within the area. An in-depth seismic hazard study needs to be performed as part of the feasibility study. A review of historical earthquake records and regional tectonics indicates that the Zafranal Project is situated in a region of high seismic activity. Information and data provided by published probabilistic seismic hazard studies and available information from other projects in the region, such as Cerro Verde, have been used to estimate appropriate seismic design parameters for the project site (Table 18-2).

Table 18-2 – Summary of probabilistic seismic hazard analysis

Return Period (Years)	Median PGA (g)
100	0.19
500	0.44
1,000	0.54
MCE[1]	0.70

18.2.3 Tailings characteristics

A laboratory testing program was conducted at the beginning of 2016 and completed in the middle of March 2016 to investigate the geotechnical characteristics of the tailings materials. CMZ provided a tailings sample for testing to Plenge Laboratory in Peru. The laboratory generated samples of tailings cyclone overflow (fine tailings) and cyclone underflow (sand) materials. Ausenco performed a testing program that included index testing to enable geotechnical classification of each of the materials. The whole tailings material can generally be described as non-plastic sand and silt with trace clay, which is suitable for use in the construction of the TMF embankments.

Detailed chemical laboratory testing and analysis of the tailings (solids and process solutions) has been conducted by pHase-Geochemistry to determine their geochemical characteristics under saturated and unsaturated conditions. The testing has indicated that the tailings are likely not to go acid during operation because lime is added to the process so the tailings discharge streams will be alkaline but are more likely to be potentially acid generating after closure of the facility. Therefore, this needs to be taken into account as part of the closure strategy. In addition, the PAG tailings stream is deemed to be suitable for production of cyclone sand fill material for embankment construction, as any surface water runoff from the facility and seepage will be collected and pumped back to the TMF for re-use in project processes.

18.2.4 TMF site geotechnical conditions

18.2.4.1 Site investigation

Ausenco conducted a geotechnical site investigation program in 2015 in the area of the TMF. The programs included 9 drill holes and 17 test pits to investigate the geotechnical characteristics of the impoundment and foundations of the two embankments. Thirteen kilometres of electrical resistivity tomography (ERT) were carried out as part of FloSolutions fieldwork in 2015 to determine the thickness of the alluvium and the underlying geology within the TMF impoundment. The geotechnical data has been used to evaluate the tailings basin and embankment foundation conditions, and to evaluate the geological and geotechnical factors affecting design of the TMF.

[1] The MCE is based on seismic studies from other mining projects in the vicinity to Zafranal. By definition the MCE has no associated Annual Exceedance Probability (AEP), it is associated with the 1:10,000 year ground motion in some jurisdictions.

18.2.4.2 Impoundment

Based on the investigation, groundwater levels within the TMF range from approximately 30 to 60 m below surface. In addition, the majority of the impoundment is underlain by the Lower Moquegua Formation (Tertiary-aged siltstone) which is characterized by low permeability (low K value) with a thickness varying from approximately 0 to 100 m. Based on the ERT geophysical work, the majority of the impoundment (approximately 60%) is covered by Tertiary and Quaternary alluvium. The alluvium ranges in depth from 0 m to 50 m with an average thickness of approximately 40 m.

18.2.4.3 Southwest embankment foundation

The base of the embankment is situated within the Quebrada Huacan, which is covered with alluvium. The alluvium is approximately 170 m wide and 50 m deep and is loose (at surface) to medium dense (at depth). The alluvium within the embankment footprint is underlain by Diorite and Chocolate basement rock.

The left abutment, looking downstream, of the southwest embankment consists of volcanoclastic and quartz diorite. The right abutment, looking downstream, of the southwest embankment consists mostly of the Chocolate formation.

18.2.4.4 Northwest embankment foundation

The limestone and calcarenites intermixed within the volcanoclastics of the Chocolate formation located within the northwest embankment area show small signs of surficial dissolution. However, the thinly bedded limestone in this area does not show signs of karstification.

18.2.5 Design basis and assumptions

18.2.5.1 Dam classification

A dam classification has been carried out to determine appropriate design earthquake and flood events parameters for the design of the TMF embankments, as part of the design criteria. The term “dam” in this instance is used in its strict definition as a barrier that impounds water or tailings, which for the TMF is an embankment. The selection of appropriate design earthquake and flood events has been based on classification of the tailings embankments using criteria provided by the Canadian Dam Association’s (CDA) “Dam Safety Guidelines 2007” (Revised 2013), which is recognized as an internationally accepted standard in the development of tailings facilities. Based on an analysis following CDA guidelines the TMF was assigned a dam class of significant (years 1 through 5) to high (years 6 through 19) looking at potential risks to communities and the environment along with economic impacts downstream.

18.2.5.2 Design flood event

Selection of an appropriate Inflow Design Flood (IDF) is required to carry out a safety assessment of the TMF and to determine storm storage requirements. The size of the IDF is dependent on the dam classification. Based on the significant and high dam classification assigned to the TMF, an appropriate IDF is an event equal to the 1,000 year event and “one third between 1,000 years and the Probable Maximum Flood (PMF)” event, respectively. In addition, for closure best practice is to design the TMF to safely pass the PMF. However, based on the discussions at a review workshop between CMZ, Teck, MMC, and Ausenco it was decided to be more conservative and reduce risk by using the PMF for both operations and closure due to social issues in southern Peru with other mining projects.

18.2.5.3 Design earthquake

An appropriate MDE for embankment design has been selected, based on the Significant (starter) and High (Years 0 to 2 and 2 to 19, respectively) dam classification defined for the TMF embankments based on the criteria for design earthquakes provided by the CDA “Dam Safety Guidelines 2007” (Revised 2013). The CDA Guidelines require that for the specified dam classification be designed for a probabilistically derived event (defined as the Earthquake Design Ground Motion) having an annual exceedance probability (AEP) of 1/1,000 (Starter) and 1/2,500 (Years 2 to 19). However, based on discussions at the same review workshop between CMZ, Teck, MMC, and Ausenco referred to above, it was decided to be more conservative and reduce risk by using the Maximum Credible Earthquake (MCE) and the MDE during both operations and closure.

18.2.5.4 Design criteria

The total tailings transported to the TMF will be passed through a cyclone sand station for a proportion of the operating time to produce suitable cyclone sand for the construction of the embankment. Initially there will be only one station required as there will be only the southwest embankment. At Year 10, a second station will be required for the northwest embankment. The cyclone sand station design has been matched with the concentrator tailings production. The annual cyclone sand fill production requirement ranges from 522,000 m^3 to 2,435,000 m^3 (809 kt to 3774 kt) annually. For estimating cyclone sand fill requirements for embankment construction, a conservative average *in situ* dry density of 1.55 t/m^3 was assumed for compacted cyclone sand fill.

The TMF has been sized to provide sufficient capacity to store approximately 396 Mt of tailings (including cyclone sand tailings used as embankment fill). Approximately 43 million tonnes of tailings will be used as cyclone sand fill for embankment construction. The remaining 353 million tonnes of tailings (156 million tonnes of cyclone overflow and 197 million tonnes of whole tailings) will report to the impoundment as slurry for permanent storage. For development of the TMF filling schedule, an average tailings dry density of 1.20 t/m^3 was assumed for the first two years of production and 1.40 t/m^3 from Year 3 to Year 19 (end of operations). These values were based on predicted average dry density values provided by tailings laboratory consolidation results. The depth-area-capacity relationship is shown on Figure 18-1.

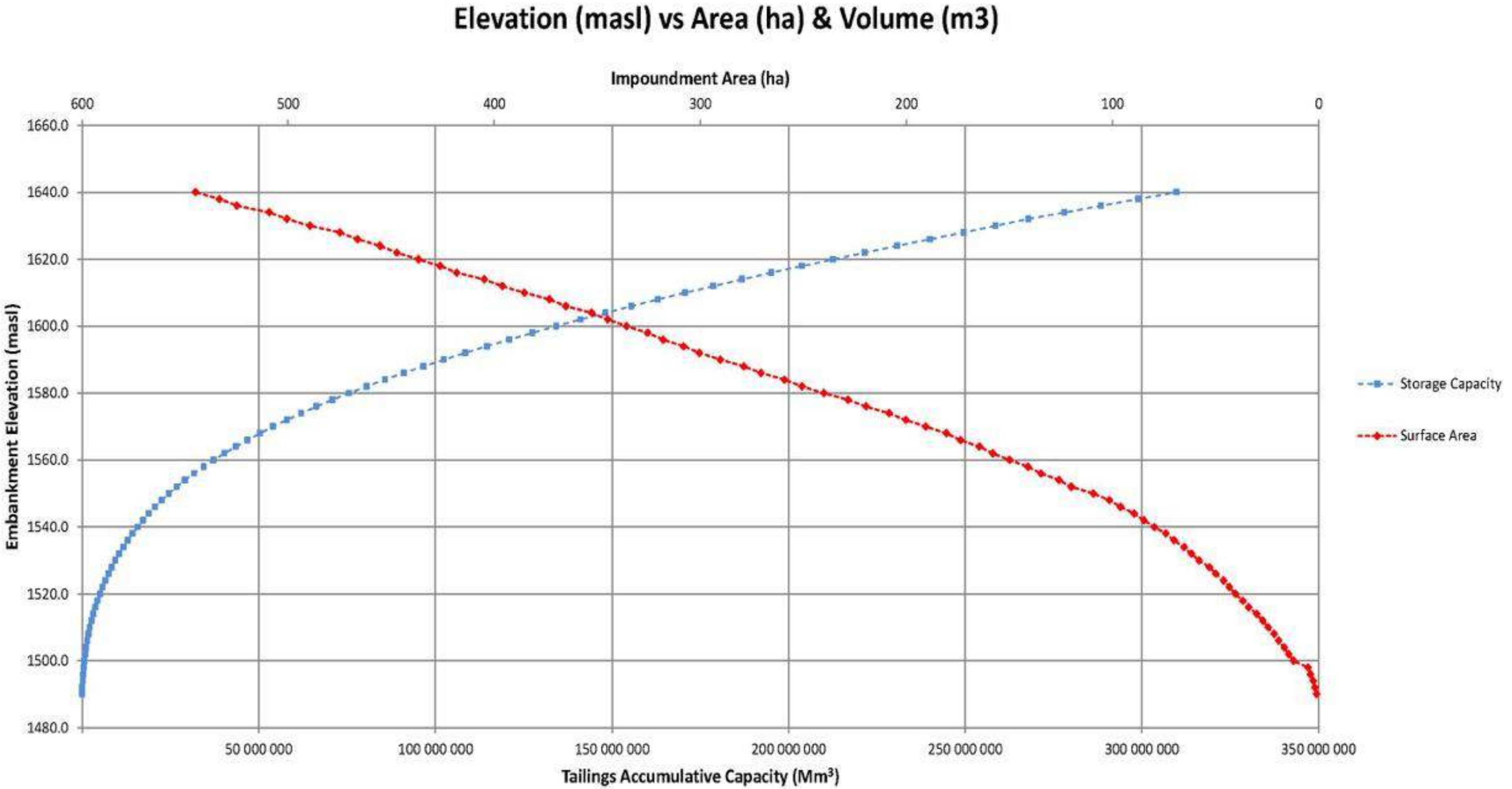


Figure 18-1 - Depth-area-capacity curve

18.2.6 TMF layout, development and operating strategy

18.2.6.1 General

Tailings will be deposited in an impoundment located in the Quebrada Huacan southeast from the concentrator. The final stage of the TMF, at the end of the LOM is shown in Figure 1-2. Specific overall features of the TMF reflected in Figure 18-2 and Figure 18-3 are:

- Two alluvium fill starter embankments, with geomembrane and geosynthetic clay liner (GCL) on the upstream sides and raised with cyclone sand, referred to as the southwest embankment and northwest embankment
- Cyclone sand station at each embankment
- Tailings distribution pipelines – bulk tailings, cyclone underflow (sands), cyclone overflow (fines)
- Reclaim water systems – barge with splitter box to convey supernatant to the plant or to the cyclone system
- Supernatant (surface water) pond
- Seepage management system – seepage collection drains and gallery, and seepage recycle system.

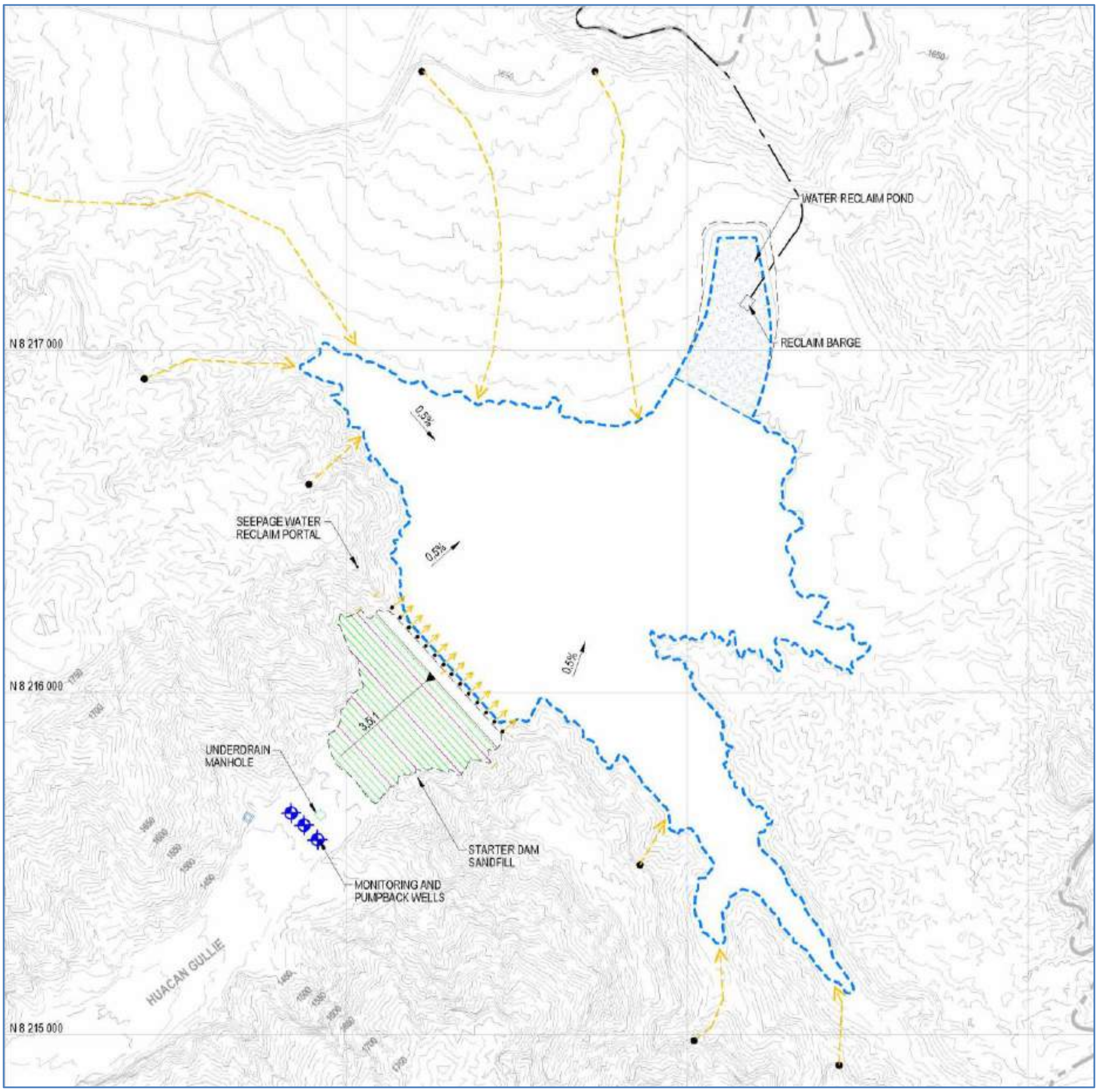


Figure 18-2 – TMF tailings deposition Year 1.5

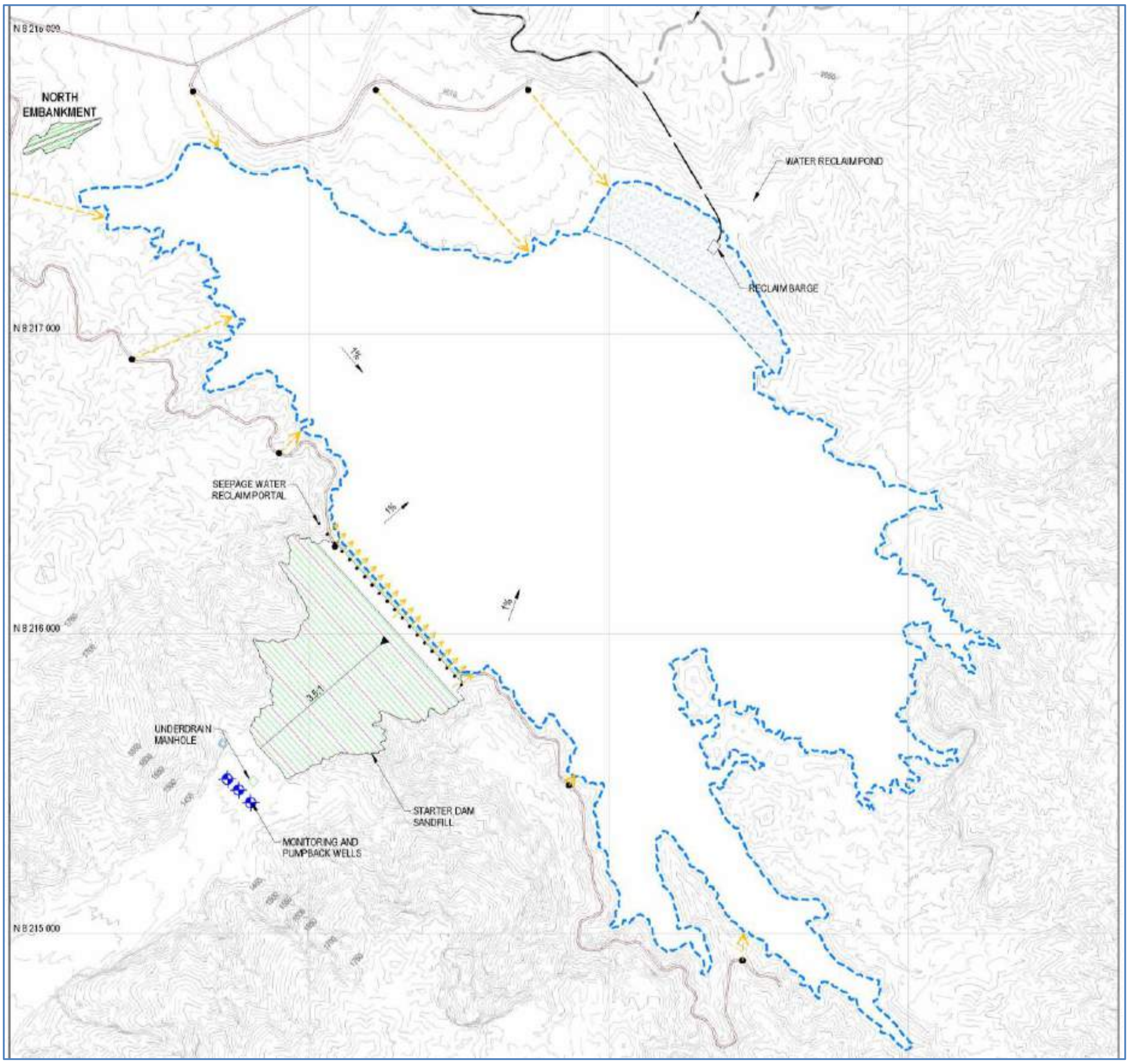


Figure 18-3 – TMF tailings deposition Year 10 showing starter northwest embankment

18.2.6.2 Cyclone sand design considerations

The particle size distribution of the Zafranal mill tailings is a key consideration for determining the suitability of the whole tailings to provide cyclone sand of suitable quality and in sufficient quantity. Coarser tailings are preferred, as higher sand fraction or 'split' can be realized. Clean sand with sufficiently low fines content (% passing 75 µm) will be required to keep up with the rate of rise requirements. The anticipated particle size distribution of the Zafranal tailings is within the range of tailings particle size distributions that have been successfully cycloned at other tailings operations.

The planned height of the southwest embankment limits the sand placement options and likely necessitates construction using sand cells, where additional vibratory compaction will be required to ensure a sufficiently dense, high strength material. It is important to provide a cyclone sand fill material that is sufficiently dense to provide adequate strength for the embankment (with consideration of the high confining stresses) and to ensure the material is not susceptible to cyclic softening or flow liquefaction under seismic loading.

18.2.6.3 Embankments

The TMF embankments will be developed in stages throughout the life of the TMF using a combination of suitable alluvium borrow materials from the impoundment and cyclone sand.

The southwest embankment will be developed across Quebrada Huacan. The alluvium-fill starter embankment is designed to accommodate tailings production, the maximum operating pond, and the PMF, and two metres of freeboard for the first 1.5 years of operations and will be approximately 92 metres 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-GCL anchored to bedrock as a cut-off barrier to reduce seepage downstream.

Cyclone sand production in the first year and half of operations will be used to extend the downstream shell of the main embankment from a 2:1 (H:V) to a 3.5:1 (H:V) slope, allowing the embankment crest to be raised after the first 1.5 years of production, which have the highest annual generation of tailings. Ongoing embankment raises will comprise centreline raises with cyclone sand. Cyclone underflow produced from the whole tailings stream diluted at the cyclone plant will be discharged directly into the cyclone sand shell zones (both upstream and downstream of the centerline of the embankment) where it will be spread in cells and compacted using bulldozers. The maximum height of the final embankment (crest to downstream toe) will be approximately 200 metres.

The northwest embankment will be required in Year 10 of concentrator operations with construction similar to that of the southwest embankment, except that due to the depth to bedrock, it is not cost effective to create a seepage cut-off barrier. For the northwest embankment, in addition to the upstream geomembrane-GCL liner, there will be a core filter zone between the GCL and alluvium with greater reliance on using cyclone plant overflow to seal the adjacent alluvium and natural walls of the impoundment with fines. Ongoing embankment raises will be developed similarly to the southwest embankment. The ultimate embankment is required to a maximum height of approximately 47 m.

18.2.6.4 Borrow materials

Local borrow sources are required to provide the following materials for embankment construction:

- Filter zone – required gradation depends on the gradation of the alluvium zone material and the tailings against the embankment. Local borrow within the TMF impoundment is the source for this material.
- Alluvial shell –shall be sourced from within the TMF impoundment.

All embankment construction materials will sourced from the TMF impoundment. The alluvium is expected to be sourced relatively easily from the impoundment based on the field geotechnical program. Some mechanical separation may be required to remove oversized material, i.e. greater than 0.6 m.

18.2.6.5 Water management

A monthly operational mine site water balance, incorporating stochastic analysis for climate conditions, indicates that the TMF 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.

The starter southwest embankment will be constructed in pre-production years, with construction scheduled to provide enough time to impound sufficient surface water from the make-up water supply system to pre-saturate the alluvium in the base of the TMF so the seepage collection system and a reclaim pond is functioning at start-up. It is estimated that to pre-saturate alluvium and generate a water pond suitable for reclaim, approximately 3 Mm^3 is required, corresponding to about 3 months of make-up water supply during the pre-production period. Process water for ongoing concentrator operations and water for cyclone plant operations will be reclaimed from the TMF supernatant pond, the seepage collection systems, and the underdrain systems.

The water balance model developed for the TMF indicates that there will still be a water deficit at closure. However, a spillway will be provided to provide safe passage of the design storm water event and to reduce ponded water within the TMF.

18.2.6.6 Tailings distribution

Two distinct tailings streams will discharge into the TMF:

- Whole tailings – when cyclone operation is not required
- Cyclone overflow (fine fraction of whole tailings) – when cyclone operation is required.

The coarser fraction of the discharged tailings will settle rapidly, tending to accumulate closer to the discharge points and forming beaches with 1% slope. Finer tailings (slimes and partial segregation from the whole tailings) particles will travel farther and create flatter beaches with 0.5% slope. Below the supernatant pond beaches will form at steeper slopes, approximately 3 to 4 percent depending on the material. Deposition strategy will be developed and managed, with the intent to seal off the alluvium with the overflow, maximize storage efficiency in the TMF, and to maintain the deeper, cleaner locations of the supernatant pond in the vicinity of the reclaim barges. In addition, a barge channel will be constructed approximately 80 m wide and 10 to 20 meters deep to provide a defined location of the supernatant pond and promote clean reclaim water for the reclaim barge.

A more detail deposition plan needs to be evaluate during the feasibility study to determine the potential impact of sloping sub-aerial and sub-aqueous beach areas.

18.2.7 TMF seepage management

Seepage within the foundation and along the southwest side of the TMF and from the northwest embankment has been identified as a risk due to the permeable nature of the geological materials and lack of hydrodynamic containment. Engineered and operational measures to reduce seepage risks include geomembrane lining of the starter embankment, in addition to primary, secondary, and tertiary seepage collection systems. The proposed seepage collection systems are summarized below, and are supported by the hydrogeological site conditions, hydrogeological conceptual model, and numerical model simulations presented in Section 20.1.2.

18.2.7.1 Primary seepage collection system

Seepage generated from the cyclone overflow and whole tailings within the TMF will be collected through the alluvial floor and tied into a main collector drain installed at the base of the starter embankment trench at the top of bedrock. Allowance has been made for a network of French drains in the TMF floor as part of the collector system.

18.2.7.2 Secondary seepage collection system

A need for secondary seepage collection has been identified at the right abutment area of the southwest embankment where high-permeability fractured bedrock was encountered. Groundwater and tailings seepage from the fractured bedrock would flow by gravity to a tunnel using a fanned network of horizontal to sub-horizontal drill holes collared within a drilling chamber located at the end of a ramp where it would be pumped to surface. No secondary seepage collection system is required for the northwest embankment.

18.2.7.3 Tertiary seepage collection system

Seepage collection wells will also be strategically located within the alluvial material down-gradient of the southwest embankment and northwest embankment and within high-angle fault zones. The alluvial material beneath the northwest embankment is too thick to remove or for construction of a hydraulic cut-off wall; therefore, seepage will be collected using a line of pumping wells located immediately down-gradient (northwest) of the northwest embankment. The locations of the seepage recovery wells is shown in Figure 18-4.

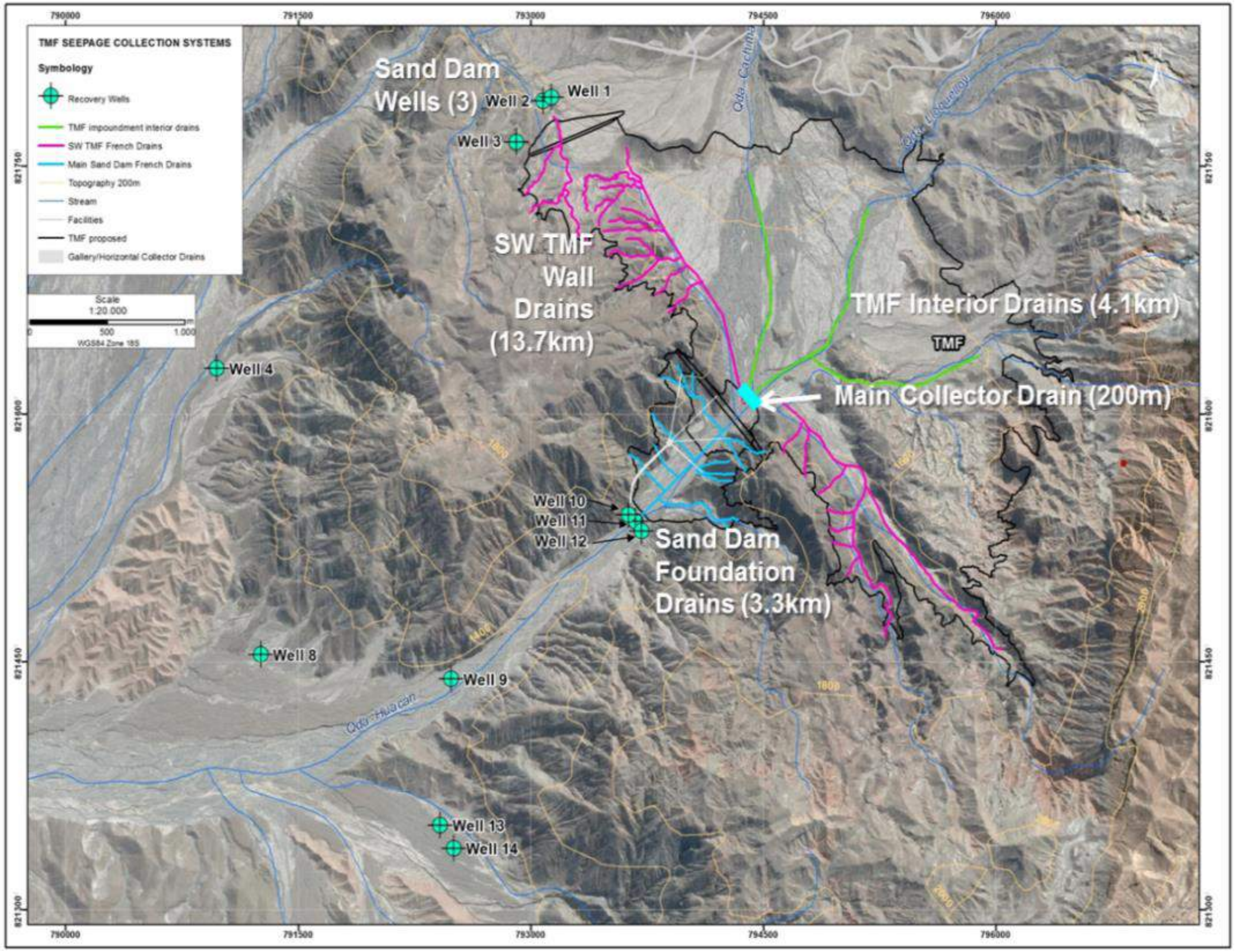


Figure 18-4 – Primary seepage collection system

18.2.8 TMF embankment stability analysis

Stability analyses were carried out to investigate the stability of the southwest and northwest embankment 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 Maximum Design Earthquake (MDE)

The stability analyses were based on a typical cross-section through both embankments for the final TMF with a supernatant pond elevation of 1630 m and an embankment crest elevation of 1635 m. The stability analyses were carried out using the limit equilibrium computer program Slide. In this program a systematic search is performed to obtain the minimum factor of safety from a number of potential slip surfaces. Factors of safety have been computed using the Spencer method. In accordance with international recommendations (ICOLD, 1995 and CDA 2007 (revised 2013) and standard industry practice, the minimum acceptable factor of safety for the tailings embankment under static conditions is 1.3 for short term operating conditions and 1.5 for the long-term post-closure condition. A factor of safety of less than 1.0 is acceptable under earthquake loading conditions provided that calculated embankment deformations resulting from seismic loading are not significant and that the post-earthquake stability of the embankment maintains a factor of safety greater than 1.2, implying there is no flow slide potential. For the post-earthquake case the tailings deposit was conservatively assumed to be fully liquefied, and an appropriate low residual strength was applied. 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.

18.2.9 Instrumentation

Geotechnical instrumentation will be installed in the tailings embankment and foundation during construction and over the life of the Project. The instrumentation will be monitored during the construction and operation of the TMF to assess embankment performance and to identify any conditions different to those determined during design and analysis. Amendments to the on-going designs and/or remediation work can be implemented to respond to the changing conditions, should the need arise.

Geotechnical instrumentation, comprising vibrating wire piezometers, slope inclinometers, internal settlement instrumentation, and exterior movement (survey) monuments will be installed at selected planes along both the southwest and northwest embankments. Groundwater wells will be installed at suitable locations downstream of the embankments.

18.2.10 Closure and reclamation

The primary objective of the closure and reclamation initiatives will be to eventually return the TMF site to a self-sustaining facility with pre-mining usage and capability. The TMF will be required to maintain long-term stability, protect the downstream environment and manage surface water. Activities that will be carried out during operations and at closure to achieve these objectives are discussed in the following sections.

Upon mine closure, surface facilities will be removed in stages and full reclamation of the TMF 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, cyclone plant 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 TMF.
- Removal and re-grading of all access roads, ditches and borrow areas not required beyond mine closure for the TMF 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 minimise dusting potential. Measures may also include tailings stabilization, i.e. adding an agent to create a trafficable crust, in the final year of operation.

18.3 Surface Water Management

The Project area is situated in the arid belt in the lower basin of the Camaná-Majes River. A relationship between precipitation and altitude is observed, with clouds coming from and rising east to west from the Amazon and descending the western slopes of the Andes, dehumidifying towards the coast in the process. On the coast the effect of the cold sea surface and the subsidence of the South Pacific anticyclone inhibit precipitation. This atmospheric phenomenon favours a regime where surface flows in the area of the mine are ephemeral, with a few days of runoff each year, and where there may be years without any runoff at all. However unusual events of short duration do occur with flows high peak flows. These relate to alterations in continental atmospheric dynamics that weaken the subsidence of the South Pacific anticyclone, which favours the intrusion of Amazonian cloud over the Andes and into the coastal desert strip causing intense rainfall that is very irregular in space and time and which results in hyper-concentrated flows.

The conditions of aridity of the area of the mine as well as the conditions of ephemeral or high peak and short-term flows, promote the design of the management of surface water by capturing ephemeral flows in conditions where this is possible, in order to minimize the entry of fresh water from other sources into the operation as well as to optimize the recirculation of contact water from within the mining process; and giving priority to the protection of mine facilities in situations of maximum precipitation.

18.3.1 Objectives

The main objectives of the management of surface water on the site are summarized as follows:

- Minimizing mine-contact water to prevent mine-contact water from entering the receiving environment by surface discharge. This is achieved by routing clean surface water runoff around disturbed areas and minimizing sediment discharge from the site to the environment by entrapping and retaining eroded sediment as close as possible to disturbed areas.
- 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 of sediment entry toward facilities and erosion at discharge points.

- Achieve environmental compliance.

18.3.2 Proposed surface water management controls

A number of water control structures have been proposed for the 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 in order 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 TMF to reduce water supply required from off site.
- Impounding as much ephemeral runoff volume as possible in water retaining ponds to diversion structures

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

Surface runoff which can be intercepted and directed by the diversion works will be considered 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.

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.

Diversion channels have been based on the final PFS configuration for the waste dumps and the concentrator; diversion channels 1 and 2 are on the western edge of the central waste dump, and diversion channels 3 and 4 on the eastern edge; upstream of the northeast waste dump and north waste dump are diversion channels 5 and 6 respectively; downstream, on the western side of the north waste dump, it is complemented by diversion channel 7; and finally, diversion channel 8 is situated northwest of the concentrator plant.

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.

18.4 Product Transport

Concentrate will be transported from the Zafranal mine site to the Port of Matarani using a standard combination of tandem drive tractor articulated with a tipper semi-trailer. The trucks will be equipped with pneumatic suspension and extra wide tires to carry a legal gross weight of

58.2 tonnes and a concentrate payload of 35 tonnes. This will require special permitting under the national regulations to allow use on the public highway system.

The corridor for concentrate transportation will follow a western route from the site to Camaná and then along the new Costanera Highway (PE 1SD) route to the Port of Matarani.

It is anticipated that the truck cycle time from the mine site to the port will be approximately 13 hours. Concentrate transport has been designed based on haulage schedule of 300 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.

Despite the western route being the longest of those investigated, it has the lowest initial capital expenditure due to use of existing paved highways for over half the distance to the port. Operating costs for the western route were similar to the other routes, as the total truck cycle time was equivalent between the options. Due to the low capital costs, the western route was determined to have the lowest present cost and was considered the best option.

18.5 Access Roads

Site access roads were developed from both the local towns of Anexo El Pedregal (Corire) and El Pedregal de Majes (Majes).

The Corire access road is an existing road which will be upgraded to suit the Project requirements. The upgraded road will be unsealed, have two lanes, and be 7.2 m in width. Additionally, a new sealed section of road (approximately 2 km in length) will be constructed to bypass the town of Anexo El Pedregal (Corire).

The Majes access road is an existing road, for approximately 29 km (from Majes), which will be upgraded to suit the project requirements. The final 5 km of road to the Project site will be new. The upgraded and new road will be unsealed, have two lanes, and be 7.2 m in width.

18.6 Power Supply

18.6.1 HV power supply

The power supply system is based on a total installed power requirement of 99 MW and peak demand of 91 MW.

A power supply alternatives study, for connection of the Zafranal operation to the Peruvian National Grid (SEIN), was conducted by GMI. The best of three alternatives developed, from a technical and cost perspective, is the construction of a single transmission line from the south connecting to the New Socabaya substation.

The main conclusions from the PFS developed by GMI, and reviewed and accepted by Ausenco, are as follows:

- The New Socabaya substation connection alternative represents a longer length of transmission line; however, its connection point to the national system is guaranteed with the construction of the new substation. The commercial operational start-up is expected by the end of 2016 and is being managed by the ISA-Transmantaro consortium.
- The other two alternatives both pose greater risks. The alternative to branch from the Cotaruse-Socabaya 220 kV transmission line will be subject to the approval of the COES during the pre-operability study. And the alternative to connect from the San Jose substation will require consent from Cerro Verde Mining Company, which is the owner of the substation.

