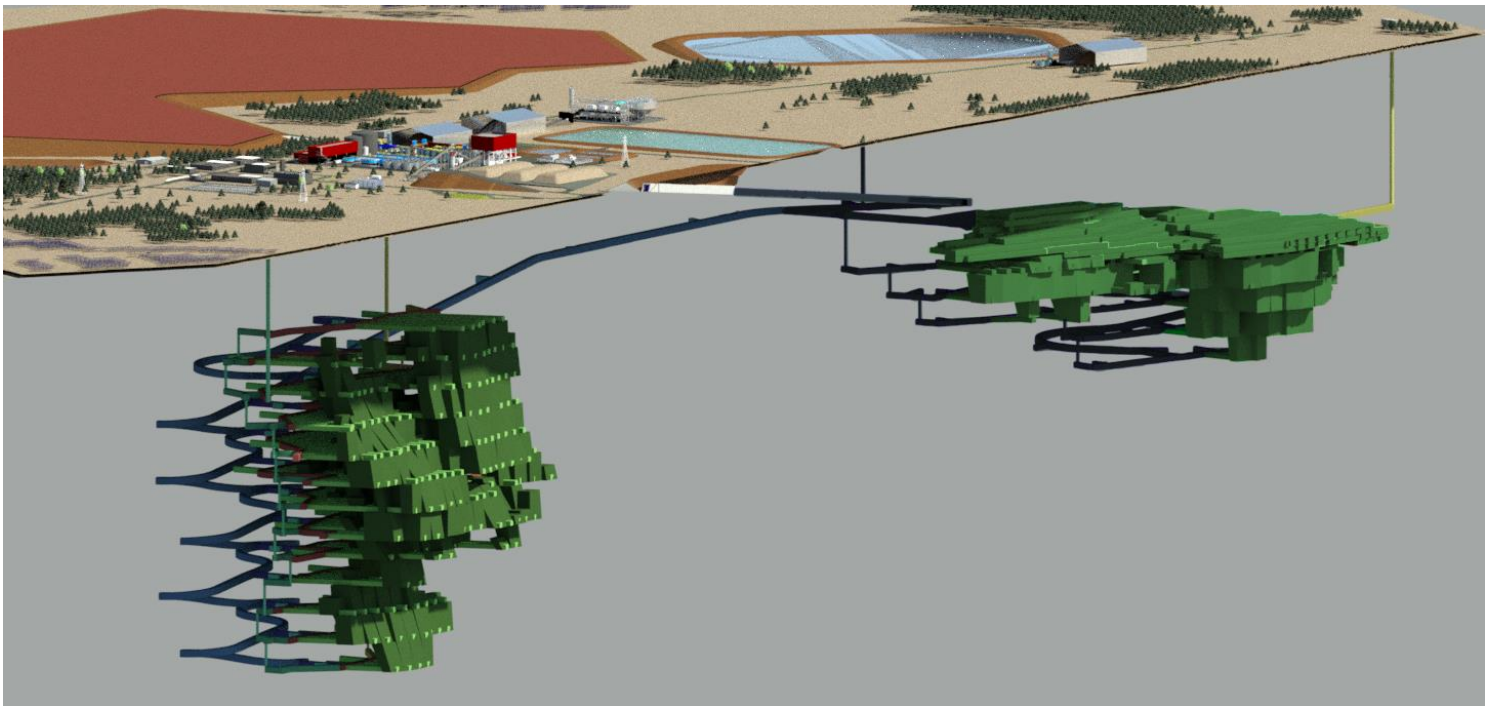
ASCENDANT RESOURCES INC.

REDCORP
Empreendimentos Mineiros, Lda.



TECHNICAL REPORT

NI 43-101 Technical Report and DFS for the Lagoa Salgada Project

Setúbal District, Portugal

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QUADRANTE

MINEPRO IGAN SLR CONSMAGA RMS micon INTERNATIONAL LIMITED wsp GOLDER

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ABBREVIATIONS AND DEFINITIONS

Term	Definition
United States dollar	$
Euro	€
Percentage	%
Greater than	>
Less than	<
Degree	°
Degrees Celsius	°C
Micron	µ
Micrometre	µm
Two-dimensional	2D
Three-dimensional	3D
Acrylonitrile butadiene styrene	ABS
Silver	Ag
Meter	m
Meter per month	m/month
Meter squared	sqm; m2
Capital Expenditures	CAPEX
Cubic meter	m3
Cubic meter per hour	m3/hr
Cubic meter per second	m3/s
Meter above sea level	Masl
Maximum	Max
Methyl isobutyl carbinol	MIBC
Micon International Limited	Micon
Minute; minimum	Min
Mineral Liberation Analysis	MLA
Milliliter	ml
Millimeter	mm
AMC Mining Consultants (Canada) Ltd.	AMC
Environmental Impact Assessment Authority	APA
Ascendant Resources Inc.	ASND; Ascendant
Gold	Au
Gold equivalent	AuEq
Best available techniques	BAT
Borehole Induced Polarization	BHIP
Capital expenditure	Capex
Alentejo Regional Coordination and Development Commission	CCDR
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Centimeter	cm
Coefficient of variation	Coeff. V / CoV

Certified reference material	CRM
Cleaner	Cl
Concentrate	Conc
Copper	Cu
Copper equivalent	CuEq
Copper sulphate	CuSO4
80% particle size distribution	D80
Direção Geral de Energia e Geologia / General Directorate of Energy and Geology	DGEG
Declaração de Impacte Ambiental – The environmental statement	DIA
Dry metric tonne	dmt
Drillcon Iberia S.A.	Drillcon
Digital terrain model	DTM
East	E
Empresa de Desenvolvimento Mineiro S.A.	EDM
Environmental Impact Assessment	EIA
Engineering, Procurement and Construction Management	EPCM
Empresa de Perfuração e Desenvolvimento Mineiro, S.A.	EPDM
Iron	Fe
Feasibility study	FS
Gram	g
General and Administration	G&A
Grams per tonne	g/t
Gossan	GO
Grinding Solutions Ltd.	GSL
Hectare	ha
Headquarters	HQ
Mercury	Hg
Hour(s)	hr(s)
Hydraulic radius	S
Indicated	I
Identification	ID
Inverse distance squared	ID2
Inverse distance cubed	ID3
Intelligent Exploration	IE
Licença ambiental – Environmental license	LA
Million cubic meters	Mm3
Manganese	Mn
Massive sulphide	MS
Million tonnes	Mt
Million tonnes per annum	Mt/y
Mega volt amperes	MVA
Millivolts per volt	mV/V

Megawatt	MW
North	N
Sodium cyanide	NaCN
Sodium sulfuret	Na2S
Sodium Hexametaphosphate	Na6((PO3)6)
National Instrument 43-101	NI 43-101
Nearest neighbor	NN
North-north-west	NNW
Net present value	NPV
Net smelter return	NSR
North-west	NW
Ordinary kriging	OK
Operating expenditure	OPEX
Operations	Op
Overhead Line	OHL
Troy ounce	oz
80% Passing	P80
Lead	Pb
Environmental Scoping Proposal	PDA
Preliminary Economic Assessment	PEA
Abbreviations & acronyms Description	Abbreviations & acronyms Description
Pre-Feasibility Study	PFS
pH is a measure of hydrogen ion concentration; a measure of the acidity or alkalinity of a solution	pH
Portex Minerals Inc.	Portex
Parts per million	ppm
Lagoa Salgada Property	LS Property/ Property
Polyvinyl chloride	PVC
Quality Assurance / Quality Control	QA/QC
Qualified Person as defined by NI 43-101	QP
Portuguese legal regime of environmental impact assessment	RJAIA
Environmental Conformity of the Detail Design Project – Relatório de Conformidade Ambiental do Projeto de Execução	RECAPE
Redcorp - Empreendimentos Mineiros, Lda	Redcorp
Redcorp Ventures Ltd.	Redcorp Ventures
Technical Report	Report
Regime de emissões industriais - Regime of integrated prevention and pollution control of industrial emissions	REI
Rock quality designation	RQD
Rougher	Ro
Run-of-mine	ROM
Rio Tinto Zinc	RTZ

South	S
System for Electronic Document Analysis and Retrieval	SEDAR
Serviço de Fomento Mineiro	SFM
Stirred media detritor	SMD
Instituto Geográfico e Cadastral de Portugal	IGCP
Instituto Geológico e Mineiro	IGM
International Geophysical Technology	IGT
Induced polarization	IP
Iberian Pyrite Belt	IPB
IRIS Instruments	IRIS
Internal rate of return	IRR
International Organization for Standardization	ISO
Joint venture	JV
Thousand	k
Kilogram	kg
Kilogram per cubic metre	kg/m3
Kilogram per tonne	kg/t
Kiloliter	kL
Kilometer	km
Thousand ounces	koz
Thousand tonnes	kt
Kilovolts	kV
Kilovolt-Ampere	kVA
Kilowatts	kW
Kilowatt-hour	kW-hr
Liter	L
Liters per second	L/s
Pound	lb
Locked-cycle test	LCT
Load-haul-dump	LHD
Lisbon International Airport	LIS
Laboratório Nacional de Energia e Geologia	LNEG
Life-of-mine	LOM
Lagoa Salgada	LS
Lower Volcanic Unit	LVU
Million; Measured	M
Sodium Iso-Propyl Xanthate	SIPX
Sub-level open stoping	SLOS
Selective mining unit	SMU
Tin	Sn
South-south-east	SSE
Short ton	st
Stockwork	SW; ST; Str/Fr

Standard deviation	Std Dev
Stringer	Str
Total Sulphur	STOTAL
Tonne	t
Transient electromagnetic	TEM
Treatment Costs / Refining Costs	TC/RC
TH Crestgate GmbH	TH Crestgate
Tonne = 1,000 kg	tonne
Tonnes per annum	t/y
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per vertical meter	t/vm
Tailings Storage Facility	TSF
Título Unico Ambiental – Single Environmental Tittle	TUA
Unconfined compressive strength	UCS
United Kingdom	UK
United States	US
United States dollar	US$
Universal Transverse Mercator	UTM
Upper Volcanic Unit	UVU
Volt	V
Volcanogenic massive sulphide	VMS
Watt, West	W
Weight	Wt
Wet metric tonne hour	Wmth
Year	y
Zinc	Zn
Zinc equivalent	ZnEq
Zinc sulphate	ZnSO4

CERTIFICATE OF QUALIFIED PERSON

I, João Miguel Cardeal Martins Horta, MIMMM, am employed as a Project Director with Quadrante, located at Estrada do Seminário 4, Edifício C, Piso 1 Sul, 2614-523 Alfragide, Portugal.

This certificate applies to the technical report entitled "NI 43-101 Technical Report and PEA for the Lagoa Salgada Project, Setúbal District, Portugal", that has an effective date of September 10, 2021 (the "technical report").

I am registered as a Senior Member of the Portuguese Engineers Association (# 50464) and am a Professional Member (MIMMM) of the Institute of Materials, Minerals and Mining (# 679050). I graduated with a M.S. degree from the Lisbon University – Higher Technical Institute in 2001. I have practiced my profession for 20 years.

I have worked in the mining industry in various positions continuously, including mining operations and mining studies, since my graduation from university. I have worked with various commodities within the Iberian Pyrite Belt.

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 Lagoa Salgada Project on multiple occasions, between December 2021 and June 2023.

I am the co-author of this report and responsible for sections 1.1, 1.2, 1.3, 1.4, 1.14, 1.17, 1.18, 1.19, 1.20, 1.21, 2, 3, 4, 5, 6, 12, 18, 21, 22, 23, 24, 25, 26 and 27.

I am independent of Ascendant Resources Inc. as independence is described by Section 1.5 of NI 43–101.

I have had prior involvement with the property that is the subject of the Technical Report in that I was a qualified person for one (1) previous Technical Reports on the Lagoa Salgada Property, dated September 2021 and titled "NI 43-101 Technical Report and PEA for the Lagoa Salgada 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 the technical report not misleading.

Dated: June 16, 2023

Signature: "Original signed"

Name: João Miguel Cardeal Martins Horta, MIMMM

CERTIFICATE OF QUALIFIED PERSON

I, David Castro López, MIMMM, am employed as a Process Engineer with MINEPRO SOLUTIONS, located at c/Fernando Vela 1, 5º D, 33001, Oviedo, Spain.

This certificate applies to the technical report entitled "NI 43-101 Technical Report and Definitive Feasibility Study for the Lagoa Salgada Project, Setúbal District, Portugal", that has an effective date of June 23, 2023 (the "technical report").

I am registered as Professional Member (MIMMM) and Qualified for Minerals Reporting (QMR) of the Institute of Materials, Minerals and Mining (# 685484). I graduated with a M.S. degree from the Oviedo University in 2005. I have practiced my profession for 12 years.

I have worked in the mining industry in various positions continuously, including mining operations and mining studies, since my graduation from university. I have worked with various commodities within the Iberian Pyrite Belt.

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 Lagoa Salgada Project on June 7, 2022.

I am the co-author of this report and responsible for sections 1.9, 1.13, 13, 17, 25.3, 25.7 and 26.1.3.

I am independent of Ascendant Resources Inc. as independence is described by Section 1.5 of NI 43–101.

I have not had prior involvement with the property that is the subject of the Technical Report.

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 the technical report not misleading.

Dated: June 23, 2023

Signature:

Name: David Castro López, MIMMM QMR

CERTIFICATE OF QUALIFIED PERSON

I, Charley Murahwi, P.Geo., am employed as a Senior Economic Geologist with Micon International Limited, located at 601 – 90 Eglinton Ave East, Toronto, ON M4P 2Y3, Canada.

This certificate applies to the technical report entitled "NI 43-101 Technical Report and Definitive Feasibility Study for the Lagoa Salgada Project, Setúbal District, Portugal", that has an effective date of June 23, 2023 (the "technical report").

I am a graduate of the University of Rhodesia in Zimbabwe (Bachelors of Geology in 1979) and Rhodes University in South Africa (Masters of Economic Geology in 1996) and possess a Diplome d'Ingénieur Expert en Techniques Minières, Nancy, France (1987). I am a member in good standing of the Association of Professional Geoscientists of Ontario (Membership #1618), the Professional Engineers and Geoscientists of Newfoundland & Labrador (Membership #05662), the South African Council for Natural Scientific Professions (Membership #400133/09), and the Australasian Institute of Mining & Metallurgy (FAusIMM) (Membership #300395).

I have worked as a mining and exploration geologist in the minerals industry for over 40 years. During this time, I have gained experience in a wide variety of deposits including gold-silver in skarn/lode/vein and shear hosted systems, and gold-copper-lead-zinc in VMS/porphyry and epithermal systems, amongst others. As an independent consultant, I have undertaken the technical and financial evaluation of mining and exploration projects in several countries in Central and Southern Africa, Canada, USA, Spain, Panama, Mexico, Bolivia, West Africa and Australia.

I do, by reason of education, experience, and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 18 years on gold, silver, copper, tin, and tantalite projects (on and off mine), 12 years on Cr-Ni-Cu-PGE deposits in layered intrusions/komatiitic environments and 10 years as a consulting geologist on precious and base metals and industrial minerals.

I visited the Lagoa Salgada Property from 16 to 19 October 2018, from 13 to 17 November 2018, from 28 to 31 May 2019, and from 24 to 27 April 2023.

I am a co-author of this report and responsible 1.5, 1.6, 1.7, 1.8, 1.10, 7, 8, 9, 10, 11, 12, 25.1, 25.2 and 23.6.1. I am independent of Ascendant Resources Inc. as independence is described by Section 1.5 of NI 43–101.

I have had prior involvement with the property that is the subject of the Technical Report in that I was a qualified person for four (4) previous Technical Reports on the Lagoa Salgada Property between 2018 and 2022. 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 the technical report not misleading.

Dated: June 23, 2023

Signature: "Original signed"

Name: Charley Murahwi, MSc., P.Geo., Pr.Sci.Nat., FAusIMM

CERTIFICATE OF QUALIFIED PERSON

I, Pablo Gancedo Mínguez, CEng MIMM QMR, am employed as a Senior Mining Engineer with IGAN Ingeniería S.L., located at Edificio CEEI, Parque Tecnológico de Asturias, 33428 Llanera (Asturias), Spain.

This certificate applies to the technical report entitled "NI 43-101 Technical Report and Definitive Feasibility Study for the Lagoa Salgada Project, Setúbal District, Portugal", that has an effective date of June 23, 2023 (the "technical report").

I am a Chartered Engineer (CEng MIMMM) with the Institute of Materials, Minerals and Mining (licence #650444). I am a graduate of the Oviedo School of Mines, University of Oviedo in 2010 where I obtained a Master's degree in Mining and Minerals Engineering.

I have practiced my profession for twelve years since graduation, including ten years in mine planning consulting and two in mining company corporate technical services roles. My relevant experience includes mine planning, cost modelling and reserve estimation for both operating and under development underground base and precious metals projects.

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 Lagoa Salgada Project on January 18th, 2022.

I am responsible for sections 1.11, 1.12, 15,16, 25.4 and 25.5 of the Technical Report.

I am independent of Ascendant Resources Inc. as independence is described by Section 1.5 of NI 43–101.

I have had prior involvement with the property that is the subject of the Technical Report in that I was a qualified person for one (1) previous Technical Reports on the Lagoa Salgada Property, dated September 2021 and titled "NI 43-101 Technical Report and PEA for the Lagoa Salgada 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 the technical report not misleading.

Dated: June 23, 2023

Signature: {signed and sealed}

Name: Pablo Gancedo Mínguez, CEng MIMMM

CERTIFICATE OF QUALIFIED PERSON

I, David Ritchie, P.Eng, am employed as a Principal Geotechnical Engineer with SLR Consulting (Canada) Ltd., with a business address at 55 University Ave Suite 501, Toronto, ON M5J 2H7 Canada..

This certificate applies to the technical report entitled "NI 43-101 Technical Report and Definitive Feasibility Study for the Lagoa Salgada Project, Setúbal District, Portugal", that has an effective date of June 23, 2023 (the "technical report").

I am registered as a I am a member in good standing of the Professional Engineers Ontario, Licence Number 90488198 since 1998. I am a graduate of the Ryerson Polytechnic University with a degree in Civil Engineering in 1995 and the University of Western Ontario with a Master of Engineering degree in Geotechnical Engineering in 2000.

I have practiced my profession in the mining industry continuously since graduation with 28 years of experience related to tailings and water management and embankment dam design.

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 Lagoa Salgada Project site on January 18, 2022.

I am responsible for input to sections 18.11.1, 18.11.2 and 18.11.3.

I am independent of Ascendant Resources Inc. as independence is described by Section 1.5 of NI 43–101.

I have not had prior involvement with the property that is the subject of the Technical Report.

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 the technical report not misleading.

Dated: June 23, 2023

Signature: "Original signed"

Name: David Grafton Ritchie, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Rafael Fernández Rubio (Spain Identity Card 01209024Y), am specialist consultant of FRASA Ingenieros Consultores, S.L., located at Rua Sariñena, 8, 1ºA, Alcobendas, 28100, Madrid , Spain.

This certificate applies to the technical report entitled "EHB - Baseline Hydrogeological Study" (or BHS) part of the ETVE - Technical and Economic Feasibility Study of the Lagoa Salgada Project, Setúbal District, Portugal" (or DFS - Definitive Feasibility Study), that has an effective date of March 23, 2023 (the "technical report").

I am Dr. Mining Engineer, Emeritus Professor of Madrid School of Mines and Energy, Polytechnic University of Madrid. Dr Honoris Causa from Lisbon University (Portugal) and from National University of Engineering of Lima (Peru). Guest Professor and Visitor Professor at 61 universities in 23 countries. Past President of the Spanish Group of the International Association of Hydrogeologists.

I have worked in more than 400 Hydrogeological missions in forty-five countries on five continents, some of them in the mining industry in the Iberian Pyrite Belt.

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 Lagoa Salgada Project on February 3, 2022.

I am the co-author and responsible for the section 20.7 of this report.

I am independent of Ascendant Resources Inc. as independence is described by Section 1.5 of NI 43–101.

I have not had prior involvement with the property that is the subject of the Technical Report.

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 (Hydrology and Hydrogeology) contain all scientific and technical information that is required in this step of the Project to be disclosed to make the technical report not misleading.

Dated: June 23, 2023

Signature:

Name: Rafael Fernández Rubio

CERTIFICATE OF QUALIFIED PERSON

I, Frank Palkovits, P.Eng., B.Eng. (Mining), am employed as President with Mine Paste Ltd., located at 26 Windsor Crescent, Sudbury ON, Canada.

This certificate applies to the technical report entitled "NI 43-101 Technical Report and Definitive Feasibility Study for the Lagoa Salgada Project, Setúbal District, Portugal", that has an effective date of June 23, 2023 (the "technical report").

I am a graduate of Laurentian University, (B.Eng,, Mining, 1988) and Cambrian College (Geological Tech., 1981). I am a member in good standing of the Association of Professional Engineers and Geoscientists of (License No. 90276379).

My relevant experience with respect to mining includes 20 years in operations (engineering/mine design, geology, underground miner and front-line supervisor) as well as 20 years in mine design and mining plants consulting and design engineering for Canadian and global interests. I have worked with various commodities within the Iberian Pyrite Belt.

I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43 101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43 101.

I visited the Lagoa Salgada Project on January 17th, 2022.

I am responsible for the preparation of tailings dewatering component for tailngs storage facility and for underground mine backfill sections within section 18.11.4.

I am independent of Ascendant Resources Inc. as independence is described by Section 1.5 of NI 43–101.

I have not had prior involvement with the property that is the subject of the Technical Report.

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 the technical report not misleading.

Dated: June 23, 2023

Signature: "Original signed"

Name: Frank Palkovits, P.Eng.

1 SUMMARY

1.1 INTRODUCTION

Ascendant Resources Inc. (Ascendant) has formally commissioned Quadrante Engenharia (Quadrante) to prepare a comprehensive technical report (referred to as the Report) for the Definition Feasibility Study (DFS) conducted on the esteemed Lagoa Salgada Project (referred to as the Project), situated in the Grândola municipality in southern Portugal.

The Definition Feasibility Study (DFS) outlines a meticulously planned underground mining operation at the Lagoa Salgada Project, employing drift & fill and long-hole stoping mining methods to extract ore at a targeted capacity of 2 million tons per annum (Mtpa). The extracted ore will undergo processing at a primary surface facility, where it will be directed to a crushing circuit featuring a semi-autogenous grinding mill (SAG) and ball mill design. Following the crushing process, the material will proceed to grinding and flotation circuits.

This comprehensive plant setup aims to produce copper, zinc, lead, and tin concentrates, as well as gold and silver dore bars. These will then be transported from the site to Sines, where they will be shipped by boat to smelters for further processing.

The Project envisaged by the DFS includes a 3,250 tpd (nominal) underground mine, onsite crushing, and mineral processing facilities, paste plant, tailing storage facility, batch plant, and all the support infrastructure for the mine and the processing plant.

This DFS is based on the Mineral Resource estimate dated May 2023, with a Reserve calculate in June 2023.

Pre-production capital costs for the project are 164.4 M$, followed by 96.9 M$ of sustaining costs for a total capital cost of 261.3 M$. An additional 6.0 M$ is required for mine closure. LOM average operating costs of 241.5 M$ for processing are expected, translating to an overall operating cost of 540.3 M$.

The project economics indicates a pre-tax net present value (NPV) at an 8.0% discount rate of 188.8 M$ and a post-tax NPV of 147.0 M$. The pre-tax internal rate of return (IRR) is 47% and the post-tax IRR is 39%. The Project generates a life-of-mine (LOM) undiscounted post-tax-free cash flow of 260.9 $M.

Preparation of the DFS included contributions from consulting firms listed below.

- IGAN Consultores – Mine planning, underground services, ventilation & underground geotechnical designs.
- Mine Pro - Process metallurgy & Process Plant design and costs and Metallurgical Testing peer review.

- Micon International Inc. – Mineral Resource Estimate peer review.
- Grinding Solutions Ltd (GSL) & Maelgwyn Mineral Services – Metallurgical test work.
- SLR Consulting – Tailings Storage Facility design and costs.
- Responsible Mining Solutions (RMS) – Paste fill design and costs & design of underground backfill reticulation.
- CONSMAGA - Geólogos e Engenheiros Consultores Lda & FRASA - Ingenieros Consultores S.L. – Baseline Hydrogeological Study (BHS).
- WSP Golder – Project peer review.

1.2 OWNERSHIP

The LS Project is covered by a single exploitation permit (mining concession) spanning approximately 10,700 hectares, held by Redcorp Empreendimentos Mineiros, LDA. The permit was granted by the Portuguese General Directorate for Energy and Geology (DGEG) on October 21, 2021, and is registered in the Diário da República, Public Register, under Contrato (extrato) 526/2021, dated December 2, 2021.

Redcorp is currently co-owned by Ascendant Resources Inc. (Ascendant), a Canadian company listed on the Toronto Stock Exchange, and Mineral & Financial Investments Limited (M&FI). Ascendant has the right to earn an 80% interest in Redcorp following the delivery of the DFS.

Under a consortium agreement, Empresa de Desenvolvimento Mineiro, SA (EDM), a Portuguese company specializing in mineral resource activities, has the option to acquire up to a 15% interest in the LS project. Ascendant has a right of first refusal to purchase the EDM Interest, ensuring its ownership is not affected if M&FI is unable to acquire the EDM Interest.

1.3 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

1.3.1 ACCESSIBILITY

The Lagoa Salgada Project is located in Portugal, approximately 80 km southeast of the capital Lisbon (approximately 120 km by road). It is situated within the Instituto Geográfico e Cadastral de Portugal (IGCP) map sheets 39-C, 39-D, 42-A, and 42-B, at approximately 38°14’ North latitude and 8°28’ West longitude in south-western Portugal.

The project is easily accessible through national highways and roads, with a travel time of approximately 1 hour and 30 minutes from Lisbon. It is positioned about 50 km southeast of Setúbal, the regional administrative center, and approximately 12 km northeast of the municipality of Grândola. The closest village, Cilha do Pascoal, is located approximately 3 km south of the project.

The property also benefits from its proximity to highway connections leading to the port facilities in Setúbal and the industrial area with deep-water port facilities in Sines. The transportation infrastructure allows for convenient movement of goods and equipment to and from the project site. Within the project area, transportation is facilitated by unpaved roads with a total length of approximately 4 km. These roads can be accessed by truck/utility vehicle or 4x4 vehicles and are regularly maintained. Some sections of the roads are suitable for standard motor vehicles as well.

Overall, the Lagoa Salgada Project enjoys a strategic location with good accessibility, providing convenient connections to major cities, regional administrative centers, and transportation infrastructure, facilitating efficient operations and logistics for the project.

1.3.2 CLIMATE

The Lagoa Salgada Project is situated in a Mediterranean climate zone (Csa), characterized by hot and dry summers and moderately cool winters with variable rainfall patterns. The area experiences a concentration of rainfall during the humid semester (October to March), which accounts for approximately 79% of the average annual precipitation. In contrast, the dry semester (April to September) receives only about 21% of the average annual precipitation.

Over the study period of 90 years, there has been a decrease in wet years and an increase in dry years. The average monthly maximum temperature ranges from 15.1 °C in January to 30.6 °C in July, while the average monthly minimum temperature varies between 3.6 °C and 16.1 °C in January and August, respectively.

1.3.3 LOCAL RESOURCES

The Lagoa Salgada Project benefits from its proximity to various local resources, providing advantages for its operations. The closest populated area to the project is Grândola, with a population of approximately 14,000 and a focus on agriculture. The Setúbal district, with an estimated population of around 90,000, is equidistant from the project and serves as a hub for industries such as cellulose, paper, and cement. Setúbal also hosts important port facilities used by the Neves-Corvo Mine and Autoeuropa, the largest automobile manufacturer in Portugal.

The nearby industrial town of Sines, located 60 km away and accessible through a well-connected highway network, plays a significant role in heavy and chemical industries and houses the primary deep-water port in Portugal. Both Setúbal and Sines ports are

integrated into the national rail network, facilitating efficient transportation of goods and services. The proximity of the Lagoa Salgada Project to the Aljustrel Mine (50 km away) and the Neves Corvo Mine (85 km away) provides access to skilled mining personnel and labour resources.

The project area boasts substantial land holdings, suitable for exploration and development. It benefits from excellent connectivity to major infrastructure systems in Portugal, including well-established road networks, railway connections, electric power lines, and port and airport facilities. The project's power requirements are expected to be met through a grid connection established via a high-voltage overhead line approximately 15.1 km away. Water sources for the survey program are available, with drilling operations currently utilizing a renovated well. In the next phase of the project, a trade-off study must be conducted to assess the need of an external water supply.

Overall, the Lagoa Salgada Project offers access to a range of local resources, including skilled labour, transportation networks, ports, and power and water sources. These resources contribute to the project's operational efficiency and support its exploration and development activities.

1.3.4 PHYSIOGRAPHY

The Property features a mostly flat terrain, with elevations ranging from 20 meters to 100 meters above sea level. The vegetation in the area reflects the typical flora found in dry Mediterranean climates, including scrub brush, tall grasses, and stands of pine trees. The land within the property boundaries is privately owned and mainly utilized for growing pine trees, eucalyptus, and sporadic cork trees.

The predominant soil composition in the project area is sandy, indicating a significant proportion of sand particles in the soil. The sandy soil is accompanied by limited exposure of the underlying tertiary sedimentary bedrock, which suggests that the bedrock remains mostly hidden beneath the surface layers.

1.4 HISTORY

1.4.1 HISTORICAL EXPLORATION

1.4.1.1 INITIAL DISCOVERY, 1992

The LS deposit was discovered in 1992 by a team from the Portuguese government geological survey, known as Serviço de Fomento Mineiro (SFM), which has now been incorporated into the Laboratório Nacional de Energia e Geologia (LNEG). During the period between 1992 and 1993, a total of 17 drillholes were completed in and around the LS Property, covering a distance of 7,588 meters (m). These drillholes were labelled LS-01 to LS-17.

The deposit itself is located beneath a thick layer of Tertiary sedimentary rock, which has an average thickness of approximately 135 meters. The discovery of the deposit was made possible using diamond drill testing, specifically targeting a gravity geophysical anomaly. This successful exploration method led to the identification and confirmation of the LS deposit.

1.4.1.2 RIO TINTO ZINC, 1994-2000

Between 1994 and 2000, the mining consortium consisting of Rio Tinto Zinc (RTZ) and Empresa de Desenvolvimento Mineiro (EDM) held the LS Property. They conducted an airborne magnetic survey, diamond drilling, downhole geophysics, electromagnetic surveys, and limited soil sampling. The consortium drilled 20 additional holes (LS-18 to LS-37) during 1994-1999, defining the North, Central, and South deposits. The historical drill core and documents are stored at LNEG facilities in Aljustrel, accessible upon request.

In 1995, RTZ commissioned metallurgical testwork on a high-grade sample from drillhole LS-22, showing significant zinc and lead content. The tests achieved a Pb cleaner concentrate grade of 34.2% at a recovery of 38.5% and a Zn cleaner concentrate grade of 44.7% at a recovery of 23.1%. However, the LS-22 sample is not representative of the current deposit definition based on 2017 mineralogical samples.

1.4.1.3 REDCORP VENTURES LTD., 2004-2008

In October 2004, Redcorp Ventures Ltd., based in Vancouver, Canada, acquired the LS Property and established its Portuguese subsidiary, Redcorp - Empreendimentos Mineiros, Lda. In 2005, Redcorp Ventures conducted a 3D inversion of geophysical data and performed diamond drilling, focusing mainly on the Rio de Moinhos Project. Lithogeochemical and petrographic samples were collected, but the results are unavailable to Quadrante/IGAN.

Redcorp Ventures carried out a drilling program in 2005, with six holes totalling 2,286 m. Further drilling took place in 2006, 2007, and 2008, comprising 16 holes totalling 8,692 m. Nearly all drillholes intersected the LS deposit, confirming the presence of polymetallic mineralization, including zinc (Zn), lead (Pb), copper (Cu), gold (Au), and silver (Ag), except for one drillhole (LS06043).

1.4.1.4 PORTEX MINERALS INC., 2009-2012

In 2009, Portex acquired full ownership of Redcorp Ventures with the aim of developing the North deposit on the LS Property. Portex conducted an exploration campaign that included a drilling program and a downhole geophysical survey. The drilling program took place between May and August 2011, with five diamond drillholes totalling 1,138 m, followed by two additional drillholes in 2012 totalling 474 m.

1.4.1.5 MINERAL AND FINANCIAL INVESTMENTS LTD 2012-2018

During the period between 2012-2018, Mineral and Financial Investments Ltd was the main holder of the Lagoa Salgada project, this company implemented a multi-approach to the project by performing a petrographic study of the ore, a downhole geophysical study, and a drilling campaign.

1.4.2 MINERAL RESOURCE ESTIMATES

There have been three previous NI 43-101 Technical Reports completed on the LS Property, each of which contained mineral resource estimates on the LS-1 deposit. The previous Technical Reports are as follows:

- Wardrop September 2007, Redcorp Ventures Ltd., Resource Estimate for the Lagoa Salgada Project. Wardrop Engineering Inc. Document No. 0752760100-REP-R0001-01. 27, 46 pages.
- Daigle, Paul January 2012, Lagoa Salgada Project, Portugal – Resource Estimate Update Document No. 1296360100-REP-R0001-02, 92 pages.
- Daigle, Paul January 2018, Revised July 2018), Technical Report for Redcorp Lda., Lagoa Salgada Project, Setubal District, Portugal, 124 pages.
- Micon February 2019, NI 43-101 Technical Report: Resource Estimate for the Lagoa Salgada Project, Setubal District, Portugal, 117 pages.
- Micon November 2019, NI 43-101 Technical Report: Resource Estimate Update for the Lagoa Salgada Project, Setubal District, Portugal, 152 pages.
- Murahwi, C., Methven, P., Zazzi, G, Malhotra, D. 2021: Updated Mineral Resource Estimate for the South Deposit and PEA for the North Deposit, Lagoa Salgada Property, Setúbal District, Portugal: Prepared by Micon International Limited for Ascendant Resources Inc., effective date 31 January 2021 (South Deposit) and 19 December 2019 (North Deposit).

1.4.3 PRODUCTION/MINING FROM LAGOA SALGADA PROJECT

There has been no mineral production on the Lagoa Salgada Project as it relates to the polymetallic mineralization for which Ascendant has been exploring and drilling.

1.5 GEOLOGICAL SETTING AND MINERALIZATION

The Lagoa Salgada project is located in the northwest region of the Iberian Pyritic Belt (IPB) in Portugal. This area is characterized by the collision of the IPB with the Ossa Morena Zone,

resulting in basins within the continental crust and submarine volcanism. The project is situated at the contact between Paleozoic and Cenozoic formations, with the Paleozoic Volcano-Sedimentary Complex (VSC) hosting massive sulphide deposits. The region exhibits Variscan and Alpine tectonic effects, with well-defined alignments of horsts and grabens. The mineralization is influenced by folding, resulting in synforms and antiforms sequences accompanied by faults.

The IPB is one of the most prolific European ore provinces, hosting one of the largest concentrations of MS in the Earth's crust; it contains more than 1,600 million tonnes (Mt) of MS ore and about 250 Mt of stockwork ore (Oliveira et al. 2005, 2006; Tornos 2006). The IPB hosts more than 90 MS deposits. The dimensions of the deposits vary from 1 to >300 Mt (e.g., Neves Corvo, Rio Tinto, and Aljustrel), including 14 world-class (>32 Mt) VMS orebodies (Laznicka 1999). Despite their large size (eight deposits with >100 Mt MS), most are particularly pyrite rich and about 11 deposits can be considered large regarding their Cu, Zn and Pb contents. In Portugal currently only Neves Corvo and Aljustrel are being exploited.

1.5.1 PROPERTY GEOLOGY AND MINERALISATION

The entire Property is covered by a paleo-fluvial fan that ranges in thickness from 100 meters up to 250 metres within the Tertiary Sado Basin, averaging 135 m over the deposit area. The Tertiary sedimentary rocks unconformably overlie rocks of the Volcano-Sedimentary Complex of the IPB. This sequence of rocks ranges in age from Upper Famenian to Middle Visean and is represented on the property by a northwest-southeast lineament which is approximately 8 km long and over 1 km wide.

The LS Project currently comprises two known deposits, the Venda Nova North and South Deposits. The deposits are folded, faulted, and interpreted to occur mostly on the subvertical-overturned and intensely faulted limb of a south-west-verging anticline (Matos et al. 2003).