- Based on the findings from the technical analysis done and given the uncertainty of the alternatives, it is recommended to use the New Socabaya substation connection alternative for the PFS.
- Power supply in southern Peru will be 30% higher than the power demand by 2021.
- Power supply is sufficient for the Zafranal Project.

18.7 Water Supply

Engineering components related to the potable, construction, and operational groundwater sources are described below. The locations of the proposed water supply sources are shown in Figure 18-5. Additional details and potential effects from development of an operational water supply are discussed as part of 20.1.2.

18.7.1 Construction (start-up) and potable water supply

The water requirement for the start-up construction phase is 20 L/s. Two potential water supply sources have been identified within close proximity to the Project: (1) a deep, confined flowing artesian aquifer within Pampa Ganchos, located 5 km southwest of the TMF; and (2) high permeability zones within the fractured bedrock in Quebrada Huacan, located immediately southwest of the TMF. The groundwater supply system in Quebrada Huacan has been integrated with the TMF seepage collection design, which will operate throughout the mine life. It is expected that groundwater could be extracted using a combination of conventional vertical wells and horizontal drains collared within an underground tunnel (gallery) within the right abutment of the southwest embankment. Groundwater from the fractured bedrock near the TMF would require treatment using reverse osmosis for potable purposes. Confirmation of the aquifer capacity and properties, and further definition of the groundwater recovery system will be carried out during the feasibility level studies.

18.7.2 Operational water supply

The operational water supply system preliminary design is for 410 L/s of make-up water based on the water balance calculations with an appropriate design allowance. A brackish groundwater resource, located beneath the Majes I irrigation area approximately 35 km south of the Project, has been identified as a potential operational water supply source, and has been selected as the basis of the current study, although further investigation of local groundwater potential, such as Quebrada Huacan, may identify supplemental or alternative sources. Additional details regarding the hydrogeology of the Majes I area and the operational water supply well field are discussed as part of Section 20.1.2.

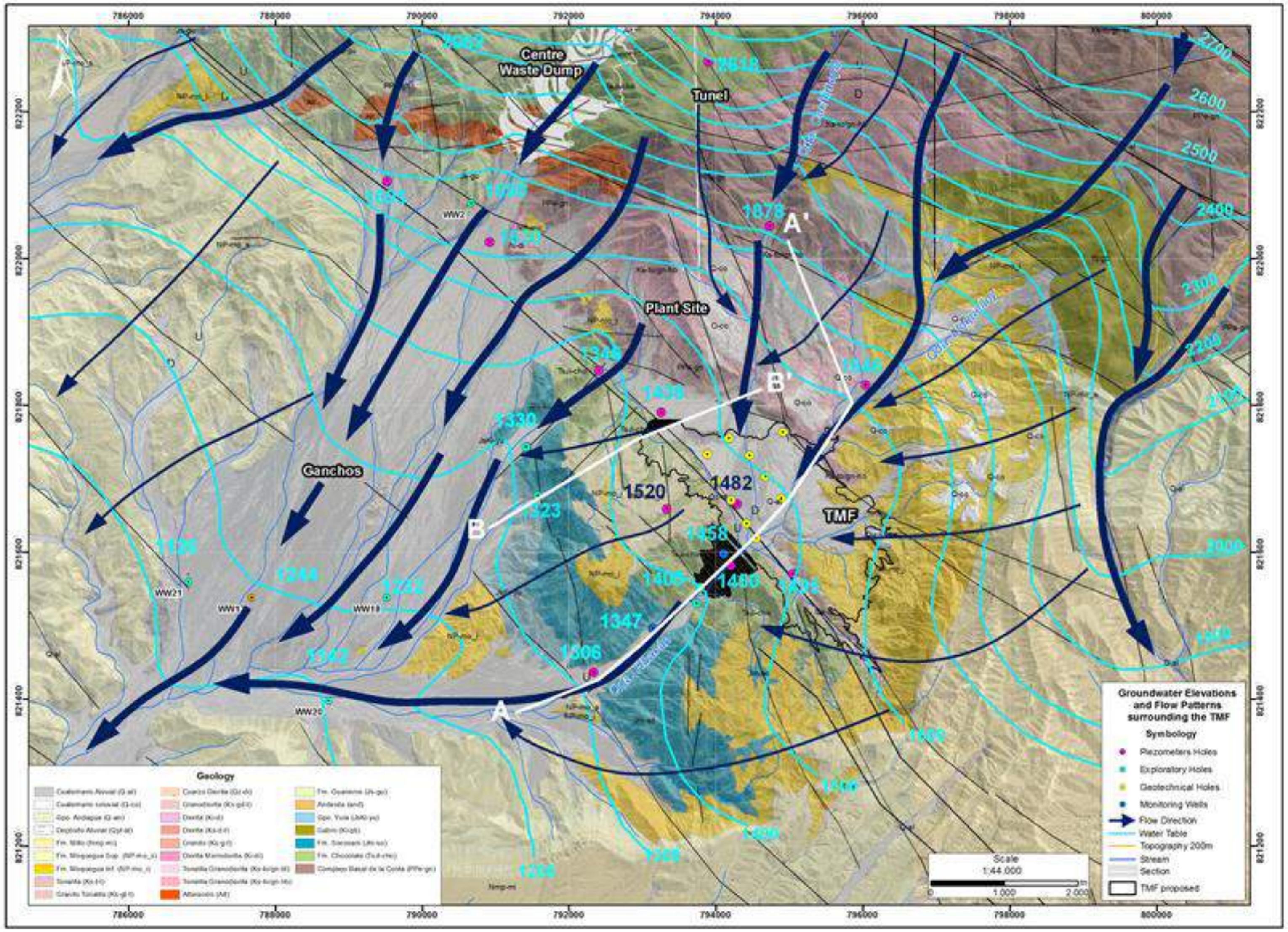


Figure 18-5 – Locations of start-up and potential operational water supplies

18.7.3 Water supply design

Table 18-3 presents a summary of the water well collection (WWC) and water supply system (WSS) components and design considerations, Figure 18-6 shows a theoretical wellfield location relative to the Project site.

The trade-off studies carried out to select the Zafranal WSS pipeline route considered as part of its design basis a buried pipeline installation along the entire route; however, to reduce capital cost, the pipeline design was changed from buried to above ground. However, there are risks associated with such an installation:

- Corrosion control problems at transitions
- External coating damage due to wear and consequential corrosion
- Community issues due to a water pipeline installed in plain sight
- Security and integrity compromised due to vandalism and accidental damage
- Greater exposure to the effects of earthquake, flooding, landslides, etc.

Table 18-3 – WWC and WSS summary

System	Pipe material and diameter	Mechanical equipment
Wellhead	8” diameter carbon steel pipe ASTM A53 Gr. B 0.25” thickness	9 x multi-stage vertical turbine deep well pumps Q = 216 m^3/h (each) P = 150 kW (each)
WWC	10”, 16”, 24”, 26” HDPE SDR 17	Normal demand 7 x operating + 2 x standby Maximum demand 8 x operating + 1 x standby
WSS	20” diameter carbon steel API 5L X65, 0.250” thickness with 0.012” fusion bonded epoxy (FBE) internal coating and three layer polyethylene (3LPE) external coating	2 x multi-stage horizontal centrifugal pumps both operating Q = 740 m3/h (each) P = 2046 kW (each)

For power supply of all WSS and WCC facilities located at Majes I, it is planned to extend a 22.9 kV overhead transmission line from the main substation near the concentrator. The exclusivity of this line from the Project provides high reliability and power quality without affecting agricultural users in the area.

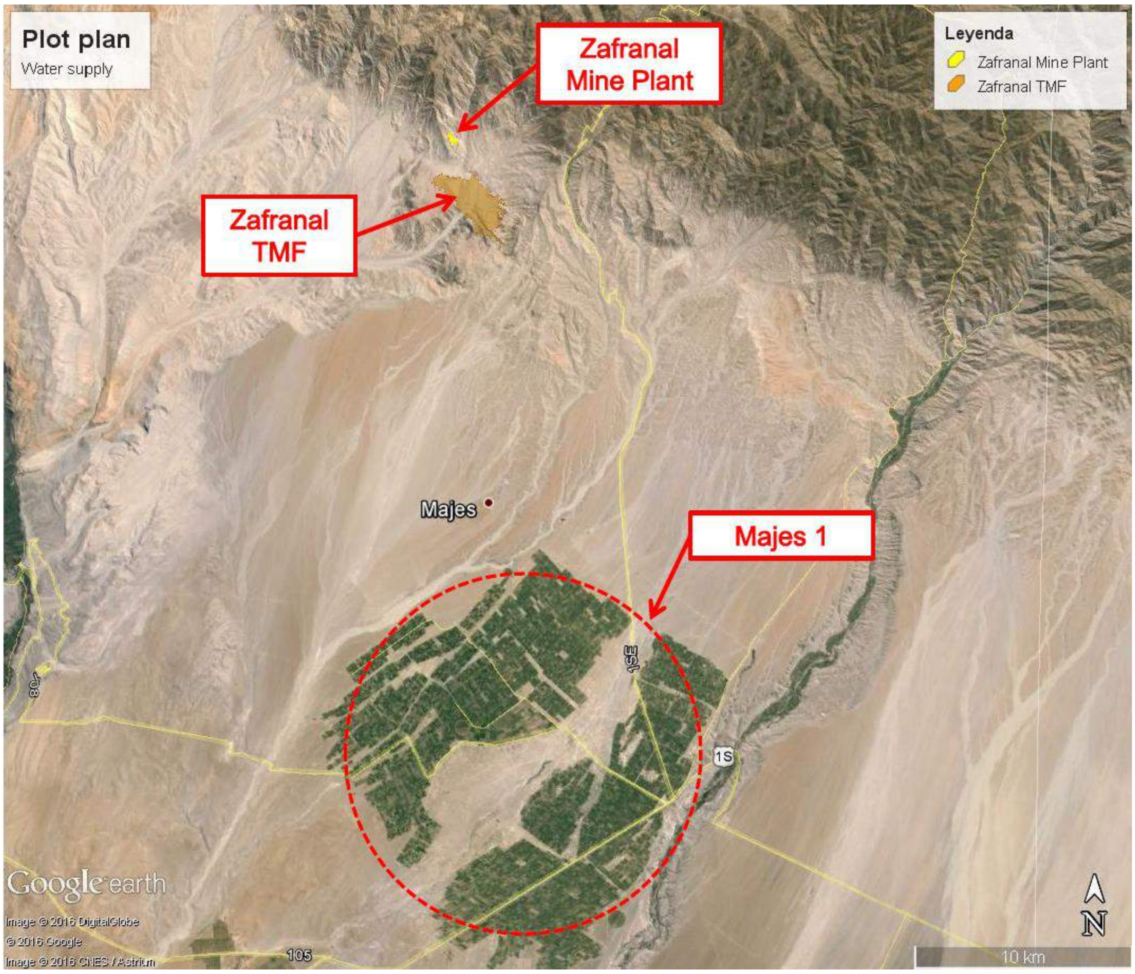


Figure 18-6 – Majes I relative to Zafranal site

18.7.4 Variable throughput

As discussed in Section 13.4.1.6, the concentrator throughput will vary through the life of mine as the relative proportions of supergene and hypogene ore in the concentrator feed varies. The 99th percentile life-of-mine average water demand for the variable throughput case is within the design capacity of the water supply system; hence, it was considered that there was no requirement to change the design or overall capital cost estimate for the water supply in the prefeasibility study.

As discussed in Section 17.4.2, later comparisons for the peak demand year showed that while the average flow is accommodated within the water supply design capacity, the 75th and higher percentiles are not. However, the exceedance, even for the 99th percentile, is within the precision of pre-feasibility capital cost estimates, and the main component of the increase in operating cost, power consumption, is incorporated in the PFS operating cost.

For the next phases of the Project, revision of the site water balance model for the variable throughput case should be a high priority, followed by review of the water supply design.

18.8 Site Accommodation

The strategy for site accommodation has two camps, one that will be provided by the contract mining company and the second to accommodate the construction phase and operations phase requirements of the project. The 3,000 person construction camp will be a temporary facility generally designed for 3 to 5 years of life: however, one component of the construction camp will be the 450 person operations camp, which will be a permanent installation designed for the life of mine.

18.9 General and Administration Facilities and Utilities

18.9.1 Buildings

The general and administration building facilities are located downhill of the concentrator water storage and distribution areas. The basic philosophy is for a modular (portable) design for office facilities. The general and administration buildings are the administration office, which includes the first aid facility, and the site gate house which includes the weighbridge.

18.9.2 Sewage treatment and effluents discharge

Sewage (gravity fed from the plant facilities, general facilities, and the camp) is treated in a vendor-supplied sewage treatment facility. The sewage treatment facility will be a conventional plant with aeration, sedimentation, clarification, and filtration. Effluent from the sewage treatment plant is discharged to the environment as irrigation water. There will be a separate sewage treatment facility for the mine facilities.

19 Copper Market Studies

19.1 Trends for Copper Prices and Forecasts

19.1.1 Recent trends for copper price and refined copper stocks

Error! Reference source not found. shows that the rise in the price of copper observed early in the past decade (2001-2010) began in the second half of 2003 when the market broke through US$1.00/lb (US$2,205/t).

Prior to the year 2000, the copper price generally was below US$0.90/lb. From 2004 to mid-2008, copper price in US dollar terms rose due to strong demand growth led primarily by the BRIC countries (Brazil, Russia, India and China) against a relatively stagnant supply base, rising input costs, a falling US dollar and a substantial flow of speculative investment funds all contributing to such commodity price rises. The growth in consumption in China was in the forefront of this growth in global demand. The net result of this rapid increase in demand was to push the refined metal balance into deficit in the middle years of the past decade and resulted in the sharp rise in prices.

At that time it became evident to market participants that global supply of refined copper from mines and secondary sources (scrap) was failing to keep up with an increasing industrial demand for copper, with refined copper stocks in metal exchange warehouses (LME + COMEX + SHFE) beginning to show a significant decline from its 1.5 million tonnes peak in May 2002.

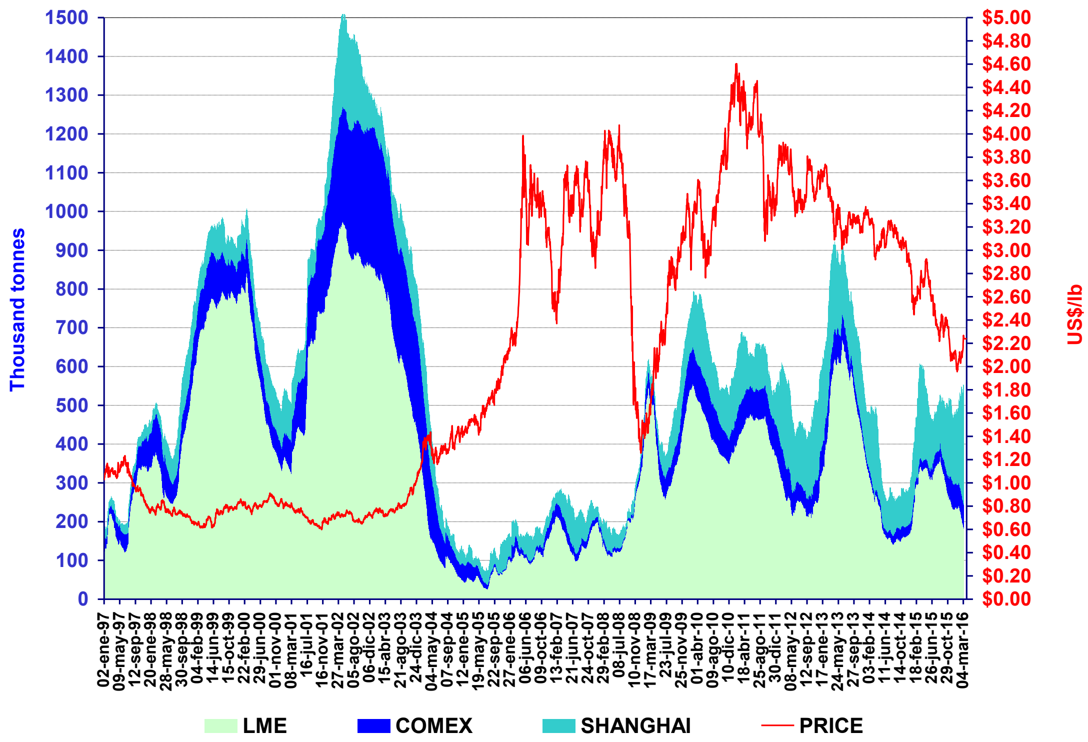


Figure 19-1 - Copper price and refined copper stocks in metal exchange warehouses

The market also became aware that the world economy was in the midst of a period of unprecedented economic growth, with China as the key driver. As a result buyers bid up to secure future deliveries of copper and a period of price increases ensued.

More fuel was added when fund managers, aware of the excellent fundamentals prevailing in the copper market and faced with the prospects of a weaker US dollar, proceeded to invest considerable sums in buying up copper futures.

Nevertheless, since October 2008 and in a relatively short period of time, the global economic scenario moved from a somewhat-contained developed country downturn to a deeper and more synchronized global economic slowdown due to the sub-prime housing crisis in the US and the ensuing impact on global financial markets. Global demand for copper was severely impacted by the downturn in manufacturing activity and destocking, with refined copper stocks increasing to the highest level in five years by February 2009. Copper prices declined very sharply to their low point by December 2008, reaching US$1.26/lb (US$2,780/t).

This downturn was somewhat short-lived and the rebound in many commodity prices, including copper, appeared very resilient. Government economic stimulus around the world led to a rebound in commodity prices after the sub-prime crisis with copper leading base metals off their lows in early 2009. Figure 19-1 shows that the copper price reached its all-time high of US$4.60/lb (US$10,140/t) in February 2011.

The surpluses that materialized over the year 2012 and first half of 2013 due to sharp increases in mine copper production replenished the stock pipeline, so that total refined copper stocks reported by metal exchanges (LME, COMEX and SHFE) reached 0.9 million tonnes by mid-2013.

Since mid-2013 refined copper stocks declined to reach very low levels by mid-2014, due mainly to strong refined copper consumption in China, but even with that, copper prices continued declining given the pessimistic scenario prevailing in the course of 2014, not only for copper but for commodities in general. This was coupled with a persistent crisis in the euro-zone in 2014.

A combination of low business and consumer confidence, lower than normal refined copper consumption in China and an extended winter in the northern hemisphere moved the global refined copper market to surplus by the beginning of 2015, with a sharp decline in copper prices in January 2015 (-13.4% in a month) and rising visible inventories. Total refined copper stocks reported by metal exchanges (LME, COMEX and SHFE) increased significantly in Q1 2015 by 293 kt, from 307 kt at the end of December 2014, to 600 kt at the end of March 2015, their highest levels since November 2003.

From hitting a trough in January 2015 at US$2.44/lb, copper prices rallied in 2015 to reach US$2.90/lb in May but prices have consistently declined since then to touch in January 2016 their lowest point since May 2009 at US$1.96/lb. Nevertheless the copper price has recovered somewhat since then.

19.1.2 Copper market forecasts up to 2020

Price-related production cutbacks announced by mining companies in the later months of 2015 provided some support to copper prices, but is it considered that these cutbacks will not allow a recovery in copper prices but only prevent further decline. A global refined copper market moving back in 2016 towards a deficit after cuts, will not be constrained enough to support the price in 2016 given the prospects for a surplus again in 2017. Under this scenario copper prices will stay at a low level both in 2016 and 2017.

An annual surplus of 468 kt of copper is estimated for 2015, followed by deficit of 95 kt in 2016, returning to surplus of 33 kt in 2017 in the case that no additional production cutbacks are announced by mining companies. This will place downward pressure on copper prices and the cash price is expected to drop to US$2.20/lb in 2016 and US$2.30/lb in 2017.

Postponement of copper mining projects announced by mining companies is estimated to lead to a global refined copper market moving to supply deficit in 2018 (-133 kt Cu), 2019 (-36 kt Cu) and 2020 (-43 kt Cu), with a low starting level of refined copper stocks allowing a recovery in copper prices from 2018 onwards (see Table 19-1).

Table 19-1 – Global refined copper supply/demand balance and price forecast

Copper Flow from Mine to Refined Copper Production

(Thousand tonnes of copper)

	2013	2014	2015	2016	2017	2018	2019	2020
Mine Production	**18,094**	**18,404**	**19,145**	**19,650**	**20,386**	**20,806**	**21,541**	**22,078**
Copper in concentrate	14,411	14,624	15,172	15,755	16,462	16,790	17,209	17,702
Copper in cathode	3,683	3,780	3,973	3,894	3,924	4,016	4,332	4,377
Processing Losses	-406	-409	-424	-443	-466	-478	-490	-503
Mine Production (Recoverable copper)	**17,688**	**17,995**	**18,721**	**19,207**	**19,920**	**20,328**	**21,051**	**21,575**
Production from Secondary Materials	3,873	3,992	3,835	3,791	3,843	3,872	4,021	4,160
Production Disruption Allowance (concentrates)	0	0	0	-574	-720	-734	-752	-774
Direct Use of Blister (Chemical Products)	10	10	10	10	10	10	10	10
Implied change in stocks of concentrates (*)	299	-135	253	61	70	51	95	72
Implied change in stocks of blister (*)	-6	4	-9	2	2	-2	24	2
Refinery Output	**21,258**	**22,108**	**22,302**	**22,350**	**22,961**	**23,407**	**24,191**	**24,877**
Production Disruption Allowance (cathodes)	0	0	0	-146	-177	-181	-195	-197
Refined Copper Production	**21,258**	**22,108**	**22,302**	**22,204**	**22,785**	**23,227**	**23,996**	**24,680**
% change year-on-year	5.3	4.0	0.9	-0.4	2.6	1.9	3.3	2.8
Refined Copper Consumption	**21,252**	**22,061**	**21,834**	**22,300**	**22,752**	**23,360**	**24,032**	**24,722**
% change year-on-year	6.1	3.8	-1.0	2.1	2.0	2.7	2.9	2.9
GLOBAL BALANCE	**6**	**47**	**468**	**-95**	**33**	**-133**	**-36**	**-43**
COPPER PRICE, US c/lb (nominal)	**332**	**311**	**249**	**220**	**230**	**240**	**260**	**290**

It is expected that 2018 will be a transition year, with the copper market starting to return to being in structural deficits. At this point, the ramp-ups at soon-to-be-commissioned greenfield projects and brownfield expansions will be slowing, and as a direct result of the low price environment, there will be a lack of follow-through projects to maintain strong supply growth. In addition, copper production at existing mines will be declining due to falling ore grades. The market is expected to react to the supply scenario, and an increase in the price is forecast in 2019 to US$2.60/lb, followed by a further increase in 2020 to US$2.90/lb.

Figure 19-2 shows copper price forecasts released during the first quarter of 2016 by a group comprising the most reputable analysts. It can be seen from this graph that analysts are forecasting in general a decline in copper prices in 2016 and 2017, with varying degrees of recovery from 2018 onwards.

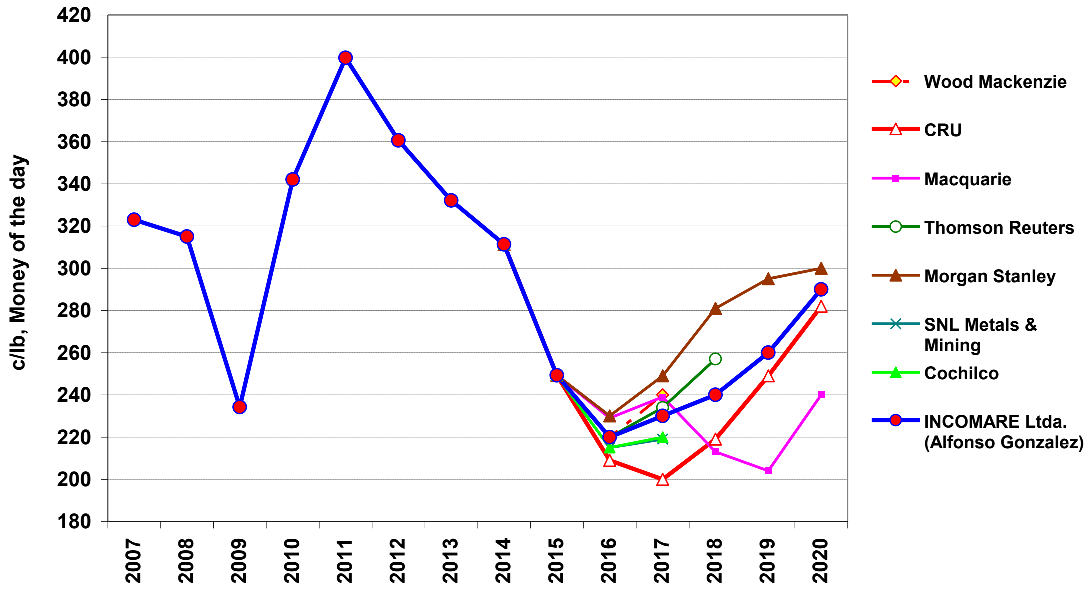


Figure 19-2 – Recent Copper Price Forecasts According to Several Sources

19.1.3 Copper Price forecasts in the longer term

For industries such as copper that are in structural deficit, that is, with long-term demand significantly in excess of base case production intentions, incentive pricing provides the most appropriate estimate of long-term cycle average prices. The incentive price analysis examines the price required to provide an investor a given rate of return for each project, and calculates the price required in theory to warrant investment in sufficient cumulative capacity to meet potential demand. In this context long-term can be considered the cycle average price over the next cycle, i.e. the decade over which a project would expect to see payback.

Assuming that for the foreseeable future copper demand will be sustained at a growth rate of 2.5% per year; roughly 700 kt of new mine copper production would be required each year (see Figure 19-3). Given the environment of high capital costs to build new projects a higher copper price will be required by mining houses to move ahead with greenfield projects. It is estimated that a long-term copper price of US$3.00/lb (US$6,614/tonne) in real US$ terms of 2016 is needed to provide sufficient incentive to ensure that copper does not slip back into a structural deficit and to encourage the development of the next wave of projects to supply expected global copper demand in the long term.

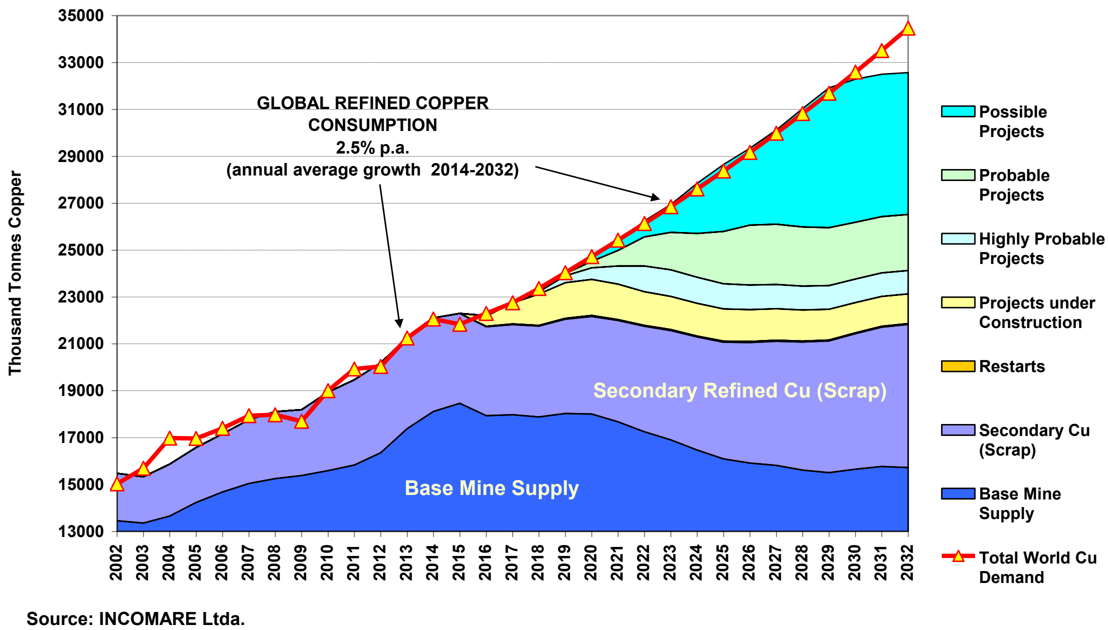
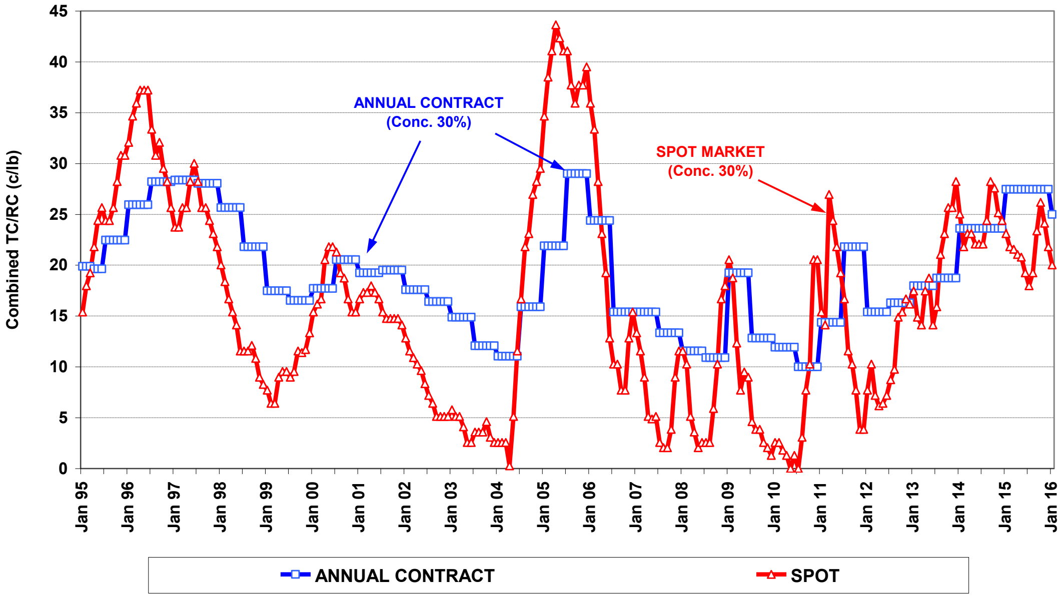


Figure 19-3– New Mine Copper Production Required in the Longer Term

19.2 Copper Concentrates Market

The market for copper concentrates reflects the price of refined copper (which is a result of the supply and demand balance for refined copper) and the supply and demand balance for copper concentrates. Payments to suppliers of copper concentrate are structured to reflect the market price of refined copper, with agreed deductions, penalties for undesirable impurities in concentrate (e.g. arsenic), and charges that historically represented the cost of treatment per tonne of concentrate (treatment charge) and the cost of refining per pound of metal (refining charge). Over time, treatment charges (TCs) and refining charges (RCs), collectively referred to as TCRCs, have devolved to a market pricing mechanism with no direct connection to the costs of treatment or refining. Producers and buyers with frame contracts meet each year to negotiate annual TCRCs. During these negotiations the parties exchange their views on the current conditions and future outlook for the supply and demand for copper concentrates. The terms are negotiated based on the exchange of each party’s views and on the review of settlements in copper concentrates agreed by the major participants in the market which are often referred to as “Benchmark Terms”.

Figure 19-4 depicts the movements in TCRCs from 1995 to January 2016 for both annual contracts and spot market. It can be seen that annual charges follow the trend of spot terms, but show less variability.



Source: INCOMARE Ltda.

Figure 19-4 – Historical Treatment and Refining Charges for Copper Concentrate Market

As shown in Figure 19-5, the balance for global copper concentrate market moved to surplus in the second half of 2013 due to a strong production performance at a group of very relevant copper concentrate producing operations, including Grasberg, Collahuasi, Codelco Norte (Ministro Hales start up), El Teniente, Antamina and Oyu Tolgoi ramp-up. There was not enough smelting capacity in the second half of 2013 to smelt all the concentrate produced in this period and concentrate stocks in hands of producer, smelters and traders increased considerably. As a result, spot terms recorded a sharp increase from US$60.3/dmt treatment charges (TCs) and 6.03c/lb refining charges (RCs) in the first quarter of 2013 to US$103.3/dmt and 10.33c/lb in the fourth quarter of 2013.

The copper concentrate market clearly moved to deficit in 2014 given that, among other factors, copper concentrate shipments from Indonesia were put on hold by many mining companies due to a new progressive export tax on copper concentrates imposed by the Indonesian Government. Freeport McMoRan and Newmont Mining halted concentrates exports from their respective Grasberg and Batu Hijau operations for several months.

Following the resumption of exports from Indonesia, concentrates were in good supply by the end of 2014, with spot terms trending up but contained within the US$85-100/dmt TC and 8.5-10¢/lb RC range by the commissioning of new smelting capacity in China and a persistent backwardation in the cash to 3-month copper price during the period, which allows traders to offer lower spot TCRCs to miners, including long quotational periods. Benchmark contract terms for 2015 were settled at US$107/dmt TC and 10.7¢/lb RC, compared to US$92/dmt TC and 9.2¢/lb RC for 2014.

Abundance of complex concentrates and lack of clean concentrates was an issue in 2014 and 2015. A significant proportion of mine output growth in 2014 and continuing into 2015 is accounted for by high arsenic concentrates mainly from Toromocho in Peru and Ministro Hales in Chile, putting pressure on the market for clean concentrates, which are required for blending, with clean concentrate producers receiving significant premiums for their material for blending purposes. This should ease in the coming years as more clean concentrates, such as Las Bambas, are brought to the market.

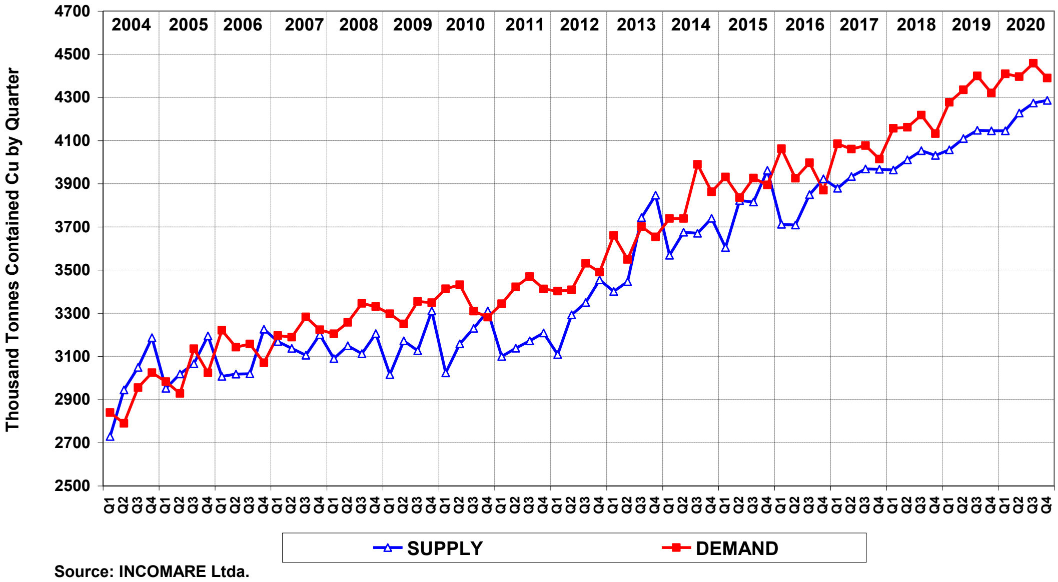


Figure 19-5 – Global Copper Concentrate Supply/Demand Balance, by Quarter

Around 800,000 tonnes per year of additional smelting capacity is expected to have come on-line in China in 2015 and 2016, placing further upward pressure on concentrate demand, although some of this additional capacity may be delayed as a result of reduced refined metal demand and low concentrate availability.

Mine closures announced by mining companies during the last few months of 2015 are expected to reduce concentrate supplies in 2016, increasing the concentrate market deficit forecasted for 2016. It is estimated that the concentrate market will record a significant deficit in the first half of 2016, with spot TCRCs remaining at very low levels in the first half of 2016 and possibly rising in the second half of the year as new copper concentrate production is exported from Peru with the production ramp-up of the Cerro Verde expansion and the commissioning of Las Bambas project.

The forecast for the copper concentrate market continues to indicate a shortfall of product availability throughout the next five years due to committed mining project delays, slow ramp-ups and additional smelting capacity to be commissioned in China. This scenario will require smelter capacity utilization adjustments from current planned levels, especially in the final years of the forecasting period. The forecast for treatment and refining charges (TCRCs) applied to the sale of copper concentrates point to an average of US$90.0/dmt and 9.0c/lb, CIF Main Japanese Port, during the next five years.

For the longer term TCRCs averaging US$90.0/dmt and 9.0c/lb are also estimated, which is equivalent to a charge of 24.1c/lb for a 28% copper grade concentrate, similar to that being produced by Zafranal.

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Environmental Baseline and Impact Assessment

As per the common terms of reference for environmental impact assessments for large-scale mining in Peru (Ministerial Resolution Nº 116-2015-MEM/DM), environmental baseline studies require two seasonal data collection periods: one corresponding to the rainy season and one corresponding to the dry season. This is so that seasonal variations in environmental conditions are considered in the impact assessment.