The Lagoa Salgada north deposit is comprised of gossan (Gos) mineralization resulting from weathering of the underlying primary massive sulphide (pMS) mineralization which is underlaid and surrounded by copper-rich stringer / fissure / stockwork mineralization (str). Recently, a new massive sulphide geometallurgical domain was identified at the transition between Gos and pMS – the transition massive sulphide (tMS)

The Lagoa Salgada south deposit comprises copper-rich stringer / fissure / stockwork and remobilized mineralization and isolated gold-rich silicified zones which appear to be structurally controlled. It is mostly hosted in a host rock known as the iV. The iV is heterogeneous, composed of intricately interleaved dacitic-andesitic-basaltic volcanic rocks. The mineralization has been remobilized into fissures/cracks arising from brittle failure during deformation, hence the name 'fissural mineralization'. Although the entire iV unit is mineralized to some extent, potentially economic grade mineralization occurs in

steeply dipping corridors; the steep dip is attributed to post mineralization east-west compressional deformation.

1.6 DEPOSIT TYPES

The known LS Property's two deposits are interpreted as components of a polymetallic, volcanogenic massive sulphide (VMS) deposit.

VMS deposits are a type of metal sulphide deposit that is associated with and created by hydrothermal events due to volcanic activity in submarine environments. They occur within environments dominated by volcanic or volcanic-derived volcano-sedimentary rocks and are coeval and coincident with the formation of the volcanic rocks. VMS deposits form on the seafloor around undersea volcanoes along many mid-ocean ridges, and within back-arc basins and forearc rifts.

Deformation and exposure help to develop epigenetic mineralization as the above-mentioned gossan, transition massive sulphides at the north deposit and fissural remobilizations at the south deposit.

At the north deposit a lower volcanic formation composed by acidic to intermediate rocks appears to be the footwall of the VMS mineralization and allocates the stringer, while another volcanic tuffitic and dacitic formation acts like a cap rock. Posterior deformation and exposure to the paleo surface determined the epigenetic gossan and transition massive sulphides mineralization.

The south deposit, despite some evidence of syngenetic stringer mineralization, important structurally controlled late to post Variscan remobilizations form the most part of the higher-grade mineralization that appears to be controlled by favourable rheology in higher altered and harder andesitic to basaltic host rocks (iV).

1.7 EXPLORATION

Since Ascendant Resources entry in 2018, the main exploration goals have been to increase and upgrade the mineral resources to the Measured/Indicated levels, and to establish mineralogical/metallurgical characteristics to pave the way for economic studies. Notable progress has been achieved on all fronts. The main exploration activities have included petrographic studies, geophysical investigations (complemented by core drilling) and metallurgical tests.

Regular petrographic studies in core, pulps and metallurgical feeds and tailings have been developed, including modal mineralogy, deportment, mineral association, liberation classes and liberation grades.

Mineralogical and geochemical studies were elaborated steadily in order to support the definition and development of the metallurgical domains and sub-domains.

Beside the reinterpretation of historical Gravimetric data, a considerable amount of regional and downhole I.P. and E.M. geophysical surveys were conducted systematically.

1.8 DRILLING

Lagoa Salgada project has been explored mainly by drilling since 1995. Due to the absence of Paleozoic outcrops, the geological cores obtained through drilling are the best source of information for the study of the various disciplines inherent in the project's progress. Accordingly, Redcorp has conducted drilling programs at the LS Property since 2005, including 2005-2009 under the direction of Redcorp Ventures, and 2011 – 2012 under the direction of Portex. The focus of drilling since 2007 has been on the North deposit and the South deposit areas. Since 2017 with the entrance of Ascendant, a total of 98 drill-holes totalling 35,795.7 m have been drilled.

The 2022 drilling campaign main goal was to perform a series of step-outs with the aim of extending and infilling the northern area of the south sector. In addition, 11 holes were executed to collect samples for metallurgical testwork.

1.9 MINERAL PROCESSING AND METALLURGICAL TESTING

Metallurgical testing of Lagoa Salgada samples began in 2019 with a series of characterisation tests completed by Grinding Solutions (GSL hereafter) in Cornwall, UK. The work was followed by additional metallurgical testing at the same laboratory in 2021, 2022 and 2023. Several mineralogical assessments were also conducted in parallel by Petrolab Laboratory, as well as reproducibility flotation testwork developed by Maelgwyn Mineral Services Africa.

The approach to flowsheet development was to prepare representative master composites for each ore type, then proceed through open circuit to identify and optimize flowsheet conditions and reagent schemes. Locked cycle tests were then conducted on some of the master composites to demonstrate the anticipated overall metallurgical performance within a continuous circuit.

Tests were completed on blends of Primary Massive Sulphide (PMS), Stockwork (STW), Gossan (GOS), Transition Massive Sulphide (TMS) and Stringer (STR) ores, to allow comparison with individual composite results and assess the viability of co-processing the ore types.

Mineralogical assessments were undertaken to provide information to refine the comminution/beneficiation process during optimization, and to provide reasonable expectations for metallurgical performance versus mineral liberation and association within each ore type. Samples from various open and locked cycle test products were used to characterise final concentrates and tailings.

Although the principal domains testing results are conclusive and well advanced, there is still work to be developed with all the domains to confirm the final grades, recoveries and process flowsheet being published in this document.

The developed metallurgical models were applied to mine production schedules as part of the financial modelling process. The resulting average recoveries over the life of the mine (LOM) are presented in Table 1.1.

Table 1.1 - LOM average metallurgical recoveries

Mine Reserve Domain	Cu	Pb	Zn	Ag	Au	Sn
GOS	-	20%	-	89%	91%	40%
TMS	30%	50%	50%	75%	60%	40%
STR	60%	50%	70%	25%	0.5%	-
PMS	30%	60%	70%	45%	30%	40%
STW	65%	65%	75%	25%	5%	-

1.10 MINERAL RESOURCE ESTIMATION

The mineral resources for the Lagoa Salgada project (North and South deposits) have been estimated using the Ordinary Kriging technique after completing geostatistical studies of the deposits to obtain the estimation parameters. The estimates for the deposits were validated using the Inverse Distance (ID) and Nearest Neighbour techniques, and visually by reviewing block model sections to confirm that the block grades honour the drill hole data.

The estimated mineral resources for the North and South deposits are summarised in Figure 1.1. All resource parameters are disclosed in Section 14 of this Report. The effective dates of the estimates are shown on the respective tables.

Figure 1.1 - LS Property Overall North and South Deposit Resources as of May 23, 2023

Deposit	Domain	Category	Density g/cm³	Mass Mt	NSR-MRE $/t	Zn %	Pb %	Cu %	Sn %	Ag g/t	Au g/t	ZnEq %	NSR-MRE M$	Zn kt	Pb kt	Cu kt	Sn kt	Ag koz	Au koz
North	Gos	**Measured + Indicated**	**2.96**	**1.89**	**109.0**	**0.46**	**2.51**	**0.10**	**0.26**	**44.4**	**0.86**	**8.38**	**206.2**	**8.8**	**47.4**	**1.9**	**4.8**	**2 700.1**	**52.5**
		Inferred	2.56	0.35	81.1	0.42	1.67	0.10	0.10	36.9	0.79	5.77	28.6	1.5	5.9	0.4	0.4	418.2	9.0
	tMS	**Measured + Indicated**	**4.06**	**0.92**	**107.7**	**0.25**	**2.47**	**1.08**	**0.20**	**121.6**	**1.24**	**13.27**	**98.9**	**2.3**	**22.7**	**9.9**	**1.8**	**3 591.2**	**36.5**
		Inferred	3.74	0.07	75.6	0.27	2.12	0.84	0.14	74.3	1.41	10.74	5.6	0.2	1.6	0.6	0.1	177.7	3.4
	pMS	**Measured + Indicated**	**4.44**	**5.46**	**143.7**	**3.79**	**2.90**	**0.28**	**0.13**	**66.5**	**0.72**	**11.58**	**784.1**	**206.8**	**158.3**	**15.0**	**7.1**	**11 677.1**	**125.6**
		Inferred	4.81	0.06	88.7	2.78	1.47	0.11	0.07	33.6	0.40	6.73	5.2	1.6	0.9	0.1	0.0	62.9	0.7
	str	**Measured + Indicated**	**2.91**	**0.63**	**45.0**	**0.75**	**0.24**	**0.48**	**0.08**	**22.1**	**0.06**	**3.82**	**28.5**	**4.8**	**1.5**	**3.0**	**0.5**	**450.8**	**1.2**
		Inferred	2.83	0.01	44.8	0.58	0.38	0.46	0.13	29.4	0.20	4.75	0.3	0.0	0.0	0.0	0.0	7.2	0.0
North	**Total**	**Measured + Indicated**	**3.85**	**8.90**	**125.6**	**2.50**	**2.58**	**0.34**	**0.16**	**64.4**	**0.75**	**10.52**	**1 117.7**	**222.7**	**229.8**	**29.9**	**14.3**	**18 419.3**	**215.8**
		Inferred	2.86	0.49	80.6	0.68	1.70	0.22	0.11	42.0	0.83	6.62	39.7	3.3	8.4	1.1	0.5	666.0	13.1

Deposit	Domain	Category	Density g/cm³	Mass Mt	NSR-MRE $/t	Cu %	Zn %	Pb %	Ag g/t	Au g/t	CuEq %	NSR-MRE M$	Cu kt	Zn kt	Pb kt	Ag koz	Au koz
South	C1 FR	**Indicated**	**2.89**	**6.43**	**63.3**	**0.40**	**1.26**	**0.75**	**13.13**	**0.07**	**1.21**	**407.0**	**25.5**	**81.2**	**48.2**	**2 715.4**	**13.7**
		Inferred	3.06	0.83	50.4	0.61	0.47	0.23	12.26	0.13	1.04	42.1	5.1	3.9	1.9	329.0	3.4
	C2 FR	**Indicated**	**2.86**	**2.50**	**67.5**	**0.50**	**1.23**	**0.64**	**15.58**	**0.04**	**1.28**	**168.5**	**12.4**	**30.8**	**16.0**	**1 251.4**	**3.6**
		Inferred	3.01	3.66	62.0	0.54	0.96	0.42	17.32	0.08	1.20	226.6	19.6	35.0	15.5	2 036.7	9.6
	C3 FR	**Indicated**	**2.85**	**1.11**	**62.5**	**0.39**	**1.17**	**0.54**	**20.77**	**0.04**	**1.16**	**69.2**	**4.3**	**13.0**	**6.0**	**740.0**	**1.3**
		Inferred	3.07	3.64	60.1	0.67	0.69	0.22	16.62	0.04	1.16	218.7	24.5	25.1	7.9	1 943.9	5.3
South	**Total**	**Indicated**	**2.88**	**10.04**	**64.2**	**0.42**	**1.24**	**0.70**	**14.58**	**0.06**	**1.22**	**644.7**	**42.2**	**124.9**	**70.2**	**4 706.8**	**18.6**
		Inferred	3.04	8.13	59.9	0.60	0.79	0.31	16.49	0.07	1.16	487.4	49.1	64.0	25.3	4 309.5	18.3

Differences may occur in totals due to rounding.

1. Mineral resources unlike mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
2. The mineral resources have been estimated in accordance with the CIM Best Practice Guidelines (2019) and the CIM Definition Standards (2014). Coherent and contiguous mining shapes backing reasonable prospect for economic extraction were considered.
3. Mineralized Zones: Gos=Gossan, tMS= transition massive sulphides pMS= primary massive sulphides, Str=Stringer, FR= -Fissural Remobilizations.
4. ZnEq% = ((Zn Grade*26.46)+(Pb Grade*22.05)+(Cu Grade * 77.16)+(Au Grade*54.66)+(Ag Grade*0.71)+(Sn Grade * 264.55))/26.46
5. CuEq% = ((Zn Grade*26.46)+(Pb Grade*22.05)+(Cu Grade * 77.16)+(Au Grade*54.66)+(Ag Grade*0.71)+(Sn Grade * 264.55))/77.16
6. Metal Prices: Cu $7,716/t, Zn $2,646/t, Pb $2,205/t, Au $1,700/oz, Ag $22.00/oz, Sn $26,455/t.
7. NSR-MRE: Gos= ((Sn%*0.40)*Sn$)+((Pb%*0.20)*Pb$)+((Au ppm*0.91)*Au$)+((Ag ppm*0.89)* Ag$)
   tMS= ((Sn%*0.40)*Sn$)+((Pb%*0.50)*Pb$)+((Ag ppm*0.65)* Ag$)+((Zn% *0.5)*Zn$)
   pMS= ((Sn%*0.40)*Sn$)+((Pb%*0.60)*Pb$)+((Ag ppm*0.45)* Ag$)+((Zn%*0.70)*Zn$)
   str= ((Cu%*0.60)*Cu$)+((Pb%*0.5)*Pb$)+((Ag ppm*0.4)* Ag$)+((Zn%*0.70)*Zn$)
   South Deposit= ((Cu%*0.65)*Cu$)+((Zn%*0.75)*Zn$)+((Pb%*0.65)*Pb$)+((Au ppm*0.05)*Au$)+((Ag ppm*0.80)*Ag$)
8. NSR-MRE cut-off: Gos= $48.24, tMS= $46.54, pMS= $41.06, str= $37.83, South Deposit= $37.83

1.11 MINERAL RESERVE ESTIMATES

The mine design and Mineral Reserve estimation have been completed to a level appropriate for feasibility studies. The Mineral Reserve estimate stated herein is consistent with the National Instrument 43-101: Standards of Disclosure for Mineral Projects and is suitable for public reporting. As such, the Mineral Reserves are based on Measured and Indicated Resources, and do not include any Inferred Resources.

The Mineral Reserve estimate consists of selected portions of the Measured and Indicated Resources that are enclosed within mineable shapes having an average grade above the Net Smelter Return (NSR) cut-off values.

NSR cut-off grades were established following Mortimer's approach to cut-off grades (Hall, 2014), which identifies a volume of rock as economically mineable when its lowest contained grade pays for itself, and its average grade provides a minimum average profit per tonne. The former ensures that all of the enclosed volume is economically mineable while the latter, or "volume cut-off", assures that the rock volume yields a certain average level of profit per treated tonne, so shared costs such as general & administration are covered by the average production grade.

Table 1.2 details the parameters and values of both cut-off values for each of the considered mining methods.

Table 1.2 - NSR cut-off values per mining method and material

Mining Method	Material	Mining Cost	Processing Cost	Boundary cut-off	G&A	Volume cut-off
LHOS	STWK	19.5	16.31	**35.81**	2.5	**38.31**
	PMS	19.5	19.62	**39.12**	2.5	**41.62**
	TMS	19.5	14.48	**33.98**	2.5	**36.48**
	GOS	19.5	16.22	**35.72**	2.5	**38.22**
	STR	19.5	16.31	**35.81**	2.5	**38.31**
CF	STWK	29.98	16.31	**46.29**	2.5	**48.79**
	PMS	29.98	19.62	**49.60**	2.5	**52.10**
	TMS	29.98	14.48	**44.46**	2.5	**46.96**
	GOS	29.98	16.22	**46.20**	2.5	**48.70**
	STR	29.98	16.31	**46.29**	2.5	**48.79**

In the estimate of the Mineral Reserves, modifying factors were applied to the tonnages and grade of all mining shapes (both LHOS and cut & fill) to account for unplanned dilution and ore losses that are anticipated in this project.

The factors used in the estimate of the Mineral Reserves are presented in Table 1.3.

Table 1.3 – Dilution and recovery factors

Mining method	Dilution factor (%)	Recovery Factor (%)
LHOS	10	85
CF	10	95

Table 1.4 and Table 1.5 present the Mineral Reserve estimate, broken down by mine zone, reserve category and lithology. The values presented herein are reported in a diluted and recovered mass and grade basis and have been rounded to an appropriate number of significant figures reflecting the uncertainty in the estimate; thus, totals may reflect small rounding errors.

Table 1.4 - Mineral Reserve estimate per mine area

Zone	Reserve category	Mass (kt)	Cu (%)	Pb (%)	Zn (%)	Sn (%)	Ag (ppm)	Au (ppm)
North	Proven	2 100	0.26	2.8	3.2	0.15	62	0.75
	Probable	4 900	0.34	2.3	2.1	0.14	63	0.74
	Total							
South	Proven	-	-	-	-	-	-	-
	Probable	7 600	0.41	0.72	1.3	0	15	0.05
	Total					-		
Total	Proven	2 100	0.26	2.8	3.2	0.15	62	0.75
	Probable	12 500	0.38	1.3	1.6	0.06	33	0.32
	Total	14 600	0.37	1.5	1.8	0.07	37	0.38

Table 1.5 – Mineral Reserve estimate per lithology

Zone	Lithology	Mass (kt)	Cu (%)	Pb (%)	Zn (%)	Sn (%)	Ag (ppm)	Au (ppm)
North	Gossan	1 200	0.1	2.4	0.41	0.25	52	1.0
	Transition	880	0.98	2.2	0.23	0.18	110	1.1
	Massive sulphides	4 400	0.25	2.7	3.6	0.12	62	0.68
	Stringer	550	0.23	0.18	0.54	0.06	13	0.06
	Total	7 100	0.31	2.4	2.4	0.15	63	0.75
South	Stockwork	7 800	0.41	0.71	1.3	-	15	0.06
	Total	7 800	0.41	0.71	1.3	-	15	0.06

1.12 MINING METHODS

The proposed underground mine design supports the annual extraction of 1.2 million ore tonnes ("Mtpa") through a combination of transverse sublevel stoping and cut&fill. Cemented paste is ought to be employed as backfill material of the mining voids, so ore recovery and productivity are maximized.

A main decline, starting from the surface portal, placed close to the processing plant, will be used to access the mine. This main decline splits into two ramps, each providing access to one of the mine zones (North and South orebodies).

The use of an independent ramp per zone, instead of one providing access to both zones, was preferred as it reduces the initial Capital Cost ("CAPEX"). Production starts earlier in the North zone and the mine economics favours the ramp design option, providing faster access to it, which in this case is achieved through a dedicated zone ramp.

A pre-production development program will be required to provide access to the initial stoping levels in the North zone during the first eighteen months. Production will start in the second year, reaching the nominal plant feed in the fourth year.

Sublevel stoping extraction levels (Fresh Massive Sulphides, Stockwork and Stringer) were designed with a 25 m vertical spacing, as determined by the stope height; all of them are accessed through the zone ramp. Footwall drives, as a rule, are designed with a minimum 35 m standoff distance from the ore contact. Cut & fill is the mining method employed for the extraction of the Gossan and Transition lithologies in the North orebody.

The primary means of backfilling at the Property will be cemented pastefill, generated from thickened mill tailings and mixed with a blend of sand and binding agents to a defined recipe.

Two paste fill mixes, one for stockwork lithology and other for the North (i.e., massive sulphide, transition zone, gossan, and stringer lithologies), will be used in the backfill of mining voids.