Environmental baseline studies were initiated for the Zafranal Project in the rainy season of 2015, having completed the required public information workshops in September and October 2014. Dry season data collection for both on-site and off-site infrastructure is programmed for June-July 2016.

The contract for the environmental impact assessment for the Zafranal Project is in the process of being evaluated. Preparation of the EIA could commence upon completion of the baseline data collection and is expected to take approximately 6 months to complete. The following sections provide an overview of the baseline data collected to date, focussing on the key components that have particular relevance to the advancement of the Project.

20.1.1 Hydrology and water balance

20.1.1.1 Regional baseline

Hydrographically, the Project area is in an arid strip located on the left margin of the Majes River, specifically in the watershed of the Huacan creek, which forms part of the Sicera creek watershed. The Huacan creek is formed from ephemeral flow contributions from the Cachimayo and Lloquelloy creeks and covers an extension of 83 km^2 at the elevation of the TMF; at the confluence with the Sicera Creek, it covers an area of 263 km^2, with altitudes that vary from 558 masl to 4,400 masl at its source. The main channel has a southwest trend; its length is approximately 42.7 km and has an average slope of 8.7%.

The average annual precipitation in the catchments containing the open pit, concentrator and TMF are 152 mm, 112 mm and 50 mm, respectively. The average annual evaporation for the TMF area is 3,085 mm.

Surface runoff into the creeks of the Project area occurs only in times of heavy rain and does not intercept groundwater, instead recharging underground flow through infiltration. The average flow for the catchments was estimated at between 2.6 L/s and 26.8 L/s.

20.1.1.2 Storm event dynamics

A relationship between precipitation and altitude is observed, with clouds rising from east to west from the Amazon and descending on the western slopes of the Andes, dehumidifying towards the coast in the process. On the coast the effect of the cold sea surface and the subsidence of the South Pacific high subtropical anticyclone inhibit precipitation.

This atmospheric phenomenon favours a regime where surface flows are ephemeral, with a few days of runoff each year, and where there may be years without any runoff at all. However unusual events of short duration do occur with high peak flows.

An example of the occurrence of these mechanisms was recorded in the Project area with the event of 23 to 29 March 2015. Light hyper-concentrated mud flows were observed in the recorded videos with depths in the order of 0.25 to 0.50 m in the beds of the streams in the area of the Project. In Huacan creek, flow traces indicate that flows had a width of 20 m and depth of 0.30 m.

The maximum 24 hour 100-year rainfall varies from 19 mm in the colluvial-alluvial plain of the TMF (1,580 masl) to 73 mm for the average altitude of the watershed upstream of the open pit (2891 masl).

The maximum flow rate for different return periods was estimated and a storm hyetograph defined using regional information. The maximum flow rate for 100-year return periods in the streams for situations "without Project" and "with Project" were calculated as being 25.1 m^3/s and 19.9 m^3/s; 15.7 m^3/s and 8.0 m^3/s; 47.0 m^3/s and 49.4 m^3/s for basins A, B and C respectively, while for basin D was calculated as being 28.0 m^3/s for both situations (refer to Section 5 for descriptions of the basins).

For the area of the Project, specifically for the C and D basins that discharge into the TMF, it has been estimated that the probable maximum flood (PMF) will be 80% of the volume of the probable maximum precipitation (PMP), being 16.9 Mm^3.

20.1.2 Hydrogeology

20.1.2.1 Mine site

Hydrogeological investigations of the mine site area consisted of geologic and wet/dry season hydrogeological mapping, 7.3 km of geophysical surveys, soil permeability testing at 18 locations, diamond (2) and reverse circulation (10) drilling, combined total of 80 hydraulic conductivity tests, construction of 20 piezometers, water level monitoring, and water quality sampling. The data were compiled and used to develop conceptual and numerical models for groundwater flow in the mine site area to investigate potential environmental impacts from the Project and to identify potential construction and operational water sources on site.

Intrusives, volcaniclastic, and mylonitic rocks dominate the highland areas at Zafranal near the mine site. Mineralization at Zafranal is hosted mostly within windows of older rocks exposed next to Cretaceous granodiorite. Exposure of these older rocks occurred due to block movement along two major east-west faults closely linked to the nearby Incapuquio Fault. Groundwater movement is almost entirely through tectonically induced high permeability fractures, with flow generally directed toward local quebradas where groundwater discharges into fault zones aligned along the quebradas. Within the mine site, pre-mining groundwater movement follows principal faults towards the Huacan and Ganchos Valleys (Figure 20-1).

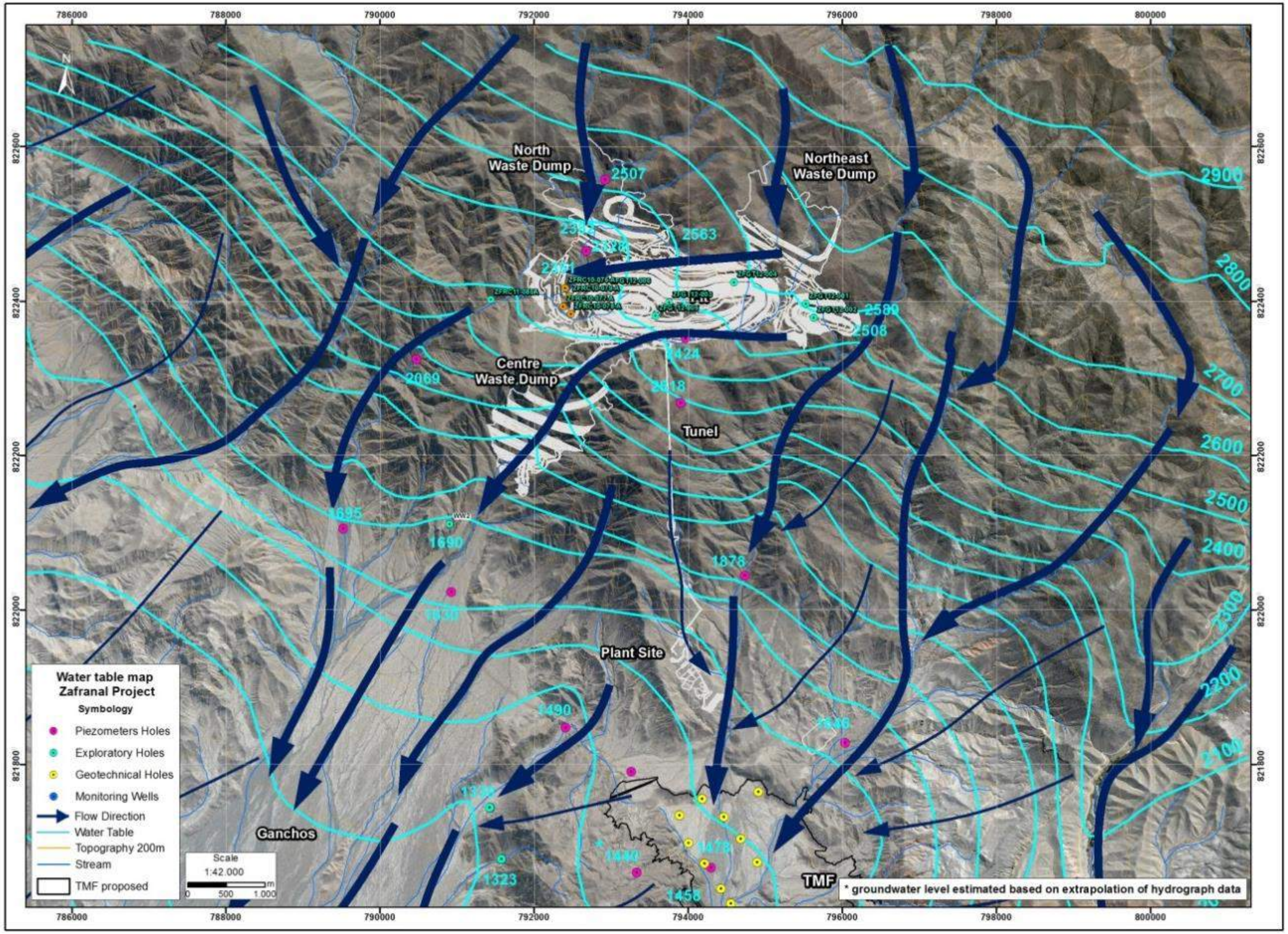


Figure 20-1 – Pre-mining groundwater flow patterns at the mine site

The groundwater flow model was used to simulate groundwater inflow rates and evaluate the potential area of influence from open pit development. The transient pit growth simulation produced an inflow rate around 8 L/s at the end of mining. The area of influence of pit dewatering extends approximately 8 km north of the open pit and approximately 6 km west, east, and south of the open pit. Drawdown is slightly elliptical near the open pit along the east-west direction, likely as a result of transverse faults. Impacts are expected to be limited to lowering the water table in the immediate area of the pit and potentially drying up local seeps located within the radius of dewatering. During operations there is limited risk of seepage from the proposed pit as groundwater flow patterns will be radially directed toward the pit.

After mine closure, a pit lake is expected to form within the lower 100 m of the pit. Pit lake development will be gradual, likely taking over 100 years to fully develop, reaching equilibrium when inflows equal outflows. The current concept is that the open pit will act as a localized hydraulic sink during operations and following closure as a result of low recharge rates, high evaporative losses, and compartmentalization effects from bounding fault zones. There is some potential for seepage to the west along the north and south fault zones which will be monitored by piezometers aligned along the quebrada. The water quality of the lake is expected to show increases in acidity, metals, and salt (e.g. sulfate) over time. In the event seepage from the pit lake occurs after closure the seepage would report to the same collection system installed for the central waste rock dump and/or TMF. Negligible seepage is expected to be generated from the mine waste storage areas due to the very low rainfall and high evaporation rates. Any seepage generated from these waste rock facilities will be monitored at the toe of the facilities within the weathered bedrock/fault zone and captured if necessary. Although there is risk for deeper seepage and bypass of the monitoring and seepage collection system, the seepage rates are expected to be very low with very long travel times to the Majes River (located 30 km to the southwest) which is the expected discharge area for regional groundwater flow.

20.1.2.2 Tailings management facility

Hydrogeological investigations of the TMF area consisted of geologic and wet/dry season hydrogeological mapping, 16 km of geophysical surveys, soil test pitting at 17 locations, 17 soil permeameter tests, completion of 46 drill holes (diamond and reverse circulation), combined total of 201 hydraulic conductivity tests, construction of 23 piezometers, water level monitoring, and water quality sampling. The data were compiled and used to develop conceptual and numerical models for the TMF area.

Within the Huacan Basin (40% of the TMF footprint) and along Quebrada Huacan (southwest embankment) high-permeability alluvium is present. Approximately 90% of the alluvial material within the TMF footprint is underlain by low permeability Lower Moquegua Formation siltstones which in turn are underlain by low-permeability intrusive rocks. Along the southwest side of the TMF, and between the southwest and northwest embankment, there is an uplifted block of fractured Jurassic carbonate and volcaniclastic rocks of the Chocolate and Socosani Formations. These rocks can have a high degree of fracturing with moderate to high permeability. There is no evidence of karstification of the carbonate rocks within the TMF foundation and dam areas. The Huacan Fault zone, which bisects the Jurassic carbonate and volcaniclastics rocks, has a moderate to high permeability based on drilling and hydraulic testing. Both the Huacan Fault zone and other fault zones are targets for groundwater exploration and seepage collection.

A pre-mining water table map near the TMF is shown in Figure 20-2, along with conceptual seepage paths local to the TMF. Within the TMF foundation and Quebrada Huacan, groundwater levels are commonly found near the interface between the alluvium and the fractured bedrock. A downward vertical gradient exists between the alluvial and upper fractured bedrock layer and the deeper bedrock in these areas. Minimal hydraulic mounding, and therefore natural hydrodynamic containment, exists within the uplifted block along the southwest boundary of the TMF due to extremely low recharge rates and assumed well-drained nature of the fractured and faulted rock mass.

Construction and operation of the TMF is anticipated to create a hydraulic mound within the underlying foundation that exceeds the pre-mining groundwater elevations during operations. To counteract this effect, the TMF design will include a number of measures to protect groundwater and minimize uncaptured seepage. The alluvial material is anticipated to serve as a natural underdrain and seepage collection system for water recovery and reuse. Operationally, spigotting of fine-grained low-permeability overflow tailings along the southwest side of the tailings impoundment and northwest embankment areas should act to reduce seepage into the volcaniclastic-carbonate complex and alluvium. Engineered measures to reduce the identified seepage risk include geomembrane lining of the southwest and northwest starter embankments, in addition to a primary collection system within the impoundment and embankments, secondary collection beneath and immediately down-gradient of the embankments, and tertiary collector wells located further down-gradient from the TMF.

Numerical modeling using the groundwater flow model developed for the Project indicates that uncollected seepage can be limited to less than 1 L/s. TMF water quality has not yet been sufficiently characterized for future operating and closure conditions, but seepage losses at these low rates are not expected to significantly affect the water quality of local or regional aquifers, which have no registered users. Shallow seepage is predicted to reach Pampa Ganchos during operations (within 20 years). Intermediate depth seepage is predicted to arrive at Pampa Ganchos between 20 and 100 years. Deeper seepage is predicted to arrive between 100 and 200 years. Perimeter monitoring and pump back wells will be required during operations. After closure, seepage will decline as the water levels at the TMF decrease and the tailings dewater. It is assumed that at the cessation of tailing deposition, seepage collected downstream from the TMF will be pumped back to the impoundment and evaporated.

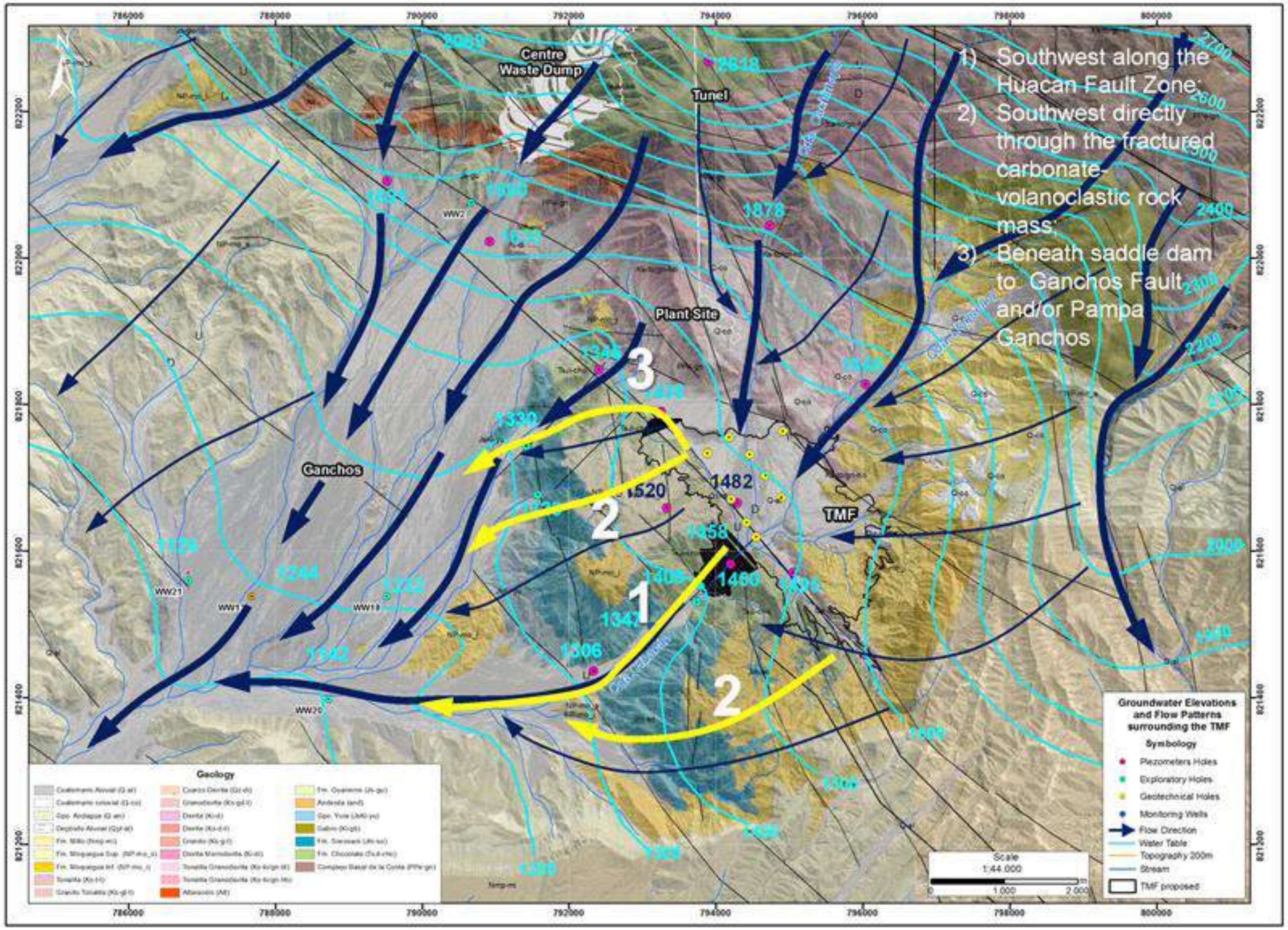


Figure 20-2 – Pre-mining groundwater flow patterns surrounding the TMF and conceptual seepage paths from the TMF

20.1.2.3 Operational water supply (Majes I irrigation area)

In addition to potential sources of water on the Zafranal site as described above, the Majes I agricultural irrigation area, located approximately 30 km south of the Zafranal Project, has been identified as a potential operational water supply source. The water would come from a brackish groundwater mound which is believed to have been formed from deep percolation of lost irrigation water into permeable soils since 1983. The hydraulic mounding of groundwater has been linked to stability issues along the west side of the Siguas River bank and potential water quality effects along the Siguas River. Pumping of brackish groundwater from beneath Majes I should have no negative effect on surface activities within the irrigation district; pumping could be beneficial in reducing pore pressure in landslide prone areas. The brackish groundwater quality appears to be unsuitable for irrigation or animal drinking water use. It is, however, suitable for industrial use at the Zafranal Project.

Hydrogeological investigations of the Majes I irrigation area consisted of local to regional geologic mapping, hydrogeological mapping along the Siguas River valley, drilling and installation of two test wells, hydraulic testing, recharge analyses, and water chemistry sampling of groundwater beneath Majes I and surface water within the Siguas River. The data were compiled and used to develop conceptual and numerical models for the Majes I and regional areas.

Exploration drilling, regional geological mapping, and geophysical information suggest that the brackish aquifer is continuous beneath the Majes I irrigation area. Brackish groundwater lies at a depth of approximately 100 m below surface and covers an estimated area of approximately 350 km^2. The average recharge rate is estimated at approximately 4,000 L/s and the estimated volume stored in the brackish groundwater mound is approximately 2-3 billion m^3. Samples of the brackish aquifer exceed Peruvian drinking water and irrigation water quality standards for a number of constituents including electrical conductivity, chloride, sulfate, boron, total dissolved solids, total coliforms, and *Escherichia coli* (*E. coli*).

Two exploration holes have been drilled during the pre-feasibility phase. Drilling and hydraulic testing at one test well located in the east sector of Majes I concluded that suitable conditions exist for the development of a well field. Similar hydraulic conditions were encountered at the second test well; however, fresh groundwater was discovered beneath the brackish zone at a depth of 240 m. The fresh groundwater, which was present prior to irrigation activities began, is apparently sourced from the Siguas River. CMZ is not considering the fresh groundwater aquifer as a water source for the Project.

The groundwater flow model was able to simulate existing groundwater levels and flow directions consistent with geophysical resistivity data, water level measurements, and estimated groundwater discharge rates to the Siguas River. The simulated depth to groundwater during 2016 is shown in Figure 20-3. A theoretical wellfield, consisting of nine wells (each pumping 46 L/s) located within the west agricultural sector at Majes I, was designed assuming a total flow rate of 410 L/s. Wells would only be installed within the brackish groundwater mound which is now believed to extend to the south, beyond the populated and irrigated areas. Subsequent studies will be undertaken to determine if the proposed wellfield can be located south of Majes I. In any event, minimal influence on the deeper, fresh groundwater is expected since water would mainly be taken from aquifer storage and ongoing recharge. Simulations of the theoretical well field showed minimal well interference effects and all wells maintained a sufficient saturated thickness to provide constant flow rates for the individual wells.

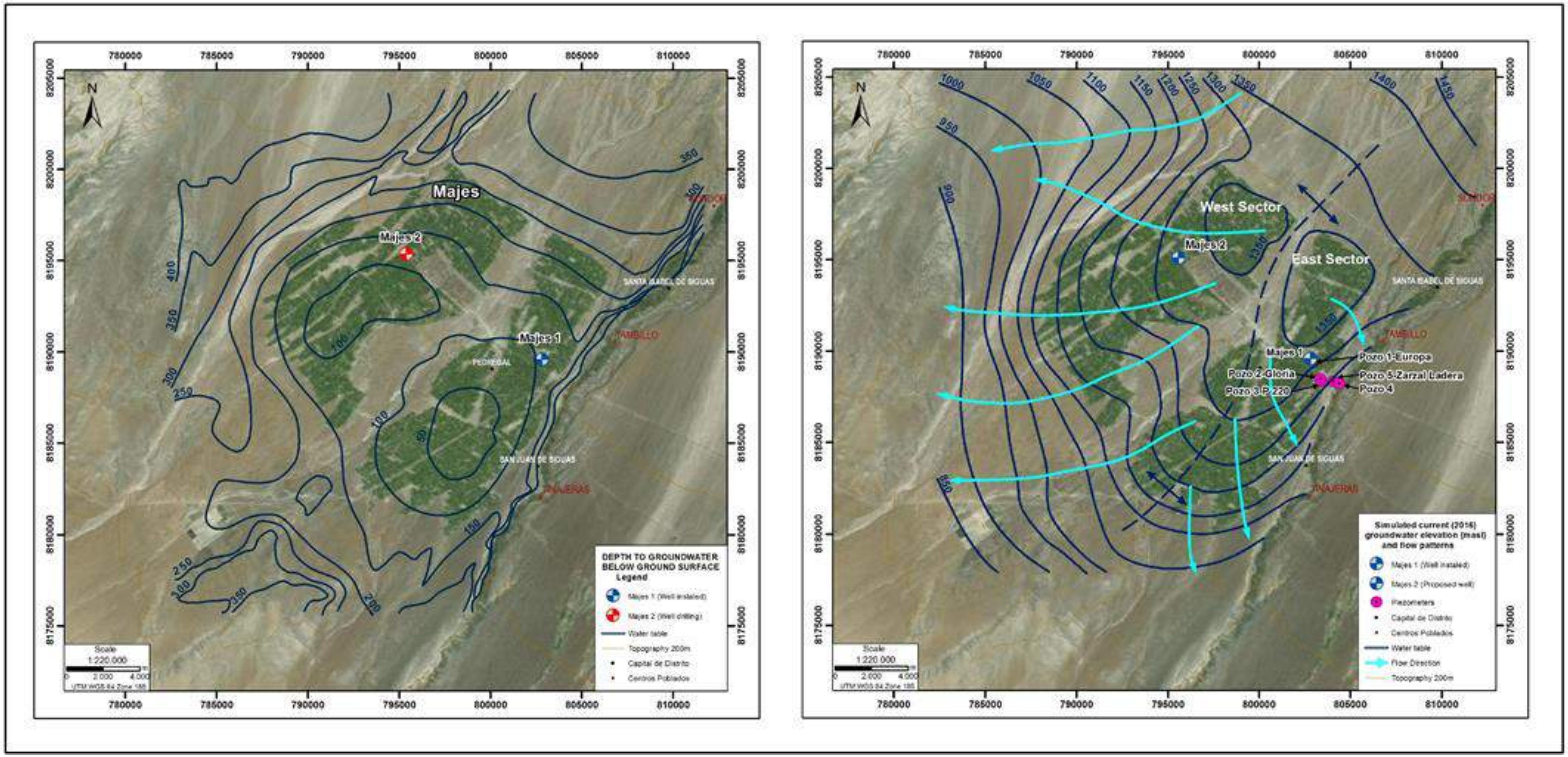


Figure 20-3 – Simulated current (2016) groundwater depths below ground surface in metres (on left) and groundwater elevations showing interpreted flow patterns (on right)

20.1.3 Geochemistry (sampling, testing, results)

A geochemical characterization program has been completed to assess the potential for acid rock drainage and metal leaching (ARD/ML) associated with wastes for the Zafranal Project. Characterization work completed on rock (expected waste and low grade ore) associated with the Zafranal Main and Victoria Zones were reported by pHase Geochemistry (2013). Work conducted on metallurgical tailings was presented by pHase Geochemistry (2016).

20.1.3.1 Waste rock and low grade ore

The geochemical sample subset was selected with emphasis on obtaining samples of the primary litho-alteration assemblages and mineralogical zones present, with representation based on the proportion that each lithology represents in the drill core database as shown in Table 20-1. Drill core assay data for key parameters was used to guide sample selection such that there was also representation of the range of key parameters (e.g. sulfur and copper) within the subset (Table 20-1).

Table 20-1 – Lithological representation in static sample set

Location	Main lithology	Code	Representation in core database	Sample numbers
Main Zone	Zafranal diorite	ZAF DIO	37%	63
	Volcaniclastic	VC	24%	44
	Microdiorite	MDIO	21%	39
	Late quartz diorite	LTQTZDIO	4%	15
	Dry diorite	DRY DIO	2%	18
	Other units		12%	10
	Total		**100%**	**189**
Victoria Zone	Mylonite	MLS	74%	31
	Gneiss	Gneiss/GNS	6%	1
	Quartz porphyry	QTZ PORPH	4%	-
	Andesite	AND	4%	2
	Zafranal diorite	ZAF DIO	3%	1
	Other units		9%	3
	Total		**100%**	**38**

Source: pHase Geochemistry, 2015

A wide distribution in sulfur, predominantly as chalcopyrite and pyrite, was reported between the various rock types at the two deposits. Mineralogical evaluations conducted in this program also identified jarosite and natrojarosite as important sulfur minerals in certain samples.

Median sulfur values typically ranged between 1% and 3%. These values are consistent with porphyry deposits at which ARD/ML is an important waste management issue. Limited carbonate within the Zafranal rocks suggests that substantial buffering capacity or carbonate mineral dissolution from the waste rock is not expected. ABA results suggest that the majority of waste rock associated with both deposits should be considered potentially acid generating (PAG), with roughly 75% to 85% of samples tested classifying as PAG. Samples that classify as clearly non-acid generating account for between approximately 15% and 25% of the samples tested and are largely sourced from the leached cap zones of the deposits and potentially from a proportion of the later stage diorite intrusives.

Kinetic testing indicates that where sulfur is mainly present as jarosite/natrojarosite, the ARD potential of the material may be over-estimated by ABA tests. However, these sulfur forms are generally found predominantly in the leach cap of the ore body (approximately 80 m in the Main zone and 30 m in the Victoria zone). Although these minerals are not soluble in water, they are soluble in acid. As they will be stripped first and will form the base layer of the waste rock dumps, it is expected that any acid generated in the waste rock dump will eventually react with the jarosite and natrojarosite, converting them into PAG materials.

Greater early-stage metal leaching potential will likely exist in rocks from the supergene and mixed zones of the deposits where soluble secondary mineral forms occur. If contact water remains buffered or is slightly alkaline, metals and metalloids such as Mo, As, Sb and Se may be mobile. If contact water becomes acidic, increased concentrations of metals such as Al, Cu, Cd, Fe, Ni, Zn and others would also be expected.

Capture and return of any contact water from the waste dumps would have to be maintained through operations. Options for closure need to be developed for the conceptual closure plan in the EIA although capture of the contact water will have to be maintained. Dispersion over the surface of the waste rock dump to promote evaporation of the contact water may be one of the more effective means of eliminating contact water discharge from the site.

20.1.3.2 Potential construction materials

Samples representing potential sources of construction materials were included in the program. These were surface grab samples and classified as predominantly non-acid generating and showed little potential for metal leaching. Confirmatory testwork on specific borrow or quarry areas will be required and has been included in the Feasibility Study program.

20.1.3.3 Tailings

Characterization of metallurgical tailings has occurred in phases following various metallurgical programs and is currently on-going. Results to date indicate that the tailings should be considered potentially acid generating. Bulk whole tailings and the cleaner scavenger tailings have higher acid generating potential than the rougher tailings stream and became acidic very quickly in kinetic testwork. With acid development, concentrations of a number of parameters were elevated (e.g. SO_4, Cu, etc.). During operations however, it is expected that supernatant water discharged with the tailings will carry sufficient alkalinity to maintain close to neutral pH conditions in the tailings facility. In these conditions, moderate concentrations of SO_4, F, Sb and Se might be expected.

20.1.4 Water quality

Water quality sampling was carried out during the baseline data collection period for the rainy season 2015 and also forms part of the participatory environmental monitoring program for the exploration EIAsd. There are seven monitoring points on the Majes River and springs for the exploration EIAsd. Ten surface water samples from the Majes River and four spring samples were taken in support of the Project EIA baseline. Additionally, one sample was collected from the Majes I irrigation area as part of the operational water supply study.

Water quality in the Majes River and the springs around the Project area was compared against the Peruvian standards for water quality for Category 3 (3A - irrigation and 3B - livestock watering) and Category 4 (aquatic habitat protection) water uses. In all locations water quality did not meet the Peruvian standards for direct use in irrigation, for livestock watering or for aquatic habitat preservation in at least one parameter.

The water quality sample from the RC-01 (Majes-1) drill hole in the Majes I irrigation area was compared against the Peruvian standards for water quality for Category 1 (human consumption – all three sub-categories) and Category 3 (3A - irrigation and 3B - livestock watering) water uses. The results of this sample indicate that the water from the Majes I aquifer is not suitable for use as irrigation water; water for livestock consumption, or for human consumption without treatment.

The limitations for water quality for Categories 1, 3 and 4 used in these analyses are not restrictive for the use of water in mining and processing. Testwork for metallurgical appropriateness of the identified water sources for the Project, including the Majes I aquifer, is discussed in Section 13.

The management of contact and non-contact water that comes off the Project facilities is discussed in Sections 16 and 18.

20.1.5 Air Quality

Due to the dry climatic conditions indicated in Section 5.4 there is a high potential for the generation of fugitive dust through aeolic erosion. Air quality sampling was carried out during the rainy season 2015 and a second round of sampling is projected for the dry season 2016 (June-August). Monitoring of air quality has also been carried out since 2011 as part of the environmental commitments for the exploration program, as specified in the exploration semi-detailed EIA (EIAsd). This monitoring program covers both the rainy and dry seasons.

All of the data collected from the analysis of PM_{10} are below the ECA of 150 µg/m^3. As part of the monitoring program for PM_{10}, wind speed and direction were recorded on an hourly basis. This has permitted the correlation of the dust concentrations with wind behaviour. The contaminant-wind roses indicate that the majority of the PM_{10} dust comes from the southwest, which would be in the direction of the tailings management facility. This indicates that an unwetted tailings surface could potentially become a dust control issue for the Zafranal Project.

The $PM_{2.5}$ component of the dust was below the ECA of 25 µg/m^3 in most of the samples, with notable exceptions in the P-01, P-02 and P-04 areas during the rainy season air quality monitoring for exploration activities (mean values 2012-2014). This is due to the fine dust content in the wind-transportable soils in the areas of the open pit, Sicera and tailings management facility. While Sicera will not form part of the Project Area, control of fine dust transport in the open pit and tailings management facility should be considered in the design of the Project operating procedures.

Arsenic and lead levels in the PM_{10} and $PM_{2.5}$ dust components were below the ECAs established for these parameters. All sampled gas concentrations were below the ECAs established for these parameters.

20.1.6 Flora and fauna

20.1.6.1 Terrestrial flora

In the Project area 174 species of terrestrial flora were identified during the rainy season sampling period 2015. The plants were associated with one or more of the eight vegetation types present in the Project area: Matorral, Matorral with Cacti, Streambed, Rocky Terrain, Bare Soil, Riparian Forest, Herbland and Agricultural Land. The study area includes a buffer zone around the proposed Project infrastructure locations.

Of the total number of species identified, 34 are considered either endemic or of conservation concern. The identification of endemic species was on the basis of their inclusion in the Red List of Endemic Plants of Peru (Libro Rojo de las Plantas Endémicas del Perú, León *et al*. 2006). The classification of plant species of conservation concern was based on their inclusion in the appendices of CITES, on the IUCN Red List of Endangered Species, or on the national list of threatened plants (as per S.D. 043-2006-AG). Table 20-2 provides a list of the 34 species and their characterization as endemic and/or of conservation concern.

Twelve species in the Project area are included in the IUCN Red List of Endangered Species (see Table 20-2). All of the cactus species identified in the Project area are included in the Appendix II of CITES. Eleven species are included in the national list of threatened plants, with three species Critically Endangered and one species Endangered (see Table 20-2). A total of 20 species are considered endemic, although with varying degrees of endemism within the Project area.

20.1.6.2 Sensitive vegetation

Sensitivity of the vegetation has been defined in this report as the number of endemic or species of conservation concern reported from each of the vegetation formations. With this in mind, the Matorral with Cacti and Matorral formations are the most sensitive, with 22 and 21 species of conservation concern or endemics registered. The least sensitive formation is the Rocky Terrain (5 species), with the remaining formations presenting between 6 and 9 species of conservation concern or endemics.

Table 20-2 – Species of endemic terrestrial flora and species of conservation concern

N°	Family	Scientific name	IUCN [1]	CITES [2]	S.D. Nº 043-2006 [3]	Endemic species distribution
1	Asteraceae	*Senecio tovarii*	--	--	--	AR, AY, LI
2	Asteraceae	*Senecio yurensis*	--	--	CR	--
3	Bromeliaceae	*Puya cylindrica*	--	--	--	AR, CU
4	Cactaceae	*Armatocereus riomajensis*	LC	II	--	AR
5	Cactaceae	*Austrocylindropuntia subulata*	LC	II	--	--
6	Cactaceae	*Browningia candelaris*	DD	II	VU	--
7	Cactaceae	*Corryocactus brevistylus*	LC	II	VU	--
8	Cactaceae	*Cumulopuntia sphaerica*	LC	II	VU	--
9	Cactaceae	*Echinopsis* cf. *peruviana*	--	II	--	--
10	Cactaceae	*Haageocereus platinospinus*	LC	II	--	AR, TA
11	Cactaceae	*Neoraimondia arequipensis*	LC	II	--	AR, IC, LI
12	Cactaceae	*Opuntia ficus-indica*	DD	II	--	--
13	Cactaceae	*Weberbauerocereus weberbaueri*	LC	II	--	--
14	Caricaceae	*Vasconcellea candicans*	--	--	CR	--
15	Caryophyllaceae	*Drymaria fasciculata*	--	--	--	AN, CU, LI
16	Ephedraceae	*Ephedra americana*	LC	--	NT	--
17	Euphorbiaceae	*Euphorbia apurimacensis*	VU	--	CR	AP, CA
18	Fabaceae	*Acacia macracantha*	--	--	NT	--
19	Fabaceae	*Dalea moquehuana*	--	--	--	AY, MO
20	Krameriaceae	*Krameria lappacea*	--	--	EN	--
21	Loasaceae	*Caiophora cirsiifolia*	--	--	--	AN, AP, AR, AY, CA, HV, LI, MO, TA
22	Loasaceae	*Prelisophytum incanum*	--	--	--	AN, AR, AY, CA, HV, IC, LL, LI, MO, PI
23	Malvaceae	*Gaya mollendoensis*	--	--	--	AR
24	Montiaceae	*Cistanthe paniculata*	--	--	--	AN, AR, LI, LL, TA
25	Piperaceae	*Peperomia peruviana*	--	--	--	CU
26	Plantaginaceae	*Plantago limensis*	--	--	--	AR, HU, JU, LI, LL, MO, TA
27	Pteridaceae	*Cheilanthes fractifera*	--	--	--	AR, AY, CA, LI
28	Pteridaceae	*Cheilanthes peruviana*	--	--	--	AM, AN, AR, CA, LL, LA, LI, MO
29	Rosaceae	*Kageneckia lanceolata*	VU	--	CR	--
30	Solanaceae	*Jaltomata diversa*	--	--	--	AP, AR, AY, CU
31	Solanaceae	*Nicotiana paniculata*	--	--	--	AN, AR, AY, CA, IC, JU, LA, LL, LI, MO
32	Verbenaceae	*Aloysia spathulata*	--	--	--	AR, CU
33	Verbenaceae	*Mulguraea arequipensis*	--	--	VU	--
34	Verbenaceae	*Lantana scabiosiflora*	--	--	--	LL, LI, PI, TU

[1] IUCN - International Union for Conservation of Nature

Critically Endangered (CR), Endangered (EN), Vulnerable (VU), Near Threatened (NT), Least Concern (LC)

Source: Knight Piésold, 2015

[2] CITES - Convention on International Trade in Endangered Species

Appendix I = Endangered Species; CITES prohibits the sale or trade of these species

Appendix II = Species that are not necessarily endangered but could become so if trade is not controlled

[3] S.D. Nº 043-2006-AG: Critically Endangered (CR), Endangered (EN), Vulnerable (VU), Near Threatened (NT)

Source : Knight Piésold, 2015

Departments:

AR:	Arequipa	CU	: Cusco	LI:	Lima
AM	: Amazonas	HU	: Huánuco	LL	: La Libertad
AN	: Ancash	HV	: Huancavelica	MO	: Moquegua
AP	: Apurímac	IC	: Ica	PI	: Piura
AY:	Ayacucho	JU	: Junín	TA	: Tacna
CA:	Cajamarca	LA	: Lambayeque	TU	: Tumbes

20.1.6.3 Terrestrial fauna

In the Project area 5 species of mammals, 48 species of birds, 4 species of reptiles and 1 species of amphibian were recorded for the rainy season 2015. Additional work will be carried out for the dry season to cover any seasonal variability in the species present.