Waste rock will need to be disposed of on an ongoing basis throughout the mine life. Waste will be hauled to surface for disposal at a designated waste dump. All the waste generated during the pre-production phase of the Project is anticipated to be used as construction material for the TSF dam and the RoM pad ramp.

A fleet of LHDs ("Load-Haul-Dump") units will be used for both material loading and tramming between production areas and dedicated level stockpiles. From the level stockpiles, ore will be loaded into 65-tonne mine trucks and hauled to a Run of Mine ("ROM") pad, located on the surface. Waste is also transported to the surface with 65-tonne mine trucks.

The ventilation system is designed as a two parallel branch system. Fresh airflow ingresses through the main ramp and a central ventilation shaft. It is then distributed into two branches, one serving a mine area (i.e., North and South). Polluted air is exhausted through

two exhaust shafts, located at the far end of each area. Two main ventilation fans are installed on the surface, one per ventilation exhaust shaft.

The pumping system is a high-pressure stage system with dewatering point located at the vent shaft, and intermediary levels draining via ventilation raises to main pumping levels. This system maximizes the pumps efficiency and reduces the number of pumps required, allows to have enough space to decant the water before pumping and minimizes the length of the pipeline, ensuring a properly maintenance. Using the Monte-Carlo method, the freshwater flow rate required to supply 95% of the possible water consumption scenarios was estimated, the required flow, including leakages, is 9.2 l/s (or 33 m^3/h) with a peak demand of 16 l/s. The average groundwater inflow was estimated, considering the available studies at the date of this report, to be 30 l/s in the North and 15 l/s in the South. These pumping flow rates ensure that the dewatering system can pump the peak demand and has an allowance to accommodate for uncertainty in the groundwater inflow.

Figure 1.2 below outlines the annual mine production by ore zone.

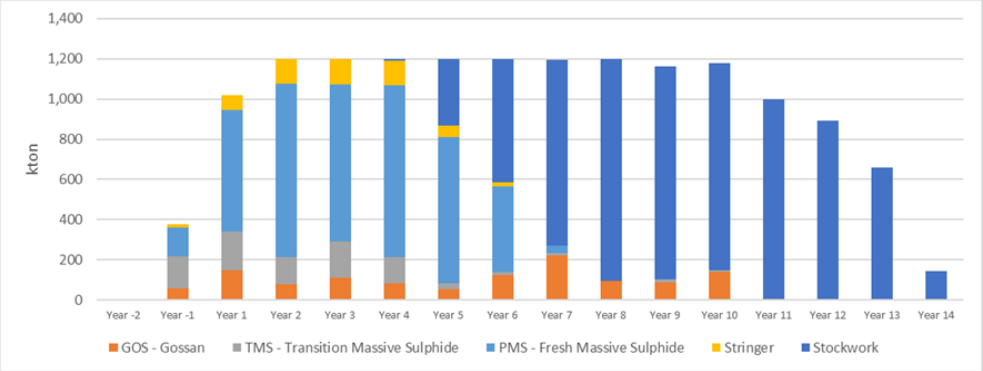


Figure 1.2 - Underground Production Profile

The mine operation labour force will consist of mine management and technical support staff, development and production crews, maintenance crews, and miscellaneous support staff. The labour schedules determined for the operation identified a peak on site complement of up to 300 personnel.

The underground mobile fleet will be BEV for the LHDs, haulage trucks and mixer trucks and the remaining will be diesel. The fleet of mobile mining equipment will include LHDs, haulage trucks, drill jumbos, bolters, cable bolters, charging platform, top hammer drills, shotcrete units, mixer trucks and ancillary equipment. Some key mobile equipment units will be purchased through a lease-to-own program. Since the mine is close to the surface and with easy access, maintenance of the UG fleet will occur in the surface shop.

1.13 RECOVERY METHODS

The mineral treatment plant consists of 10 process areas designed for crushing, grinding, and flotation processes to produce various concentrates such as Cu, Pb, Zn, Sn, and Au/Ag dore bars. The process areas are tailored to the specific mineral domains being processed.

Common process areas for all mineral domains include crushing, grinding, dewatering, reagents, and auxiliary services. Specific process areas for Fresh Massive Sulphide, Transition Sulphide, Stockwork and Stringer, and Gossan are configured based on the mineral domain.

In the crushing area (Area 100), ore is trucked from underground and dumped into a RoM Bin or fed via the ROM pad. The material goes through a grizzly feeder and primary jaw crusher, and then onto a conveyor belt leading to a silo. An emergency bypass flopper gate directs material either to the silo or a bypass conveyor and stockpile for contingency.

The grinding circuit (Area 200) receives material from the crushing area and produces a final grinded product. It consists of a SAG mill, ball mill, and vertical mill in closed circuit with hydrocyclones. The material is discharged into a vibrating screen, with rejected pebbles recirculated to a pebble crusher. The grinded product is collected in pump boxes, and the overflow represents the final grinding stream for flotation.

The copper and lead flotation circuit (Area 300) includes aeration and conditioning tanks, rougher cells, regrinding mill, and cleaning stages. The circuit can produce bulk or separate Cu and Pb concentrates depending on the mineral domain. Flotation volume is adjusted based on the domain being processed, with slurry flowing through all cells even when not in use.

The zinc flotation circuit (Area 400) consists of conditioning tanks, rougher cells, regrinding mill, and cleaning stages. Except for Fresh Massive Sulphide, only the rougher and cleaner circuit is used. Flotation volume is adjusted based on the mineral domain, and slurry flows through all cells for flexibility.

The sulphide flotation circuit (Area 500) removes sulphides before concentrating tin minerals. It includes conditioning tanks, rougher cells, and cleaner stages. Rougher tailings flow to the next area, while the concentrate is pumped to the tailings management area.

Area 600 - flotation circuit combines flotation and gravimetric concentration technologies. It includes conditioner and aeration tanks, rougher and cleaner stages. The intermediate tin concentrate is further processed using multi-gravity separators to increase tin content.

The process area 700 aims to recover Au and Ag from the Gossan domain using a conventional cyanide leaching process, followed by CCD and Merrill-Crowe precipitation. The equipment will be separate from the main process building. The material from the grinding circuit undergoes screening to remove damaging elements before being pumped to the leaching area. The leaching tanks receive a cyanide solution to leach the metals. The slurry proceeds to CCD thickeners in a counter-current flow pattern, maximizing separation efficiency and mineral concentration. The overflow from the first thickener goes to a

storage tank feeding the Merrill-Crowe and refinery area, where zinc is added for efficient metal precipitation and smelting to obtain dore bars. The entire area is on a concrete basin with a negative slope for efficient slurry collection and return to the circuit via sump pumps.

The mineral treatment plant is designed with concrete basins to collect high or low-grade materials, and spillages are returned to the circuit through sump pumps. Various equipment such as crushers, conveyors, screens, pumps, and hydrocyclones are used in the different process areas.

Each process area's mechanical equipment and flowsheet configuration are described in detail in the subsequent chapters.

1.14 BIO-LEACHING

Bio-leaching of the TMS test results achieved following (needs to be pursued in more detail moving forward):

- Mass reduction of tailings by 70%/ reduction of CN leach feed by 70%;
- Generation of H2SO4 from bio-leach;
- Potentially increasing Au and Ag extraction from TMS feed as well as PMS Zn Ro Tails; and
- Potential Sn recovery from CN leach residue through gravity on both TMS and PMS.

1.15 PROJECT INFRASTRUCTURE

The Lagoa Salgada Project will be developed on a greenfield site located in close proximity to Grândola, Setúbal district, which benefits from well-established infrastructure including road and rail transport, power, and water supply services. The site will be accessed via an upgraded tarmac road connected to the municipal road EM-543. Three access routes have been studied, with one preferred access for light vehicles (Secondary Road access) and two alternative access routes for heavy vehicles. Transportation of supplies will be facilitated by trucks from Portugal or Spanish locations, while concentrate products will initially be shipped to the Sines port by road and subsequently by ship to final destinations.

The project site will have a compact layout that incorporates essential components such as the tailings storage facility, ore and waste dumps, water treatment infrastructure, and various buildings including administration, warehouse, laboratory, gatehouse, and mobile equipment workshop. The processing facilities will consist of a primary crusher building, ore stockpiles on the ROM pad, a mill building, and a paste plant building.

The mine access, known as the mine portal, will be implemented using a Cut & Cover solution. The ore will be brought to the surface and stored as stockpiles on the ROM pad near the Mine Portal, while waste stockpiles will be utilized for constructing the embankments of the Tailings Storage Facility (TSF). These stockpiles will serve for ore

processing or permanent storage, with the material being crushed at the surface by the Primary Crusher and transported to the processing plant via conveyor belts.

During the construction phase, power supply for the site will be obtained by connecting to the existing grid, while an overhead transmission line will be installed during operations to replace the grid connection. The primary source of water supply for the project will be the underground mine dewatering system. In the next phase of the project, a trade-off study must be conducted to assess the need for an external water supply.

The mine plan outlines the processing of 14.8 million tonnes (Mt) of ore and the generation of 1.9 Mt of waste rock. After accounting for concentrate and underground backfill, a total of 11.3 Mt of tailings, along with 1.0 Mt of development rock, will be deposited in the TSF. The TSF design provides an estimated storage capacity of 5.9 million cubic meters (Mm^3) for tailings and development rock.

1.16 MARKET STUDIES AND CONTRACTS

The Lagoa Salgada Project will produce a gold/silver dore and four (4) saleable metal concentrates including:

- Zinc concentrates.
- Copper concentrates.
- Lead concentrates.
- Tin concentrate.

The Project will contribute a small portion of the global market's supply for each concentrate, which is characterized by a large and liquid market with transparent daily pricing. Consequently, the project's production is expected to have minimal impact on the overall supply and demand dynamics of the global markets for these concentrates. While the concentrates will be delivered to smelters and refineries in Europe, they can be sold in any major market or smelter worldwide, similar to products from existing mines in the Iberian Pyrite Belt.

Currently, no offtake contracts have been signed for any of the products. However, it is anticipated that these contracts will be negotiated during future financing decisions, which is customary for a project at this stage of development. The company has already confirmed the marketability of the products with several large commodity traders and smelters in Europe. The findings from these discussions have been integrated into the study, particularly concerning market terms for payment, credits for by-products, treatment and refining charges, and penalty rates for any harmful elements.

Additionally, the study incorporates price forecasts for the Lagoa Salgada Project, which are based on long-term consensus among analysts. The assumed prices for financial modelling align with these forecasts. Table 1.6 provides a summary of the analyst estimates, the prices

used for financial modelling, and the current spot prices for the project's primary commodities.

Table 1.6 – Price Forecasts and Assumptions

	Units	Long Term[1]	Spot Prices[2,3]	Modelled Price
Copper Price	US$/lb	$3.71	$3.87	$3.75
Zinc Price	US$/lb	$1.19	$1.18	$1.20
Lead Price	US$/lb	$0.95	$0.96	$1.05
Tin Price	US$/lb	N/A	$12.20	$15.00
Silver Price	US$/oz	$22.01	$25.39	$22.50
Gold Price	US$/oz	$1,672	$2,008	$1,700

Source: 1. CIBC Global Mining Group, Analyst Consensus Prices, April 2023. 2. LME cash contract bid price dated 5 May 2023. 3. Kitco.com dated 5 May 2023.

The transportation costs for the smelter were assumed at US$35 per tonne for Lead and Copper concentrate, and US60$ per tonne for Zinc concentrate, and the sale will deduct treatment and refining charges. Penalties are deducted on final price. The assumptions used for Net Smelter Return calculations and financial modelling are summarized in Chapter 19 based on average concentrate grades.

1.17 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL / COMMUNITY IMPACT

To comply with the Canadian Norm NI-43-101, an environmental section was included in this DFS report. However, it should be noted that the Environmental Impact Assessment (EIA) is still ongoing under the Portuguese Legal Environmental Regime. The main EIA report can only be submitted after the completion of the DFS, which will be evaluated during the Environmental Impact Assessment Procedure.

The project area is located in the Alentejo Region of Portugal, specifically in the Municipality of Grândola. The study area considered for the EIA analysis extends beyond the project site to encompass relevant factors such as water resources, landscape, noise, vibrations, and social impacts. This comprehensive approach ensures a thorough assessment of the Lagoa Salgada Project's environmental impacts.

Environmental and heritage baseline studies have been conducted as part of the Preliminary Study phase. An initial EIA Scoping Proposal (PDA) was conducted in 2019 based on the Preliminary Economic Assessment (PEA) of that year. Subsequently, a new PDA was submitted in 2021 after the decision was made to proceed with the exploitation of the Lagoa Salgada deposit. Ongoing environmental studies are based on the proposed study areas and methodologies from 2022.

The assessment of potential environmental risks and issues considers the construction, operation, and closure stages of the project. Moderate environmental impacts are anticipated, including vegetation removal, alteration of wildlife habitat, impacts on forest areas, changes in land use, and increased noise levels. However, positive impacts at the

social level, such as Portugal's positioning in European mining policies and socioeconomic development, have also been identified. Mitigation measures have been proposed to minimize adverse impacts and compensate for any negative effects.

The environmental compliance and permitting process in Portugal involves several steps. These include the Legal Regime of Single Environmental Permitting (LUA), Environmental Impact Assessment (RJAIA), and the Legal Regime for Industrial Emissions (REI) applicable to Integrated Prevention and Pollution Control (IPPC). The specific path chosen for the Lagoa Salgada Project was identified, along with the applicable legal requirements and options available to obtain the necessary environmental permits.

The project's social and community impacts have been carefully considered, recognizing the importance of public opinion and acceptance by local communities. Social impact studies have been conducted to identify the main social issues and perceptions associated with the mining development of Lagoa Salgada. Key Informant Interviews (KIIs) were conducted with stakeholders, including service providers, interest groups, government authorities, and affected communities. Efforts will be made to align the project with the needs and concerns of neighbouring populations and to minimize negative impacts while enhancing the region's socioeconomic aspects.

In conclusion, this chapter provides an overview of the environmental studies, permitting process, and social/community impact assessment for the Lagoa Salgada Project. It highlights the ongoing EIA, baseline studies, identified risks/issues, mitigation measures, and the importance of addressing social and community concerns. The ultimate goal is to ensure environmental compliance, obtain necessary permits, and foster positive social and community interactions throughout the project's lifecycle.

In addition to the previous, it is also described in the report the major findings of the Baseline Hydrological Study (BHS) and Closure Plan.

1.18 CAPITAL AND OPERATING COSTS

The capital expenditure (Capex) required for implementing the Lagoa Salgada Project has been estimated to a level of accuracy of +/-10% to 15% using the outcomes of the basic engineering study completed by Quadrante, MinePro, IGAN, SLR and RMS (i.e., equipment selection, sizing, and specification; civil, concrete, mechanical, structural, electrical and instrumentation material takeoffs; construction and management services; taxes, etc.) following both international and Portuguese standards.

In what regards to the operating Costs, Ascendant, Quadrante, IGAN and MinePro collaborated to develop the estimates, which serve to determine the annual production costs. These unit costs are presented as total Life of Mine (LOM) expenses and expressed in dollars per tonne processed. All costs included in the estimate are reported in US dollars.

For all the presented costs, the conversion rate applied was 1 EUR€ = USD$ 1.06815.

1.18.1 CAPITAL COSTS

Capital cost estimates were developed for the construction of an underground mining and processing operation capable of treating 1,200,000 tonnes per annum of ore to produce copper, zinc, lead, and tin concentrates, also including credits for gold and silver. Gold and silver bullion will be also produced.

Total capital expenditures of $255.34 million are estimated for the LOM, $267.24 million with contingency. The initial CAPEX (first eighteen construction months) is $152.45 million, $164.35 million with contingency. The capital cost estimate is divided into ten broad-scope areas:

- Support infrastructure.
- Tailings storage facility.
- Paste-fill plant.
- Mining processing plant.
- Underground capital.
- Mobile equipment.
- External connections.
- Box cut.
- Closure cost.

A summary of the estimated LOM capital costs is presented in Table 1.7 and details of the cost breakdowns are summarised in the sub-sections that follow.

The initial capital cost incurred a contingency of 10%, excluding mobile equipment.

Table 1.7 - LOM capital costs ($ Millions)

Description	Initial Capital Cost	Sustaining Capital Cost	Total Capital Cost
Support Infrastructure	18.51	1.05	19.56
Tailings Storage Facility	5.54	29.21	34.75
Pastefill Plant	5.30	3.90	9.20
Process Plant	67.37	0.94	68.31
Underground Capital	25.50	36.89	62.40
Mobile Equipment	24.91	24.90	49.80

Description	Initial Capital Cost	Sustaining Capital Cost	Total Capital Cost
External Connections	3.92	0.00	3.92
Box Cut	1.39	0.00	1.39
Closure Cost	0.00	6.00	6.00
Total	**152.45**	**102.89**	**255.34**
Contingency	11.90	0.00	11.90
Total with Contingency	**164.35**	**102.89**	**267.24**

The Capex excludes:

- Any additional exploration costs.
- Any financing costs, including capitalised interest and commitment fees.
- Costs incurred to establish the operations and conduct feasibility studies prior to project financing.
- Working capital, which refers to operating costs beyond the scope of Capex that are incurred during the construction and ramp-up of the project, net of revenue generated in the same period.
- bridging costs required to prepare the project for construction, such as detailed engineering FEED, land acquisition, and further geotechnical work.

1.18.2 OPERATING COSTS

The estimated total operating expenditures over the LOM period amount to $540.3 million. However, certain non-capital expenditures occurring before the processing plant becomes operational over the first eighteen months are categorised as Pre-production operating costs, totalling $8.5 million.

The operating cost estimate is divided into three main areas: Mining, Processing and General and Administrative (G&A). The detailed operating costs presented here are based on design criteria for the mine and process plant, as well as engineering information and unit cost data obtained from various sources.

The operating cost estimate meets the criteria to be classified as a Class 3 estimate, as defined by AACE International, with an approximate accuracy level of ±15%. It is important to note that all costs included in the estimate are reported in US dollars.

The operating cost estimate has been meticulously calculated using a comprehensive bottom-up approach, which relies on crucial engineering deliverables that define the scope

of the project. The scope itself was described and quantified through a series of line items within material takeoffs (MTOs).

Table 1.8 provides a summarised overview of the estimated Life of Mine (LOM) operating costs. For a more detailed breakdown of the costs, please refer to the subsequent sub-sections. These sub-sections provide a comprehensive summary of the cost breakdowns, allowing for a closer examination of the various components involved.