Of these, 3 species of mammals, 11 species of birds, 2 species of reptile and 1 species of amphibian are considered of conservation concern by CITES, the IUCN, or the national list of threatened fauna (as per (S.D. Nº 004-2014-MINAG). Table 20-3 provides a list of the 17 species and their classification of conservation concern.

20.1.6.4 Sensitive terrestrial fauna

The four species of hummingbird in the Project area and one species of bat play an important role in the propagation of vegetation, with some species of cacti dependent exclusively on these species for pollination.

Guanacos are large South American camelids. There are an estimated 3,500 individuals left in the Peruvian population of these animals (Baldi *et al*., 2008). Local residents in Lluta and Huancarqui have indicated that the population of guanacos in the Project area has decreased in recent years. Studies at the Cerro Verde mine have identified an altitudinal migration of guanacos (Knight Piésold, 2014). These animals are subject to poaching pressure.

20.1.6.5 Aquatic flora and fauna

Rainy season baseline data collection has been undertaken for aquatic flora (phytoplankton and periphyton) as well as aquatic fauna (benthic macro-invertebrates, zooplankton and fish species). There is very little permanent surface water in the Project area, therefore the majority of this study focussed on the Majes River.

The most important component of aquatic flora and fauna in the area is the commercially extracted river shrimp population in the Majes River. An ecological base flow evaluation of the Majes River water levels is currently in preparation by Ausenco with a focus on this species. This will permit a more complete evaluation of any potential risks to the commercial fishery by the Zafranal Project.

Table 20-3 –Species of fauna of conservation concern

Family	Scientific name	Local common name	S.D. Nº 004-2014-MINAG[4]	IUCN[5]	CITES[6]	Endemism[7]	EBAs	Migratory
Mammals								
Phyllostomidae	*Platalina genvensium*	Murciélago longirostro peruano	EN	NT				
Perissodactyla	*Lama guanicoe*	Guanaco, huanaco	CR	LC	II			
Canidae	*Lycalopex culpaeus*	Zorro colorado		LC	II			
Birds								
Accipitridae	*Geranoaetus melanoleucus*	Aguilucho de Pecho Negro			II			
Strigidae	*Bubo virginianus*	Búho Americano			II			
Strigidae	*Glaucidium peruanum*	Lechucita Peruana			II			
Strigidae	*Athene cunicularia*	Lechuza Terrestre			II			
Trochilidae	*Patagona gigas*	Colibrí Gigante			II			
Trochilidae	*Myrtis fanny*	Estrellita de Collar Púrpura			II			
Trochilidae	*Rhodopis vesper*	Colibrí de Oasis			II			
Trochilidae	*Thaumastura cora*	Colibrí de Cora			II			
Falconidae	*Falco sparverius*	Cernícalo Americano			II			
Falconidae	*Falco femoralis*	Halcón Aplomado			II			
Falconidae	*Falco peregrinus*	Halcón Peregrino	NT		II			BM
Reptiles								
Tropiduridae	*Microlophus* cf. *tigris*		NT*	-	-	Nacional[8]		
Colubridae	*Pseudalsophis elegans*		-	LC	-	-		
Amphibians								
Bufonidae	*Rhinella spinulosa*	Sapo espinoso	-	LC	-	-		

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[4] IUCN - International Union for Conservation of Nature
Critically Endangered (CR), Endangered (EN), Vulnerable (VU), Near Threatened (NT), Least Concern (LC)

[5] CITES - Convention on International Trade in Endangered Species
Appendix I = Endangered Species; CITES prohibits the sale or trade of these species
Appendix II = Species that are not necessarily endangered but could become so if trade is not controlled

[6] S.D. Nº 043-2006-AG: Critically Endangered (CR), Endangered (EN), Vulnerable (VU), Near Threatened (NT)

[7] Aves de Perú (Schulenberg *et al.*, 2010)

[8] Carrillo e Icochea, 1995
BM = Boreal Migrant

20.1.7 Archaeology

Several studies have been carried out since the start of exploration activities to determine the presence of archaeological remains within the expanding exploration areas. Additional studies are underway to determine the presence of any additional archaeological remains within the definitive Project area and obtain official government clearance, through the Ministry of Culture (MINCULT), for all areas that are free of archaeological remains. This official clearance, denominated Certificate for the Inexistence of Archaeological Remains (or CIRA using the Spanish acronym) has been obtained in two separate processes, one for the TMF and one for the majority of the rest of the Project area. The CIRA areas are shown as green polygons in Figure 20-4. The remainder of the Project area that currently does not have a CIRA (outside the green polygons) will require additional archaeological studies (Archaeological Exploration Project or PEA for the Spanish acronym). These are scheduled for inclusion in the Feasibility Stage.

It is important to note that near the current access road to the Project, on the public road from El Pedregal de Majes to Lluta, there is an archaeological complex called Huacán. This site will not be affected by the Project development but is a sensitive area for the local residents. It is currently expected that salvage works will be required for the archaeological remains in the Victoria pit and east access road areas, as well as on the pre-Hispanic road that will be crossed by the water supply pipeline from the Majes I irrigation area. The extent of these salvage works is currently under study.

Where Project installations will affect any archaeological remains, these must be salvaged by a registered archaeologist, through an Archaeological Salvage Plan. This plan must be approved by the MINCULT before undertaking the salvage. All salvaged archaeological remains must be documented and submitted to the MINCULT with a report on the findings. Additionally, an archaeologist must be on site during all earthworks in the event archaeological “chance finds” are encountered. Any “chance finds” must be documented, salvaged and then submitted to the MINCULT with a report on the findings.

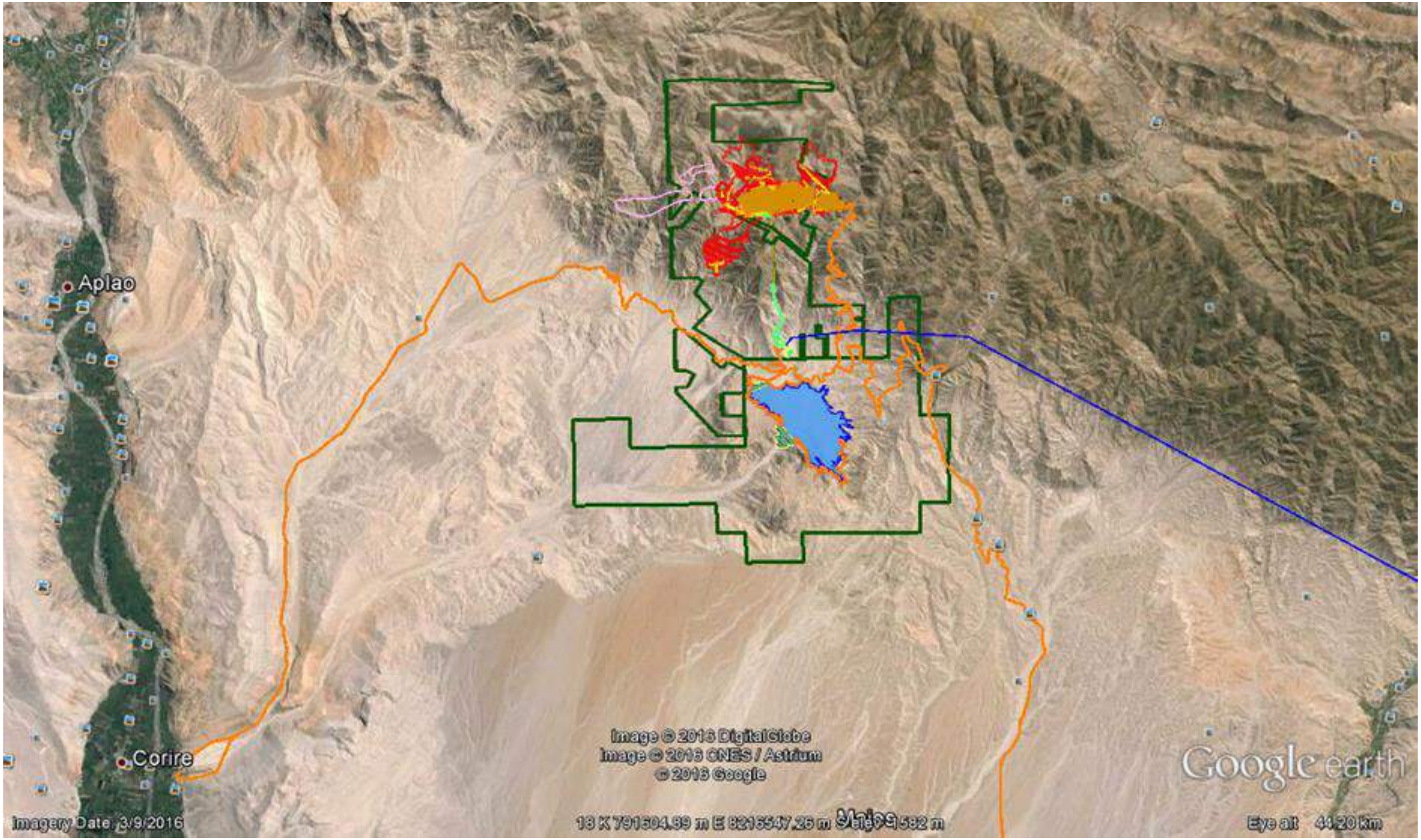

Figure 20-4 – CIRA areas (within green polygons)

20.1.8 Environmental impacts

Environmental impacts will be evaluated once the baseline data collection and project description have been completed for the detailed EIA.

20.2 Social Baseline and Impact Assessment

The social baseline data collection was carried out in the communities of the direct area of influence of the Project (Huancarqui, Lluta, Anexo El Pedregal de Uraca-Corire) in March 2015. Data collection will be carried out for the off-site infrastructure in conjunction with the rainy season environmental baseline data collection period in February-March 2016 including El Pedregal de Majes; which, as the potential water source is the brackish aquifer in the Majes I irrigation area, would also form part of the direct area of influence of the Project. Additional work will be done to identify and engage landholders along the alignment for the power line. These people form part of the indirect area of influence of the Project. Figure 20-5 presents the direct and indirect areas of influence of the Project. The communities included in each area of influence (direct and indirect) are presented in Table 20-4.

Table 20-4 – Communities in the direct and indirect areas of influence of the Zafranal Project

Influence	Province	District	Municipality
Direct	Caylloma	Lluta	Lluta Population Centre
		Majes	City of El Pedregal de Majes
	Castilla	Huancarqui	Town of Huancarqui
		Uraca	Anexo El Pedregal (Corire Population Centre)
Indirect	Caylloma	Lluta	Taya Population Centre
		Huambo	District level
	Castilla	Aplao	District level
	Camaná	Camaná	City of Camaná
	Arequipa	Santa Isabel de Siguas	Power line alignment
		Vitor	Power line alignment
		Yura	Power line alignment
		Uchumayo	Power line alignment
		Tiabaya	Power line alignment
		Jacobo Hunter	Power line alignment
		Socabaya	Power line alignment
		Yarabamba	Power line alignment

Source: CMZ

The contract for the social impact assessment for the Project has not yet been awarded. Preparation of the social components of the EIA could commence upon completion of the baseline data collection. The social baseline has been prepared using primary information collected in the communities using a variety of tools aimed at optimum data collection. Secondary information from public entities has also been used in the preparation of the baseline report.

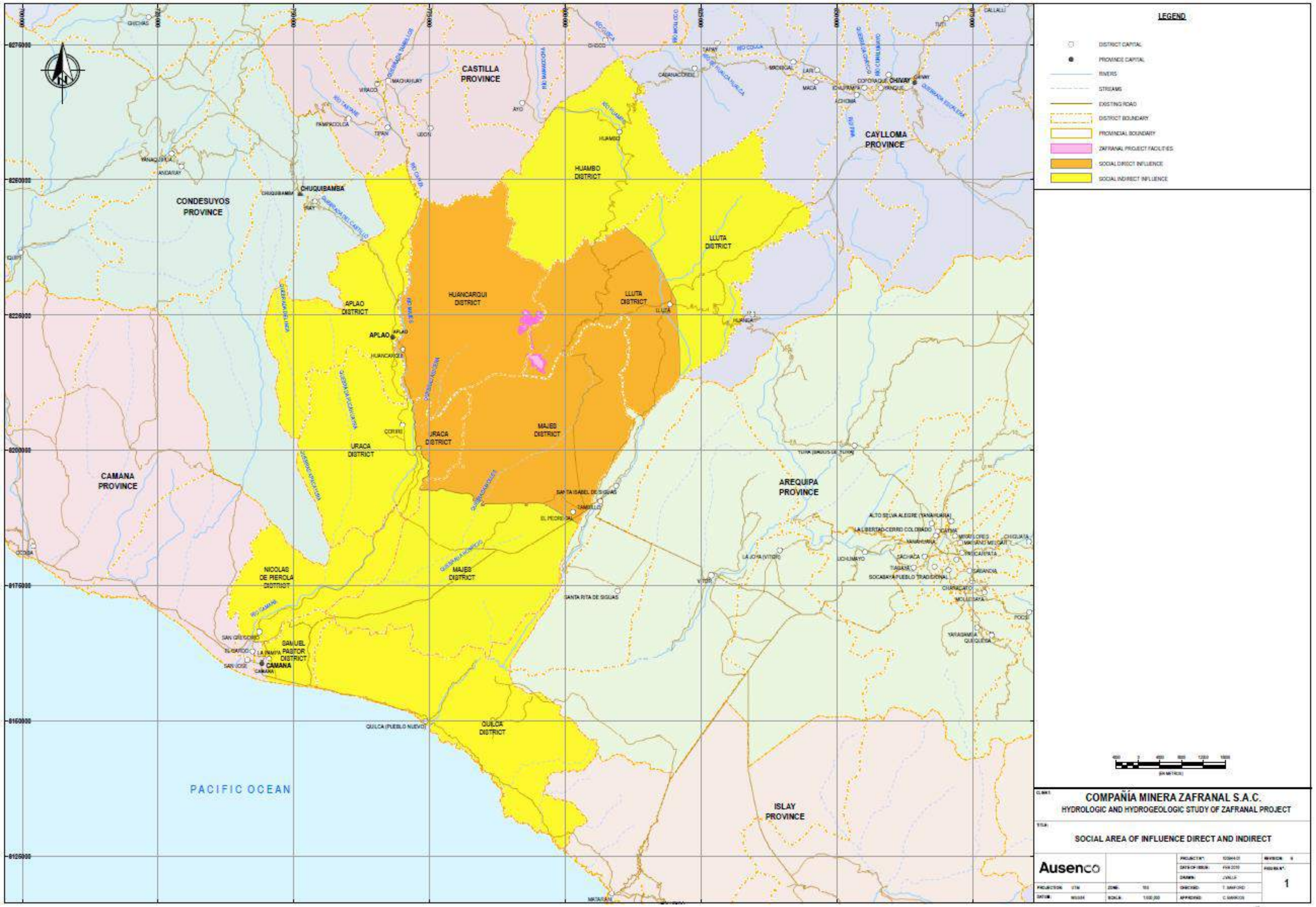

Figure 20-5 – Direct and indirect areas of influence of the Project

20.2.1 Population demographics

The population demographics in the direct area influence of the Zafranal Project are based on the results of the official population census from 2007 (INEI, 2007). In the District of Huancarqui the total population was 1,445 people with a slight dominance by women (52%). The population shows little variation in the distribution of people over the age groups, indicating an aging population. The change in population dynamics between the 1993 and 2007 censuses show significant migration of the younger population out of the District of Huancarqui; this has also been observed in the results of the questionnaires. The large majority of the population is urban (82.3%).

The total population of the District of Lluta, as per the 2007 census (INEI, 2007), was 1,417 people with a slight dominance by women (54.5%). The population also shows little variation in the distribution of people over the age groups, indicating an aging population. The change in population dynamics between the 1993 and 2007 censuses shows significant migration of the younger population out of the District of Lluta. The majority of the population is rural (64.3%).

The total population of the District of Majes, as per the 2007 census (INEI, 2007), was 39,445 people with a slight dominance by women (51.4%). The population shows a standard structure and distribution of the population over the age groups. Information was not available from the 1993 census to show changes with time. The majority of the population is urban (61.5%).

The total population of the District of Uraca, as per the 2007 census (INEI, 2007), was 7,182 people with a slight dominance by women (53.8%). The population shows growth in the age groups between 35 and 69 years but a reduction in the younger population. The majority of the population is rural (66.4%).

Migration of the younger population out of the Lluta, Huancarqui and Uraca districts is predominantly for educational purposes (Metis Gaia, 2015). Migration of the working population into these districts from outside areas is generally related to a requirement for personnel to work the agricultural lands. These people replace the younger population, which is less motivated to work in agriculture, preferring a better education for other technical or professional opportunities.

20.2.2 Public infrastructure and services

The supply of water to most houses in the direct area of influence is through a public system and is direct to tap in house; the notable exception is in El Pedregal de Majes where drinking water supply is an issue identified by the municipal and regional authorities. However, it must be noted that the distribution of water in house by a public system does not guarantee a potable quality of water.

The majority of houses in the direct area of influence are not connected to a public sewerage system, although this is currently being improved in Lluta by the Municipality of Lluta. The lack of sewerage systems and/or adequate sewage treatment is a potential issue for public and environmental health.

In the larger population areas (Districts of Uraca and Majes) there is a broad availability of educational services and infrastructure. In the smaller populations (Districts of Huancarqui and Lluta) the services are significantly less diverse and the infrastructure is limited. The level of interaction between the teachers and students is also dependent on the availability of services. In the larger populations more of the schools have independent classrooms per grade (single grade) with teachers by discipline (multiple teachers) whereas in the smaller populations the schools tend to be multigrade classrooms with one teacher for each classroom. This makes the workload greater for the teachers and reduces the one-on-one interaction between the teachers and the students.

There is limited government health infrastructure and services in the Project direct area of influence. The number of beds per 1,000 persons in the majority of the Project area of influence is below the national average of 1.7 beds per 1,000 population and the regional average for Arequipa of 1.9. The exception to this is Uraca, which has 2.9 beds per 1,000 persons.

20.2.3 Economic activities

The population in the direct area of influence of the Project maintains relatively high levels of employment activity and there are low levels of unemployment. Agriculture is the primary economic activity in all of the districts within the direct area of influence. There is very little formal mining activity in the economically active population.

20.2.4 Interest groups

The primary interest groups in the Project area of direct influence are identified through stakeholder mapping, which is updated on a monthly basis. The majority of the interest groups are aligned with the development of the Project at this time. Measures to engage all stakeholders have been implemented. The measures undertaken to maintain or improve relations with all stakeholders are included in the Community Relations Plan (see Section 20.4).

20.3 Environmental Management Plan

The contract for the environmental management plan for the detailed EIA has not yet been awarded. This will form an integral part of the EIA report. The specifics of the EMP for the detailed EIA will depend on the impacts identified and evaluated for the construction, operations, and closure phases. However, the general content of the EMP will comply with the requirements for the Environmental Management Strategy indicated in the Common Terms of Reference for Environmental Impact Assessments for large-scale mining in Peru, indicated below:

Environmental Management Strategy

1. Environmental Management Plan
    a) Air quality
    b) Noise and vibrations
    c) Soils
    d) Surface water
    e) Groundwater
    f) Process and domestic effluents
    g) Biological components
    h) Other components as required
2. Environmental Vigilance
    a) Environmental Monitoring Plan
        i. Effluents
        ii. Emissions
        iii. Surface water quality
        iv. Groundwater quality
        v. Air quality

vi. Vibrations

vii. Soils

viii. Biological monitoring

3. Waste Management Plan
4. Environmental Compensation Plan (this is not currently expected to be required)
5. Social Management Plan (see Section 20.4)
6. Contingency Plan
7. Conceptual Closure Plan (see Section 20.5)
8. Schedule and budget for implementation of the Environmental Management Strategy
9. Summary table of environmental commitments.

20.4 Social Infrastructure and Community Relations Plan

The community relations plan is currently in progress. This consists of a series of mechanisms for informing the public in the communities of the direct area of influence regarding the Project activities and promoting sustainable development in the area of direct influence. The community relations plan is a living document and its implementation is guided by a complete and current stakeholder mapping process and a monthly evaluation of social risks. The implementation of the community relations plan includes the following components as a part of a social management system:

1. Social Investment Program
2. Local Hiring Program
3. Local Supply Program (services and goods)
4. Grievance Resolution Program
5. Attention Request Program
6. Commitments Monitoring Program
7. Communication Program.

In support of the community relations plan, CMZ maintains community relations offices in Huancarqui, Lluta and Anexo El Pedregal. CMZ has staff full-time in these offices. There are full-time staff located in El Pedregal de Majes and also staff or consultants that assist with community relations as needed in Camaná.

Additional engagement with the Huancarqui, Lluta and Uraca-Corire Districts has been developed through the implementation of a Working Table. This Working Table is conformed of representatives of the Municipality of Huancarqui, the Huancarqui Defense Front (as a representative of the Huancarqui society), the Municipality of Lluta, representatives of the Lluta society, the Municipality of Uraca-Corire and the Anexo El Pedregal Defense Front (as a representative of the Anexo El Pedregal society). The primary objective of the Working Table is to construct the foundation upon which a Framework Agreement for each District can be developed. The Working Table is advised by Technical Working Groups for environmental and sustainable development issues. The technical working groups as well as the working table are moderated by a representative of the General Office of Social Management (OGGS) from the Ministry of Energy and Mines; the Office of Conflict Avoidance of the Regional Government of Arequipa acts as Technical Secretary. The incorporation of Pedregal de Majes as an additional member of the existing Working Table or in a separate Working Table is projected as El Pedregal de Majes is now being considered as part of the direct influence area of the Project.

Where other opportunities arise to aid in the development of the local communities, these are looked for, evaluated and implemented in accordance with Project needs and available budget.

The Environmental Impact Assessment will also be accompanied by a Community Relations Plan for the development stage of the Zafranal Project. This document is submitted and approved by SENACE prior to the submission of the EIA. Preparation of this document will follow the requirements established in the Supreme Decree Nº 028-2008-EM and its regulation (Ministerial Resolution Nº 304-2008-MEM/DM).

As part of the Environmental Management Strategy included in the detailed EIA for the Zafranal Project, the measures for social management will be specified. These will follow the Common Terms of Reference for Environmental Impact Assessments for large-scale mining projects in Peru, which include the management of social impacts, as follows:

Social Management Plan

1. Community Relations Plan
    a) Communications Plan
    b) Social Relations Protocol
    c) Worker Code of Conduct
2. Public Participation Plan
3. Social Mitigation Plan
    a) Mitigation of negative social impacts
    b) Compensation Program (if applicable)
    c) Social Contingency Plan
    d) Resettlement Plan (if applicable)
4. Community Development Plan
    a) Local Employment Program
    b) Local Economic Development Program
    c) Local Training Program
    d) Social Investment Program.

20.5 Mine Closure Plan

Mine closure planning is covered at various stages of a mining project development in Peru. The first stage is during the approval process for the exploration program and details the progressive and final closure objectives and activities for the exploration program. The Zafranal Project is currently at this stage of mine closure planning. Progressive closure is under way.

The second stage of mine closure planning is the conceptual mine closure plan in the detailed EIA for the mining project. This stage has not yet been initiated and will form an integral part of the scope for the EIA.

The criteria for closure, as established in the mine closure law for Peru (S.D. Nº 033-2005-EM and modifications S.D. Nº 035-2006-EM and S.D. Nº 045-2006-EM), are to ensure short, medium and long term physical and chemical stability for all closed areas and installations. Where the possibility exists for remediating areas to a land use capability commensurate with the land use prior to the Project, this will be done.

The third stage of mine closure planning is the definitive mine closure plan, which must be submitted to the Ministry of Energy and Mines within one year of obtaining approval of the EIA for the Project, and is at feasibility study level. The definitive closure plan must include scenarios for temporary closure, progressive closure and final closure. The temporary closure scenario provides for a shut-down of greater than 6 months; the progressive closure scenario provides for closure of areas or infrastructure that are no longer required for mine operations; the final closure scenario is for end of mine life. This definitive mine closure plan must be first updated after three years following its approval and from thereafter every five years. In the definitive mine closure plan the calculation for closure costs and the mechanism for establishing a closure bond must be included.

Specific considerations for closure of individual facilities are presented in Sections 16 and 18.

21 Capital and Operating Costs

21.1 Capital Costs

The capital cost of the project over a three-year construction and pre-mining period, and sustaining expenditures over the mine life, have been estimated primarily by Ausenco, with NCL for mining, CMZ for fuel storage (by supplier) and Owner's costs, and GMI for power supply.

21.1.1 Basis of estimate

The following basic data pertains to the estimate

- The estimate base date is 1st quarter 2016.
- The estimate is expressed in United States dollars (USD or US$).
- The estimate is Ausenco Class 4 (based on AACEI Class 4) at PFS precision ±20-25%.
- No allowance has been made in the estimate for escalation from the base date or changes in currency exchange rates.
- All import duties and taxes are excluded from the estimate (not expected to apply).
- Each element of the estimate is developed initially as a bare cost only. A growth allowance has then been allocated to each element of the costs to reflect the level of definition in pricing and engineering maturity relating to that element.
- Estimate contingency is included to address anticipated variances between the specific items contained in the estimate and the final actual project cost assessed as a percentage of direct and indirect costs to arrive at a total project estimate with the required confidence level.
- The following items were not considered in the capital cost estimate:
    - working capital
    - replacement capital other than in sustaining capital
    - finance charges
    - residual value of temporary equipment and facilities
    - residual value of any redundant equipment
    - cost of any downtime
    - environmental approvals
    - legal costs
    - any further studies
    - force majeure issues
    - future scope changes
    - special incentives (schedule, safety or others)
    - allowance for loss of productivity and/or disruption due to religious, social and/or cultural activities
    - risk management reserve (Owner's contingency).

21.1.2 Exchange rates

Table 21-1 shows the long-term exchange rates used to develop the capital cost estimate.

Table 21-1 – Exchange rates

USD	Exchange rate
USD 1	AUD 1.09
USD 1	USD 1.00
USD 1	EUR 0.74
USD 1	CAD 1.15
USD 1	PEN 2.90

21.1.3 Work breakdown structure (WBS)

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

21.2 Capital Cost Estimate Summary

The capital cost estimate has been summarized at the levels indicated in Table 21-2 and Table 21-3.

Table 21-2 – WBS level one cost summary

WBS LVL 1	LVL 1 Description	Total US$M	Total Labour ,000 hours
Initial capital		**1,157**	**14,048**
	Initial direct cost	**703**	**13,998**
1000	Mine	142	168
2000	Concentrator	430	12,016
4000	On-site infrastructure utilities and facilities	33	362
5000	Off-site infrastructure utilities and facilities	98	1,452
	Initial indirect cost	**454**	**50**
6000	Project preliminaries	103	0
7000	Indirect costs (EPCM, vendors, commissioning, spares, etc.)	132	50
8000	Provisions	151	0
9000	Owner's costs	68	0
Sustaining capital		**263**	**345**
	Sustaining direct cost	**228**	**345**
1000	Mine	213	69
2000	Concentrator	15	267
4000	On-site infrastructure utilities and facilities	1	9
	Sustaining indirect cost	**34**	**0**
8000	Provisions	34	0
Grand Total		1,420	**14,393**

Table 21-3 – Major facility cost summary

WBS LVL 1	LVL 1	LVL 2 Description	Total
Initial capital			1,157
1000	Mine		142
	1100	Mine	113
	1200	Haul and access roads	8
	1300	Mine support utilities	2
	1400	Mine support facilities	3
	1700	Waste dumps and mineral stockpiles	16
2000	Concentrator		430
	2100	Primary crushing and crushed ore handling	81
	2200	Concentrator	229
	2300	Concentrator utilities	21
	2400	Concentrator facilities	3
	2500	Tailings management facility	92
	2900	Concentrator site earthworks and drainage	4
4000	On-site infrastructure utilities and facilities		33
	4100	On-site water storage, treatment, and distribution	1
	4300	On-site general utilities	6
	4400	On-site general facilities	1
	4500	On-site accommodation camp	1
	4600	On-site roads	3
	4700	HV switchyard and on-site power distribution	14
	4900	Mobile vehicle fleet (operations other than mining)	7
5000	Off-site infrastructure utilities and facilities		98
	5100	Water supply	42
	5200	Power supply	34
	5600	Access roads	22
6000	Project preliminaries		103
	6000	Project preliminaries	28
	6100	Mobilization / demobilization	3
	6200	Site establishment	62
	6300	Temporary utilities	9
	6400	Temporary facilities	1
7000	Indirect costs		132
	7300	EPCM	110
	7400	Vendor representatives	9
	7500	Commissioning / operations readiness	1
	7700	Spares and first fills	12
8000	Provisions		151
	8000	Provisions	151
9000	Owners costs		68
	9000	Owners costs	68
Sustaining capital			263
1000	Mine		213
	1200	Haul and access roads	5
	1400	Mine support facilities	3
	1500	Mining fleet	205
2000	Concentrator		15
	2500	Tailings management facility	15
4000	On-site infrastructure utilities and facilities		1
	4400	On-site general facilities	0
	4600	On-site roads	0
8000	Provisions		34
	8000	Provisions	34
Grand Total			1,420

21.3 Capital Cost Estimate Development

A detailed explanation of how each discipline has been estimated is contained in the individual discipline sections that follow in this document.

21.3.1 Definition of costs

The direct costs are those costs that pertain to the permanent equipment, materials and labour associated with the physical construction of the facilities and utilities.

The indirect costs include all costs associated with development of the project and incurred by the owner, engineer or consultants in the engineering design, procurement, construction, and commissioning of the project.

Construction contractor's indirect costs are either included explicitly in the direct costs or are in the rates.

21.3.2 Market availability

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

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

21.3.3 Estimate methodology

The estimate was derived using the methodologies set out in Ausenco Class 4 (AACEI Class 4) preliminary feasibility study standards.

The general methodology applied to develop the estimate is as follows:

- A defined scope of work was prepared
- The work was quantified in accordance with standard commodities
- The estimate structure is in accordance with an agreed WBS
- Labour wage rates are based on contractor received data and data from similar current projects in the same region
- Contractor distributable costs were calculated and comparisons made with received pricing for construction work
- Installation cost for equipment and commodities were estimated based on contractor evaluated data, current projects or Ausenco historical data
- Major and some minor equipment pricing were based on detailed budget quotations
- Pricing for other minor items were based on recent or current projects and escalated to the base date of this estimate
- Allowances for freight were included
- Import taxes and duties have not been included.

- Foreign exchange content and exchange rates for all imported materials, equipment and services were converted to USD from the native currency based on CMZ's advised currency rates.

21.4 Direct costs

Direct costs are generally quantity based and will include all permanent equipment, materials associated with the physical construction of the facilities.

The structure of the estimate is a build-up of the direct cost of the quantities; this includes the installation/construction hours, unit labour rates and contractor distributable costs, bulk and miscellaneous material and equipment costs, subcontractor costs and factored allowances for various scopes, freight and growth.

The following subsections describe the cost development of the various components of the estimate.

21.4.1 Mining and mining fleet

The mining scope of works has been compiled by NCL and reviewed and issued to Ausenco by CMZ. The total estimated mining capital costs are summarized in Table 21-4. The estimate reflects that:

- The mine pre-production period is from Project approval to the start-up of processing and metals production.
- The initial capital for the mine pre-production period includes all road construction, waste pre-stripping, ore and other mineral stockpiling and mine support facilities and utilities.
- The estimate considers the use of a mining contractor and was based on quotes obtained by NCL during second-quarter 2015. This value also includes the Owners cost of supervising the contractor operation and some minor items not included in the contractors quotes.
- Sustaining capital covers the period from the start of commercial production to the end of the life of mine.
- Mine sustaining capital expenditures include but are not limited to the equipment fleet to achieve the required material movement during the owner-mining period, from Production Year 4, although most of this amount is during Production Years 3 and 4.

The initial capital costs contained within the estimate includes all costs required up to the start of initial production from the open pit mine, including pre-stripping, preproduction mining and owner's costs. The Initial phase of the mining work will be performed by a contract miner supplying, all equipment utilities facilities and camp costs. Owners mining equipment capital will be procured after construction and has been treated as sustaining capital and procurement will be in Year 3 onwards during operations; the pre-stripping performed by the mining contractor will start in Year -2.

Total bare cost for Owner´s mine equipment (purchase and rebuild) is US$170M over the mine life. The costs for the major equipment are based on vendor quotes obtained by NCL during second-quarter 2015. The costs for some minor equipment are based on quotations obtained by NCL during second-quarter 2015 and others from NCL's database. Estimated freight costs for shipping the mine equipment to site and all costs to prepare the equipment for work are included in the estimates.

NCL has scheduled the purchase of initial mining equipment six months before it is needed, so as to maintain production continuity. The purchase and replacement schedule for the equipment is based on the estimated life of the equipment in operating hours and the number of operating hours that the equipment is scheduled for each production year during the mine life. Supplier estimates of equipment replacement life were adjusted based on NCL experience.

New pieces of equipment were considered up to the maximum requirement. Equipment lives were extended with rebuilds where required assuming a cost of 60% of a new unit.

NCL has included costs for spares for the shovel, FEL, mining trucks, drills, and auxiliary equipment. Costs for tires for the FEL, trucks, wheel dozers and grader have been included in sustaining costs.

Other costs, related to spare parts not included with the main equipment, minor equipment and a dispatch system are based on NCL's database.

Table 21-4 – Mine capital costs

		Initial capital			Sustaining capital	Total capital
		Y-2	Y-1	Total		
Total capital	**US$M**	**27**	**81**	**108**	**184**	**292**
Equipment purchase	US$M				156	156
Equipment rebuild	US$M				14	14
Other Investments	US$M	3		3	8	11
Dispatch	US$M	1		1	2	3
Mining Haul Roads	US$M	7	0	7	4	11
Pre-stripping (owners cost)	US$M	1	2	2		2
Pre-stripping (contractor other costs)	US$M	7	6	13		13
Pre-stripping (contractor opex)	US$M	9	72	81		81

The final estimated initial capital expenditure for the mine is US$142M, the difference from the US$108M shown in Table 21-4 is accounted for by growth allowance; mine support facilities and utilities, and waste dump underdrain systems, all estimated by Ausenco.

The final estimated sustaining capital expenditure for the mine is US$213M, the difference from the US$184M shown in Table 21-4 is accounted for by growth in the estimate and refurbishment of mine support facilities when the mining contractor terminates, both estimated by Ausenco.

21.4.2 Earthworks and site preparation

This scope covers all the concentrator and off-site and on-site infrastructure for roads, pads, embankments, diversion channels, overland conveyor corridors and tunnels, quarry development. The main earthworks contractor will free issue material such as aggregates and sand to other project contractors.

The mining contractor, in addition to works in the mining area, will complete the internal access road from the concentrator wet areas to the mining areas and bulk earthworks for the primary crusher and the TMF with moisture conditioning and compaction by the main earthworks contractor. The crushed ore transfer tunnel will be developed by a tunnelling or underground mining specialist. The eastern and western site access roads will be completed by a specialist road contractor.

Material take-offs (MTOs) have been taken from the general arrangement (GA) drawings, design sketches and civil drawings. Earthwork quantities for the concentrator have been generated by overlaying an AutoCAD/12d model of the layout against a contour map of the site.

Rates were derived from contractor budget quotes compared with data sourced from previous projects or studies.

21.4.3 Concrete

This scope allows for all concrete work in the concentrator, mining support facilities and utilities, and on-site and off-site infrastructure. The concrete supplier will operate a batch plant on site and free issue concrete to the installation contractor.

MTOs were based on preliminary GA designs for major structures, and typical design drawings and sketches for some minor structures.

Pricing for supply and rates for installation were evaluated from quotes from contractors for current or recent enquires and benchmarked against internal historical data to derive all-in unit rates for each type of concrete structure.

21.4.4 Structural steel

This scope allows for all cladding, conveyor steelwork, flooring, structural sections and all associated items for the concentrator, mining support facilities and utilities, and on-site and off-site infrastructure. The structural steel supplier will free issue fabricated steel for assembly, erection and installation by the structural, mechanical and piping (SMP) installation contractor.

MTOs were based on typical designs for all structures. Structural steel quantities were prepared based on design calculations and drawings and factored quantities, with some minor development from sketches.

Pricing for supply and rates for installation were evaluated from quotes from contractors for current or recent enquires and benchmarked against internal historical data to derive all-in unit rates for each type of structural steel.

21.4.5 Architectural and non-process buildings

This scope allows for pre-fabricated modular, semi-portable and tent-style (tensioned fabric) buildings for the concentrator, mining support facilities and utilities, and on-site and off-site infrastructure. The mining support buildings will be initially provided by the mining contractor and the owner mining structures will be part of the sustaining capital.

Dimensions and general specifications of requirements were developed with inputs from CMZ, Ausenco and NCL. Concrete slab quantities based on square metre requirements for each building have been calculated with some preliminary development from drawings and sketches and from suppliers' drawings.

Quotes for supply, fabrication, and installation of each building based on a design and construct contract for the required buildings were received from multiple vendors and assessed on pricing and scope inclusions, and benchmarked against internal historical data.

21.4.6 Mechanical equipment

This scope allows for the supply and installation of all mechanical equipment for the concentrator, mining support facilities and utilities, and on-site and off-site infrastructure. The equipment will be free issued for assembly and installation by the SMP installation contractor.

Mechanical equipment was selected based on an overarching principle of using proven designs that Ausenco has used on similar projects. The quantities are primarily based on the mechanical equipment list.

Equipment pricing obtained was a combination of detailed budget quotations, budget prices from supplier's evaluation of the technical data, and recent historical data. Installation rates quoted by various contractors were evaluated and benchmarked against internal historical data to derive the various rates used in the capex estimate.

21.4.7 Mechanical bulks

This scope allows for supply and installation of all mechanical plate work and liners for the concentrator, mining support facilities and utilities, and on-site and off-site infrastructure. The mechanical plate work supplier will free issue fabricated plate work (predominantly lined) and liners for assembly and installation by the SMP installation contractor.

MTOs were based on on design calculations, previous similar designs, factored quantities, and some items were developed from drawings and sketches.

Pricing for supply and rates for installation were evaluated from quotes from contractors for current or recent enquires and benchmarked against internal historical data to derive all-in unit rates for each type of mechanical plate work and for metal and rubber lining.

21.4.8 Pipework

This scope allows for supply and installation of all pipework, fittings and supports for the concentrator, mining support utilities, and on-site and off-site infrastructure. The suppliers of piping and valves will free issue fabricated spools, bulk pipe, fittings and supports for assembly and installation on-site by the SMP installation contractor and off-site by a specialist pipeline contractor.

All of the piping supply and installation within the concentrator, mining support utilities and on-site infrastructure has been factored at a facility level based on previous projects utilizing in-house historical data, based on the supply and installation cost of mechanical equipment.

MTOs for the pipelines between the concentrator, mining support utilities and on-site infrastructure, including the tailings pipeline and distribution pipework to and return water pipeline system from the TMF, are based on the PFDs, GAs and topography. MTOs for the water well collection and water supply system are based on the pipeline routes, flow modelling and GAs.

Pricing for supply and rates for installation were evaluated from quotes from contractors for current or recent enquires and benchmarked against internal historical data to derive all-in unit rates for each type of pipe.

21.4.9 Electrical

This scope allows for supply and installation of all of all electrical equipment and electrical bulks for the concentrator, mining support facilities and utilities, and on-site and off-site infrastructure. The suppliers will free issue equipment, cable, tray, conduit and fittings for installation by the electrical, instrumentation and control (EIC) installation contractor or specialist overhead power line contractor as appropriate.

Electrical equipment was selected based on an overarching principle of using proven designs that Ausenco has used on similar projects. The quantities are primarily based on the high-voltage (HV) single-line diagram (SLD) and mechanical equipment list.

Equipment pricing obtained was a combination of detailed budget quotations, budget prices from supplier's evaluation of the technical data, and recent historical data. Installation rates quoted by various contractors were evaluated and benchmarked against internal historical data to derive the various rates used in the capex estimate.

All of the electrical bulks supply and installation within the concentrator, mining support utilities and on-site and off-site infrastructure has been factored by cost at a facility level from previous projects utilizing in-house historical data, based on the supply and installation cost of mechanical equipment.

MTOs for overhead power lines to and within the Majes I wellfield, were generated from the project, site, and water supply GAs and HV SLD. Pricing for supply and rates for installation were from in-house historical data.

GMI S.A. Ingenieros Consultores (GMi) completed a power supply study for the project, including development of a capital cost estimate. The estimate was reviewed and found to have been developed on a similar basis to the other electrical scope for the Project.

21.4.10 Mobile plant and equipment

The number and type of units required for the support mobile equipment has been evaluated between NCL, Ausenco and CMZ. Equipment costs, including price ex-factory, freight and, if required, assembly at site, have been based on information from recent quotations and historical data.

21.4.11 Contractor labour and distributable rates

Fully burdened craft-based wages were built up from recent project and studies data reflecting a project workforce using predominantly local labour and Peruvian nationals from surrounding regions. Bulk earthworks have been included in the estimate as subcontractor rates without a separate labour component.

The distributable costs for contractors were developed based on contractor data for all the major contract packages and benchmarked against historical data from within the region. The rates cover provision of construction equipment and other indirect scope required to support and deploy installation labour. Distributable costs were considered to be covered within the unit rates where a subcontract price has been provided by the contractor, e.g. for bulk earthworks. The contractor distributable costs do not include the cost of camp establishment and operation.

21.4.12 Site construction labour hours and productivity

Unit labour-hours were based on Ausenco's standard hours to install categorized equipment and bulk materials with adjustment made for productivity. Where labour hours are not available due to specific design and construct packages, sub-contractor rates are factored and labour hours were back-calculated from a percentage of the installed cost and the labour rates, or estimated from typical industry averages for the task.

The total labour hours were used to estimate construction manning schedules and construction camp size. The unit labour-hours were combined with the labour rates to estimate the total labour costs.

The work schedule for the construction contractors is based on 7 days, 10 hours per day for 6 days and 5 hours on the 7th day for a 65-hour working week. A 3 weeks on and 1 week off rotational roster has been adopted.

The work schedule of the construction contractors' supervision and the EPCM contractor's personnel is based on the same 65-hour working week and the same 3 weeks on and 1 week off rotational roster.

Productivity factors used in the estimate are typical of Peruvian projects of this location and type. The determined productivity factors have been benchmarked against projects in Ausenco's database.

21.4.13 Freight

Freight costs are deemed to include inland transportation, export packing, all forwarder costs, ocean freight, air freight, insurance, receiving port custom agent fees, local inland freight to the project site. They exclude taxes and custom duties.

The freight costs have been determined by applying a factor to the applicable items' direct cost and then including it as a separate value on each line item. The factors used have been determined from region-specific freight costs and in-house historical data.

21.5 Indirect Capital Costs

Indirect costs include items that are necessary for project completion, but are not related to the direct construction cost.

21.5.1 Project preliminaries

Project preliminaries are items or services which are not directly attributable to the construction of specific physical facilities but are required to be provided as support during the construction period.

The allocated costs for project preliminaries are a combination of costed major items such as the camp accommodation, operational costs, and heavy lift equipment and an additional allowance for common on-site project requirements for the Owner and EPCM contractor. The overall project preliminaries cost was derived from benchmark analysis of similar type projects within the region.

Project preliminaries include but are not limited to:

- Temporary site works and roads
- Temporary construction facilities
- Temporary construction utilities
- EPCM construction equipment and vehicles
- Temporary construction services to support operation and maintenance of the shared construction facilities, utilities, and equipment and to support the workforce
- Construction camp purchase and operating cost

21.5.2 Engineering, procurement and construction management (EPCM)

EPCM costs cover time charges of office and site staff, office facilities, IT support, staff transfer expenses, sub-consultants, field inspection and expediting, business travel, corporate overhead and fees.

The overall engineering, procurement, and construction management budget has been factored from direct costs based on the various scope types. This has been benchmarked against similar projects and in-house historical data and is within the expected range.

21.5.3 Vendor site supervision

The estimated costs for vendor representation on site for supervision are factored from permanent mechanical and electrical equipment costs.

21.5.4 Capital spare parts

The costs for capital spare parts for commissioning and critical or insurance spares are factored from permanent mechanical and electrical equipment costs

21.5.5 First fills

The costs for first fills of fuels, lubricants and chemicals, etc. for installed equipment and process reagents are factored from permanent mechanical and electrical equipment costs.

21.5.6 Pre-commissioning and commissioning

Most costs for pre-commissioning and commissioning are included in EPCM, vendor site supervision and Owner’s costs. An allowance has been included for a modification squad to assist the commissioning team to make minor modifications.

21.5.7 Third party engineering, testing and inspection services

Costs for specialist engineering; site surveying; testing, inspection and quality assurance for materials and fabrication; and expediting are included in EPCM and Owner’s costs.

21.5.8 Owner's costs

Owner’s costs are included in the capital cost estimate to cover the Owner’s incurred costs for the period starting from the project approval date up to the project completion date. The Owners’ costs have been prepared by CMZ and are within typical industry range.

21.6 Estimate Provisions

Each element of the estimate is developed initially as a bare cost only. A growth allowance has then been allocated to each element of the costs to reflect the level of definition in pricing and engineering maturity relating to that element. Growth allowances are an integral part of the capital cost estimate. Without the inclusion of these, the estimate is incomplete.

Estimate contingency is included to address anticipated variances between the specific items contained in the estimate and the final actual project cost. It is assessed with due consideration of the preliminary level of design work, the manner in which pricing was derived and the preliminary nature of the project execution plan. Variances are applied to material cost, material quantity, labour price, labour productivity, indirect cost and indirect quantity. The estimate contingency was assessed as a percentage of direct and indirect costs to arrive at a total project estimate with the required confidence level.

Risk management reserve (Owner’s contingency) is capital that may be spent if a risk event occurs as opposed to growth and estimate contingency which are expected to be spent. Risk management reserve is covered by positive variance in capital cost in the sensitivity analyses in Section 22.

21.7 Sustaining Capital

Sustaining capital has been estimated for the mining area (see Section 21.4.1) and for the TMF in the same way as for initial direct capital costs. Growth and contingencies have been allowed for sustaining capital costs; however, no allowance has been made for other indirect costs, in the expectation that the operation will manage all sustaining capital activities with the then existing facilities, utilities and services, and within operating budgets.

21.8 Operating Costs

The operating cost of the project has been estimated on the same basis as the capital cost estimate except that there is no growth allowance or estimate contingency included. The estimate is considered to be at a preliminary feasibility study level with a precision of ±20-25%.

21.8.1 Operating costs summary

Project operating costs are composed of four major cost centers:

- Mining operating costs were estimated by NCL with input from CMZ.
- Processing costs covering the concentrator and on-site and off-site infrastructure were estimated by Ausenco with input from CMZ.
- General and administration (G&A) operating costs were estimated by Ausenco in conjunction with CMZ.
- Concentrate road transport costs from site to the Port of Matarani were estimated by Ausenco.

A summary of the average operating cost over the life of mine including road transport costs is show in Table 21-5.

Table 21-5 – Summary of average production period operating costs

Description	Production period average	
	$/t ore	($/lb Cu produced)
Mining	4.25	0.54
Processing	4.58	0.59
General and administration	1.22	0.16
Concentrate road transport	0.45	0.06
Total Cost	**10.50**	**1.35**

Operating costs vary with time according to total material mined and concentrator throughput which is provided in the production schedules in Section 16. There is also some minor variation related to concentrate output.

21.8.2 Exclusions

The operating costs exclude:

- Escalation or exchange rate fluctuations (long-term Peruvian New Sol exchange rate per USD used for cost estimates is approximately 12% lower than current levels)

- Exploration costs
- Contingency
- Import duty and taxes (not expected)
- First fill reagents and consumables (in capital cost estimate)
- Sustaining capital (in capital cost estimate)
- Interest and financing charges (assumed 100% equity basis)
- Concentrate shipping, port, insurance and marketing costs (see Section 19)
- Treatment and refining charges (see Section 19)
- Corporate overheads (other than offices in Arequipa and Lima which are included)
- Rehabilitation activities (included in financial analysis).

21.9 Mining Operating Cost Estimate

21.9.1 Summary of mine operating costs

The basis of the estimate for the preliminary feasibility study was that the Zafranal open pit will be initially operated by a qualified mining contractor during the pre-production period and the first three years of commercial production. Year 4 is considered to be a transition period with contractor operation winding down and owner operation ramping up, with owner operation from Year 5 to the end of the life of mine.

The following assumptions were made and inputs used:

- Contractor open pit mining costs are the results of applying unit rates by operation to the mined materials as per the production schedule, plus fixed and demobilization costs, as provided for in the contractor quotations
- Owner open pit mining costs include the sum of operating and maintenance labour, supervisory labour, parts and consumables, fuel and miscellaneous operating supplies
- Personnel are divided into salaried and hourly personnel
- Parts, non-energy consumables, fuel and miscellaneous operating costs were based on the projected mining fleet requirements
- Full maintenance and repair contract (MARC) for the first six years of owner operation (Year 4 through Year 9), provided by the vendors
- Recent quotes from explosives suppliers and equipment suppliers
- Diesel fuel cost of $0.718/L delivered to site
- Mine fleet replacement was considered to be part of the sustaining capital estimate.

Mine operating cost forecasts are presented in Table 21-6.

Table 21-6 - Mine operating costs

	Contractor - mining (Y1-Y4)			Owner - mining (Y4-Y19)			Total mining cost		
	$ M	$/t mined	$/t total	$M	$/t mined	$/t total	$M	$/t mined	$/t total
Operating cost	484	1.95	1.87	1,256	1.82	1.66	1,740	1.85	1.71
Loading	64	0.26	0.25	162	0.23	0.21	226	0.24	0.22
Hauling	181	0.73	0.70	558	0.81	0.74	738	0.79	0.73
Drilling	22	0.09	0.09	90	0.13	0.12	112	0.12	0.11
Blasting	65	0.26	0.25	216	0.31	0.28	280	0.30	0.28
Ancillary	56	0.22	0.22	138	0.20	0.18	194	0.21	0.19
Support				36	0.05	0.05	36	0.04	0.03
Engineering and administration	5	0.02	0.02	47	0.07	0.06	52	0.06	0.05
Dewatering	3	0.01	0.01	11	0.02	0.01	13	0.01	0.01
General	56	0.22	0.22				56	0.06	0.06
Re-handle	7	0.03	0.03				7	0.01	0.01
Fixed	25	0.10	0.10				25	0.03	0.02
Demobilisation	1	0.00	0.00				1	0.00	0.00
Tonnes mined	249	Mt		690	Mt		939	Mt	
Tonnes re-handled	10	Mt		67	Mt		77	Mt	
Tonnes total	259	Mt		757	Mt		1,016	Mt	

Mine unit operating costs that were used in the financial model are slightly lower (3 – 6 cents/t mined) primarily due to adjustments in escalation of labour rates.

21.10 Processing Operating Cost Estimate

21.10.1 Basis of estimate

Processing operating costs were estimated based on:

- CMZ recommendations for labour rates, fuel costs, electricity, miscellaneous expenses, reviewed against Ausenco's database for reasonableness
- Process design criteria
- Zafranal mine production schedule initially for the original 55 kt/d fixed throughput case and then finally scaled for the variable throughput case through the financial model
- Calculations by Ausenco based on test work results or benchmarking similar operations.
- Equipment and reagent supplier quotations.

21.10.2 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, TMF and water supply
- Labour for operating and maintaining concentrator, TMF and water supply including mobile equipment and light vehicles
- Fuel, reagents, consumables and maintenance materials for concentrator, TMF and water supply
- Fuel, lubricants, tyres and maintenance materials used in operating and maintaining the concentrator, TMF and water supply mobile equipment and light vehicles
- Operation of the TMF, including tailings cycloning, cyclone overflow and whole tailings slurry discharge management spreading and compaction of cyclone underflow sand for on-going embankment raises, supernatant water reclaim, seepage water recovery and recycle of return water to tailings cyclone feed dilution and concentrator process water
- Power supplied to the site from the main site substation
- Raw water supply from Majes I wellfield
- Power and contractor operation costs for sample preparation and assay and metallurgical laboratory.

21.10.3 Estimating method

The processing operating cost was initially developed based on the original 55 kt/d fixed throughput case to a level sufficient for fixed and variable costs to be estimated and for consumable, reagent and power costs to be estimated according to mineral zone: supergene or hypogene. These costs were then applied to the variable throughput production schedule developed by CMZ through the financial model with input from NCL and Ausenco. The resulting LOM operating cost was checked by Ausenco prior to inclusion in the report.

21.10.4 Summary

Processing costs per tonne are presented as the average costs over the life of mine for the 55 kt/d fixed throughput case unless nominated otherwise. Annual processing costs are presented for the 55 kt/d fixed throughput case unless nominated otherwise.

Table 21-7 provides average production period processing costs per tonne by mineral zone. Processing costs include power, reagents, consumables, mobile equipment, maintenance spares, as well as labour.

Table 21-7 – Average processing operating cost summary (life of mine) by mineral zone

Description	Hypogene US$/t ore	Supergene US$/t Ore
Power	1.57	1.49
Reagents	0.58	0.58
Consumables	1.32	1.13
Maintenance	0.26	0.26
Labour	0.02	0.02
Mobile equipment	0.72	0.72
Tailings sand spread and compaction	0.16	0.16
Total cost (US$/t ore)	**4.63**	**4.36**

21.10.4.1 Power

Power costs were calculated using the grid power rate of US$0.066/kWh supplied by CMZ. Power was calculated by facility, 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. An allowance of 2% of the total calculated power was allowed for small power consumers not identified in the equipment list. The power consumption was split based on the average specific energy values calculated for hypogene and supergene.

21.10.4.2 Reagents and consumables

Concentrator reagent and consumable costs were estimated based on the throughput and mineral zone. The costs were based on calculated consumption rates and unit costs supplied by vendors. Reagents costs include transport to site.

SAG mill media consumption was calculated based on the SAG mill power, ball loading and yearly weighted abrasion index. Ball mill media consumption was calculated by mineral zones based on the average annual abrasion index and power consumption of the ball mills.

The consumption of reagents were calculated based on consumption rates from metallurgical test work or estimated from first principles. These rates only varied according to concentrator feed tonnage as no differences were determined between supergene and hypogene zones.

Mill and crusher liner consumption rates were estimated based on vendor information and benchmarking similar plants. The liner consumptions were based on the 55 kt/d fixed throughput case and scaled to the variable throughput case in the financial model (see Section 22).

A refund was allowed at 5% of mill liner costs for liner and grinding media scrap.

Costs for filter cloths were based on vendor information. The filter cloth consumption was based on concentrate tonnage production.

Other, smaller, general consumables were factored based on the tonnage milled. The rate of US$0.01/t was used was based on benchmark data from similar sites. In addition, an annual allowance of US$75,000 was made for personal protective equipment for process employees.

21.10.4.3 Labour

Labour costs for all operations, maintenance, service and support roles were based on salaries and labour rosters provided by CMZ. The rosters and salaries were benchmarked against similar operations, and include on-costs.

Transportation, recruitment and training costs are included as separate cost items in the G&A costs. Staff profit sharing is included in the financial model.

21.10.4.4 Light vehicle and mobile equipment fleet

Costs were estimated for the concentrator and infrastructure light vehicle and mobile equipment fleet 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, service and support activities.

The fuel consumption rate was calculated for vehicles in the fleet. 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.

21.10.4.5 Maintenance spares and consumables

Annual maintenance spares and consumables costs were estimated as a factor of the total installed mechanical equipment, plate work, electrical and instrumentation equipment cost for the concentrator and infrastructure.

Annual maintenance contractor costs were estimated as a factor of the mechanical, electrical and instrumentation equipment cost for the concentrator and infrastructure.

21.10.4.6 Tailings cyclone underflow sand spread and compaction cost

Annual tailings cyclone underflow sand spread and compaction cost were calculated based on unit costs per tonne provided by Stracon and the embankment raising schedule.

21.11 General and Administration Costs

21.11.1 Estimating method

Operating cost estimates were prepared for G&A by CMZ and were checked by Ausenco. The estimate for G&A was broken down by department in which the cost was incurred:

- Presidency
- Vice President Operations
- Finance, logistics and information technology
- HR, camp and security
- External affairs
- Health, safety and environment
- Technical services.

21.11.2 Summary

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 US$25.6M was estimated. Costs are summarized by organization area in Table 21-8.

Table 21-8 - G&A cost summary

Item	Cost (US$/y)
Presidency	1,227,120
Vice President Operations	3,291,293
Finance , logistics and information technology	7,949,992
HR, camp and security	3,794,737
External affairs	6,170,918
Health, safety and environment	1,304,801
Technical services	1,914,509
Total	**25,653,370**

The majority of G&A costs are based on current costs being incurred by CMZ as well as benchmarked data from similar projects in Peru. The following sections describe the build-up of department's cost in the G&A area.

21.11.3 Presidency

The Presidency department cost is made up of the costs for the Arequipa office.

Costs for the Arequipa office are based on the current costs incurred by CMZ which include lease, stationary and utility costs.

Light vehicle costs for the office have been based on a five vehicle pool being rented for the office. These costs are benchmarked from CMZ's current office costs.

An allowance was made for corporate travel. This allowance includes all travel and conferences for senior personnel.

21.11.4 Vice President Operations

Allowances were made under the Vice President Operations department for general and administration costs. These costs include:

- Labour
- Consultants
- G&A site maintenance
- Light vehicles.

An allowance has been made for external metallurgical test work, vendor and other consultants in G&A costs.

Maintenance in G&A under the Vice President Operations consists of:

- Road maintenance
- G&A building maintenance
- Equipment hire
- Power supply maintenance.

G&A building maintenance costs were based on a factor of architectural and camp capital costs per year. In addition, an allowance was made for 400 t crane hire for heavy lifts in the plant.

Road maintenance costs were calculated based on an allowance per kilometre per year of roads. The allowance included internal roads, western access road, eastern access road, TMF access road, mine eastern access road and camp access road. Labour, equipment and material costs are included in road maintenance cost; reconstruction and improvements are excluded.

Power supply maintenance was based on information from GMI S.A. Consulting Engineers supplied to CMZ. Power supply maintenance includes the costs for operation, control, security, insurance, and maintenance of the transmission lines to site. The costs are based on the costs incurred by the company for the provision of the transmission service under the standards required by regulations of the Peruvian Electric Sector.

Light vehicle costs for the Vice President Operations department were based on a five vehicle pool being rented.

21.11.5 Finance, logistics and information technology

Costs for the finance, logistics and IT department include:

- Labour
- Information technology
- Lima office lease
- External financial audits
- Insurance costs.

Costs for the Lima office are based on the current costs incurred by CMZ which include lease, stationary and utility costs.

IT support costs were based on an all-in rate for support, hardware, software and communications per employee per year. The total cost for information technology was based on the personnel who require network access.

Costs for financial auditing and insurances were based on quotes from service providers who currently work with CMZ. Insurances include all operational property and general liability insurances for the operational phase.

Mobile equipment costs included an allowance for a forklift for the warehouse.

21.11.6 Human resources, camp and security

The human resources, camp and security department includes costs associated with recruitment functions, camp, site transport and site security functions.

The cost of the recruitment service is based on a proportion of annual costs of the positions being recruited during the year. Costs were calculated by position grade according to the Hays survey. The annual number of positions being recruited was scheduled by CMZ according to operational requirements and demobilization of the mining contractor.

Camp lodging and catering costs were calculated based on the organizational structure and rosters. Total number of annual person-days in camp was calculated by grade and roster.

Bus transport costs were based on personnel locations and rosters along with a rate for each round trip. Trip rates were based on benchmark data. An additional cost per person per day was allowed for local bus costs for local personnel.

Security contract costs for the site were based on a contracted amount which included security personnel, management, camp lodging and catering as well as transport of security personnel to site.

21.11.7 External affairs

The external affairs department includes costs associated with community relation programs, legal functions and permitting.

Permits and license costs were developed by CMZ and based on costs for maintaining operational and environmental permits and licenses required by Peruvian Legislation.

Community relations programs were based on a proportion of the profit.

Land lease payments were based on a benchmarked exploitation rate for a calculated total area.

21.11.8 Health, safety and environment

The cost for health, safety and environment department includes labour, environmental sampling and compliance programs along with mobile equipment costs..

Environmental compliance, sampling and external audit costs were based on benchmarked costs from similar projects in Peru.

Mobile equipment was allowed in the health, safety and environment department for site emergency services. The costs for mobile equipment were based on a fuel consumption rate and an average maintenance rate per engine hour.

21.11.9 Technical services

The cost for technical services includes costs for ore control, in-fill drilling and planning for the mine.

Light vehicle costs for the technical services department were based on a seven vehicle pool being rented. The costs for the fleet are based on the average monthly costs.

Personal safety equipment for the department was based on the number of employees and a rate per person per month.

In-fill drilling costs were calculated based on a rate provided by NCL and the schedule of total tonnage mined. A drilling rate per tonne mined was applied based on information supplied by NCL to calculate the in-fill drilling cost. Costs for in-fill drilling start in Year 4 of operations as the mining contractors scope is transferred to the Owner as operator.

As the mine is planned to be initially run as a contract operation, the costs for ore control were only applied from Year 4 of operation. These costs are covered by the contract mine operating cost contained in Section 21.9 prior to Year 4.

Costs for ore control samples were based on the schedule of ore and waste tonnage mined, the proportion of ore and waste drill holes that will be sampled, the tonnage miner per drill hole for ore and waste, and a cost per sample.

21.11.10 Labour summary

The basis for the labour cost estimate for G&A is the organizational structure which is discussed in Section 24.2.3. Remuneration for each position is based on Hays' benchmarked data by responsibility. The salaries and wages are inclusive of all on-costs.

21.12 Concentrate Road Transport

Costs for concentrate transport were developed by Ausenco based on a concentrate road transport route trade-off study. The cost of concentrate road transport is variable according to the tonnage of concentrate produced and does not consider transport and handling losses. Costs provided are based on average concentrate moisture of 9.5%.

Concentrate road transport costs consist of operating, maintenance and fuel costs, fleet capital amortization costs and profit margin for a contractor truck fleet and a road maintenance cost. The operating cost for the trucks was based on a 216 km (one-way route). The road maintenance cost was allowed for new road. No additional cost was allowed for maintenance of public highways. Costs for concentrate road transport were based on the contractor buying a new fleet for Zafranal, as Peruvian concentrate production is anticipated to still be expanding by the time Zafranal starts production.

Table 21-9 – Concentrate road transport unit cost summary

Item	Unit cost (US$/WMT concentrate)	Unit cost (US$/t concentrate dry)
Truck to Matarani port	37.55	41.49

Other costs associated with concentrate transport and delivery to the customer are presented in Section 19.

22 Economic Analysis

22.1 Cash Flow, Net Present Value (NPV), Internal Rate of Return (IRR) and Payback Period

22.1.1 Investment valuation results

The base valuation for the Zafranal project has been determined using a discounted cash flow (DCF) analysis, conducted in real term US dollars and derived from first principles. Net Present Value (NPV), Internal Rate of Return (IRR) and Payback Period are utilized as the valuation metrics for decision-making.

The DCF was derived by means of a techno-economic model that generated five basic cash streams: revenues, realization charges, operating costs, capital expenditures and taxes. Each of these cash flow streams were calculated from first principles, that is on the basis of production parameters. Additionally, working capital (WC) calculations are also included in the valuation.

The valuation included deterministic sensitivity analyses. The key variables and assumptions used in the calculation of the cash flows are based on CMZ's mid-range (i.e. expected) estimates. Range analyses were carried out to determine the possible range of values for the key variables.

Zafranal is a greenfield project consisting of an open pit mine that feeds a flotation concentrator at rates from 55 to 64 thousand tonnes per day (after initial ramp-up) depending on the grinding characteristics of the feed. It is expected to produce an average of 104 kt per year of copper metal during the first 10 years of production.

Table 22-1 summarizes the results from the deterministic valuation analysis on which the valuation is based. It also incorporates the results of the range analysis performed on the most relevant assumptions and project variables.

Table 22-1 – Summary of valuation results[1]

Valuation results: mid-range case prices	Deterministic
Total undiscounted revenues (US$M real)	9,627
Total undiscounted free cash flows (US$M real)	1,649
Initial capex with contingency (US$M real)	1,157
Mid-range NPV@ 8% (US$M)	496
Mid-range IRR (%)	15.9%
Payback period - start of mill operations (years)	5.1
Mid-range C1 Costs (US$/Cu lb real)	1.59
Mid-range C2 Costs (US$/Cu lb real)	2.07
Mid-range C3 Costs (US$/Cu lb real)	2.14
NPV@ 8% (US$M) - range min-max	(706) to 1,425
Decision/Valuation date	01-Jan-2019

The deterministic analysis results for the mid-range case show an NPV@8% of US$496M, an IRR of 16% and a payback period of 5.1 years counted from start of mill operations. This valuation assumes life of mine average prices of US$3.00/lb Cu, US$1,200/oz Au. Mid-range values for copper and gold head grades and recoveries, operating costs and capital expenditures are used in the valuation. During the life of the project, C1 costs are expected to average US$1.59 per pound of payable copper, enabling the project to retain a significant margin between cash costs and copper prices.

22.1.2 Cash flows forecast

The cash flows are summarized in Table 22-2 and presented graphically in Figure 22-1. Undiscounted free cash flow of US$1,649M is generated during the life of the mine.

Table 22-2 – Deterministic cash flows

Cash flow comparison (US$M real)	Undiscounted	Discounted
Total gross revenues	9,627	4,865
Total realization charges	-1,311	-652
Total operating costs	-4,026	-1,830
Total capital expenditures (net of WC Items)	-1,556	-1,252
Total royalty, workers' profit sharing and income tax	-1,062	-588
Changes in working capital	-23	-47
Total net cash flows	**1,649**	**496**

[1] C1, C2 and C3 are as defined by Wood Mackenzie. C1 includes direct operating costs (mine, mill, site G&A), TCRC, transportation and freight, marketing less by-product credits; C2 includes C1 plus depreciation and amortization; and C3 includes C2 plus royalties and interest charges.

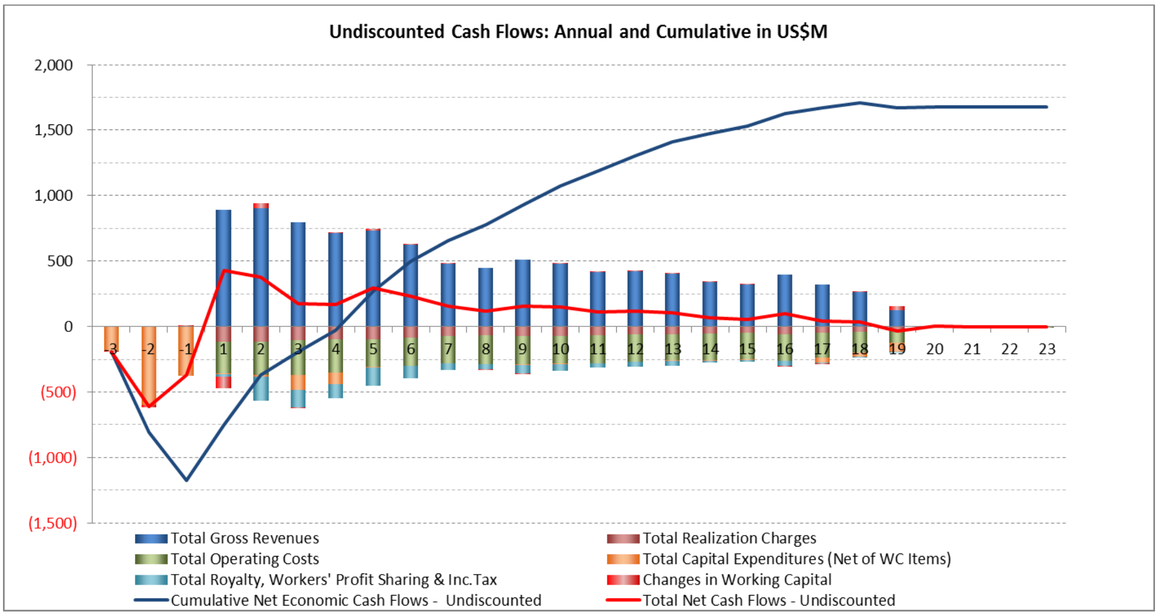

Figure 22-1 – Undiscounted cash flows

22.2 Principal Assumptions

The key assumptions employed in the valuation (discount rates, metal prices and foreign exchange) are based on internally developed forecasts. These forecasts incorporate applicable general economic assumptions as derived from numerous sources including forecasts by Teck Resources Limited and Mitsubishi Materials Corporation, industry studies and financial institutions (i.e. CIBC, Macquarie Bank and the IMF).

The NPV is based on the valuation of CMZ as a Peruvian unleveraged stand-alone asset, on an after-tax basis as of the currently expected decision point of 01 January 2019. A real US$ discount rate of 8.00% has been applied to calculate the NPV. The life-of-mine (LOM) price forecast for copper is US$3.00 per pound and price forecast for gold is US$1,200 per troy ounce. Other forecasts include US$85 per barrel of oil and PEN:USD exchange rate of 2.90. CMZ assumed a diesel price of US$0.72/L and an all-in power price of US$0.066/kwh.

CMZ expects to produce copper concentrate product with an average grade of 32.5% Cu that is to be trucked from site to the Port of Matarani. The marketing of this product is to be handled externally at a cost of US$12/dmt of concentrate. CMZ expects realization charges to comprise a treatment charge (TC) of US$90/dmt of concentrate and a refining charge (RC) of US$0.90/lb of copper and US$4/ounce of gold .

Other realization charges include US$37.55/dmt land transportation, US$20/dmt port handling and insurance, and US$65/dmt for shipping costs.

Forecasts for copper prices and treatment and refining charges were benchmarked by an independent consultant, reviewed and approved by the relevant QPs.

This NPV valuation was developed both as net economic cash flows in Peru and as dividends paid out net of withholding taxes.

22.3 Taxes and Royalties

The taxation assumptions used in the economic model were reviewed for reasonableness by Zuzunaga, Assereto & Zegarra Asociados S. Civil de R.L., CMZ's tax advisors. CMZ's tax advisors concur with the logic employed in the economic valuation.

22.3.1 Depreciation and amortization

The economic valuation incorporates calculations to determine the various taxes and other levies. CMZ does not currently have a tax stability agreement. Tax depreciation is limited to the maximum rate allowed by law and the useful life of the asset.

For simplicity, the capital expenditures have been allocated, as appropriately determined, to five main tax depreciation asset classes:

- LOM asset life: for fixed assets such as buildings, constructions, hydraulic shovels, drills, and other assets expected to last the 19 life of the project.
- 13 year asset life: for haul trucks, mine equipment and other assets expected to be replaced at the end of 13 years of service.
- 10 year asset life: mine ancillary and support equipment expected to be replaced at the end of 10 years of service.
- 5 year asset life: mine ancillary and support equipment expected to be replaced at the end of 10 years of service.
- 100% depreciation in Year 1: for pre-operating cost capitalized during pre-stripping and construction, depreciated in full at the start of operations.

Organization expenses, operating expenses and interest accrued during the pre-production period may be deducted in the first year of operation or amortized using the straight-line method over a maximum period of ten years. The tax model considers that these costs are deducted in the first period of operation.

Mine development expenses may be expensed in the first year of operation, or may be amortized over a maximum of three years. The tax model considers that these costs are deducted in the first year of operation.

22.3.2 Value added tax

CMZ is subject to an 18% value added tax, known in Peru as Impuesto General a las Ventas (IGV), against the value of goods and services purchased and then balanced against the value of goods sold. For exported goods, the IGV paid to produce them is refunded. Consequently, there is no net effect for CMZ, except for the working capital involved to administer the rolling payment and refund cycle. Additionally, CMZ benefits from the early recovery of IGV paid during the development phase. In general, IGV is recovered 90 days after payment.

IGV is also applicable on capital expenditures. The economic valuation model allows for the calculation of working capital associated with the payment and recovery of the IGV on the capital expenditures.

22.3.3 Mining royalty and special mining tax

Introduced in September 2011, mining companies are subject to both a mining royalty and a special mining tax. The royalty and special taxes are calculated using a marginal tax rate scale applied at different rates depending on seventeen levels of operating margin.

The royalty rate starts at 1% (for operating margins up to 10%) and is capped at 12% (for operating margins of 80% and above). There is a minimum 1% royalty payment that is applicable to mining companies.

The special tax rate starts at 2.0% (for operating margins up to 10%) and is capped at 8.4% (for operating margins of 85% and above).

The mining royalty and special mining tax are deductible for the purposes of calculating taxable income and thus reduce the Workers' Profit Sharing and corporate income tax. Over the life of the project, the royalty and special mining tax are projected to be 4.2% and 3.1% of operating income.

22.3.4 Workers' profit sharing

Employees are entitled to a statutory profit sharing which is calculated considering the employer's taxable income. For mining companies, the profit sharing rate is 8%. The profit sharing tax is deductible by the employer for tax purposes and is distributed to all workers after year-end.

This distribution to the employees has a set maximum and if the distribution exceeds that maximum to the employees, the remainder is returned to the Peruvian Government.

22.3.5 Corporate income tax and dividend withholding tax

Income tax of 26% is applicable to CMZ. A withholding tax of 9.3% is applied to dividends paid to owners. According to current tax legislation, these rates apply starting in 2019. As CMZ does not have a tax stability agreement, these will be the rates applicable to the project. The economic valuation model allows for the calculation of both taxes.