Table 1.8 - Operating Cost Estimate

Operating Costs	Cost per unit processed ($/t)	LOM Total ($ Millions)
Mining Costs – Drift & Fill	25.24	269.96
Mining Costs – LHOS	17.72	
Processing Costs – Gossan	15.58	241.47
Processing Costs – TMS	13.71	
Processing Costs – PMS	18.72	
Processing Costs – Stringer	15.37	
Processing Costs – Stockwork	15.37	
G&A Costs	2.00	28.89
Total	**36.45**	**540.32**

1.19 ECONOMIC ANALYSIS

The outcomes of the economic analysis provide insights into future prospects but are influenced by numerous known and unknown risks, uncertainties, and other factors that could result in actual results differing significantly from those outlined here. This report contains forward-looking statements, including, but not limited to, projections regarding future metal prices, currency exchange rates, estimations of Mineral Resources and Mineral Reserves, projected mine production and metals recovery, estimated capital and operating expenses, and anticipated cash flows generated from planned mining operations.

The following factors may impact actual results:

- Variances in estimated initial capital costs and development time compared to assumptions.
- Unforeseen fluctuations in ore quantity, grade, or recovery rates, or the presence of harmful elements that could affect the process plant or waste disposal.
- Unexpected discrepancies in geotechnical and hydrogeological conditions compared to assumptions made in mine designs, including water management during construction, mine operations, and post-mine closure.

- Differences in the timing and quantity of projected metal production, future production costs, requirements for sustaining capital, future operating costs, assumed currency exchange rates, the need for additional capital, unexpected equipment or process failures, or equipment not performing as expected.

- Changes in government regulations pertaining to mining operations, the environment, and taxes.

- Unanticipated social risks, increased closure costs, unforeseen closure requirements, disputes over mineral titles, or delays in obtaining surface access to the property.

At present, production mine development and licensing approvals are not yet secured. Prior to the commencement of mining operations, it is necessary to obtain the necessary statutory permits, including environmental permits.

The Project has undergone evaluation through discounted cash flow analysis, which involves assessing the present value of future cash flows. Cash inflows are derived from annual revenue projections, while cash outflows include project capital expenditures, sustaining capital costs, operating costs, taxes, royalties, and obligations to other stakeholders. These outflows are deducted from the revenues to determine the annual cash projections.

Cash flows are assumed to take place at the midpoint of each period. In order to account for the time value of money, annual cash flow projections are discounted to the Project valuation date using the annual discount rate. The choice of discount rate for a particular project can vary based on several factors, such as the nature of the commodity, the cost of capital for the project, and the level of risks associated with the project (e.g., market risk, environmental risk, technical risk, and political risk) compared to the expected returns from equity and money markets.

The project utilizes a base case discount rate of 8%. By summing the discounted present values of the cash flows, the project's Net Present Value (NPV) is determined. Alongside NPV, the Internal Rate of Return (IRR) and payback period are also calculated. The IRR represents the discount rate at which the NPV becomes zero. The payback period is calculated as the duration necessary to attain a positive cumulative cash flow for the project, starting from the initiation of mineral processing.

Taxes and depreciation for the Project were modelled using input from Ascendant.

On a pre-tax basis, the NPV at 8% is 188.8 million, the IRR is 47%. On a post-tax basis, the NPV at 8% is $147.1 million, the IRR is 39% and the payback period is 2 years.

A summary of the LOM cashflow is provided in Table 1.9 and Figure 1.3. Table 1.10 summarizes the economic results with the NPV at 8%.

Table 1.9 - LOM Cashflow Forecast Summary Table

Description	Units	Value
Revenue	$ million	1.386
Penalties	$ million	(29)
TC/RC	$ million	(138)
Freight Costs	$ million	(47)
NSR	$ million	1.172
Royalties	$ million	(40)
Total OPEX	$ million	(540)
EBITDA	$ million	592
Total CAPEX	$ million	(261)
Closure Costs	$ million	(6)
Pre-Tax CashFlow	$ million	325
Taxes	$ million	(65)
Post-tax CashFlow	$ million	260

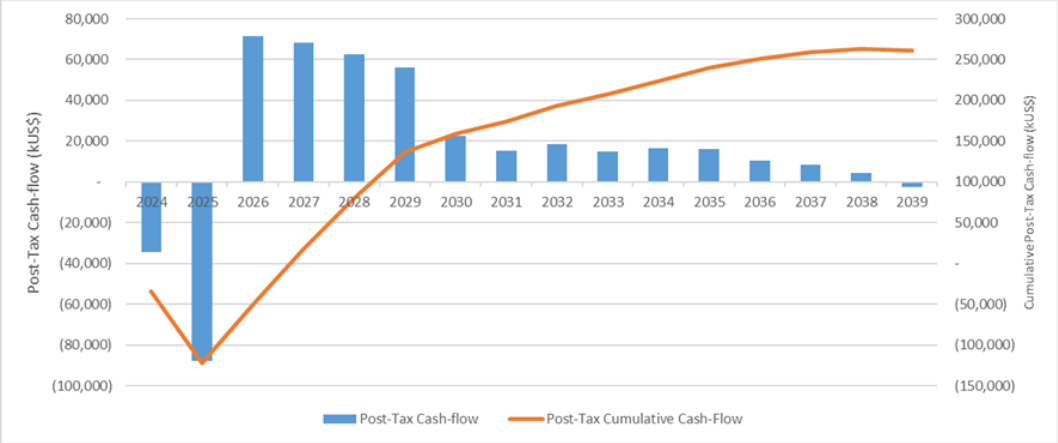


Figure 1.3 - Discounted Cash-Flow After-Tax

Table 1.10 - Forecast Economic Results

Description	Units	Value
Pre-Tax		
NPV at 8%	$ million	188.8
IRR	%	47
After-Tax		
NPV at 8%	$ million	147.1
IRR	%	39

Description	Units	Value
Payback Period	Years	2

1.20 ADJACENT PROPERTIES

There are no properties of any significance directly adjacent to the site.

The following mining operations are currently active in the IPB:

- Aljustrel Mine (Almina).
- Neves-Corvo Mine (Somincor – Lundin Mining).
- Aguas Teñidas, Sotiel, and Magdalena Mines (Matsa – Trafigura/Mubadala).
- Rio Tinto Mine (Atalaya Mining).
- Cobre Las Cruces Mine (CLC – First Quantum Minerals) – Currently reprocessing mined tailings.

In addition to these active mining operations, several projects are at the exploration or permitting stages.

1.21 INTERPRETATION AND CONCLUSIONS

Based on the assumptions outlined in this report, the Project demonstrates favourable economics. The projected cash flow of the Project is particularly influenced by fluctuations in the OPEX costs and the price of zinc.

1.22 RECOMMENDATIONS

Based on the positive economics, it is highly recommended to progress the Lagoa Salgada Project to the next phase of engineering. The recommended development strategy entails continuing the advancement of environmental assessment and licensing activities, alongside the completion of the detailed engineering program. This program will provide additional project definition and help mitigate potential risks that may impact the project's execution timeline.

To ensure accurate cost estimation and project scheduling for permit approvals and the Final Investment Decision (FID), it is essential to intensify engineering and field investigations. This will enhance certainty regarding costs and timelines, thus facilitating smoother permit approvals and the FID process.

It is important to note that the Financial Study (FS) does not include expenses incurred prior to project approval, such as those related to the pre-development program or early engineering costs.

The subsequent subsections provide a comprehensive list of programs recommended for the upcoming phase of engineering work for the Lagoa Salgada Project.

1.22.1 PROCESS PLANT & METALLURGICAL TESTING

All reported mineral concentrates satisfy the specifications of a number of refiners and commodity traders. However, further optimisation work should be undertaken to improve on the results recorded to date. This refers specifically to, the zinc concentrate on Primary Massive Sulphide, Stockwork and Transition Massive Sulphide.

Ascendant is committed to continue metallurgical program test work on Maelgwyn laboratory on the following months to robust the result to be used on the detailed engineering phase.

1.22.2 MINING

Quadrante QP has identified various opportunities that have the potential to further improve project economics:

- Further optimization of the mine plan and stope sequence to identify and realize higher NPV.
- Further optimization of the cut-off NSR, based on the final FS costs, to increase resource conversion and extend the mine life.
- Continue to investigate reducing the addition of sand in the backfill to reduce cost. Further test work is required.
- Recommends checking the plant feed in a smaller time window than the one used for the LOMP. This will ensure that mine output, ROM capacity, and plant changes between processes are correctly aligned to process all the ore produced by the mine.
- Further investigate expanding electrification of underground mining equipment to include major other underground mobile equipment. BEV equipment was determined to be economically beneficial to the project (trucks, loaders and mixer trucks), but as the BEV technology improves, the cost of the equipment is becoming more competitive with the standard diesel equipment with potential project benefits.

1.22.3 INFRASTRUCTURE

The following potential opportunities related to surface infrastructure should be investigated as new technology or information becomes available.

- With the deepening of the hydrogeological study, review the water management system to minimize water consumption, treatment, and discharge needs into the environment.
- With the constant advancement of technology and the consequent reduction in costs, continue to investigate the use of batteries for energy storage in the PV plant to maximize the utilization of solar energy and reduce energy costs.
- Develop a ground investigation program to the TSF; this investigation is intended to provide both hydrogeological and geotechnical data for the TSF area. Additional investigation elements may be required based on the conditions encountered. Hence, a multi-technique and phased approach is recommended, this will allow the efficient coverage of the footprint and follow-up targeting of areas of ground that are of interest. Subsequent to the ground investigation monitoring of the instruments installed will be required.

1.22.4 FURTHER STUDIES

It is recommended that Ascendant proceeds with detailed engineering, which will serve as the foundation for procurement, construction, and commissioning activities. The scope of work will encompass the following:

- Complete engineering tasks to achieve a 40-45% level of completion, in accordance with the AACE Class 2 estimate requirements.
- Prepare Request for Proposal (RFP) packages for construction quotations.
- Clearly define long-lead procurement items and initiate the procurement process for critical path items.
- Create a Level 4 Implementation Schedule for the project.

It is important to note that the feasibility study does not include expenses related to pre-project approval, such as those associated with the previously announced pre-development program (surface preparation and land acquisition), as well as early engineering costs.

2 INTRODUCTION

2.1 OWNERSHIP

The Lagoa Salgada Project ("LS Project") is covered by a single exploitation permit (mining concession) with an area of approximately 10,700 hectares (ha). The exploitation permit, Contrato C-168, is held by Redcorp Empreendimentos Mineiros, LDA. (Redcorp). The Portuguese General Directorate for Energy and Geology (Direção Geral de Energia e Geologia - DGEG) granted the mining concession's signature on the 21st of October 2021. The mining concession is registered in the Diário da República, Public Register, under Contrato (extrato) 526/2021, of December 2nd, 2021.

Redcorp holds an 85% stake in the Lagoa Salgada Project and also acts as the operating entity. The remaining 15% of the stake is held by the Mining Development Company ("EDM"), which is a Portuguese state-owned company that aims, among other things, to carry out activities related to prospecting, research, and exploitation of mineral resources.

Prior to the delivery of the definitive feasibility study ("DFS"), Redcorp is owned 50% by Ascendant Resources Inc. ("Ascendant"), a Canadian company listed on the Toronto Stock Exchange (TSX:ASND), and 50% by Mineral & Financial Investments Limited ("M&FI"), an investment company listed on the Alternative Investment Market (AIM:MAFL). Following the filing of this DFS, Ascendant has the right to earn an 80% interest in Redcorp.

Pursuant to the consortium agreement between Redcorp and EDM, following the delivery of the DFS, EDM has the option to participate in up to a 15% interest (the "EDM Interest") in the LS project and has provided Redcorp with a right of first refusal to purchase the EDM Interest. Ascendant has certain contractual rights set out in agreements with M&FI, which ensure that it will own an effective 80% interest in the LS Project if M&FI is not successful in acquiring the EDM Interest through Redcorp, thus safeguarding its ownership in the LS Project from any impact of the EDM Interest.

2.2 TERMS OF REFERENCE

All measurement units used in this Report are metric unless otherwise noted. The currency used is United States (US) dollars (US$). The Portuguese currency is the Euro (EUR or €). The Report uses UK English.

2.3 SITE VISITS AND SCOPE OF PERSONAL INSPECTION

The authors of this Technical Report possess the necessary qualifications and expertise in various fields such as Mineral Resource/Reserve Estimation, mining, geology, metallurgical process engineering, civil engineering, mechanical engineering, electrical engineering, cost estimating (capital and operating), and mineral economics. Their educational background, professional experience, and affiliations make them Qualified Persons. It is important to

note that none of the Qualified Persons involved in preparing this report have any personal stake or beneficial interest in the Lagoa Salgada Project.

Table 2.1 presents a comprehensive roster of the diligent Qualified Persons (QPs) responsible for meticulously preparing this report. The table meticulously delineates their respective titles, areas of expertise, and documented dates of site visits.

Added to the report are the QP Certificates, duly endorsing their professional competence.

Table 2.1 – Qualified Persons (QPs)

Qualified Person	Title	Area of Responsibility	Site Visit
João Horta, M.Sc., MIMMM	Project Director, Quadrante.	1.1, 1.2, 1.3, 1.4, 1.14, 1.17, 1.18, 1.19, 1.20, 1.21, 2, 3, 4, 5, 6, 12, 18, 21, 22, 23, 24, 25, 26 and 27.	Several, between December 2021 and June 2023.
David Castro López, MIMMM	Process Engineer, Minepro Solutions.	1.9, 1.13, 13, 17, 25.3, 25.7 and 26.1.3	June 7, 2022.
Pablo Gancedo Mínguez, CEng MIMMM QMR	Senior Mining Engineer, IGAN.	1.11, 1.12, 15, 16, 25.4 and 25.5	January 18, 2022.
Charley Murahwi, Msc, P.Geo., FAusIMM	Senior Economic Geologist, Micon.	1.5, 1.6, 1.7, 1.8, 1.10, 7, 8, 9, 10, 11, 12, 25.1, 25.2 and 23.6.1.	16-19 October 2018. 13-17 November 2018. 28-31 May 2019. 24-27 April 2023.
David Ritchie, P.Eng	Principal Geotechnical Engineer, SLR.	18.11.1, 18.11.2 and 18.11.3.	January 18, 2022.
Frank Palkovits, P.Eng., B.Eng.	Pastefill Specialist, Mine Paste.	18.11.4.	January 17, 2022.
Rafael Fernández Rubio, PhD	Specialist consultant, FRASA.	20.7	February 3, 2022.

2.4 EFFECTIVE DATES

- The effective date of the Resource Estimate is May 2023.
- The effective date of the Reserve Estimate is June 2023.
- The effective date of the Feasibility Study is 28 July 2023.

2.5 INFORMATION SOURCES AND REFERENCES

This report draws on several important sources of information, outlined as follows:

- Section 2.7 provides a comprehensive list of previous technical reports.
- References to relevant documents can be found in Chapter 27 of this report.

2.6 UNITS OF MEASURE

The currency units used in this study and report are American Dollars (USD$). Any reference to “dollars” or “$” within this report means American Dollars. The metric system has been used throughout the study and this report. Unless otherwise noted, any reference to “t” or “tonnes” means dry metric tonnes. One metric tonne equals 1,000 kg or 2,204.6 lb

2.7 PREVIOUS TECHNICAL REPORTS

Ascendant has previously filed the following technical reports:

- Murahwi, C., 2019: NI 43-101 Technical Report Resource Estimate for the Lagoa Salgada Project, Setúbal District, Portugal: Prepared by Micon International Limited for Redcorp – Empreendimentos Mineiros, LDA and Ascendant Resources Inc., effective date 8 February 2019.
- Murahwi, C. and Gowans, R. 2019: Technical Report on the Resource Estimate Update for the Lagoa Salgada Project, Setúbal District, Portugal: Prepared by Micon International Limited for Redcorp – Empreendimentos Mineiros, LDA and Ascendant Resources Inc., effective date 5 September 2019.
- Murahwi, C., Methven, P., Zazzi, G, Malhotra, D. 2019: Technical Report and PEA for the Lagoa Salgada Property, Setúbal District, Portugal: Prepared by AMC Mining Consultants (Canada) Ltd. For Ascendant Resources Inc., effective date 19 December 2019.
- Murahwi, C., Methven, P., Zazzi, G, Malhotra, D. 2021: Updated Mineral Resource Estimate for the South Deposit and PEA for the North Deposit, Lagoa Salgada Property, Setúbal District, Portugal: Prepared by Micon International Limited for Ascendant Resources Inc., effective date 31 January 2021 (South Deposit) and 19 December 2019 (North Deposit).
- Nunes, J., Horta, J., Gancedo, P., Murahwi, C., Gowans, R., Doundarov, G.: NI 43-101 Technical Report and PEA for the Lagoa Salgada Project. Prepared by Quadrante for Ascendant Resources Inc., effective date 21 September 2021.

3 RELIANCE ON OTHER EXPERTS

3.1 MINERAL TENURE, SURFACE RIGHTS, AND ENCUMBRANCES

This Technical Report is based on information and materials provided by Ascendant, including discussions on royalties, permitting, taxation, and environmental matters. The discussions on these topics rely on the expertise of Ascendant staff and experts retained by Ascendant. The Qualified Persons (QPs) and Quadrante, however, do not possess the qualifications to provide commentary on such matters. Therefore, the QPs and Quadrante have relied on the representations and documentation provided by Ascendant for these discussions.

All the data utilized in this report were originally supplied by Ascendant. The QPs have thoroughly reviewed and analyzed this data, and they have formed their own conclusions based on it.

The QPs and Quadrante do not provide any legal opinion regarding the validity of the title to the mineral concessions claimed by Ascendant. In this regard, they have relied on the information provided by Ascendant.

Information pertaining to royalties, permitting, taxation, environmental matters, and the validity of the title to the mineral concessions claimed by Ascendant has been extracted from previous NI 43-101 Technical Reports. Ascendant has updated this information through personal communication with the QPs. The previous NI 43-101 Technical Reports and other references used in the compilation of this report can be found in Chapter 2.

3.2 TAXATION

The Qualified Persons (QPs) have explicitly acknowledged their reliance on information provided by Ascendant staff and experts retained by Ascendant regarding taxation, as follows. They have also disclaimed responsibility for this information.

- Portuguese Tax Code, "Código dos Impostos" - Portuguese Official Gazette (*Diário da República*).

The utilization of this information is primarily for the purpose of supporting the economic analysis presented in Chapter 22.

3.3 ROYALTIES

Regarding royalties, the QP has similarly relied on information supplied by Ascendant staff and has explicitly disclaimed responsibility for this information. The specific details regarding the relied-upon information are as follows:

- Exploitation permit, Contrato C-168Y - Portuguese Official Gazette (*Diário da República*), under Contrato (extrato) 526/2021, of December, 2nd, 2021.