During the feasibility study, CMZ will analyze the requirements of and the potential benefits under a tax stability agreement with the Peruvian government. Among other benefits, a tax stability agreement allows the use of an accelerated depreciation regime, stabilization of tax rates and ability to keep the accounting of the company in US dollars.

Figure 22-2 shows the various tax payments per year and in aggregate. Over the life of the mine, CMZ will have paid in real terms US$1,062M or PEN 3,080M. This equates to US$1,324M or PEN 3,840M in nominal terms, which are the basis of actual tax payments. Overall, the Zafranal project will be significant contributor of taxes to the Peruvian economy.

Royalties and Special Mining Taxes will account for US$253M in nominal terms (US$202M real terms) equivalent to PEN 734M over the life of the mine.

Workers' Profit Sharing is expected to total in nominal terms US$269M (US$215M real terms) equivalent to PEN 780M over the life of the mine.

Corporate Income Tax is projected to total US$803M in nominal terms (US$644M real terms) equivalent to PEN 2,329M over the life of the mine. Of this amount, 50% is directed to the regional and local governments as mining canon.

Table 22-2 shows annual and cumulative mining royalty, special mining tax, workers' profit sharing, corporate income tax and the total of these in US$ real terms.

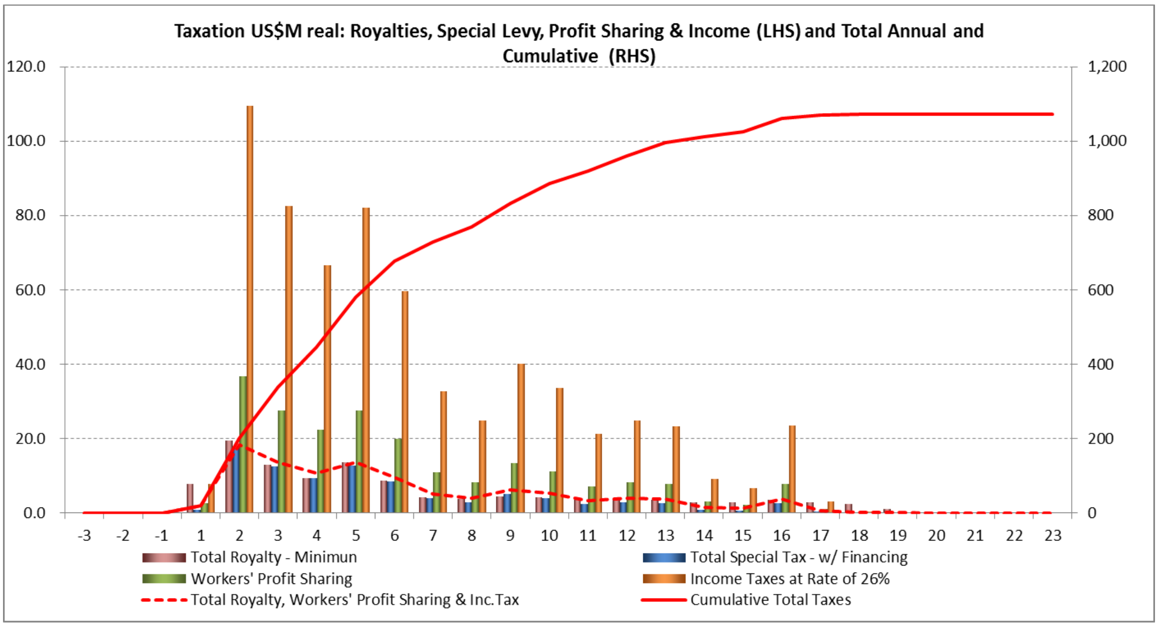

Figure 22-2 – Mining royalty, special mining tax, workers' profit sharing, corporate income tax and total

22.4 Sensitivity Analysis

The valuation was undertaken deterministically. The key variables and assumptions used in the calculation of the cash flows are based on CMZ's mid-range estimates. Range analyses were carried out to estimate the range of values for the key variables and assumptions.

The range analysis performed on the mid-range case was based on the use of minimum and maximum values for the critical variables: metal and oil prices, foreign exchange rates, head grades and capital expenditures. The minima and maxima values for metal prices and foreign exchange are based on the forecast sensitivity ranges used by CMZ shareholders, reviewed and approved by the relevant QPs. Minima and maxima values for head grades are based on the QPs advice. Minimum and maximum values for capital expenditures were determined on the basis of the contingency analysis by Ausenco. NPVs were calculated for these extreme cases although no likelihood of occurrence is expected. The objective of this analysis was to test the limits of the valuation and provide insight as to the potential extreme downside and upside values. The range of deterministic $NPV_{8\%}$ outcomes is negative US$706M to positive US$1,425M.

The results from a simple sensitivity analysis are presented on the next two tables. Table 22-3 shows the copper prices sensitivities on undiscounted cash flows, after tax net present values and internal rate of return. Table 22-4 shows the NPV sensitivity to changes from the base case values for selected project variables.

Table 22-3 – Copper price sensitivity

Copper price sensitivity		Copper prices (US$/lb)				
		2.00	2.50	3.00	3.50	4.00
Undiscounted cash flows US$M	Pre-Tax	-311	1,204	**2,711**	4,236	5,753
	After-Tax	-478	648	**1,649**	2,632	3,574
After-tax NPV US$M at discount rates	5%	-515	180	813	1,426	2,029
	8%	-547	-5	**496**	978	1,454
	10%	-567	-101	333	749	1,162
Internal rate of return	IRR %	0.0%	7.9%	**15.9%**	22.1%	27.6%

Table 22-4 – Net present value sensitivity

NPV Sensitivity	Base case	Changes from $NPV_{8\%}$ (US$M)				
		-25%	-10%	-	10%	25%
Initial capital (US$M real)	1,157	745	595	**496**	394	241
Total operating costs (excluding. concentrate road transport) (US$/t milled)	10.05	788	613	**496**	374	188
Head grade - feed (Cu %)	0.401	-155	239	**496**	745	1,118
Head grade - feed (Au g/t)	0.069	455	479	**496**	510	534

For the valuation, a more detailed sensitivity analysis was performed on NPV and IRR. Inputs were varied by 20% for metal prices and 7% for foreign exchange from the mid-range values. The following key inputs were used in the sensitivity analysis:

- Copper prices: mid-range case US$3.00/lb
- Gold prices: mid-range case US$1,200/ troy ounce
- Oil prices: mid-range case US$85.00/bbl
- PEN:US: mid-range case 2.90.

Key variables that were investigated were:

- Copper head grade: mid-range case 0.401% Cu
- Gold head grade: mid-case 0.069g/t Au
- Initial capital expenditures: Mid-range case US$1,006M excluding contingency.

Table 22-5 shows the range of sensitivities applied to the mid-range case values in the deterministic sensitivity analysis.

Table 22-5 – Deterministic sensitivity analysis ranges

Tables of ranges	Deterministic		
	Min	Mid-range	Max
Head grade - Cu (Leaching)	95%	100%	105%
Head grade - Cu (Flotation)	95%	100%	105%
Head grade - Au (Flotation)	90%	100%	110%
Capex - all	83%	100%	144%
Commodity prices and FX			
Copper	2.40	3.00	3.60
Gold	960	1,200	1,440
PEN:USD	3.10	2.90	2.70
Oil	68.00	85.00	102.00

As part of the deterministic sensitivity analysis, cases were developed based on the ranges of the key assumptions and variables. Each of these cases was valued under various commodity price and FX assumptions. The objective of this analysis is to gain insight as to the extreme values of the valuation range. This analysis however does not shed light into the likelihood of occurrence of each of these cases or any permutation thereof.

The cases are defined as follows:

- Mid-range case: All key assumptions and key variables at their mid-range values.
- High case: Scenario in which the project benefits lower than expected capital expenditures during construction while enjoying head grades for copper and gold are higher than expected.
- Low case: Scenario in which initial capital expenditures are higher than expected. Head grades for copper and gold are lower than expected.

Table 22-6 summarizes the findings of this deterministic case analysis. Of particular interest are the High Case under high commodity prices and the Low Case under low commodity prices.

Table 22-6 – Deterministic cases

Deterministic cases (8%WACC)		Commodity prices and FX		
		Low	Mid-range	High
Cu/Au/PEN:USD/Oil		2.40/ 960/ 3.10/ 68	3.00/ 1200/ 2.90/ 85	3.60/ 1440/ 2.70/ 102
High case: high head grades and low capex	NPV US$M	174	809	1,425
	IRR %	11.8%	22.5%	30.9%
Mid-range case: mid-range head grades and mid-range capex	NPV US$M	-119	496	1,083
	IRR %	5.6%	15.9%	23.4%
Low case: low head grades and high capex	NPV US$M	-706	-103	466
	IRR %	0.0%	6.8%	13.0%

Under the high case with high commodity prices, in addition to lower capital and higher head grades, the project enjoys higher than expected metal prices when production commences. Alongside higher metal prices, the PEN appreciates and oil price increases therefore operating costs increase as a result of higher payroll costs in US$ terms and higher fuel costs. This scenario yields a potential NPV of US$1,425M and an IRR of 30.9%.

In contrast, the low case with low commodity prices scenario generates a negative NPV of US$706M and a negative/nil IRR as initial capital of the project is costlier than expected, head grades prove lower and metal prices drop at the time production commences. Offsetting these losses are lower payroll costs on the back of a depreciating PEN and lower fuel costs as the barrel oil drops.

These two extremes help understand the extent within which the valuation could oscillate.

23 Adjacent Properties

There are no relevant adjacent properties.

24 Other Relevant Data and Information

24.1 Project Implementation Plan

24.1.1 General

The implementation strategy is based on an organization with an Owner's project execution team and an EPCM contractor both reporting to a Project Director. The team also includes an Vice President Operations, whose role is to manage pre-production mining activities and prepare for operations, and VPs of Finance, Human Resources and External Affairs, whose primary roles are to manage the off-site service and support functions that will transition from project execution to operations. The Project Director, and Vice Presidents all report to the CMZ President, who has ultimate accountability for delivery of the project.

The Owner's team has overall accountability for the project including management of the EPCM contractor and direct accountability for the delivery of key components of the project.

Owner and EPCM project organization chart are shown respectively in Figure 24-1 and Figure 24-2.

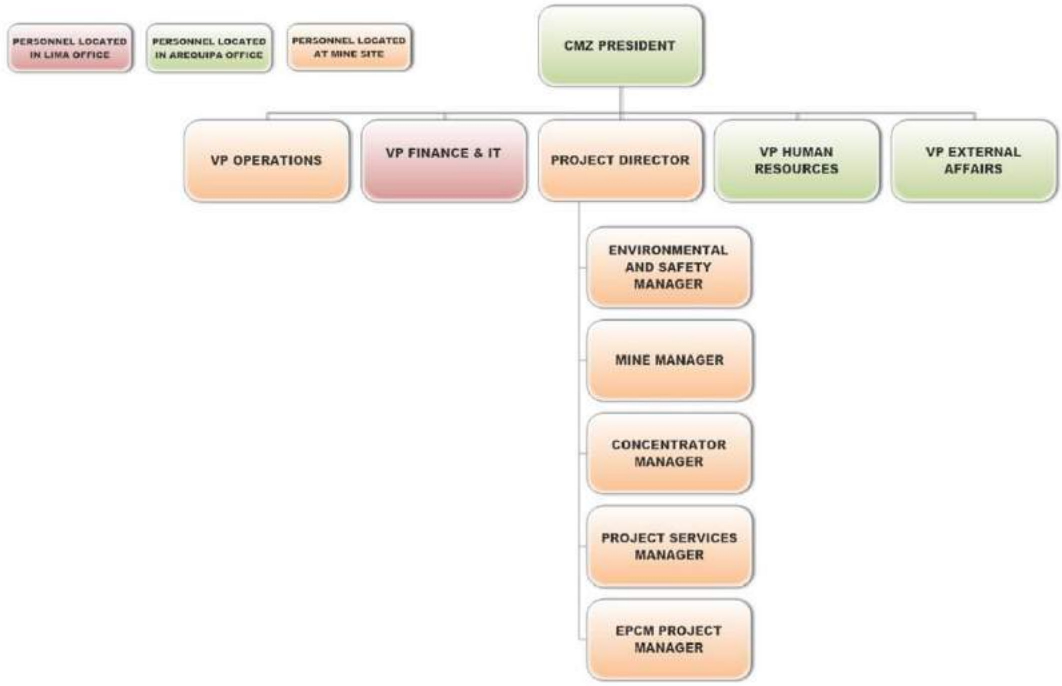

Figure 24-1 – Owner´s project organization chart

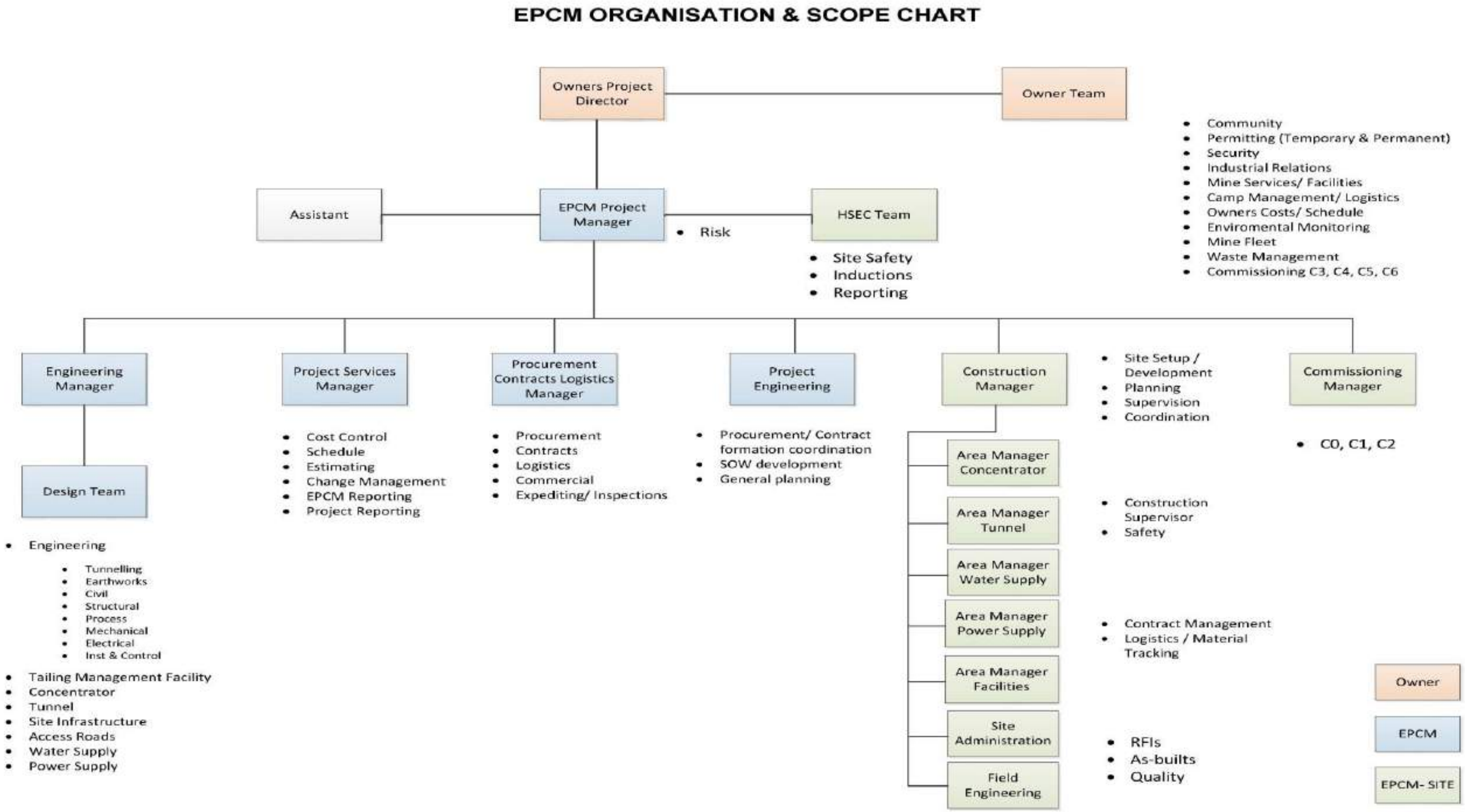

Figure 24-2 – EPCM organisation and scope chart

The CMZ President´s (Owner's) team will provide specialist commercial, legal, environmental, industrial relations and other services and support to the EPCM contractor. They will also ensure consistency is provided across the full project and ensure the effective implementation of policy across the project.

The overall philosophy is to source as many skilled personnel from Peru, especially those who are local to the project site. If insufficient trained and experienced people from Peru are available, then personnel will be sourced from elsewhere in South America. It is anticipated that senior management will also include expatriates with extensive experience in project delivery. It is likely that some design will be performed in offices in other countries due to the specialized experience required for specific areas. Initial control of the project is from a Lima office until the focus of the project moves from engineering and procurement to construction on site.

24.1.2 Engineering

Detailed engineering for the project will include all engineering disciplines and all design and drafting capabilities. The engineering team will have available all previous study information and detail, and be able to utilize all current technologies at the time of execution.

Once CMZ has authorized the project to proceed, the EPCM contractor will establish the engineering organization and assemble the necessary engineering teams to meet the project requirements. The design will be completed in accordance with local and international standards.

24.1.3 Procurement plan

The EPCM procurement team will provide procurement, expediting, inspection and delivery of all equipment and materials. Purchase orders will be placed with qualified vendors based on technical compliance, quality, price and delivery.

Equipment will typically be sourced from recognized international vendors, where practicable via 'the vendors' Peruvian or South American agents to ensure continuity of warranty and service support during operation.

Fabricated items (structural steel and plate work) will typically be sourced from within Peru. However, consideration will also be given to vendors located in other countries.

A freight forwarding agency will be engaged to manage transportation of equipment. Internationally supplied equipment requiring sea freight will be shipped from major ports. The preferred destination ports are Matarani in Islay and San Martin in Pisco, due to their proximity to the project site. However, the Port of Callao, adjacent to Lima, is an alternative option.

Every effort will be made to maximize Peruvian content, but this will be on a competitive basis.

24.1.4 Contracting plan

The contracting plan summarizes the contracting strategy, approach and management plan for for site construction works and other site services.

The Zafranal Project will be delivered using an integrated approach by partitioning various project scopes involving site services into horizontal, vertical, and geographical pieces into manageable packages.

Vertical packages, where contracts include all disciplines for a particular facility, are preferred for minor works such as buildings, complete with design, i.e. design and construct packages.

Horizontal packaging is a contracting strategy where each package tends to reflect single or multiple disciplines across multiple areas, e.g. concrete for all facilities on the site; structural, mechanical and piping for all areas of the concentrator.

The major part of the Project will be implemented with a general strategy of multiple horizontal discipline-oriented contract packages (for earthworks, concrete, structural/mechanical/piping installation, tank erection, electrical/instrumentation works, HV electrical works, process control, etc.) with the inter-discipline construction interfaces managed by an EPCM contractor and free issue of equipment, structural steel, plate work and spooled piping.

The mining contractor and some other contracts will be managed directly by CMZ with some major items possibly under build, own and operate (BOO) or build, own, operate and transfer (BOOT) approaches.

24.1.5 Construction plan

The EPCM contractor will manage the construction activities on behalf of CMZ, other than the mine development and others managed directly by CMZ.

The primary objective of the EPCM construction team is to safely manage the delivery and coordination of the Zafranal Project field activities up to the point of mechanical completion. The team ensures compliance with HSEC procedures, quality plans, schedule, contracts management; and manages site construction logistics and coordination between construction contractors to ensure a safe and efficient delivery of the project.

Construction will be undertaken by suitably-experienced contractors operating under the management of the EPCM contractor’s site teams.

24.1.6 Commissioning plan

The EPCM contractor will be responsible for commissioning up to wet commissioning. After wet commissioning, the CMZ’s operations team would take responsibility for ore commissioning, any performance testing, and transition to operations. Control of the site and HSEC responsibility follows the responsibility of each commissioning stage.

24.1.7 Execution schedule

24.1.7.1 Summary schedule and critical path

Figure 24-3 shows the Level 1 summary schedule and the critical path for the Project, the latter of which includes the following activities:

- EIA detailed – submission and approval
- Early funding finance after EIA approval
- Early works (contractor mobilization, camp construction, access roads)
- Concentrator and infrastructure construction and commissioning.

Activity ID	Activity Name	Current Float Start	Current Float Finish
Milestone		**04-May-16**	**23-Aug-19**
A4510	Board Approval to Start EIA Advance Engineering, Met & FS Hydro Program (2016)	04-May-16	
3	Board Approval to start Feasibility (MEC, Infill-2017)	03-Dec-16	01-Jan-17*
5	Board Approval Project Sanction: Early Financing	08-Oct-18	05-Jan-19
A1040	Board Approval Project Sanction: Construction	28-Jul-19	23-Aug-19
Financing		**02-Jan-18**	**23-Aug-19**
A2150	Financing Phase 2: Due diligence	02-Jan-18	01-May-18
A2160	Financing Phase 3: Commitments & Definitive documents	08-Oct-18	06-Apr-19
A2730	Financing Phase 3:Satisfaction of conditions precedent	27-May-19	23-Aug-19
A2740	Financing Phase 3:First Disbursement		23-Aug-19
EIA & Engineering for EIA		**26-May-16**	**07-Oct-18**
A4480	FS Hydro Studies (Drilling & Report)	26-May-16	16-Mar-17
557	Waste Dump & TMF - Geotechnical Drill & Report	14-Oct-16	01-Jan-17
A4490	Engineering for EIA (Waste, TMF, Leach, Water)	24-Dec-16	10-Jun-17
A4750	Open Pit - Geotechnical Drill & Report	02-Jan-17	13-Sep-17
A4760	Water Supply - Geotechnical test pits & Report	02-Jan-17	12-Mar-17
187	EIA Elaboration	11-Jun-17	07-Dec-17
188	EIA Approval	08-Dec-17	07-Oct-18
Feasibility Engineering		**02-Jan-17**	**31-Dec-17**
556	FS drilling Infill Program	02-Jan-17	01-Apr-17
A4470	FS plant & infrastructure Engineering	02-Jan-17	31-Dec-17
561	FS Geology & Resource model	02-May-17	30-Jul-17
562	FS Mine Design	14-Sep-17	12-Nov-17
Power Line - OffSite		**02-Mar-17**	**28-Jul-20**
A3970	Power Line Feasibility & Engineering for Procurement	02-Mar-17	29-Jul-17
A3980	Power Line Pre-Operativity Permit (Elaboration & approval)	30-Jul-17	26-Dec-17
A1840	Power Line Negotiation & Acqusition - Easement (Imposicion de Servidumbre)	27-Dec-17	24-Jun-18
A4001	Power Line Consession Approval	27-Dec-17	25-May-18
A4040	Power Line Major Equipment Purchase (Trafo, Eq Patio Llaves)	06-Jan-19	31-Dec-19
A4120	Power Line Construction: Substation Civil & Mechanical Electrical	06-Mar-19	02-Oct-19
A4200	Power Line Construction: Substation Equipment Installation	01-Jan-20	28-Jun-20
A4130	Power Line Commissioning (Pruebas y puesta en servicio)	30-May-20	28-Jul-20
Water Infra - OffSite		**11-Apr-18**	**18-Jun-20**
2191	Permit 1 - Water Availability Permit (Certificado de Disponibilidad Hidrica despues de estudios)	11-Apr-18	07-Oct-18
2192	Permit 2 - Water Infrastructure construction Permit (AEO Autorización de Ejecución de Obra)	08-Oct-18	05-Apr-19
220	Permit 3 - Water use for construction permit (AAA permiso acuifero Zafranal) - Elaboration and Approval	06-Apr-19	23-Aug-19
592	OffSite Infra - Water and sewage supply (for Operations)	24-Aug-19	18-Jun-20
Mine		**08-Oct-18**	**24-Jul-21**
225	Permit - Mining Construction Permit (Autorización de Inicio 1) - Elaboration & Approval	08-Oct-18	05-May-19
597	Early works Mine (Access Roads, Truck Shop)	02-Jan-20	29-Jun-20
A3440	Pre stripping Mine	30-Jun-20	24-Jun-21
228	Permit - Mine Operation Permit (Autorización de Inicio 2 Actividades de Operación) - Elaboration & Approval	26-Jan-21	24-Jul-21
Detail Engineering & Early Works		**13-Mar-17**	**25-Nov-20**
A1960	Permit - EIAsd 4th Modification Approval	13-Mar-17	08-Oct-17
226	Permit - Explosives for Construction	23-Nov-17	17-Nov-18
580	Detail Engineering	02-Jan-18	25-Feb-19
601	Construction Tunnel and conveyor	06-Jan-19	25-Nov-20
596	Early works Plant & Infrastructure construction	06-Jan-19	23-Oct-19
Plant & Infra Construction		**08-Oct-18**	**24-Dec-21**
233	Permit - Plant Construction Permit (Concesión de Beneficio Etaba B)	08-Oct-18	06-Mar-19
600	Construction Tailings Storage (TMF)	24-Aug-19	04-Jan-21
599	Construction Plant and Infrastructure	24-Oct-19	10-May-21
605	Permit - Plant Operation Permit (Concesión de Beneficio Etaba C) - Approval	26-Apr-21	24-Jul-21
607	Commissioning - after receipt of CdB - (Stage 3) Exploitation	25-Jul-21	07-Nov-21
609	First Concentrate	24-Dec-21	

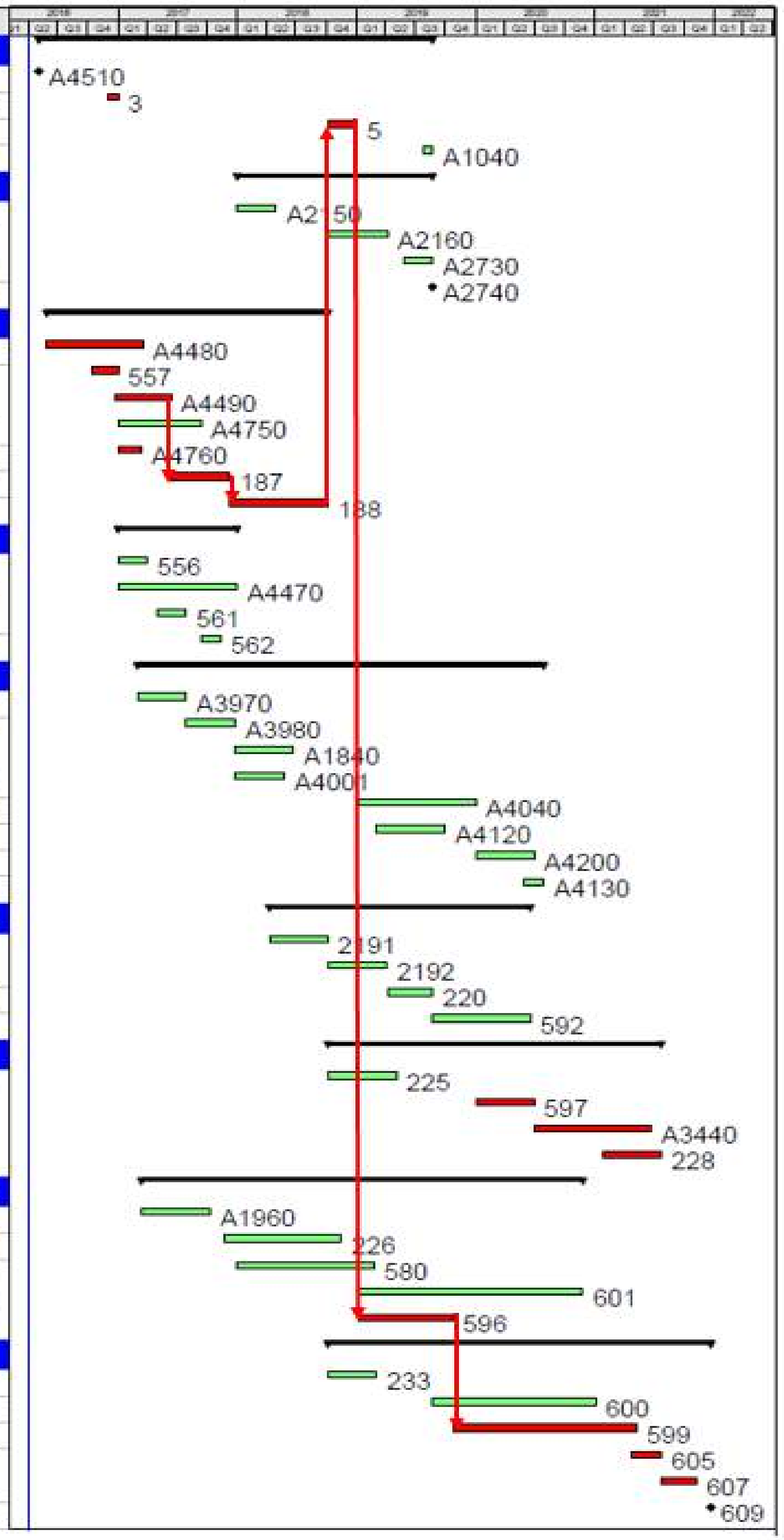

Figure 24-3 – Summary schedule

24.1.7.2 Assumptions

Early funding is granted after the Feasibility study and EIA approvals. With the early funding the early works would start.

Early works activities (accommodation camp, access roads) will be carried out after being granted the EIA semi-detailed 4th modification.

Archaeological permit process starts during feasibility study preliminary phase to finish before the initiation of major permits, the beneficiation concession and the mine operation permit; these two are required to start the concentrator and infrastructure construction and the mine pre-stripping. Water permits and EIA start in the feasibility study preliminary phase before applying for the construction permits mentioned above.

Engineering required for permits is generally a separate exercise to the feasibility study and detailed engineering. In some cases, such engineering will occur during the feasibility study or at an early stage of the project execution phase to allow early submission of a permit.

Procurement of items defined as 'long-lead' will commence when early funding is approved. Long-lead items will thus require datasheets etc. prepared during the feasibility study. All other procurement items will require full financing to be in place. Durations for supply of procurement packages have been provided by vendors.

24.1.7.3 Key project milestones

The key project milestones are summarized in Table 24-1 below:

Table 24-1 – Key project milestones

Deliverable	Revision 0 End Date
PFS complete	Q2 2016
Gate 2 approval – start FS	Q4 2016
FS complete	Q4 2017
EIA detailed	Q3 2018
Gate 3: approval - FS and EIA	Q4 2018
Board approval project sanction: early funding	Q4 2018
Financing Phases 2 and 3; board approval project sanction: construction	Q3 2019
Beneficiation construction authorization	Q1 2019
Detailed engineering complete	Q1 2019
Construction of concentrator complete	Q2 2021
Pre-stripping complete	Q2 2021
First ore	Q4 2021

24.2 Operations Management Plan

24.2.1 General

CMZ will have three administration centres in Peru, at the Project site, in Arequipa and a small office in Lima. The Vice President of Operations will be located at the Project site, reporting to the President, and be responsible for the health, safety and performance of the personnel on site including mining, concentrator, technical services; health, safety and environment; as well as on-site representatives of human resources, purchasing and warehouse, camp administration, labour relations, security, and cost accounting. The President will be located in Arequipa and be the legal representative for the Company with responsibilities including overall Company performance, external affairs (including community relations), human resources, and finance. The Finance Manager will be located in Lima, reporting to the President, and will be responsible for information technology, marketing, procurement, treasury and finance.

24.2.2 Organization structure

CMZ will structure the organization in a matrix style to minimize the number of senior management required to oversee the activities of its employees and contractors. As a result, management will be by location (Zafranal, Arequipa and Lima), which will require some positions to have dual reporting responsibilities. Supervisory personnel will be assigned to approximate a span of control of 12 personnel per supervisor. Figure 24-4 illustrates the proposed organizational structure for Compañía Minera Zafranal.

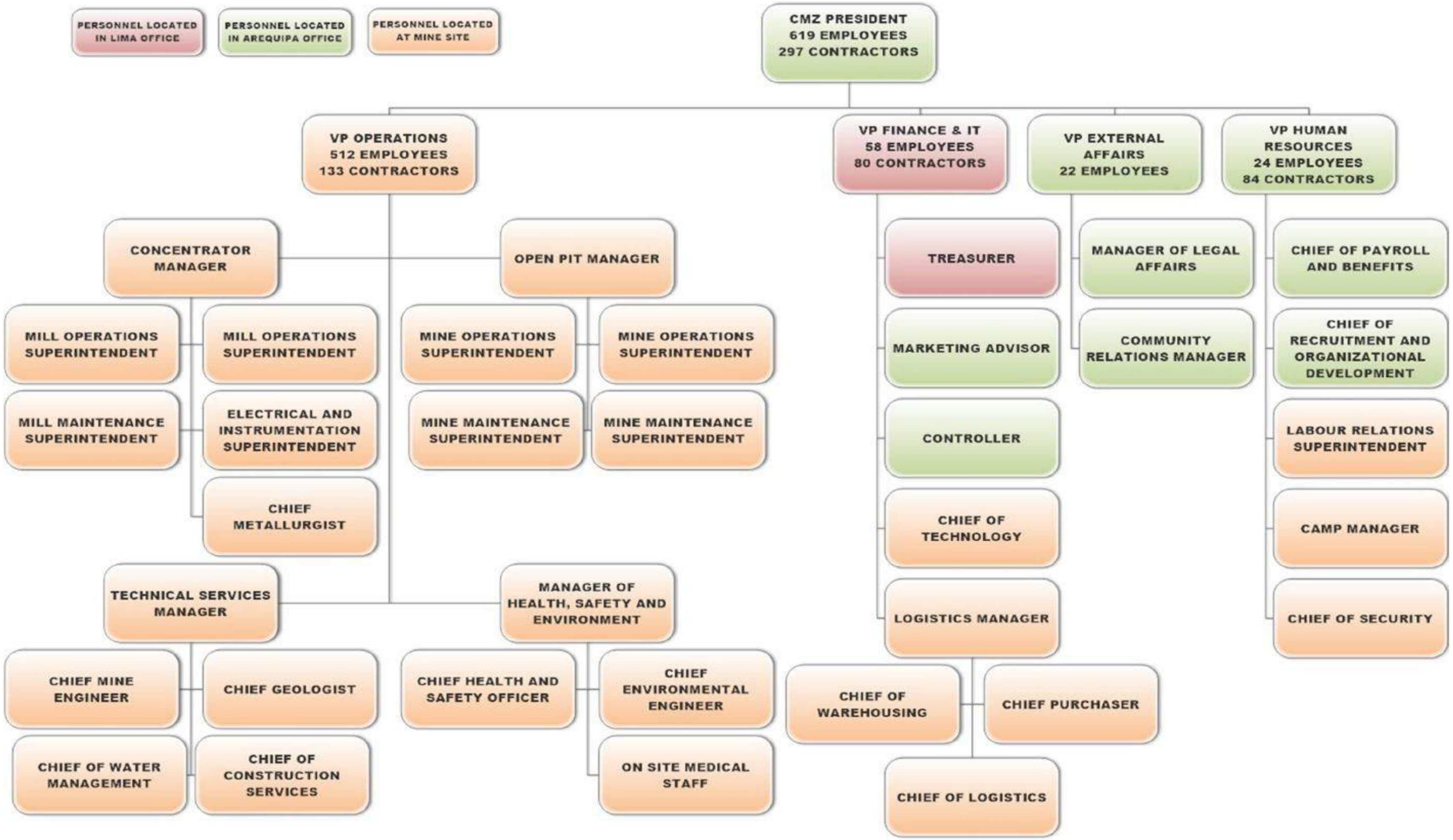

Figure 24-4 – CMZ organization chart (production Year 4)

24.2.3 Labour sources

CMZ has an established general and administrative (G&A) and field workforce with several years of service and corporate history. While not all current employees will be retained, the basic structure will remain in place and serve as a solid starting point for building the permanent workforce. Additional CMZ labour will be recruited primarily from Arequipa, and Lima.

CMZ skilled labour will be recruited from existing mines within the Arequipa and neighbouring regions or from non-mining sectors in the larger populated areas of Peru. The Project will be an attractive place to work given its location relative to neighbouring agricultural communities and the Peruvian coast, and its moderate elevation of 1,400 to 2,900 metres above sea level.

Senior management personnel will be bilingual and be sourced from the mining sector and major population centres in Peru or recruited internationally. A mix of national and expatriate senior personnel will be encouraged to facilitate the adoption of world´s best practices by the organization. Over time the number of expatriates will decrease as national employees gain more experience. CMZ will adhere to Degree Law 689 that governs the recruitment of foreign workers in Peru.

Administrative employees will be recruited locally where possible, with most being recruited from Arequipa.

A stripping contractor will be used in the mine for the pre-production period and for the first three and a half years of mine production. It is anticipated that the majority of contractor personnel will stay on as CMZ employees once the contract work is completed. The contractor will be responsible for the maintenance of its own equipment on site.

The selected fuel and lubricant vendor will be required to construct and operate the mine fuel and lubrication station and the explosives vendor will be required to construct and operate the explosives plant and magazine. A separate contractor will provide and maintain small vehicles and minor equipment for the site. CMZ supervision of these activities will be done at the Company level to avoid assuming contract worker liabilities.

CMZ will purchase its own mining equipment when it takes over operation of the mine and its employees will operate the equipment. During the second three years of production CMZ will contract and supervise mine maintenance activities that will be provided by equipment vendors under a modified Maintenance and Repair Contract (MARC). Over time the MARC will be replaced with a Labour Plus Parts Contract (LPP) and eventually the labour component of the LPP will be replaced with CMZ employees.

24.2.4 Roster schedules

24.2.4.1 Lima and Arequipa offices

The CMZ offices in Lima and Arequipa will open at 8.00 am and close at 6.00 pm, Monday to Friday. Office employees will have a one-hour lunch break and work 45 hours per week. The Company will not provide transportation to and from work and both offices will have limited parking facilities.

24.2.4.2 Zafranal site

Workers and staff will be encouraged to live in the neighbouring towns and where travel time permits, daily bus transport to and from their place of work in lieu of camp accommodation will be provided by CMZ. It is anticipated that 30% of the workers and 10% of staff will be located initially in the neighbouring towns and this number will increase to 60% and 20% respectively, over a five-year period.

The roster schedule for CMZ workers and team supervisors will be 4 days on and 4 days off (4 X 4), 12-hour shifts, and most senior management and technical employees (staff) will work 4 X 3, 12-hour shifts, except where 7 days per week coverage is required to support the operations.

Table 24-2 illustrates the composition and work location of the proposed CMZ employee and contractor workforce as well as initial camp requirements:

Table 24-2 - Composition and location of CMZ workforce and camp requirements

Description	Total personnel	Work location			New Zafranal Camp	
		Zafranal	Arequipa	Lima	On-site personnel(1)	No. of Rooms
Total employees	619	569	45	5	569	232
Total contractors	297	297			297	97
Total site visitors	22	22			22	22
Total personnel	**938**	**888**	**45**	**5**	**888**	**351**
Site camp requirements						
Final camp requirements	938	888	45	5	888	351
Less stripping contractor camp	(284)	(284)			(284)	(91)
Total to Year 3 of production	**654**	**604**	**45**	**5**	**604**	**260**

Notes:

1. The number of personnel requiring camp accommodation could decrease with local community hires
2. Existing Camps — Capacity
   - Zafranal Exploration Camp — 50
   - Ganchos Exploration Camp — 40
3. Event Sequence
   - 3.1 Stripping contractor to occupy Zafranal and Ganchos camps to facilitate construction of eastern access road and contractor camp.
   - 3.2 Owner's mining team to occupy Zafranal camp to supervise stripping contractors activities.
   - 3.3 TMF contractor to occupy Ganchos camp to facilitate early start of earth-filled starter embankment using alluvial material.
   - 3.4 Stripping contractor to occupy its own camp located near mine stripping operation (plateau areas around Zafranal camp).

3.5 Construction group to occupy Ganchos camp while building construction camp and CMZ permanent camp.

3.6 Construction group leaves sufficient camp accommodation for mine operations and maintenance personnel as stripping contractor demobilizes.

24.2.5 Industrial relations

All candidates for supervisory staff positions will undergo psychological profiling and an evaluation of emotional intelligence, to ensure the appropriate recruitment of supervisors.

The distance between the Arequipa office and the Zafranal dictates the requirement for a strong human resource presence on site with seven-day coverage in key areas of industrial relations, camp operations, payroll and benefits. In addition, a mine superintendent and mill superintendent will always be present on site to deal with situations before they become industrial relations issues.

As per Peruvian law, workers and management will form a joint safety and health committee to actively address safety and health issues as they arise and to monitor remedial actions taken by management. Families will be invited to participate in safety in the home campaigns and other activities orientated to increase safety awareness in all employees and their families. Partners of employees will also be invited to participate in tours of the operation and company-sponsored functions and events.

24.2.6 Recruitment

Ownership of the recruitment process resides with the Human Resources Department that will provide the position description and outsource appropriate parts of the process. Recruitment of employees will be based on the selection of healthy individuals of either gender with adequate skills and emotional intelligence to perform the requirements of the position. Preference will be made for local hiring in neighbouring communities, followed by recruitment from within the Arequipa region. Special attention will be paid to achieving agreed-upon hiring levels within local communities with the commitment to provide suitable training for local candidates without the necessary skill levels for the position.

Hiring of operations personnel during construction and commissioning will be undertaken where the knowledge obtained will facilitate the proper operation and maintenance of the process equipment. Special attention will be paid to the assessment of individual safety awareness before assigning the person to the work environment. Every employee and contractor will have the right to refuse a task, which he or she believes to be unsafe.

24.2.7 Remuneration and performance management systems

CMZ has adopted the job matching methodology of the Hay Group in determining the content and responsibilities and remuneration of each position within the current and future organization.

24.2.8 Training systems

Proper training of employees is a critical component to achieve a safe and efficient operation. Two knowledgeable training professionals will be recruited to coordinate the training programs adopted by the chief of recruitment and organizational design to ensure a smooth start-up of activities following the project construction. The trainers will focus on the concentrator and related infrastructure, as the mine will start with a stripping contractor who will operate and maintain its own equipment until Year 3 of production.

Training activities will begin 6 to 12 months before the start of commissioning as this process will provide these employees with the trouble-shooting skills required to operate and maintain complex and sophisticated equipment. The training professionals will make extensive use of simulators, teaching aids and courses provided by the main equipment vendors both on site as well as at the factory location.

24.2.9 Information systems

24.2.9.1 Management software

All mining operations are challenged to manage the business and improve profitability without compromising worker safety or efficiency. Professional management ERP (Enterprise Resource Planning) software for mining will be implemented to assist in achieving these objectives by integrating the business process, providing real-time reporting and advice for decision-making without duplicating data entry and information.

Real-time reporting capability will be an integral part of the ERP software, meaning the software updates information throughout the entire enterprise instantaneously. The ERP software shall provide CMZ with real-time strategic data to stimulate and plan the company future development and generate the required financial statements for the organization. Benefits include flexibility, control, and visibility for each operational area, while reducing wasted time and resources.

24.2.9.2 Communication network

CMZ’s communication network can be categorized into two different types: office and field. The office network corresponds to the local area network (LAN) that connects computers and devices in a limited geographical area such as office buildings. The office network will be available in wireless and wired mode for registered CMZ workers only.

The field network is aimed at supporting the work done by field staff: operations, maintenance, security and other production areas that require communication outside of offices and other buildings. Machinery communication will also be included in this network type. The field communication network will be built over a microwave communication technology such as WiMAX.

CMZ’s communication network will support real-time data acquisition for hard-to-reach assets, high-speed and long-range performance for monitoring and control purposes, low maintenance and flexibility of expansion to meet changing requirements.

24.3 Risk Management

24.3.1 Introduction

The Zafranal Preliminary Feasibility Study (PFS) has been subject to proactive risk management from July 2014.

This report section summarizes the status of the project risk register at 24 March 2016. This register combines the project level risks with inputs of all risk activities from the PFS phase.

24.3.2 Summary of high severity risks

Table 24-3 includes the current high severity risks listed in order of severity rating.

Table 24-3 – Summary of high severity risks

ID	Name	Cause	Effect	Categories	Likelihood	Driving impact	Current rating
15792	Study activities and field work delayed by community resistance	Political situation with communities in the region; Project fails to generate sufficient broad-based communication, engagement and transparency required to enable and promote achievement of Project milestones and schedule	Potential for a delay in the social engagement workshop which would delay the project; Social baseline development for EIA has time windows that must be achieved - if one is missed it has a six month delay implication	Community	Likely	Major	High: 21
14916	Archaeological discovery during the land purchase process or during Construction	Remains of Peruvian cultures in Project area	High cost and delay associated with relocating, salvaging or extracting the land affected	Site Conditions	Possible	Major	High: 18
14933	Delayed land rights (corridors, easements rather than mine site) due to disputes	Opportunistic land owners	Schedule delays; delayed finalization of base case scope	Permits and Approvals	Possible	Major	High: 18
14928	Geotech uncertainty around the TMF	Lack of test drilling; Presence of karst and limestone caving issues	Increased cost of managing the TMF due to construction complexities and/or the need to line the TMF	Engineering	Possible	Major	High: 18
15936	Hydrogeologic al model form TMF site may change as additional information from field test work becomes available	Hydrogeological drilling in abutment and surrounding areas yields unexpected results; Lack of field drilling during PFS due to permitting and site access issues	Design rework; Potential for additional cost; Potential revision of TMF design philosophy	Engineering	Possible	Major	High: 18
15865	Invasion of CMZ property	Publication of details regarding the project and land use	Delays, costs reputation	Community	Possible	Major	High: 18

ID	Name	Cause	Effect	Categories	Likelihood	Driving impact	Current rating
15976	Mine plan assumes the contractor will complete pre-stripping and initial three years of mining without passing on a new fleet purchase cost amortization	Assumes contractor will not charge for a new fleet over the life of the contract (four years)	If the Contractor includes the fleet cost the cost would rise above current budget	Budget	Possible	Major	High: 18
15864	Project fails to negotiate access to water within schedule and budget due to community opposition	Unrealistic expectations regarding compensation for water prevent a viable or timely deal; Pedregal de Majes mobilizes opposition to EIA approval;	Delay of overall project	Permits and Approvals	Possible	Major	High: 18
15081	Tia Maria Project protests developing anti-mining sentiment in the local region	On-going protest and NGO activity building mistrust and anti-mining sentiment	Increased exposure to delays associated with community engagement activities; Increased objections against permit applications; Potential targeting by NGOs	Community	Possible	Major	High: 18

25 Interpretation and Conclusions

25.1 Geology and Mineral Resources

25.1.1 Geology

- The Zafranal Property is underlain by highly deformed metamorphic rocks of indeterminate age and Mesozoic sedimentary and volcanic rocks, all intruded by large intrusive complexes emplaced during the Andean orogeny. Mineralization in the Zafranal Main Zone and Victoria deposits occurs within an east-west trending fault-bounded block associated to the regional Incapuquio fault zone and is cut by a complex series of reactivated faults. Primary copper mineralization in Zafranal is of late Cretaceous age.
- Porphyry copper-gold mineralization in the Zafranal Main Zone and Victoria deposits occurs as veins, stockworks and as disseminations, extending at least 3.3 km, up to 600 m width and depths of up to 500 metres, and is hosted mainly in Cretaceous aged diorite and microdiorite intrusive rocks. Primary copper mineralization is overlain by a laterally continuous secondary supergene enrichment blanket and by irregularly shaped zones with copper oxides.

25.1.2 Mineral resources

- The construction of the 2015 CMZ resource model has followed 2014 CIM definition standards for mineral resources and CIM estimation of mineral resources and mineral reserves best practice guidelines.
- The modeling and grade estimation process used is appropriate for a porphyry-style copper-gold deposit and the resource model is reasonable and is appropriate for the declaration of mineral resources to support mine planning at a pre-feasibility study level for a large-scale open pit mine.
- Other than the risks identified in this report, AMEC Foster Wheeler is not aware of any other environmental, permitting, legal, title, taxation, socio-economic or political factors that could materially affect the mineral resource estimate.

25.2 Mining and Mineral Reserves

- Open pit mineral reserves were constrained within an operative mine design based on a Lerchs-Grossman shell. The mineral reserves were defined through the application of a simultaneous 0.15%Cu grade cut-off and a value cut-off of US$5.72/t for feed to a concentrator for all supergene and mixed materials and US$6.00/t for hypogene material.
- A mine plan was developed for the Project to process a variable throughput of 55 kt/d to 64 kt/d. The mined material movement peaks at 75 Mt/y in Year 1. The production is limited by the number of benches that it is possible to mine in a single phase in a year, or the amount of vertical development per phase. Inferred mineral resources were treated as waste in the mine plan.
- NCL performed pit optimization and mine planning without introducing any additional factors to account for dilution, as the block model was considered to be a fully diluted resource model. NCL considered a 100% mining recovery to be appropriate due to the disseminated characteristics of the ore.
- Proven and probable mineral reserves total 401 Mt grading 0.40% Cu and 0.07 g/t Au.

- The main factors that may affect the Mineral Reserve estimates are metallurgical recoveries and operating costs (fuel, energy and labour). The base price of copper, even though it is the most important factor for revenue calculation, does not affect the mineral reserves estimate as a manual cut-off of 0.15% Cu is being considered, which is higher than the grade cut-off obtained from an economical evaluation. The selected grade cut-off allows for a broad swing in metal prices before they have an effect on the mineral reserves estimate.
- Other than the risks identified in this report, NCL is not aware of any other environmental, permitting, legal, title, taxation, socio-economic or political factors that could materially affect the Mineral Reserve estimate.
- Seven geotechnical design sectors have been defined for the Zafranal pit and four for the Victoria pit. Kinematic analyses of bench and inter-ramp slopes assumed a friction angle of 33° for minor structures (joints) and 30° for major structures. The results of the kinematic analyses indicate that variable inter-ramp slopes between 45° and 48° can be achieved. The exposure of fault zones along the upper pit walls would result in flatter slope angles.
- The results of the limit equilibrium analyses indicate that all design sectors for the Zafranal and Victoria pits are stable (assuming a blasting disturbance factor (D) of 0.85), except for two sectors, where the calculated factor of safety is below the acceptable minimum. Analysis indicates that the targeted factor of safety could be achieved when a lower disturbance factor (D) of 0.7 is applied; therefore, it is recommended to use good controlled blasting practices and implement an instrumentation installation and slope monitoring program.
- Mining of the Zafranal open pit will occur beneath the water table and require dewatering infrastructure to manage water during operations. Mining is expected to advance below the phreatic surface within 2 to 3 years. Groundwater inflow is expected to be low (on the order of 5 to 15 L/s). To increase slope stability and provide dry working conditions a pit dewatering and depressurization strategy has been developed. The Victoria pit is not expected to be mined below the water table.
- Management of surface water run-off outside the pit and around the waste dumps is summarized in Section 25.4.3.
- Eight pit phases are planned for the Mthe ne iranal i Mtg t thco

- The pre-feasibility study assumes that the mining operation will use 34 m^3 hydraulic excavators and trucks with a capacity of 220 t. The fleet will be complemented with drill rigs for ore and waste delineation. Auxiliary equipment will include track dozers, wheel dozers, motor graders and water trucks.

25.3 Metallurgy and Processing

25.3.1 Concentrator metallurgy

- Analysis of the comminution database including results from Phase 5 testwork has identified the following:
  - SMC and JKMRC DWI testing show hypogene and supergene to have low Axb parameters indicating high competence, but with hypogene higher than supergene.
  - No significant difference between hypogene and supergene ball mill work index (BWI) with both moderately hard.
  - Mixed ore is significantly less competent and less hard than hypogene and supergene but is a very minor proportion of total ore.
  - There are no significant differences between lithologies or alterations.
  - All ore types are amenable to comminution by primary crushing followed by a SAG mill and ball mill circuit with recycle pebble crushing.
- A copper recovery of 89% and a final concentrate grade of 28% Cu were used for design based on locked cycle tests results conducted in the Phase 3 metallurgical testwork program.
- The rougher flotation testwork indicates an optimum primary grind size P_{80} of 150 µm.
- A nominal P_{80} of 40 µm is the optimum regrind size.
- A combined rougher and scavenger circuit with three-stage cleaning at elevated pH, and a cleaner-scavenger, all in conventional flotation cells, is adequate to achieve target concentrate grades and recoveries.
- Elemental analysis of the copper concentrate indicates low penalty element levels.
- The concentrator has been designed to treat a maximum throughput of 2,915 t/h (63.7 kt/d) and a copper production rate of 20.5 t/h at 91.3% (8,000 h/y) availability, based on a maximum head grade of 1.01% Cu.
- The average of the variable throughput over the life of the concentrator is 59,300 dry metric tonnes per day (t/d) for 365 days per year at a copper head grade of 0.40%

25.3.2 Concentrator recovery and concentrate grade estimates

- From geometallurgical analysis, the following flotation domains were found useful for copper recovery prediction:
  - Low sulfur
    Lo_S, or low S/Fe ratio, for S/Fe<0.20
  - Low dilute sulfuric acid soluble copper
    Lo_CuSS, or low CuSS/CuT ratio, for CuSS/CuT<15%
  - Mid-range dilute sulfuric acid soluble copper
    Mid_CuSS, or mid CuSS/CuT ratio, for CuSS/CuT=15%-30%
  - High dilute sulfuric acid soluble copper
    Hi_CuSS, or low CuSS/CuT ratio, for CuSS/CuT≥30%
  - Low cyanide soluble copper
    Lo_CuCn, or low CuCn/CuT ratio, for CuCn/CuT<30%
- The recovery and concentrate grade models for each of the geometallurgical flotation domains are summarized in Table 25-1. For the overall copper recovery calculation, a cleaner stage copper recovery loss of 2% has been assumed, based on the locked cycle test work.

Table 25-1 – Recovery and grade model algorithms for geomet flotation domains

Domain name	Algorithm	Cu conc grade (%)	Au conc. grade (g/t)	Cu rec. final (%)	Au rec. final (%)
Lo_S	S/Fe < 0.2	0	0	0	0
Lo_CuCn	CuCn/CuT <= 0.3	28	by calculation	90.5	56
Lo_CuSS	CuSS/CuT <= 0.15	37		89	52
Mid_CuSS	0.15 < CuSS/CuT <= 0.3	34		84	55
Hi_CuSS	Rest	32		77	52

25.3.3 Concentrator metallurgy sample representativity

- All of the major rock types, regions and mineralization zones have been sampled and tested at an appropriate level for a PFS.
- Not all of the geometallurgical domains have an adequate number of samples, so more variability testing is recommended to reduce risk in the feasibility study, e.g.:
  - In Phase 5, supergene samples were over-represented in the flotation program, in particular the Mid_CuSS and Hi_CuSS samples.
  - No samples from Victoria zone were evaluated.
  - For further studies it is recommended to select more samples from mixed minzone and Years 1 to 5 of production for flotation testing.
- Further flotation studies should include sampling based on geometallurgical domains rather than lithology and alteration.
- Further comminution testwork should focus on rock quality data, fracturing, depth and weathering within the major minzones, rather than lithology and alteration.

25.3.4 Concentrator unit processes and facilities

The Zafranal process plant includes the following unit processes and main features:

- Primary gyratory crusher, crusher discharge conveyor and crushed ore transfer conveyor, the latter of which passes through a tunnel to the stockpile feed conveyor:
  - The total tunnel length, not including immediate portal structures is 3515 m, of which 688 m occurs in mylonite and/or gouge fill requiring steel sets and shotcrete fill for roof support. The remaining 2,827 m occurs in rock projected to be supported by bolting.
  - The first 2,265 m of tunnel geotechnical investigation is at prefeasibility study level but, due to the lack of drilling and sampling, the ground control design for the tunnel extension is based on geological information that is not sufficient for a pre-feasibility level capital cost estimate; however, in the context of the overall Project capital cost estimate it is not a material impact and can be mitigated by an appropriate cost allowance. Based on the available drill hole data and local geology, there is no significant likelihood of a potential fatal flaw.
- Open conical crushed ore stockpile with two reclaim tunnels and two reclaim feeders in each tunnel to feed two grinding lines
- Dual SABC grinding lines each consisting of a 16 MW SAG mill and an 8 MW ball mill with a recycle pebble crushing system
- Flotation comprising rougher flotation in nine 300 m^3 tank cells, three-stage cleaning and cleaner-scavenging comprising fourteen 100 m^3 tank cells and three 38 m^3 conventional flotation cells, and an M15000 IsaMill™ for rougher and cleaner-scavenger concentrate regrind
- Concentrate thickening in a 20 m diameter high rate thickener
- Concentrate filtration utilizing two horizontal plate pressure filters with dedicated air compressors
- Tailings thickening in a 62 m diameter high rate thickener.

25.3.5 Site-wide water balance

- A site-wide water balance was developed in GoldSim for the 55 kt/d fixed throughput case. The inflow of water to the TMF area is 451 L/s on average, of which 410 L/s (91%) is water from tailings, 15 L/s (3%) is runoff and 26 L/s is groundwater through-flow. TMF outputs are 453 L/s, of which 122 L/s (27%) is reclaimed and 331 L/s (73%) are losses. The main losses come from pore retention, 239 L/s (52% of losses); followed by evaporation, 87 L/s (19%), and infiltration/seepage, 6 L/s (1%).
- The site-wide water balance results indicate that of the total average water requirement of 362 L/s to the site, 84% is expected be supplied from off-site (abstracted from the Majes I wellfield), 14% is expected to be supplied from on-site water capture, both groundwater and runoff, and 2% comes as moisture in ore.
- For design purposes, and due to the uncertainty in this stage of study, it was recommended to use the 75th percentile for the nominal supply of raw water, and the 90th percentile as the maximum. At the 75th percentile, the monthly peak demand of raw water for the mine is 370 L/s, and at the 99th percentile it is 397 L/s (in Years 3 to 6).

- Due to parallel development of the site water balance and water supply engineering, the design capacity of the water supply exceeded the final water demand modelled for the original base case for the 99th percentile of the life-of-mine average by a sufficient margin to accommodate the increase in water demand pro-rated from the increased throughputs of the variable throughput case. On that basis it was considered unnecessary to update the site water balance for the variable throughput case for the prefeasibility study.
- Later comparisons for the peak demand year (Year 3) showed that while the pro-rated average flow is accommodated within the water supply design capacity, the 75th and higher percentiles are not. However, the exceedance, even for the 99th percentile, is within the precision of PFS capital cost estimates, and the main component of the increase in operating cost, power consumption, is incorporated in the operating cost.

25.4 Project Infrastructure

25.4.1 Waste dumps and mineral stockpiles

- Three waste rock storage areas, to be located to the north and south of the pits, were designed for the Project. According to Piteau geotechnical recommendation, the facilities were designed in lifts of a maximum 100 m height. Each lift will be constructed at an approximate angle of repose of 37°. A set-back between each lift will maintain the overall slope at 2:1 to facilitate reclamation and long term stability. A constant 2.0 t/m^3 loose density (after natural compaction) was assumed in the design.
- During the pre-production period, the ROM stockpile area will be constructed to the west of the initial pit for later re-handling to the primary crusher. The total ore to be stockpiled during this period amounts to 2.5 Mt. The low grade ore stockpile will be located to the west of the pit, in the same location of the initial ROM stockpile area. The oxide stockpile will be placed to the south west of the main pit, over a platform built on the central waste dump. The stockpiles are designed with 25 m lifts and minimum 17 m set-backs to facilitate later re-handling.
- A geochemical characterization program is on-going to assess the potential for acid rock drainage and metal leaching (ARD/ML) associated with wastes from the open pit. However median sulfur values are typically between 1% and 3%, indicating that open pit waste will be potentially acid generating (PAG) and that limited carbonate within the Zafranal rocks suggests that substantial buffering capacity or carbonate mineral dissolution from the waste rock is not expected.
- The surface water management system is designed to minimize contact water and to prevent contact water from entering the environment. This is achieved by routing clean, non-contact surface water runoff around disturbed areas and minimizing contact water and sediment discharge from the site. Diversion channels are incorporated in the design for this purpose around the north, northeast, and central waste dump areas, the mine workshop, stockpile areas and the open pit area.
- Dumps will be maintained with compacted surfaces to minimize any rain water or dust suppression water infiltrating into the dump. In addition, dump surfaces will be graded to shed rain water to shallow, lined evaporation ponds; water from these ponds will be used for dust control on the waste dumps or evaporated by natural methods. Collection ponds will be maintained downstream of the central waste dump and water quality will be monitored through laboratory testing. Water that is unsuitable for release to the environment will be used for on-site purposes during operations or evaporated. The volumes of contact water from this source are not expected to be high and are expected to be naturally evaporated so no provision for treatment has been allowed.

25.4.2 Tailings management facility

- The tailings management facility (TMF) design will be capable of managing the Probable Maximum Precipitation (PMP) and its consequent Probable Maximum Flood (PMF). The Maximum Credible Earthquake (MCE) has been used as the Maximum Design Earthquake (MDE) for the embankments. Stability analyses were carried out on the southwest and northwest embankment under both static and seismic loading conditions during operations and after closure.
- Geochemical testing by pHase-Geochemistry has indicated that the tailings are likely not to go acid during operation because lime is added to the process so the tailings discharge streams will be alkaline but are more likely to be potentially acid generating after closure of the facility. The PAG tailings stream is deemed to be suitable for production of cyclone sand fill material for embankment construction.
- The TMF location was selected after a trade-off study evaluated three alternative sites to the preferred site. In an earlier assessment of the preferred site, no seismic or geotechnical fatal flaws were identified; however, there are very complex hydrogeological conditions. A tailings disposal options trade-off study concluded that the preferred option was discharge of conventionally thickened tailings slurry to a minimally lined impoundment. Consideration of embankment raising methods and construction materials concluded that the centerline method using underflow from cycloning tailings as sand fill was most cost-effective while meeting design standards and statutory requirements.
- The thickened tailings (60% solids content) will flow by gravity pipelines from the concentrator to the TMF. A starter southwest embankment will be constructed from alluvium fill from within the impoundment prior to mine start-up and will provide approximately 1.5 years of tailings storage capacity. A starter northwest embankment will be constructed from locally borrowed fill material in Year 10. The upstream slopes of the starter embankments will be lined with a geomembrane to minimize the potential for seepage through the embankments.
- Instrumentation, such as piezometers and inclinometers, will be installed in the TMF embankments and foundation during construction of the starter embankments to monitor the performance of the embankments to assess their performance, such as internal pore water pressure, settlement, and deformation to detect any conditions different to those assumed during design and analysis. Amendments to the ongoing designs and/or remediation work can be implemented to respond to the changed conditions, should the need arise.
- Expansion of both embankments will utilize underflow sand from cyclones fed by tailings from the concentrator, diluted with water from the supernatant pond in the impoundment. Production of sand from a cyclone plant for the southwest embankment must commence immediately following start-up to meet the required raising rate. A second cyclone plant will be constructed to support the raise of the northwest embankment.
- Two streams of tailings will discharge into the TMF from points around the perimeter: whole thickened tailings directly from the concentrator (60% solids) and cyclone overflow slimes (29% solids content). The discharge points will be managed to provide deposition of a lower permeability barrier over the highly permeable alluvium base and southwest hillside of the impoundment to reduce seepage from the impoundment into the underlying alluvium and host rock. Whole tailings will be discharged on top of the slimes
- Reclaimed water from the decant pond is recovered by pontoon-mounted pumps and, together with recovered seepage water, reports to the return water pumping station. A controlled portion of this water is gravity fed to the tailings cyclone dilution water tank and the rest is pumped back to the process water tank for re-use in the plant.

- There are several design measures to reduce and collect seepage from the TMF. The numerical groundwater flow model indicates that uncollected seepage would be <1 L/s with the following seepage collection systems:
  - A low permeability geomembrane on the upstream face of the starter embankments.
  - A primary seepage collection system consisting of a network of French drains within the foundation of the impoundment and embankments.
  - A secondary seepage collection system consisting of a tunnel (gallery) and fanned network of drainage holes within Quebrada Huacan. The drain holes will intercept high angle, permeable fault zones aligned along Quebrada Huacan and high permeability fractured bedrock located within the right abutment of the TMF. This secondary seepage collection system would also provide an additional water supply source for construction.
  - A tertiary seepage collection system consisting of vertical pumping wells strategically located within alluvium down-gradient of the dams and along permeable fractured bedrock.
- Underdrain discharge from the TMF embankments are collected in seepage ponds and pumped back into the TMF.
- The primary objective of reclamation and closure activities for the TMF will be to ensure physical and chemical stability of the TMF by establishing an environmentally and physically stable facility with a landscape and habitat consistent with adjacent areas. In this environment and at this location the focus will be on stabilizing 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 emergency spillway located next to the northwest embankment.

25.4.3 Surface water management

- The developed analyses identified dependence on altitude of precipitation to areas exceeding 1,200 masl along with identifying that the average annual rainfall is approximately 5 mm below 1,200 metres.
- Annual rainfall frequency analysis was developed for wet and dry years for 5 to 500 year return periods along with extreme runoff event flows for those same return periods to develop surface water management systems for site facilities.
- Surface flows in the area of the mine are ephemeral, of the order of 10 days of runoff per year. However, unusual events of short duration with high peak flows occur.
- The concept of the management of surface water for the project is to capture the ephemeral flows as much as possible to reduce the water supply requirement for operations and to protect facilities by safely diverting runoff from extreme events.

25.4.4 Product transport

- Concentrate will be transported from the Zafranal mine site to the Port of Matarani using a standard combination of tandem drive tractor articulated with a tipper semi-trailer. The trucks will be equipped with pneumatic suspension and extra wide tires to carry a legal gross weight of 58.2 tonnes and a concentrate payload of 35 tonnes. This will require special permitting under the national regulations to allow use on the public highway system.

- The corridor for concentrate transportation will follow a western route from the site which follows the existing road west to Corire and then south along the eastern side of the Rio Camaná valley. The haulage road then follows south from Mesana along Highway PE 1SG1 to Morro Sihuas and Highway PE 1S1. From Morro Sihuas, the transport route follows to Camaná and then along the coast on the new Camaná – Division Quilca - Matarani Highway (PE 1SD) to the Port of Matarani.

25.4.5 Access roads

- The site access roads from Anexo El Pedregal (Corire) and El Pedregal de Majes (Majes) have been clearly defined and captured in the Project capex. The internal access roads are also defined and sufficient for the project requirements.

25.4.6 Power supply

- The power supply system is based on a total installed power requirement of 99 MW and peak demand of 91 MW.
- The alternative of connection to the New Socabaya substation represents the longer length of transmission line (98 km); however, its connection point is guaranteed with the construction of the New Socabaya substation managed by Red de Energía del Perú S.A. (REP), with its operational start-up being expected by the end of 2016.
- The connection of Zafranal to the 220 kV New Socabaya substation will not affect the national grid system operation because the voltage levels are within the allowed range.
- The other national transmission lines and transformers are not affected by Zafranal because their use would be below carrying capacity.

25.4.7 Water supply

- Two groundwater supply sources close to the mine site have been identified to supply the construction start-up water for Zafranal. The water requirement for start-up is estimated to be 20 L/s. Confirmation of the aquifer properties and optimization of the groundwater recovery system will be carried out during the feasibility level studies.
- A brackish groundwater resource located beneath the Majes I irrigation area and 35 km from the process plant site was identified as the preferred make-up water source. The quality of the water is poor and not suitable for irrigation given its very high salinity hazard and high alkalinization potential. The estimated volume of stored brackish groundwater is approximately 3 billion m^3, which is at least ten times greater than the total life of mine water requirement volume.
- A wellfield, consisting of nine wells located within the northwest agricultural sector of Majes I was designed assuming a make-up water requirement of 410 L/s. Groundwater extraction will require the approval of the water authorities and the acceptance of the local authorities and community residents.
- After route trade-off studies, a predominantly above-ground water pipeline in carbon steel lined with fusion bonded epoxy (FBE) from the wellfield collection tank at Majes I to the Zafranal has been engineered and costed.
- Social impacts and community relations risks associated with the water supply system are discussed in Section 25.5.1.

25.4.8 Site accommodation

- The strategy for site accommodation is to have two camps, one that will be provided by the contract mining company and the second to accommodate the short-term construction phase and long-term operations phase requirements of the project.

25.4.9 General and administration facilities and utilities

- The strategy for general and administration facilities has been to use modular portable buildings for office facilities and tent-style (tensioned fabric) structures for warehousing. The utilities to support these facilities are also well defined and consolidated between the concentrator, administration, and camp.

25.5 Environmental and Social

25.5.1 Design considerations

From the evaluation of baseline conditions and early identification of potential risks the design considerations in the following sub-sections should be included in further development of the Project.

25.5.1.1 Water controls

One of the key risks is perceptions of the local community around the use and potential effect on water that could be used for agricultural purposes. This is one of the principal risks related to the Project and could be used by third parties to lever sentiment against the Project. The permanent water source for the Project should be from a source that will minimize this risk; the Majes I brackish groundwater resource could meet this criteria. Studies to date indicate that the groundwater under Majes I is not suitable for agriculture or drinking water and would not be a likely source of competition between the Project and community requirements.

Contact water and seepage from the Project must be contained during operations and adequately treated for closure to avoid the risk of affecting surface water quality in the Majes River or the perception that the surface and/or groundwater that drains to the Majes River is affected by Project mining activities. Further studies are required downstream of the tailings storage facility to be able to appropriately characterize the groundwater flows and potential to intercept seepage from this facility.

Preliminary modeling of the open pit indicates that there is a potential for a pit lake to be formed post closure of the Zafranal pit. This is not expected to occur with the Victoria pit. The pit lake is anticipated to reach steady state conditions at 100 m depth around 100 years post closure. Closure planning must take into account the potential for pit lake formation and seepage collection from the lake for perceptions of the Majes River communities regarding potential effects on the river water quality to be addressed.

25.5.1.2 Dust controls

Although the populations surrounding the Project are at a sufficient distance from the Project facilities to not be affected by dust dispersion and the surrounding topography is expected to precipitate fallout of any dust content in the wind before reaching the local communities, this is still considered a concern by the local population; for the potential to affect human health but also potential effects to pasturelands and agricultural crops. The design of the crushing and milling circuits must include dust control measures. These will also mitigate potential health risks from dust inhalation for the Project workforce. Maintaining an adequate level of moisture on the tailings surface through managed deposition would help mitigate wind erosion off that surface during operations. Injecting a concreting agent into the tailings stream in the final depositional layers in the TMF would aid in reducing the potential of wind erosion from the final tailings surface and would allow for the activity of heavy machinery in the selective placement of alluvial materials on the surface of the final TMF.

25.5.1.3 Management of sensitive vegetation

There are a number of species of plants of conservation concern that will need to be evaluated further as part of the EIA process. Management of these species will involve collection of seeds, asexual propagation of plants and/or transplanting of individuals. The specific measures will be defined in the EIA.

25.5.1.4 Management of local fauna

There is a local population of guanacos that is of interest to the local residents, as well as being a protected species at the national and international level. During the development of the EIA and the feasibility study design, measures to avoid Project impacts to this species and the other species of fauna of ecological interest need to be investigated. Studies from nearby mining operations with similar conditions and fauna present may be used to guide the design of the Zafranal Project. A specific management plan will be developed for guanacos and included in the EIA.

25.5.1.5 Archaeology

There are archaeological remains of local interest in the vicinity of the Project. While the location of these remains has been included in the design of the Project at this preliminary feasibility level, with the objective of avoiding them, this criterion must be continued through the remaining design stages of the Zafranal Project.

25.5.2 Stakeholder engagement

All stakeholders in the direct area of influence have been mapped at this time and stakeholder mapping is updated regularly. Stakeholder mapping in the indirect area of influence has also been completed. It is important to maintain at least the current level of engagement with all stakeholders to avoid disruption of the Project by any potential local resistance.

Additionally, there are groups acting at the national level with objectives that do not coincide with the development of mining or other extractive activities. These groups look for opportunities to derail extractive industry projects through well-timed misinformation. To manage this risk it is important to keep the local stakeholders informed of Project activities and engaged in a meaningful manner. Keeping the local stakeholders well-informed and meaningfully engaged will reduce the likelihood of external groups gaining support in the local environment.

Attention must also be given to stakeholders in the indirect area of influence, since if not appropriately engaged they could feel marginalized by the Project. If this occurs, there is a risk that these stakeholders may seek opportunities (real or perceived) to benefit from the Project.

As part of the stakeholder engagement, the Project will implement a series of public workshops in support of the EIA. These workshops will include the official public workshops coordinated with the MINEM and SENACE but will also include a series of voluntary information spaces.

25.5.3 Social responsibility

Social responsibility is being managed through a series of programs and plans:

- Local employment program
- Local supply and service program
- Social investment program
- Communications plan and programs

- Social commitment monitoring program
- Program for attention to grievances
- Program for attention to requests for donations
- Social climate monitoring and risk prevention program
- Social crisis management program.

25.6 Capital and Operating Costs

25.6.1 Capital costs

The capital cost of the project over a three-year construction and pre-mining period, and sustaining expenditures over the 19 year mine life, have been estimated. The following basic data pertains to the estimate:

- The base date is 1st quarter 2016
- All costs are expressed in United States dollars (USD or US$)
- The estimate is at pre-feasibility precision ±20-25%
- No allowance has been made in the estimate for escalation from the base date or changes in currency exchange rates.
- All import duties and taxes are excluded from the estimate (not expected to apply).
- Each element of the estimate is developed initially as a bare cost only. A growth allowance has then been allocated to each element of the costs to reflect the level of definition in pricing and engineering maturity relating to that element.
- Estimate contingency is included to address anticipated variances between the specific items contained in the estimate and the final actual project cost assessed as a percentage of direct and indirect costs to arrive at a total project estimate with the required confidence level.
- The following items are excluded from the capital cost estimate:
  - working capital
  - replacement capital other than in sustaining capital
  - finance charges
  - residual value of temporary equipment and facilities
  - residual value of any redundant equipment
  - cost of any downtime
  - environmental approvals
  - legal costs
  - any further studies
  - force majeure issues
  - future scope changes
  - special incentives (schedule, safety or others)
  - allowance for loss of productivity and/or disruption due to religious, social and/or cultural activities
  - risk management reserve (Owner's contingency).