3.4 MARKET AND METAL PRICES

The QP explicitly disclaims responsibility for the marketing and metal price forecast information included in the report, as they have relied on experts retained by Ascendant for this information.

Within the project's economic analysis, the QP have relied on bank consensus pricing provided by Ascendant for long-term metal prices, as forecasted in the report. The sources for the marketing and metal pricing information are referenced in Chapter 19 of the report.

Additionally, this information is utilized to support the Mineral Reserve estimate presented in Chapter 15 and the economic analysis in Chapter 22.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 DESCRIPTION AND LOCATION

The Project is in Portugal, approximately 80 km southeast of the capital Lisbon (approximately 120 km by road). It is located approximately 50 km south-east of Setúbal, the regional administrative centre, 12 km north-east of the municipality of Grândola, approximately 2.5 km northeast of the village of Cilha do Pascoal and 3 km northwest of the village of Água Derramada.

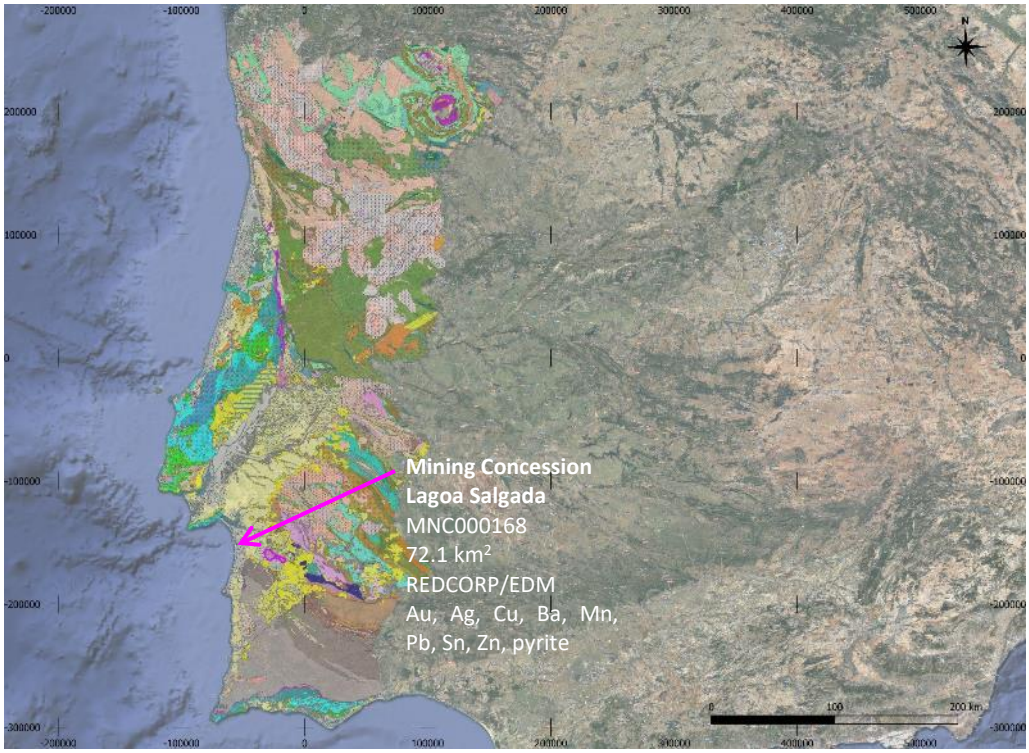


Figure 4.1 – Area of the Lagoa Salgada mining concession within the Iberian Peninsula

Geographically, the LS Property is situated as follows:

- Within the Instituto Geográfico e Cadastral de Portugal (IGCP) map sheets 39-C, 39-D, 42-A, and 42-B (1:50,000 scale maps).
- At approximately 38°14’ North latitude and 8°28’ West longitude in south-western Portugal.
- At approximately 548,000 E; 4,229,000 N, Zone 29 (European Datum 1950) Universal Transverse Mercator (UTM) coordinates.

4.2 LAGOA SALGADA EXPLORATION PERMIT AND PORTUGUESE MINING LAWS

The Project is contained in a single *Contrato de Concessão de Exploração de Depósitos Minerais Metálicos* (Concession Agreement for the Exploitation of Metallic Mineral Deposits – mining permit) with the registration number C-168, covering a total area of approximately 72.1 km^2 signed on 28 October 2021.

The concession is valid for a period of 20 years and, under certain legal and contractual obligations, can be extended 2 times for a maximum of 15 years per extension.

The mining permit, C-168, is held by a joint venture between Redcorp and EDM, a Portuguese Government owned company for the mining sector. Redcorp holds an 85% interest and EDM holds a 15% interest. The *Direção Geral de Energia e Geologi*a (the Portuguese General Directorate for Energy and Geology – DGEG) granted the mining permit. The mining permit is registered in the *Diário da República*, Public Register, under Contrato (extrato) 526/2021, of December 2nd, 2021.

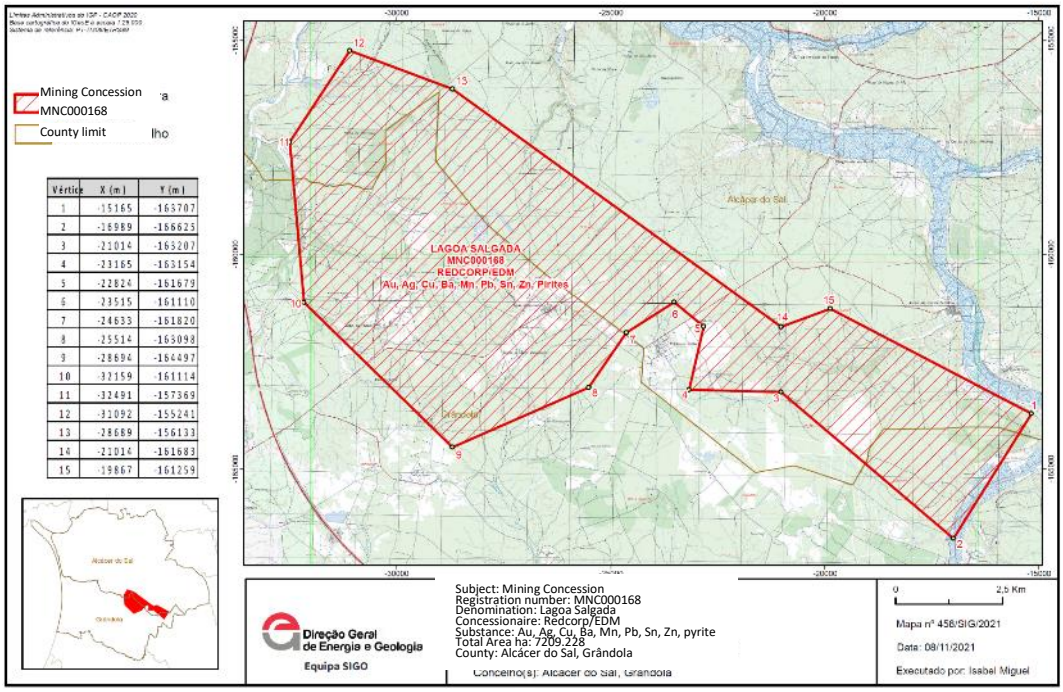
The original exploration permit had an effective expiry date of 20 June 2017, but it was extended to 20 June 2019 after a mining permit valid until October 2041 was signed. Table 4.1 summarises the information regarding the original exploration permit, the renewal and the mining concession information. Figure 4.2 shows the outline of the mining permit. Note that this permit also encompasses the Rio de Moinhos sector at the southwest edge of the area, which is yet to be fully explored and is not part of the resources declared in this Technical Report.

Table 4.1 – Details of the Property Permits

Name	Permit	Expiry Date	Area (km^2)
Lagoa Salgada	Exploration MN/PP/009/08	20 June 2017	133.3
Lagoa Salgada	Exploration MN/PP/009/08	20 June 2019	107.0
Lagoa Salgada	Mining MNC000168	28 October 2041	72.1

In April 2019, Redcorp and EDM applied to the Portuguese Government, through the Secretary of State for Energy, for the definitive mining concession for the LS Property, for an area of approximately 72.1 km^2. The application was accepted, and the decision was published in the Official Gazette of Portugal (*Diário da República – Aviso nº 13907/2019, DR nº 171, 2ª Série de 06 Set.*) and signed on 28 October 2021, as mentioned previously.

Publications in the journal of the Municipality and in a National Journal have also been made.



Vértice	X (m)	Y (m)
1	-15165	-163707
2	-18989	-166625
3	-21014	-165207
4	-23165	-163154
5	-22824	-161679
6	-23515	-161110
7	-24633	-161820
8	-25514	-163098
9	-28694	-164497
10	-32159	-161114
11	-32491	-157369
12	-31092	-155241
13	-28689	-156133
14	-21014	-161683
15	-19867	-161259

Source: DGEG, but legend has been translated into English, November 2021

Figure 4.2 – Area of the Lagoa Salgada mining concession

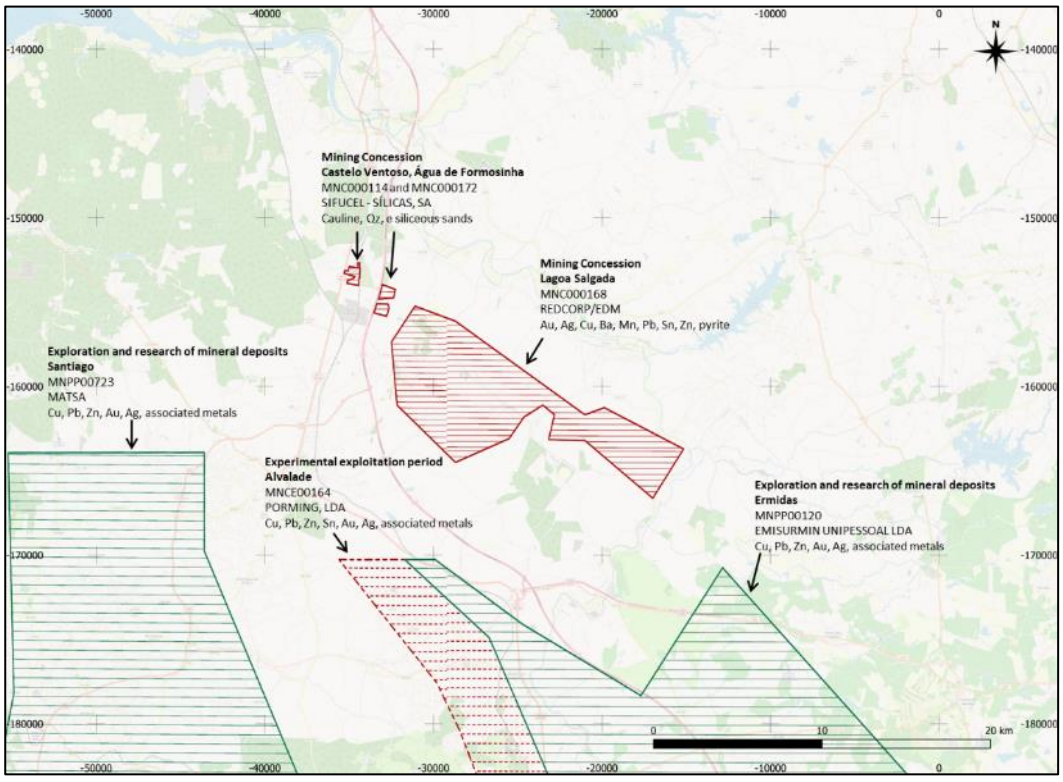


Figure 4.3 – Property Plan

4.3 PROPERTY OWNERSHIP AND AGREEMENTS

On October 28, 2021, Redcorp - Empreendimentos Mineiros, Lda. ("Redcorp") signed the definitive mining concession contract for its Lagoa Salgada VMS project in the prolific Iberian Pyrite Belt, Portugal, with the Portuguese Government through the General Directorate for Energy and Geology ("DGEG").

Prior to the delivery of the DFS, Redcorp is owned 50% by Ascendant, and 50% by Mineral & Financial Investments Limited ("M&FI"). Following the filing of this DFS, Ascendant will earn an 80% interest in Redcorp.

Pursuant to a consortium agreement between Redcorp and Empresa de Desenvolvimento Mineiro, SA ("EDM"), a Portuguese company with public capital whose object is, among others, the pursuit of prospecting, research, and exploration activities of mineral resources, following delivery of a feasibility study, EDM has the option to participate in up to a 15% interest (the "EDM Interest") in the LS project and has provided Redcorp with a right of first refusal to purchase the EDM interest. Ascendant has certain contractual rights set out in agreement with M&FI which ensure that it will own an effective 80% interest in the LS Project if M&FI is not successful in acquiring the EDM Interest through Redcorp, such that its ownership in the LS Project is not impacted by the EDM Interest.

4.3.1 TRANSACTION SUMMARY – KEY OPTIONS TERMS

- On June 22, 2018, Ascendant entered into an agreement with Mineral & Financial Investment AG (formerly TH Crestgate GMBH, "M&FI") to acquire an initial 25% interest, with an option to earn up to an 80% interest in Redcorp. Ascendant acquired the initial effective 25% interest in the Lagoa Salgada Project for an upfront payment of $2.5 million composed of $0.8 million in cash and 2,053,546 Ascendant shares with a value of $1.7 million.

- On May 25, 2022, Ascendant satisfied conditions to increase its effective ownership interest of the Lagoa Salgada Project from 25% to 50%. Ascendant had spent more than $9.0 million in qualifying project expenditures on the Lagoa Salgada Project and made the last $1.0 million milestone payment to the optionor, M&FI, as required to earn a 50% interest in the Lagoa Salgada Project.

- On December 14, 2022, Ascendant satisfied its last financial condition required to increase its ownership percentage in Redcorp from 50% to 80% by making a $2.5 million payment to M&FI.

- Ascendant will increase its interest in Redcorp from 50% to 80% by completing the DFS by June 23, 2023.

- Ascendant will fund all development and future construction costs and recoup M&FI's share of investment through shareholder distributions until all qualifying project expenditures incurred by Ascendant in the Lagoa Salgada Project have been repaid.

- Ascendant will retain a right of first offer on the remaining equity held by TH Crestgate.

4.4 SURFACE RIGHTS, PERMITTING, AND ENVIRONMENTAL LIABILITIES

4.4.1 SURFACE RIGHTS

The surface rights covering the Property are held by three main landowners: Mr Manuel Rocha, Mr Carlos Caiado, and Mr António Rocha. The LS Property is situated within the surface rights of Mr Manuel Rocha and Mr António Rocha. Relations with both landowners are favourable, with an agreement having been made to conduct exploration activities on the Property as well as to negotiate their lands for the construction of the project.

The core logging and sampling facility is in a rented warehouse located approximately 10 km southwest (by road) of the North Deposit (former LS-1 deposit).

4.4.2 PERMITTING

To the QP's knowledge, all the required permits and permissions to access and conduct exploration activities have been obtained from the holders of the surface rights. The exploration activities conducted on the Property do not require any additional permits; however, the proposed exploration programmes are subject to approval by DGEG.

4.4.3 ENVIRONMENTAL LIABILITIES

The QPs are unaware of any environmental liabilities that would prevent Redcorp from conducting exploration activities on the Property.

4.5 ROYALTIES

Royalties are whichever of the following values is the highest, at the discretion of DGEG (Mining Bureau):

- 10% of the net profit.
- 3% of the NSR during the first 3 years of production.
- 4% of the NSR for the fourth year of production.
- 5% of the NSR from year five onwards.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The LS project (Figure 5.1) is situated approximately 80 kilometers (km) southeast and approximately 120 km by road from the capital city of Portugal, Lisbon. It is positioned about 50 km southeast of Setúbal, which serves as the regional administrative center, and approximately 12 km northeast of the municipality of Grândola. The closest village, Cilha do Pascoal, is located approximately 3 km south of the LS project. Access to the Property is facilitated through national highways and roads, with a travel time of approximately 1 hour and 30 minutes from Lisbon.

Furthermore, the property benefits from highway connections to the port facilities in Setúbal and the industrial area and deep-water port facilities in Sines. The nearest town, Grândola, is located 12 km from the Property.

Transportation within the Property is achieved via unpaved roads with a total length of approximately 4 km, accessible by truck/utility vehicle or 4x4 vehicles. These roads are regularly maintained, and some sections can be accessed by standard automobiles.

Figure 5.1 - Lagoa Salgada Project – 3D Masterplan

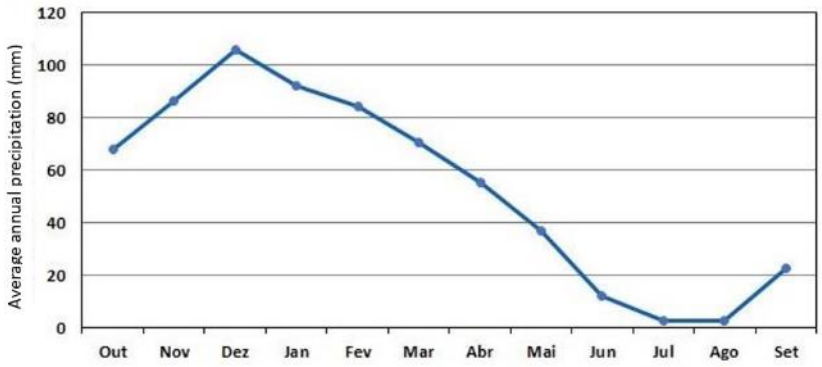
5.2 CLIMATE

The Property is situated within a Mediterranean climate zone (Csa; Köppen climate classification), characterised by a temperate climate with hot and dry summers, as well as moderately cool winters with variable rainfall patterns.

Regarding precipitation, the series correspond to 78 years of observation, from 1939 to 2020, available on the APA website, at http://snirh.pt/. The years 2010 to 2014 were excluded as there were no data that could be analysed. There were other gaps in precipitation data for the years 2001, 2002, and 2009. Out of the various existing methods for filling gaps, it was opted for the method that uses simple linear regressions due to the

scarce number of climatological stations with monthly precipitation data corresponding to the months of the years in which there were gaps in the climatological station (CS) of Grândola. To fill these gaps, it was applied simple linear regression to the records in the month in question, in the post with gaps and in another geographically close post that correlated with the post with the monthly series to be filled. The precipitation data from the geographic posts of Torrão (24H/01UG), Ferreira do Alentejo (25I/01UG) and Azinheira Barros (25G/01UG), near the climatological station (CS) of Grândola, was consulted to ensure that they simulated the characteristics of precipitation in the region and with the aim of finding monthly rainfall data in the years with gaps in the previously-mentioned station; it was also consulted observation years that would allow for a correlation index of over 0.95 between the series of monthly records in this station and in the faulty station.

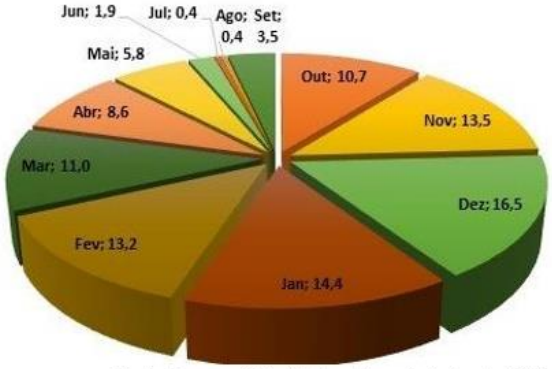
The evolution of monthly rainfall throughout the year in the Project implementation area is almost null in July and August, with December and January being the rainiest months (Figure 5.2).



Data Source: APA. Elaboration: P. Balsa. In BHS.

Figure 5.2 – Monthly average rainfall evolution, in the Grândola AE. (October 1939 - September 2020)

The rainfall in December and January makes up about 31% of the average annual rainfall (Figure 5.3). The precipitation is concentrated mainly in the humid season (October to March), totalling about 507.7 mm, 79% of the average annual precipitation. In the dry season, precipitation totals about 132.8 mm, 21% of the average annual precipitation.

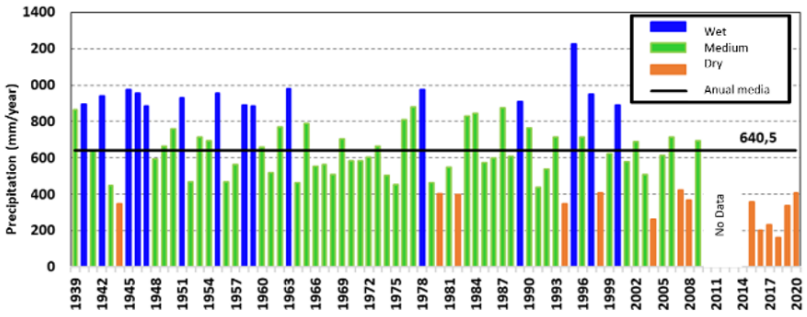


Data Source: APA. Elaboration: P. Balsa. In BHS.

Figure 5.3 - Distribution of the monthly average rainfall in the ND of Grândola

In the evolution of annual precipitation, for the study period considered (90 years), there is a decrease in the occurrence of wet years and an increase in the occurrence of dry years, especially in the first decades of the twenty-first century.

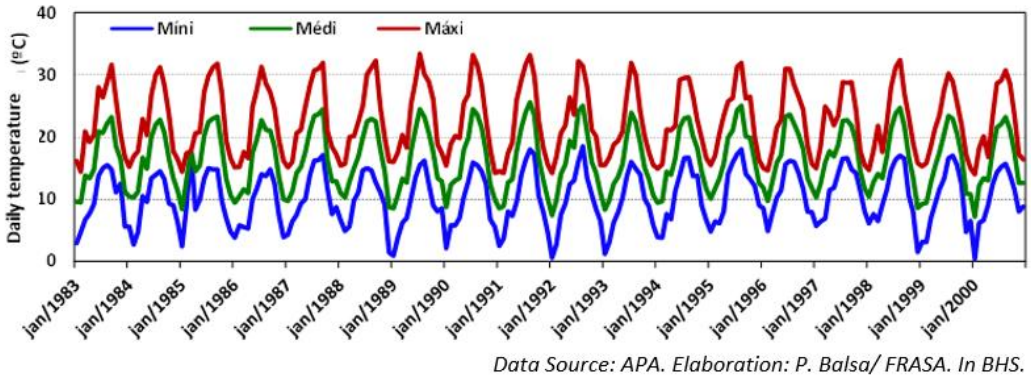
The years were then classified as dry, medium, or wet. The years with precipitation below the 20th percentile value and above the 80th percentile value of the weighted precipitation series in the hydrographic region were considered as dry and wet years, respectively. Thus, the hydrological year is classified as dry, medium, or wet if the mean annual precipitation is in the precipitation percentile ranges of 0% to 20%, 20% to 80% and 80% to 100%, respectively. The dry year precipitation is 433.1 mm, about 68% of the mean annual precipitation, and the wet year precipitation is 881.7 mm, about 138% of the mean annual precipitation. The minimum annual precipitation value, 159.7 mm was recorded in the hydrological year 2018/2019, and the maximum annual value, 1 227.8 mm, occurred in the year 1995/96 (Figure 5.4).



Data Source: APA. Elaboration: P. Balsa/FRASA. In BHS.

Figure 5.4 - Annual precipitation in the CS of Grândola

For the classification of the air temperature in the Project implementation area, the temperature series provided by APA and available at http://snirh.pt/ were used. The available series correspond to 18 years of observation from 1983 to 2000 (Figure 5.5).



Data Source: APA. Elaboration: P. Balsa/ FRASA. In BHS.

Figure 5.5 - Daily minimum, medium and maximum temperatures

The minimum, average, and maximum temperature values collected are shown.

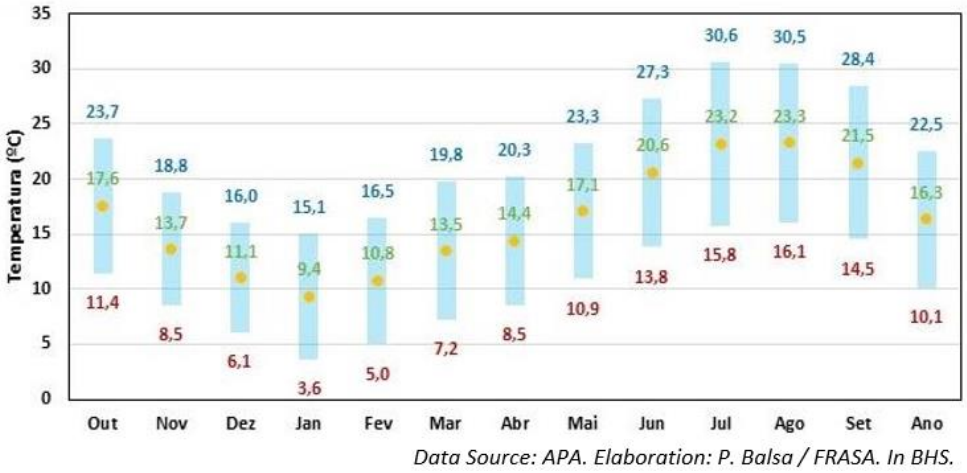
The variation of the maximum, medium and minimum average monthly temperatures, in ºC, for the CS of Grândola are represented. Table 5.1 shows the variation of the maximum, medium and minimum temperatures, as well as the average monthly thermic amplitude for the CS of Grândola.

Table 5.1 – Variation of the maximum, medium, minimum temperatures and average monthly temperature range (ºC) for Grândola CS

Temperature	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sept	Oct	Nov	Dec	Year
Maximum	15.1	16.5	19.8	20.3	23.3	27.3	30.6	30.5	28.4	23.7	18.8	16.0	22.5
Medium	9.4	10.8	13.5	14.4	17.1	20.6	23.2	23.3	21.5	17.6	13.7	11.1	16.3
Minimum	3.6	5.0	7.2	8.5	10.9	13.8	15.8	16.1	14.5	11.4	8.5	6.1	10.1
Thermic amplitude	11.4	11.5	12.6	11.7	12.3	13.5	14.9	14.4	13.9	12.3	10.2	9.9	12.4

Data Source: APA. Prepared by: P. Balsa. In BHS

Figure 5.6 displays the graphical representation for the variation of the maximum, medium and minimum average monthly temperatures, in ºC, for the CS of Grândola.



Data Source: APA. Elaboration: P. Balsa / FRASA. In BHS.

Figure 5.6 - Monthly and annual variations of the maximum, medium and minimum average temperatures, in ºC, in the CS of Grândola

The average monthly maximum temperature varies between 15.1 °C in January and 30.6 °C in July, while the average monthly minimum temperature varies between 3.6 °C and 16.1 °C in January and August, respectively. The average monthly temperature range varies between 9.9 °C in December and 14.9 °C in July. Regarding the average monthly temperature, the year is divided into two seasons, a colder one, from November to April, in which the average monthly temperature is lower than the average annual temperature, and a warmer one, from May to October, in which the average monthly temperature is higher than the average annual temperature. The average monthly temperature varies between 9.4 °C and 23.3 °C in January and August, respectively.

5.3 PHYSIOGRAPHY

The property consists of predominantly flat terrain, with elevations varying from 20 meters to 100 meters above sea level. The vegetation found on the Property is representative of the characteristic flora observed in dry Mediterranean climates. It comprises scrub brush, tall grasses and stands of pine trees. The land within the Property boundaries is privately owned and primarily dedicated to the cultivation of pine trees, eucalyptus and sporadic cork trees.

The soil composition in the Project area is predominantly sandy, indicating a high proportion of sand particles in the soil matrix. The sandy soil has resulted in limited exposure of the underlying Tertiary sedimentary bedrock, suggesting that the bedrock remains largely concealed beneath the surface layers.

Source: Photo taken by Quadrante, August 2021

Figure 5.7 - View of the Lagoa Salgada Property

5.4 LOCAL RESOURCES

is the closest populated area to the Project is Grândola, which has an approximate population of 14,000 and a predominantly agricultural economic sector. Grândola is located within the Setúbal district, which has an estimated population of around 90,000 and is positioned equidistantly between the LS Project and Lisbon. Notably, the Setúbal district accommodates port facilities utilised by the Neves-Corvo Mine and Autoeuropa, the largest automobile manufacturer in Portugal and a prominent exporter. The local economy in Setúbal revolves around industries related to cellulose, paper, and cement.

Adjacent to the Project lies the industrial town of Sines, situated 60 km away and accessible via a well-connected highway network. Sines is renowned for its heavy and chemical industries, housing the primary deep-water port in Portugal. Both the Setúbal and Sines ports are integrated into the national rail network, facilitating efficient transportation of goods. These towns serve as convenient sources of essential services and supplies.

The LS Property is geographically positioned 50 km away from the Aljustrel Mine, which focuses on zinc and lead extraction, and 85 km from the Neves Corvo Mine, primarily engaged in zinc and copper mining. The proximity to these mines signifies the availability of locally accessible skilled mining personnel. In addition, labor resources can be recruited from nearby towns, villages, other regions within Portugal, or even the Estremadura region in Spain, where several active mines are situated along the extension of the Iberian Pyrite Belt.

The Project is endowed with substantial land holdings suitable for exploration and development activities. As previously highlighted, the Property benefits from excellent

connectivity to major infrastructure systems in Portugal, including well-established road networks, railway connections and electric power lines, as well as port and airport facilities.

In terms of power requirements for the Lagoa Salgada Project, a grid connection is expected to be established through a high-voltage overhead line intersecting the Project at a location approximately 15.1 km away. For further information, please refer to Chapter 18 of this Report.

The Property also features available water sources for the survey programme, with current drilling operations sourcing water from a renovated well. To ensure a dependable water supply for the Project, a study was conducted to evaluate the feasibility of a pipeline connection to the nearest dam. Detailed findings can be found in Chapter 18 of this Report.

6 HISTORY

6.1 PRIOR OWNERSHIP

The prior ownership and ownership changes of the LS Property are summarised as follows:

- 1992-1993: Discovery by the Portuguese government geological survey team.
- 1994-2000: The project was held under a consortium consisting of Rio Tinto Zinc (RTZ) and EDM, a Portuguese government agency.
- 2001-2003: The project was free for acquisition.
- 2004-2008: Redcorp Ventures Inc. was granted an exploration permit.
- 2009-2012: Portex Minerals Inc. (Portex) following 100% interest acquisition in Redcorp Ventures Inc.
- 2012-2014: Redcorp - Empreendimentos Mineiros, Lda. (Redcorp), 100% Portuguese subsidiary of Portex was able to maintain the property in good standing through office work and marketing to find a new partner.
- 2015-2017: In July 2015, M&FI acquired a 100% stake in Redcorp. Redcorp then signed an addendum to the current contract for a period of 5 years. Redcorp entered a Consortium Contract with EDM whereby it has a right to participate with up to a 15% interest in the LS Project or assign its right to participate.
- 2018: In June 2018, Ascendant entered into an agreement with TH Crestgate to acquire an initial 25% interest in its Portuguese subsidiary Redcorp - Empreendimentos Mineiros, Lda (Redcorp), which holds a 100% interest in the polymetallic LS volcanogenic massive sulphide (VMS) Project, as well as an option to earn up to an 80% interest in Redcorp upon completion of certain milestones. Ascendant has certain contractual rights under agreements with M&FI which guarantee that it will own an effective 80% interest in the LS Project if M&FI is not successful in acquiring the EDM Interest through Redcorp, such that Ascendant's effective interest in LS is not impacted by the EDM Interest.
- 2021 – Ascendant filed a new NI 43-101 Technical Report and Preliminary Economic Assessment for LS reflecting Post tax NPV8% of $246 Million and Post-tax IRR of 55%.
- 2022 – Ascendant earned a 50% interest in Redcorp and made the final payment required under the option agreement to earn an 80% interest. Upon completing the DFS, Ascendant may exercise its right to earn an 80% interest in Redcorp.

6.2 HISTORICAL EXPLORATION

6.2.1 INITIAL DISCOVERY, 1992

In 1992, the LS deposit was discovered by a team from the Portuguese government geological survey, known as Serviço de Fomento Mineiro (SFM). Nowadays the SFM has been incorporated into the Laboratório Nacional de Energia e Geologia (LNEG). Between 1992 and 1993, 17 drillholes were completed in and around the LS Property for a total of 7,588 metres (m); LS-01 to LS-17.

The deposit is completely covered by a thick sequence of Tertiary sedimentary rock with an average thickness of 135 m; the discovery was made through diamond drill testing of a gravity geophysical anomaly.

6.2.2 RIO TINTO ZINC, 1994-2000

In 1994, the area was awarded to a mining consortium composed of Rio Tinto Zinc (RTZ) and Empresa de Desenvolvimento Mineiro (EDM), a Portuguese government agency, who held the property from 1994 to 2000.

The consortium completed an airborne magnetic survey of the property and completed several widely spaced diamond drillholes. In addition to the magnetic survey, RTZ performed limited downhole geophysics, electromagnetic surveys, and limited soil sampling.

6.2.2.1 DRILLING

Between 1994 and 1999, the consortium drilled 20 additional drillholes (LS-18 to LS-37) which successfully defined the broad outlines of the North (formerly LS-1), Central, and South (formerly LS-1 Central) deposits.

Nowadays, the historic RTZ/EDM drill core is stored in the new LNEG facilities in Aljustrel village, located approximately 55 km south of Grândola, and is easily accessible upon request at the Aljustrel office of the LNEG.

In Portugal, two years after the completion of a drill campaign, the drill core becomes the property of the government. It becomes the responsibility of the LNEG to collect the drill core and accompanying documents, drill logs and drill assays. The historic drill core, from southern Portugal, is stored at LNEG facilities in Aljustrel.

In 2016, Redcorp was given permission to transport and store some of the historic RTZ drill cores on the property.

In 2005, Carmichael noted in his report that: “No information is available regarding sample preparation or quality control measures for the historical sampling. The work was carried out by a major mining company, RTZ, and the author has no reason to assume that the

sample results do not accurately reflect the true values of metals in the mineralized sections."

6.2.2.2 METALLURGICAL TESTWORK, ANAMET, 1995

In 1995, RTZ commissioned a preliminary metallurgical testwork programme on a MS from the LS Property. The sample tested was a relatively high-grade composite from drillhole LS-22 containing 9.45% Zn, 6.7% Pb, 0.27% Cu, 62 ppm Ag, and 1.47 ppm Au. The best results from a series of Pb Zn differential flotation tests produced a Pb cleaner concentrate grade of 34.2% Pb at a recovery of 38.5%. A Zn cleaner concentrate grade of 44.7% Zn was achieved at a recovery of 23.1%. It was not possible to produce an acceptable bulk concentration in one stage of flotation.

The sample from drillhole LS-22 is not representative of the deposit as it is currently defined by the 2017 mineralogical samples.

6.2.3 REDCORP VENTURES LTD., 2004-2008

In October 2004, the LS Property was acquired by Redcorp Ventures Ltd. (Redcorp Ventures) of Vancouver, Canada. Redcorp Ventures established its Portuguese subsidiary, Redcorp - Empreendimentos Mineiros, Lda.

In 2005, Redcorp Ventures conducted a three-dimensional (3D) inversion of existing geophysical data followed up by a diamond drilling programme and the re-logging of the historic RTZ-EDM drill core. Most of this work covered the Rio de Moinhos Project to the south-west of the LS Property and therefore the results are not discussed in detail.

Lithogeochemical and petrographic samples were collected by Dr Tim Barrett of Ore Systems Consulting (Wardrop 2007) but the results are not available to Quadrante/IGAN.

In 2005, Redcorp Ventures' drilling programme consisted of six holes totalling 2,286 m. Drilling continued in 2006, 2007, and 2008 for a total of 16 holes totalling 8,692 m. All but one (LS06043) of the drillholes intersected the LS deposit and confirmed polymetallic mineralization comprising Zn, Pb, Cu, Au, and Ag.

6.2.4 PORTEX MINERALS INC., 2009-2012

In 2009, Portex acquired a 100% interest in Redcorp Ventures to develop the North deposit on the property. Portex's exploration activities included a drilling programme and a downhole geophysical survey programme.

6.2.4.1 DRILLING PROGRAMME

From May to August 2011, Portex completed five diamond drillholes on the LS deposit totalling 1,138 m. This was followed by a further two drillholes in 2012 totalling 474 m.

The following information regarding the drilling programmes was partly summarized from Daigle (2012).

Portex contracted Drillcon Iberia S.A. (Drillcon), a Portuguese subsidiary of the Drillcon Group, to conduct the drilling. Drillcon used one drill with a tri-cone bit to pre-collar the drillholes through the Tertiary sedimentary units. The drillholes were cased using a steel casing for the entire length of the drillhole within the Tertiary sedimentary units.

A second drill was then brought in to continue drilling with a diamond core drilling rig using HQ size core. Once the drill rods showed signs of stress, the drill core size was dropped to NQ. Most of the drillholes were cored using HQ.