The capital cost estimate has been summarized at the levels indicated in Table 25-2

Table 25-2 – WBS level one cost summary

WBS LVL 1	LVL 1 Description	Total $M USD	Total Labour ,000 hours
Initial capital		**1,157**	**14,048**
	Initial direct cost	**703**	**13,998**
1000	Mine	142	168
2000	Concentrator	430	12,016
4000	On-site infrastructure utilities and facilities	33	362
5000	Off-site infrastructure utilities and facilities	98	1,452
	Initial indirect cost	**454**	**50**
6000	Project preliminaries	103	0
7000	Indirect costs (EPCM, vendors, commissioning, spares, etc.)	132	50
8000	Provisions	151	0
9000	Owner's Costs	68	0
Sustaining capital		**263**	**345**
	Sustaining direct cost	**228**	**345**
1000	Mine	213	69
2000	Concentrator	15	267
4000	On-site infrastructure utilities and facilities	1	9
	Sustaining indirect cost	**34**	**0**
8000	Provisions	34	0
Grand Total		1,420	**14,393**

25.6.2 Operating costs

The operating cost of the project has been estimated on the same basis as the capital cost estimate except that there is no growth allowance, estimate contingency or Owner's contingency included. A summary of the average production period operating costs is shown in Table 25-3.

Table 25-3 – Summary of average production period operating costs

Description	Production period average	
	$/t ore	**($/lb Cu produced)**
Mining	4.25	0.54
Processing	4.58	0.59
General and administration	1.22	0.16
Concentrate road transport	0.45	0.06
Total Cost	**10.50**	**1.35**

Operating costs vary with time according to total material mined and concentrator throughput. There is also some minor variation related to concentrate output. These are reflected in the economic analysis.

25.7 Economic Analysis

The Project has an NPV@8% of US$496M, an IRR of 16% and a payback period of 5.1 years from the start of production. This valuation assumes life of mine average prices of US$3.00/lb Cu, US$1,200/oz Au. Mid-range values for copper and gold head grades and recoveries, operating costs and capital expenditures are used in the valuation.

The valuation model fully incorporates the Peruvian tax regime which includes royalties, special mining tax, 18% value added tax (i.e. Impuesto General Ventas or IGV), 8% Workers' profit sharing and income tax.

The valuation was undertaken both deterministically and probabilistically (i.e. Monte Carlo simulations). Range analyses were carried out to determine the possible range of values for the key variables. The results of probabilistic analysis revealed a 91% probability of a positive NPV.

The results of the economic analysis in general underline that robustness of the investment across a range of various scenarios.

CMZ does not currently have a tax stability agreement and this is reflected in valuation results. During the feasibility study, CMZ will analyze the requirements of and the potential benefits under a tax stability agreement with the Peruvian government.

25.8 Other Relevant Data and Information

25.8.1 Project implementation plan

The implementation strategy is based on an organization with an Owner's project execution team and an EPCM contractor both reporting to a Project Director. The team also includes a Vice President Operations, whose role is to manage pre-production mining activities and prepare for operations, and administrative VPs, whose primary roles are to manage the off-site service and support functions that will support project execution and facilitate the transition from construction to operations. The Project Director, VP Operations and the administrative VPs all report to the CMZ President, who has ultimate accountability for delivery of the project.

The key project milestones are summarized in Table 25-4:

Table 25-4 – Key project milestones

Deliverable	Revision 0 End Date
PFS complete	Q2 2016
Gate 2 approval – start FS	Q4 2016
FS complete	Q4 2017
EIA detailed	Q3 2018
Gate 3: approval - FS and EIA	Q4 2018
Board approval project sanction: early funding	Q4 2018
Financing Phases 2 and 3; board approval project sanction: construction	Q3 2019
Beneficiation construction authorization	Q1 2019
Detailed engineering complete	Q1 2019
Construction concentrator complete	Q2 2021
Pre-stripping complete	Q2 2021
First ore	Q4 2021

The critical path for the Project includes the following activities:

- Detailed EIA– submission and approval
- Early funding finance after EIA approval
- Early works (contractor mobilization, camp construction, access roads)
- Concentrator and infrastructure construction
- Ore commissioning.

25.8.2 Operations management plan

CMZ will have three administration centres in Peru, one at the Project site, one in Arequipa and a small office in Lima. The Vice President of Operations will be located at the Project site, reporting to the President, and be responsible for the health, safety and performance of the personnel on site including mining, concentrator, technical services; health, safety and environment; as well as on-site representatives of human resources, purchasing and warehouse, camp administration, labour relations, security, and cost accounting. The President will be located in Arequipa and be the legal representative for the Company with responsibilities including overall Company performance, external affairs (including community relations), human resources, finance and accounting. The Finance Manager will be located in Lima, reporting to the President, and will be responsible for information technology, marketing, procurement, treasury and finance activities.

CMZ has an established general and administrative (G&A) and field workforce with several years of service and corporate history. While not all current employees will be retained, the basic structure will remain in place and serve as a solid starting point for building the permanent workforce. Additional CMZ labour will be recruited primarily from Caylloma Province, Castilla Province, Arequipa, and Lima.

25.8.3 Risk management

The Zafranal Project Pre-feasibility Study (PFS) has been subject to proactive risk management from July 2014. The key risks (rated as “high”) to the Project at the end of the PFS phase fall into the three categories described below.

Community, access and approvals refer to the social and community relations component of the Project, which includes most of the current high risks to the Project. This needs to be actively managed by the Owner with guidance from suitably qualified consultants to manage the risks that have been identified and those that might manifest through the life of the Project.

Technical risk associated with site conditions refers to the inability to gain timely access to some field sites in the area of the TMF during the PFS phase which has led to the possibility that geotechnical and hydrogeological modeling may change as more information becomes available. These risks are also associated with the community, access and approvals risks above. An archaeological discovery during the land purchase process or during construction could also affect the Project schedule.

Commercial and budget risk refers to the assumption that the contractor will complete pre-stripping and the initial three years of mining without passing on a new fleet purchase amortization, which is based on the assumption that the contractor will not charge for a new fleet over the six year life of contract, due to the contractors having large current fleets paid off on other projects.

25.9 Overall

The overall conclusion from the PFS is that the Project is suitable to progress to the next phase of investigation and evaluation.

26 Recommendations

The next phase of Project development includes the feasibility study and the elaboration of the detailed Environmental Impact Assessment. Some feasibility-level field investigations and engineering will be required to commence early to provide the necessary inputs to the EIA, which is on the critical path of the project execution schedule, and then integrate with the FS. Hence the FS and EIA are considered as an integrated next phase. Typical expenditure for the FS phase, including EIA, is 2-5% of project capital cost, in this case US$30-70M.

The subsequent phase of project development is the implementation or execution phase, the cost of which is the initial capital cost estimate presented in Section 21.

Where the recommendations for a particular area or discipline would do no more than detail the standard requirements of an FS or EIA, they have generally not been presented.

26.1 Geology and Mineral Resources

26.1.1 Geology

- Exploration work outside of the existing Main zone and Victoria deposits should be ramped up, including detailed mapping and sampling focusing on altered areas along the favourable structural corridor. Initial diamond drilling is recommended on the Campanero (6 drill holes) and Zafranalito (3 drill holes) targets already identified by previous exploration work. Two diamond drill holes should also test the southern geochemical anomaly at Moly Sur. The overall 2016-2017 recommended drilling program totals 3,000 metres and the overall cost is estimated to be US$1,013,000.

26.1.2 Mineral resources

- The CuT grade estimate is over-smooth at the resource model block scale; HERCO change-of-support grade-tonnage curves indicate that the model over-predicts tonnage by 6% and under-predicts grade by 5% relative to an ideal distribution at a grade cut-off of 0.15 CuT%. There may be an opportunity to increase net present value (NPV) and cash flows in the future by mining a lesser tonnage at a higher average grade above cut-off keeping the contained copper content essentially constant. For future resource updates, the modellers should optimize the kriging parameters, introduce lithological control, or use local uniform conditioning to develop a recoverable mineral resource model.

- The soluble copper, CuSS and CuCn models are less well developed than the CuT model as a result of limited sampling and assaying for soluble copper. These models are reasonable in the supergene zone. For future resource updates, additional assays should be made available and the CuS model developed from the composite sample (CuSS+CuCn)/CuT ratio and applied to the full CuT model as the preferred methodology.

- The decision to develop estimation domains from mineralization zones (minzone) and lithology is appropriate. The highest impact on the reduction of cross-validation (CV) errors occurs by using minzone-based domains. There still exist low to moderate differences in means and CVs between lithologies within estimation domains and, for future resource updates, study and consideration should be given to instituting lithological control within minzones in the grade estimation.

- Examination of the cross-sections shows that the spacing at which continuity of grade is confirmed in the supergene domain can vary from 50 to 100 m. For the next level of study, conditional simulation is recommended to establish the recommended drill spacing to declare measured mineral resources on a local basis. In-fill drilling should be planned to increase confidence, so that at least 80% of the mineralization to be mined in the first five years of production, could support a measured classification.

- Appropriate measures should be implemented to ensure data integrity that should include use of proper database management software to ensure the integrity of the database. It is understood that the selection process for a suitable data management solution is underway.

26.2 Mining and Mineral Reserves

To achieve feasibility level mineral reserves and mine design the following are recommended:

- Geotechnical update by additional drilling, sampling, analysis and design, targeting eight years of mine life. Explore pit slope optimization, considering double benching for fresh rock material.
- Complete geotechnical and hydrogeological studies with a focus on dewatering and pit wall depressurization.
- Complete geochemical characterization of waste material to establish a dumping plan to minimize the potential of acid rock drainage during the mine life and at mine closure.
- Develop a monthly mine schedule based on blasting polygons and loading equipment productivity to ensure the concentrator feed during the payback period.
- Improve characterization of material hardness for mining throughput studies (results from next stage geotechnical and in-fill drilling programs).
- Validate density and moisture content.
- Throughput rationalization study update to confirm the selected option.
- Blasting study to characterize drill and blast patterns, powder factors and crusher feed size.
- Update the mining costs model with equipment and contractor cost quotes to confirm the mining operations strategy. Contractor quotes should be in the form of a frame contract.
- Additional geotechnical drilling, test pits and laboratory testing of resulting samples to investigate and confirm foundation conditions for the mine support facilities and utilities.
- Benchmarking of key mining indices from operating mines in Peru.
- Risk analysis.

26.3 Metallurgy and Processing

- Conduct further comminution test work to confirm comminution characterization of ore types and determine potential gains in concentrator throughput and assess potential opex and capex savings.
- Analyse likely feed size distribution based on drill and blast modelling by the mine.
- Further laboratory flotation test work, including locked cycle tests and variability tests using Majes I brackish groundwater, is recommended to confirm copper recoveries and concentrate grades in the rougher and cleaner flotation circuits and overall flotation circuit.
- Include samples from the Victoria zone in the next phase of metallurgical test work.

- Progress geometallurgical analysis, in particular weathering and oxidation profiles and further developing their relationship with flotation copper recovery by selecting samples for flotation testing by geomet flotation domain.
- Conduct transportable moisture limit (TML) and auto-ignition test work of copper concentrate to confirm benchmark data used in process design criteria.
- Additional geotechnical drilling and test pits and laboratory testing of resulting samples are required to investigate and confirm foundation conditions for the concentrator areas, in particular the crushed ore transfer conveyor tunnel and directly under significant dynamic loads such as the primary crusher and grinding mills and significant static loads such as the crushed ore stockpile and tailings thickener.
- The crushed ore transfer conveyor tunnel's specific geotechnical recommendations are:
  - Perform an over-coring test to measure horizontal stress in two of the first 2,265 m of tunnel and two in the bulk mylonite zone in the extension. High horizontal stress cannot be ruled out until this occurs.
  - Drill a horizontal geotechnical core hole through the alignment of the tunnel at a bearing and dip in the centre of the tunnel and tunnel extension. This will give specific information for feasibility–level and detailed engineering-level design. The hole will also give specific information on fault intercepts.
  - Drill four geotechnical holes to delineate the mylonite and granodiorite zones in the tunnel extension for characteristic and strength variation vertically and horizontally.
  - Perform physical strength testing on the major rock types of the tunnel extension.

26.4 Project Infrastructure

26.4.1 Waste dumps and mineral stockpiles

To achieve feasibility study level design of the waste dumps and mineral stockpiles, the following are recommended:

- Geotechnical investigations and laboratory test work:
  - Additional geotechnical and hydrogeological drilling and test pits in the waste dumps (especially the northeast waste dump) and stockpiles area to investigate and confirm foundation conditions
  - Additional laboratory index, and strength, permeability tests on waste and stockpile materials along with foundation materials.
- Design studies and analyses:
  - Perform a formal seismic risk assessment to develop design earthquake criteria for various facilities (including the open pit and waste dumps). As part of this risk assessment, identified faults within the area of the project will be further investigated to determine potential activity.
  - Stability analyses for the waste dumps and stockpiles, including static, pseudo-static, and deformation analyses.

26.4.2 Tailings management facility

To achieve feasibility study level design of the TMF, the following are recommended:

- Geotechnical investigations and laboratory test work:
  - Additional geotechnical and hydrogeological drilling and test pits in the TMF 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 field mapping and testing of carbonate rock in the south wall of the TMF to verify that karstification is not an issue.
  - Additional laboratory index testing (including compaction tests) and strength and permeability tests on potential borrow materials for embankment construction.
  - Laboratory testing to confirm the physical characteristics of the cyclone materials (material classification, strength and permeability tests should include strength and permeability tests at very high confining stresses, representative of the large height of the main embankment, and testing to examine the influence of cyclone sand fines content on available percentage of sand recovery and permeability and consolidation testing of the different tailings streams to determine settlement and permeability characteristics, and beach slope testing.
  - Examination of the influence of cyclone underflow solids content on the compaction characteristics of in situ cyclone sand fill. Large scale generation of a cyclone sand sample (using a laboratory cyclone circuit) is recommended to provide sufficient quantity of representative sand material for laboratory testing.
- Design studies and analyses:
  - Confirmation of geochemical characterisation and potential reactivity of tailings and from on-going waste characterisation 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.
  - Assessment of the sequencing of sand cell construction in relation to embankment raising requirements to optimise material demand and placement strategies.
  - 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.
  - Perform a formal seismic risk assessment to develop design earthquake criteria for various facilities (especially the TMF). As part of the risk assessment, identified faults within the area of the project will be further investigated determine potential activity.
  - Stability analyses for the TMF, including static and pseudo-static. Perform a rigorous deformation analysis for the TMF using FLAC or similar software along with stability analyses for the TMF.
  - Perform a failure mode and effect analysis (FMEA) on the TMF 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 TMF. This will crust the top of the TMF and allow for the placement of the alluvium with machinery as well as reducing the potential for aeolian erosion and capital costs.

26.4.3 Hydrology and surface water management

26.4.3.1 Hydrology

To achieve feasibility study level hydrology inputs, the following are recommended:

- Carry out a quality control program for each weather station at Zafranal on a monthly basis to verify its operation.
- Install at least one more weather station located at an altitude similar to the pit.
- Perform lysimeter testing to evaluate the infiltration depth due to the presence of fog in the area of the Project and refine the recharge calculation.
- Perform infiltration testing to evaluate the soil moisture to refine the calculation of recharge.
- Carry out inspections to the streams in the area of the Project with emphasis on the Cachimayo and Lloquelloy rivers during successive days of rain. These inspections may be performed by CMZ as part of its routine inspections. A photographic record of the events should be taken and flow rates calculated by measuring the width and height of the flow along with speed. The measurements and registration of these events are to refine the estimate of surface runoff.

26.4.3.2 Surface 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 TMF to refine on-site control measures.
- The designs proposed for surface water management have been carried out on topography with 5 m spacing and in some areas using National Charter. 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.

26.4.4 Power supply

- The Zafranal project should be formally presented to the Comité de Operación Económica del Sistema, (COES), so that the Project's power requirements are considered in future analysis and planning of transmission infrastructure.
- Progress is required towards a contract for power pricing, preferably to be agreed by the end of the feasibility study, subject to the Project approval.

26.4.5 Water supply

- Modelling of the water balance should be considered for Years -1 (pre-stripping), 1, 3, 5, 10, 15 and end of mine life. For the next phases of the Project, revision of the site water balance model for the variable throughput case should be a high priority.
- Confirmation of the aquifer properties and optimization of the groundwater recovery system and water supply source for construction are required during the feasibility level studies.
- Feasibility level hydrogeological studies are recommended which will include drilling and installation of additional pumping wells and piezometers throughout the Majes I irrigation area. Long-term pumping tests should be competed to confirm the conceptual and numerical models. The results of these analyses will be used to optimize the wellfield design.
- A risk assessment on keeping the above ground pipeline design should be carried out and the cost associated with additional risks compared to buried pipeline incorporated into the decision matrix.

- A stress analysis of the pipeline is recommended for the next phase, especially if an above ground design will be still considered for FS engineering, to locate and design supports and anchoring structures according to the results.
- Treatment of the Majes I aquifer water for potable water use in the Project would require reverse osmosis. A test program for this should be included in the next stage of Project development.

26.5 Environmental and Social

- The Project should continue with the Environmental Impact Assessment process. Additional interruption in the EIA process would potentially affect the credibility of the Project team with the area stakeholders. The focus of the Project facilities design should be to cover the risks and considerations identified in Section 25.5.2 and should be carried through to the feasibility level for inclusion in the EIA.
- The EIA baseline work is currently being developed to meet the requirements of the national regulatory agencies (MINEM and SENACE) and also to meet the requirements of the IFC Performance Standards and Equator Principles. The preparation of the EIA should also be designed to meet the international as well as national standards.
- Further geochemical, hydrological and hydrogeological studies are required to evaluate the potential for pit lake formation and seepage from the pit after closure.
- An annual census of guanacos in the Project area is also recommended.
- The Working Table for the Sustainable Development of the Huancarqui and Lluta Districts that has been initiated should be maintained as means of securing social acceptance for the project, managing expectations and to distribute project information to the stakeholders.
- Other current measures to engage the local stakeholders in a meaningful manner must be continued and increased where needed, as identified through continuous evaluation.
- The specific programs and plans identified in Section 25.5.3 need to be implemented for social acceptance of the Project to be solidified.

27 References

27.1 Summary

No reference specified for this section.

27.2 Introduction

No reference specified for this section.

27.3 Reliance on Other Experts

No reference specified for this section.

27.4 Property Description and Location

No reference specified for this section.

27.5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

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27.6 History

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27.7 Geological Setting and Mineralization

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27.9 Exploration (excluding drilling)

27.9.1 Manfrino, A., Harbort, G., & McCrea, J. (2010). Technical Report, Zafranal Copper Project.

27.10 Drilling

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27.10.2 Bruna Novillo, J. (2014). Geotechnical Logging Protocol Zafranal_2014. Department of Geology CMZ. Politax S.A. August, 2014. Internal report. Page 1-17.

27.10.3 Bruna Novillo, J. (2015). MemoCMZ002_2015_Re_Logueo_MGeo. Department of Geology CMZ. Politax S.A. February 20, 2015. Internal report. Page 1-20.

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27.12 Data Verification

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27.13 Mineral Processing and Metallurgical Testing

No reference specified for this section.

27.14 Mineral Resource Estimates

No reference specified for this section

27.15 Mineral Reserve Estimates

27.15.1 Knight Piésold; Zafranal and Victoria Open Pit Slope Design; Prefeasibility Study; Final Report; 27 May 2013.

27.15.2 Piteau Associates, Memorandum, Zafranal Project – Review of Knight Piésold Prefeasibility Study; 26 June 2014.

27.15.3 Piteau Associates, Memorandum, Review of Prefeasibility-Level (PFS) Requirements for the Zafranal Project; 1 May 2015.

27.16 Mining Methods

27.16.1 Piteau Associates, Memorandum, Zafranal Project – Review of Knight Piésold Prefeasibility Study; 26 June 2014.

27.16.2 Piteau Associates, Memorandum, Review of Prefeasibility-Level (PFS) Requirements for the Zafranal Project; 1 May 2015.

27.16.3 Knight Piésold; Zafranal and Victoria Open Pit Slope Design; Prefeasibility Study; Final Report; 27 May 2013.

27.17 Recovery Methods

No reference specified for this section

27.18 Project Infrastructure

No reference specified for this section

27.19 Market Studies and Contracts

No reference specified for this section.

27.20 Environmental Studies, Permitting and Social or Community Impact

No reference specified for this section.

27.21 Capital and Operating Costs

No reference specified for this section

27.22 Economic Analysis

No reference specified for this section.

27.23 Adjacent Properties

No reference specified for this section.

27.24 Other Relevant Data and Information

No reference specified for this section.

27.25 Interpretation and Conclusions

No reference specified for this section.

27.26 Recommendations

No reference specified for this section.

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28 Qualified Person Certificates & Consents

I Alvaro Fernandez-Baca, QP for deposit type, exploration history, geology, mineralization, structure and alteration and exploration outside of the main bodies, certify that

I am Chief Geologist at Barracuda Exploraciones S.A.C, Calle Kenko 188, Santiago de Surco, Lima 33, Perú.

This certificate applies to the Technical Report titled Technical Report on the Pre-Feasibility Study for the Zafranal Project, Peru, dated 10 June 2016.

My qualifications and relevant experiences are that:

1. I graduated with a Bachelor of Science (Honours) degree from the University of Edinburgh in 1996
2. I am a member of the Association of Professional Engineers and Geoscientists of British Columbia
3. I have worked as an exploration geologist for a total of 19 years primarily on epithermal gold and porphyry copper-gold deposits and belts, including the Heruga copper-gold porphyry deposit in Mongolia, the Paleocene copper porphyry belt in southern Peru, the Miocene copper-gold porphyry belt in northern Peru, the copper porphyry belt of northern Chile and northern Argentina, the Virgen epithermal gold deposit in northern Peru and the Sinchao/Antakori copper-gold skarn-porphyry deposit in northern Peru.
4. I have read the definition of Qualified Person set out in Nation Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101 for those parts of the Technical Report for which I am responsible.
5. I have visited the Property for a total of 3 days in November 2015 as well as extensively before 2013.
6. I am responsible for the preparation of sections 6.2, 6.3, 7, 8, 9, and 10.6 of the Technical Report.
7. I am independent of the issuer per Section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. I was the Vice President of Exploration and General Manager for AQM Copper Peru between 2009 and 2012, during which time I was in charge of all exploration efforts in the property.
9. I have read National Instrument 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
10. As of the date of the certificate, to the best of my knowledge, information and belief, those parts of the Technical Report for which I am responsible contain all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: July 7th, 2016

Signature:

Name: Alvaro Fernandez-Baca

PROFESSIONAL
PROVINCE OF
#143115
BRITISH COLUMBIA
GEOSCIENTIST

I Andrew P. Schissler, QP for geotechnical design of the crushed ore transfer conveyor tunnel, certify that

I am Global Lead: Mining Engineering at Ausenco Inc., Vancouver, Canada, and Lima, Peru.

This certificate applies to the Technical Report titled Technical Report on the Pre-Feasibility Study for the Zafranal Project, Peru, with an effective date of 10 June 2016.

My qualifications and relevant experiences are that:

1. I graduated with a B.Sc. in Mining Engineering from The Colorado School of Mines, May, 1975; and a PhD in Mining and Earth Systems Engineering from The Colorado School of Mines Dec. 2002.
2. I am a Founding Registered Member member of the Society for Mining, Metallurgy, and Exploration Inc. I am a licensed Professional Engineer in the States of Utah and Colorado.
3. I have worked as a Mining Geotechnical Engineer for a total of 24 years, including haulage tunnel projects having similar geotechnical conditions in Europe, Greenland, Nevada, and the Yukon.
4. I have read the definition of Qualified Person set out in Nation Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
5. I have visited the Property for a total of 4 days in August 2015.
6. I am responsible for the preparation of the haulage tunnel geotechnical, Section 17.1.2 of the Technical Report.
7. I am independent of the issuer per Section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with that instrument.
10. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: **June 10, 2016**

Signature: *Andrew P. Schissler*

Name: Andrew P. Schissler

I, Anthony Sanford, QP for Environmental, Permitting and Social or Community Impact, certify that:

I am Environmental Services and Water Resources Manager at Ausenco, Calle Esquilache 371, 6th Floor, San Isidro, Lima 27, Peru.

This certificate applies to the Technical Report titled Technical Report on the Pre-Feasibility Study for the Zafranal Project, Peru, dated 10 June 2016.

My qualifications and relevant experiences are that:

1. I graduated with a Bachelor of Science (Geology and Applied Geology) in 1984 from the University of Natal, a Bachelor of Science (Hons.) in Geology in 1985 from the University of Natal, and a Masters in Business Administration in Natural Resource Management from Dundee University in 1998.
2. I am a member of the South African Council for Natural Scientific Professions (Pr.Sci.Nat.), License # 400089/93.
3. I have worked as a as a Geologist continuously since 1985 and as an Environmental Scientist since 2004 for a total of 31 years. For the past 12 years I have been employed with Ausenco. During this period I have fulfilled roles as Senior Geologist, Environmental Services Manager, and Environmental Services and Water Resources Manager. I have been involved in the following projects in a similar capacity during this time: Shahuindo Project, Sulliden Resources (Peru); Zamin Resources, Valentines Project (Uruguay); Galeno Project, Northern Peru Copper (Peru).
4. I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43 -101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101 for those parts of the Technical Report for which I am responsible.
5. I have visited the Property for a total of 2 days on 24-25 June 2015. I am responsible for the preparation of sections 4, 5, 6.1 and 20 of the Technical Report (excluding hydrology and hydrogeology) and ensured that sections 1, 2, 3, 25, 26 and 27 accurately reflect the contributions of the QPs responsible for inputs to these sections.
6. I am independent of the issuer per Section 1.5 of NI 43-101.
7. I have not had prior involvement with the property that is the subject of the Technical Report.
8. I have read National Instrument 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
9. As of the date of the certificate, to the best of my knowledge, information and belief, those parts of the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: 08 July 2016

Signature:

Name: Anthony Sanford

Ausenco

AQM COPPER INC.

I, Carlos Guzman, QP for the mineral reserve estimate certify that:

I am Principal and Project director at NCL Ingeniería y Construcción SpA, General del Canto 235, Providencia, Santiago, Chile.

This certificate applies to the Technical Report titled Technical Report on the Pre-Feasibility Study, dated 10 June 2016.

My qualifications and relevant experiences are that:

1. I am a Graduate of the Universidad de Chile and hold a Mining Engineer title (1995).
2. I am a practicing Mining Engineer and a Fellow Member of the Australasian Institute of Mining and Metallurgy (FAusIMM, N° 229036); and a Registered Member of the Chilean Mining Commission (RM CMC 0119).
3. I have worked as a mining engineer for a total of 19 years. My relevant experience for the purpose of the Technical Report is:
    - Review and report as a consultant on numerous exploration, mining operation and projects around the world for due diligence and regulatory requirements.
    - I have extensive experience in mining engineering. I have worked on mining engineering assignments.
4. I have read the definition of Qualified Person set out in Nation Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
5. I have visited the Property for a total of two days in April 2015. I am responsible for the preparation of sections 15, 16, 18.1.1, 21.4.1 and 21.11 of the Technical Report.
6. I am independent of the issuer per Sections15, 16, 18.1.1, 21.4.1 and 21.11 of NI 43-101.
7. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is as a co-author of the Technical Report(NI 43-101) of the Preliminary Economic Assessment of the Zafranal Project, Perú, for AQM Copper Inc, effective January 16, 2013.
8. I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with that instrument.
9. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: **July 8, 2016**

Signature:

Name: **Carlos Guzmán**
Minming Engineer, FAusIMM, RM CMC

Ausenco

I, Gregory Lane, QP for mineral processing, concentrator design, infrastructure, product marketing, capital and operating costs (other than mining), economic analysis, project implementation plan (other than mining), and operations management plan (other than mining), certify that

I am Chief Technical Officer at Ausenco, 144 Montague Road, South Brisbane, Queensland, Australia.

This certificate applies to the Technical Report titled Technical Report on the Pre-Feasibility Study, dated 10 June 2016.

My qualifications and relevant experiences are that:

1. I graduated with a M.Sc. from the University of Tasmania in 1987.
2. I am a Fellow of the Australian Institute of Mining and Metallurgy (AusIMM).
3. I have worked as a process engineer for a total of 27 years.
4. I have read the definition of Qualified Person set out in Nation Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
5. I have experience in process design and management from over 20 similar studies or projects including the 17 Mt/y Cadia Project in Australia (1994 to 1998) the 12 Mt/y Phu Kham Project in Laos (2005 to 2008) and the 26 Mt/y Constancia Project in Peru (2010 to 2015).
6. I have visited the Property for a total of 2 days in July 2014. I am responsible for the preparation of Sections 13, 17, 19, 21, 22 and 24 of the Technical Report.
7. I am independent of the issuer.
8. I have not had prior involvement with the property that is the subject of this Technical Report.
9. I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with that instrument.
10. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: **8 July 2016**

Signature: Greg Lane

Name: Gregory Lane

I, Julio Bruna Novillo, AusIMM(CP) and Member CIM certify that:

I am Consultant Geologist at Patagonia Geosciences (PGSc), Apt. 601, 1305 Juncal Street, Montevideo (11300), Uruguay.

This certificate applies to the Technical Report titled Technical Report on the Pre-Feasibility Study for the Zafranal Project, Peru, with an effective date of 10 June 2016.

My qualifications and relevant experiences are that:

1. I graduated with a Bachelor of Science in Geology from the National University of Córdoba, 1993
2. I am a member of the Geology Professional Board of Cordoba (membership # 518); as well, I am a member of the Australasian Institute of Mining and Metallurgy as a Chartered Professional in the discipline of Geology, and of the Canadian Institute of Mining, Metallurgy and Petroleum.
3. I have worked as a geologist in the minerals industry for over 22 years. My work experience includes 10 years as an exploration geologist for gold and base metal deposits, 10 years as a mine geologist in open pit mining and more than 2 years as a consulting geologist working in base metals and precious metals. I was involved in the QA/QC process and geological modelling of the Bajo de La Alumbrera Mine, Bajo El Durazno Mine and Agua Rica Project in Catamarca - Argentina, all porphyry style deposits Cu-Au.
4. I have read the definition of Qualified Person set out in Nation Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101 for those parts of the Technical Report for which I am responsible.
5. I have visited the Property for a total of seven days in November 2013, January 2014 and July 2015.
6. I am responsible for the preparation of sections 10, 11 and 12 of the Technical Report.
7. I am independent of the issuer as described in Section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is the quality assurance and quality control of the sampling (QA-QC), and the optimization and development of criteria for the models of mineralization and geology.
9. I have read National Instrument 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
10. As of the date of the certificate, to the best of my knowledge, information and belief, those parts of the Technical Report for which I am responsible contain all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: **June 09, 2016**

Signature:

I, Leonard Paul Staples, QP for the Mining Support Facilities and Utilities (design) and Project Infrastructure (other than Waste Dumps, Tailings Storage, Surface Water Management and Water Supply Hydrogeology), certify that:

I am VP and Global Practice Lead at Ausenco, 855 Homer Street, Vancouver, British Columbia, Canada.

This certificate applies to the Technical Report titled Technical Report on the Pre-Feasibility Study, dated 10 June 2016.

My qualifications and relevant experiences are that:

1. I graduated with a Bachelor of Science degree in Materials and Metallurgical Engineering from Queens University in 1993.
2. I am a registered Professional Engineer of New Brunswick (memebership number 4832).
3. I have worked as a Metallurgist and process engineer continuously for a total of 23 years.
4. I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
5. I have experience in process operation, design and management from over 15 similar studies or projects including the 80 Mt/y Grasberg complex in Indonesia (1998-2003), the 20 Mt/y Lumwana Project in Zambia (2005-2007), the 26 Mt/y Constancia Project in Peru (2010-2015) and the 38 Mt/y Dumont feasibility study in Canada (2010-2016).
6. I have not visited the Property.
7. I am responsible for the preparation of subsection 16.9 of the Technical Report and for Section 18 (other than Waste Dumps, Tailings Storage, Surface Water Management, and Water Supply Hydrogeology), as well as the related portions of the the Summary, Interpretations and Conclusions, and Recommendations.
8. I am independent of the issuer.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read National Instrument 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
11. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: **8 July 2016**

Signature:

Name: Leonard Paul Staples

CERTIFICATE OF QUALIFIED PERSON

I, Peter Anthony Oshust, P.Geo., am employed as a Principal Geologist with Amec Foster Wheeler Americas Ltd. (Amec Foster Wheeler).

This certificate applies to the technical report titled "Technical Report on the Pre-Feasibility Study for the Zafranal Project, Peru", that has an effective date of 10 June 2016 (the "technical report").

I am registered as a Professional Geoscientist with the Association of Professional Engineers and Geoscientists of British Columbia and the Association of Professional Geoscientists of Ontario. I graduated with a Bachelor of Science (Specialist) degree in Geology and Economics from Brandon University in 1987.

I have practiced in my profession since 1988 and have been involved in geological modelling and resource estimation for a variety of base and precious metals and diamond deposits since 2001. Prior to the Zafranal Project, I was involved in the geological modelling and resource estimation of the Ann Mason project in Nevada, U.S.A. and the Oyut and Hugo North porphyry-style deposits at Oyu Tolgoi in Mongolia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I visited the Zafranal Project (the "Project") from 17–18 November, 2015.

I am responsible for the preparation of Section 14.0 of the technical report and for subsection 1.9 of the Summary, subsection 25.1.2 of the Interpretations and Conclusions, and subsection 26.2 of the Recommendations.

I am independent of AQM Copper Inc. as independence is described by Section 1.5 of NI 43–101.

I have no previous involvement with the Project.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 13 June, 2016

"Signed and sealed"

Peter Oshust, P.Geo.

I Ryan Jakubowski, QP for hydrogeology certify that

I am a Principal Hydrogeologist at FloSolutions, 160 Cantuarias, Office 704, Miraflores, Lima, Peru.

This certificate applies to the Technical Report titled Technical Report on the Pre-Feasibility Study for the Zafranal Project, Peru, with an effective date of 10 June 2016.

My qualifications and relevant experiences are that:

1. I graduated with a Bachelor of Science in Geology and Geophysics in May 2000. I obtained a Master of Science in Hydrology from the New Mexico Institue of Mining and Technology in December 2006.
2. I am a licensed Professional Geologist from the State of Wyoming.
3. I have worked as a hydrogeologist for a total of 13 years.
4. I have read the definition of Qualified Person set out in Nation Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
5. I have visited the Property for a total of 2 days in March 2016. I am responsible for the preparation of sections 5.8.2, 16.2.1, 18.2.9, 18.7.1, 18.7.2 and 20.1.2 of the Technical Report.
6. I am independent of the issuer per Sections 5.8.2, 16.2.1, 18.2.9, 18.7.1, 18.7.2 and 20.1.2 of NI 43-101.
7. I have not had prior involvement with the property that is the subject of the Technical Report.
8. I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with that instrument.
9. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: **08 July 2016**

Signature: Ryan T. Jakubowski

Name: Ryan Jakubowski

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28 Qualified Person Certificates & Consents

I Scott Elfen, QP for site geotechnical, site hydrology, tailings management facility, certify that

I am VP VP, Global Practice Lead and Director at Ausenco Peru S.A.C. Calle Esquilache 371, Piso 6 San Isidro I Lima 27 I Perú.

This certificate applies to the Technical Report titled Technical Report on the Pre-Feasibility Study, dated 10 June 2016.

My qualifications and relevant experiences are that:

1. I graduated with a Bachelor of Science degree in Civil Engineering from the University of Californina, Davis in 1991.
2. I am a Registered Civil Engineer in the State of California by exam since 1996 (No. C56527). I am also a member of the American Society of Civil Engineers (ASCE), Society for Mining, Metallurgy & Exploration (SME), and Canadian Geotechnical Society.
3. I have worked as a engineer in the mineral sector internationally for a total of 22 years. I have been directly involved in the mining and evaluation of mineral properties internationally for both precious and base metals, i.e. similar project such as Corani Project in Peru for Bear Creek Minng, La Arena Project in Peru for Rio Alto, El Porvenir Project in Peru for Milpo, Pascua Lama in Argentina for Barrick Gold, and Minas Rio in Brasil for Anglo American.
4. I have read the definition of Qualified Person set out in Nation Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
5. I have visited the Property for a total of 2 days in June 2015. I am responsible for the preparation of sections 1.3.1, 1.11.2, 1.11.3, 5.8.1, 5.8.3, 16.2.2, 18.1.3, 18.2 (except 18.2.7), 18.3, 20.1.1, 25.3.5, 25.4.2, 25.4.3, 26.4.2, and 26.4.3 of the Technical Report.

1. I am independent of the issuer per Sections 1.5 of NI 43-101.
2. I have not had prior involvement with the property that is the subject of the Technical Report.
3. I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with that instrument.
4. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated:

Signature:

Name: Scott Cameron Elfen

REGISTERED PROFESSIONAL ENGINEER
SCOTT C. ELFEN
No. C056527
Exp. 06/30/17
CIVIL
STATE OF CALIFORNIA