Once the drillhole was completed, acrylonitrile butadiene styrene (ABS) polyvinyl chloride (PVC) pipe (NQ) was inserted down the entire length of the drillholes. This was done to prevent the drillhole wall from collapsing in anticipation of conducting future downhole geophysical surveys.

The drillhole steel collars were cemented in place and a steel cap was welded to the collars to allow for a hinged cap to cover the drillhole and be locked with a padlock.

The diamond drill core was collected by Portex geologists at the drill site and brought to the drill core logging and sampling facility. The drill core was roughly logged on paper and transcribed into a Microsoft Excel® spreadsheet.

Sample tags were inserted on 1.0 m sample intervals respecting the contacts between lithologies. The sample tags were standard tags from ALS Laboratories, with a sample number and bar code, and were inserted into a small sealable plastic bag and stapled into the core box at the beginning of the sample interval.

Lead and zinc standards were inserted roughly every 15 samples within the Gossan (GO) and MS lithologies. Gold and copper standards were inserted in roughly the same intervals in the stockwork lithologies. Duplicates were collected from the drill core by quartering the half core and submitting the sample.

6.2.4.2 DOWNHOLE TEM GEOPHYSICAL SURVEY

In August 2012, Portex retained International Geophysical Technology (IGT) to conduct a downhole transient electromagnetic (TEM) geophysical survey in drillholes PX-02 and PX-05.

The results from PX-02 did not produce any significant anomalies and may not be part of the MS body. However, results from PX-05, where the MS were intersected, showed two independent anomalies, one which pertains to the intersected MS body, and a second anomaly, possibly 30 m to the west. This second anomaly may lie within the interpreted MS.

6.2.5 MINERAL AND FINANCIAL INVESTMENTS LTD 2012-2018

During the period between 2012-2018, Mineral and Financial Investments Ltd was the main holder of the Lagoa Salgada project. This company implemented a varied approach to the project by performing a petrographic study of the ore, a downhole geophysical study, and a drilling campaign.

6.2.5.1 PETROGRAPHIC STUDY

The petrographic study was performed by the Faculty of Sciences of Porto University by analysing 20 samples belonging to the different stratigraphic domains, namely the Gossan, the Massive Sulphides, Cherts/Jasper, and the stockwork. The main goal of this study was identifying the most interesting mineralogical phases, as well as how these minerals occur. This study allowed us to conclude that for the massive sulphides, the sphalerite and the galena occurred together with some arsenopyrite and chalcopyrite. The sphalerite, just like the galena, may occur in several granulometric sizes with the biggest crystals of sphalerite showing inclusions of chalcopyrite, and the crystals of galena showing an intergrowth with chalcopyrite. In the massive sulphides, the chalcopyrite never exceeds 25 µ, occurring mostly in the outside border of the pyrite or filling small fractures. Intergrowth with the sphalerite was also observed in fine crystals of cassiterite.

6.2.5.2 DOWNHOLE GEOPHYSICAL SURVEY

In 2017, the International Geophysical Technology (IGT) performed a downhole geophysical survey. The subjects of this study were three drillholes, namely LS_MS_01, LS_MS_03, and LS_MS_06. The main conclusions of this study revealed that the anomalies resulting from the massive sulphides are as sharp as the theoretical model would indicate. The study also concluded that despite the tertiary cover that may relax the potential values at the surface, this effect does not mask the conductivity of the orebodies where the electrodes were earthed in the surveyed drillholes. The interpretation of the conductor results obtained from the three drill holes leads us to believe that the conductor is the same, with LS_MS_01 intersecting it close to its SE end, LS_MS_03 entering where the orebody shows its maximum thickness in the central section and LS_MS_06 showing it at its NW end. This means that this conductive ore body has an average trend of N160E with an approximate length of 500 m.

6.2.5.3 DRILLING CAMPAIGN

From 2012 to 2018, Mineral and Financial Investments, Ltd (M&FI), through its Portuguese subsidiary Redcorp - Empreendimentos Mineiros, Lda (Redcorp), subcontracted the Drillcon Group to conduct a drilling campaign. During this period, the drilling was mainly conducted to increase the resources as well as increase the reconnaissance of the limits of the Lagoa Salgada deposit. To achieve these goals, 26 drill holes were made with a total of 12,641.1 m of drill core. Out of these, 10 drill holes were made in the North sector (3,463.85 m), and 16 drill holes in the South sector (totalling 9,177.25 m).

These drill holes were performed in two different phases. In the first stage, a drilling rig coupled with a destructive tri-cone bit was used to pre-collar the drill holes through the Tertiary sedimentary units. Afterwards, the drillholes were cased using a steel casing for the entire length of the hole within the Tertiary sedimentary units. Later, a second drilling rig was placed at the drill hole location. This rig was coupled with a diamond core drilling bit using HQ size core and it was then brought in to continue drilling. Once the drill rods showed signs of stress, the drill core size was reduced to NQ, although most of the core drilling was conducted using HQ.

6.3 MINERAL RESOURCE ESTIMATES

There have been three previous NI 43-101 Technical Reports completed on the LS Property, each of which contained mineral resource estimates on the LS-1 deposit. The previous Technical Reports are as follows:

- Wardrop September 2007, Redcorp Ventures Ltd., Resource Estimate for the Lagoa Salgada Project. Wardrop Engineering Inc. Document No. 0752760100-REP-R0001-01. 27, 46 pages.
- Daigle, Paul January 2012, Lagoa Salgada Project, Portugal – Resource Estimate Update Document No. 1296360100-REP-R0001-02, 92 pages.
- Daigle, Paul January 2018, Revised July 2018), Technical Report for Redcorp Lda., Lagoa Salgada Project, Setubal District, Portugal, 124 pages.
- Micon February 2019, NI 43-101 Technical Report: Resource Estimate for the Lagoa Salgada Project, Setubal District, Portugal, 117 pages.
- Micon November 2019, NI 43-101 Technical Report: Resource Estimate Update for the Lagoa Salgada Project, Setubal District, Portugal, 152 pages.
- Murahwi, C., Methven, P., Zazzi, G, Malhotra, D. 2021: Updated Mineral Resource Estimate for the South Deposit and PEA for the North Deposit, Lagoa Salgada Property, Setúbal District, Portugal: Prepared by Micon International Limited for Ascendant Resources Inc., effective date 31 January 2021 (South Deposit) and 19 December 2019 (North Deposit).

Other than the January 2018, February and November 2019 reports and the Updated Mineral Resource Estimate for the South Deposit and PEA for the North Deposit with an effective date of 31 January 2021 (South Deposit) and 19 December 2019 (North Deposit), the prior Mineral Resource Estimates were conducted under previous versions of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves, and/or prior versions of the National Instrument NI 43-101 Standards of Disclosure for Mineral Projects. All the previous Mineral Resource estimates are superseded by the current estimate of the Mineral Resources contained in Section 2.7 of this Technical Report. As a result, they will not be further discussed herein.

6.4 HISTORICAL MINING

No historical mining has been conducted at the LS Property.

It can be stated that the local populations and the Municipality of Grândola, and certainly the Municipality of Alcácer do Sal, are accustomed to mining activity, the local mining activity being divided between non-metallic mineral resources (sand quarrying, aggregates, clays, etc.), which maintain significant activity in the region, and the mining activity of metallic mineral resources, as they are the most significant and historical ones.

As for metallic resources, the most important were the old mines of Lousal and Canal Caveira, mainly known for producing pyrites, and both being underground mines that were exploited in the Municipality of Grândola. The Lousal mine operated for about 90 years since the early 1900s and was an important centre of employment in the region.

There are other small-scale mines that were exploited during the 1920s.

The potential in non-metallic mineral resources is high, especially sandy deposits.

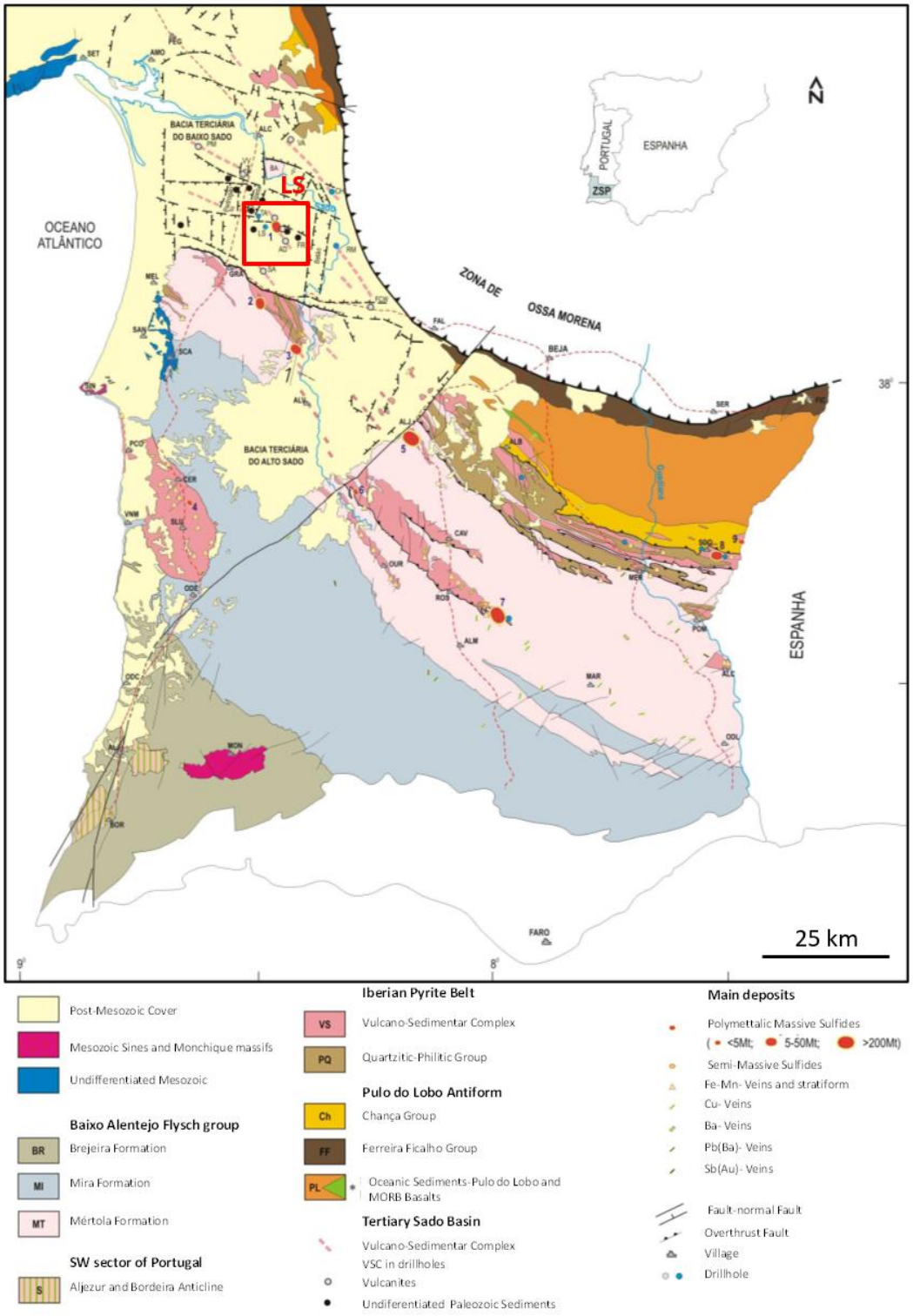
7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGICAL SETTINGS AND MINERALIZATION

The Lagoa Salgada project is located in the northwest edge of the Iberian Pyritic Belt (IPB). The IPB is one of the main domains of the Portuguese south Zone, being the southernmost expression of the Variscan orogeny in Portugal. It is believed that this region collided with the terrains of the Ossa Morena Zone (Oliveira et al 2006), resulting in creation of basins within the continental crust of the Portuguese south Zone, which triggered the submarine volcanism of the IPB (Silva et al. 1990; Quesada 1991; Tornos et al. 2002). The oldest rocks in the region are assigned to the Phyllite-Quartzite group, consisting of alternating grey schists and sandstones. Overlying these rocks is the Volcano-Sedimentary Complex (VSC), which is characterized by felsic to mafic sequences interspersed with metasediments (Oliveira 1990). The VSC group hosts the massive sulphide deposits of volcanic origin in this region that were formed in different ways involving either effusive or explosive volcanic episodes. The volcanic centers on its borders gradually pass into schists with a non-volcanic component. The Lower Alentejo Flysch group, a turbiditic sequence containing lithified sandstone schists and rare conglomerates, was subsequently deposited above the VSC.

This region is also marked by post-Mesozoic sedimentary deposition that make up the Sado Tertiary Basin and by some Paleozoic outcrops that occur mostly isolated and related to horst structures, which can be observed in Figure 7.1 showing the alignments of the VSC where, among others, sector 1 corresponds to the Lagoa Salgada deposit (Oliveira et al, 2001).

In the area of the Lagoa Salgada deposit, the contact between Paleozoic and Cenozoic formations occurs, according to gravity surveys and confirmed by drilling, at a depth of between 110 and 250 m, corresponding to a temporal hiatus of over 200 Ma. The Paleozoic formations correspond exclusively to VSC formations, mostly rhyodacites, dacites, and intermediate to basic rocks. These lithologies host various types of mineralization, including syngenetic polymetallic massive sulphides as well as the stringer and associated epigenetic mineralization related to fissural remobilizations. Weathering has resulted in the gossan and oxidized massive sulphides zones, which occur in the upper part of these formations, where there is a supergene enrichment. From a structural point of view, the main characteristics of the region are related to the Variscan tectonics that affected the surface of the Paleozoic formations through horsts and grabens with well-defined alignments, which can be observed on sheet 42 B of the Geological Map of Portugal at a scale of 1:50,000 and in Oliveira et al, 1998a. In addition to this, Alpine tectonics also had effects in this region, having reactivated Variscan structures and also affecting the most recent sedimentary formations, especially in Miocene lithologies, with the presence of fractures with movement stretch marks. In addition to the main lithologies, the different types of mineralization were also affected by tectonics, undergoing folding resulting in synforms and antiforms sequences, often accompanied by faults vergence towards the SW, as observed throughout the VSC (Oliveira et al, 2013).



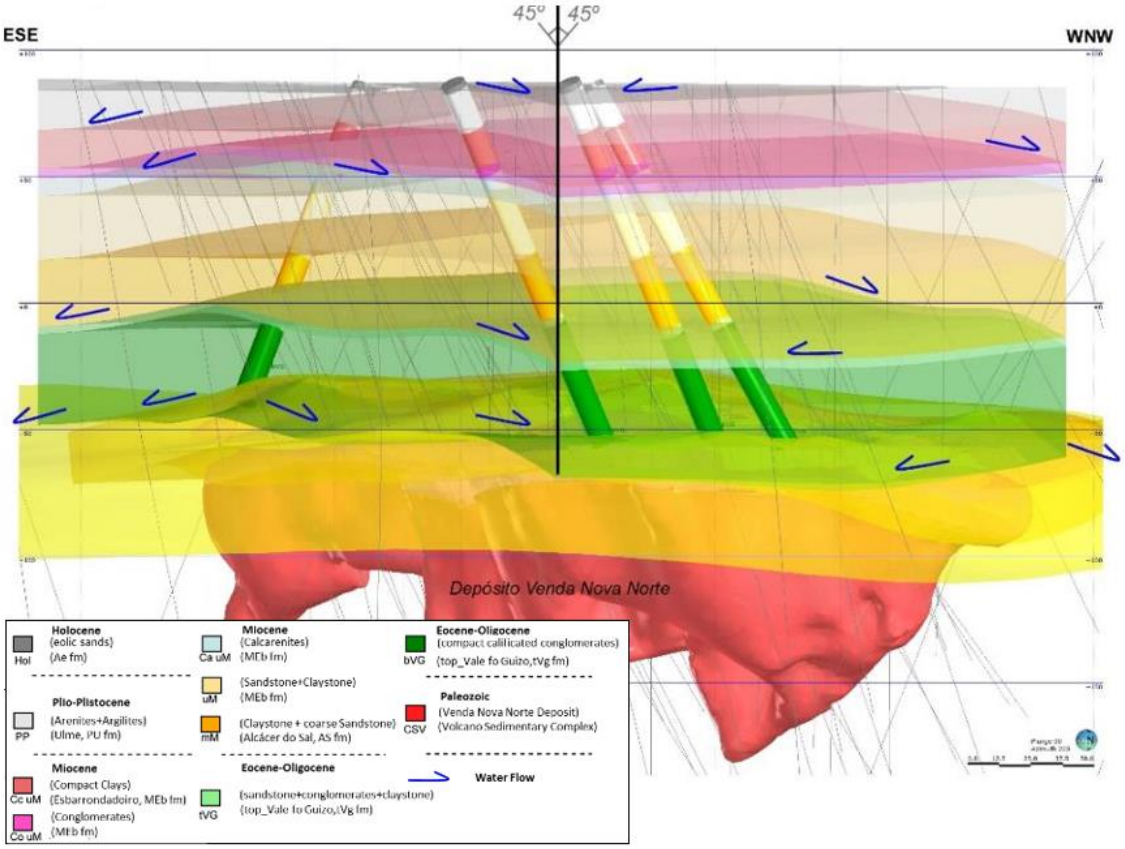
Source: Adapted from Oliveira 1990, Barriga et al, 1997, V. Oliveira et al, 1998a, b, 2001, Carvalho et al, 1999, Leistel et al, 1998, Rosa et al, 2001, Matos et al, 2000, 2003.

Figure 7.1 - Geological Map of the south Portuguese Zone showing the main deposits

7.2 LOCAL GEOLOGY

From a general point of view, the local geology of the Lagoa Salgada deposit area can be divided into a set of Cenozoic formations and Paleozoic formations. The first are derived from the harmonization of information taken from map sheets 39-C, 39-D, 42-A, 42-B of the Geological Map at a scale of 1:50000, and their comparison with samples obtained from core drilling.

Thus, in the project area, from top to bottom of the Cenozoic stratigraphic sequence, Holocene formations appear with meter size thicknesses, followed by Plio-Pleistocene formations, namely the Ulme Formation with thicknesses of up to about 20 m. At the base of these formations appear the equivalent Esbarrondadoiro formations with a thickness of about 30 to 35 m and the Alcácer do Sal Formation with a thickness between 20-25 m, both from the Middle to Upper Miocene. Finally, the oldest Cenozoic formation present in the Lagoa Salgada area is the Vale do Guizo Formation, with no consensus on its age in the available literature, Eocene to the Oligocene or Lower. This formation has a thickness of 50 to 60 meters and rests on Paleozoic rocks. This stratigraphic sequence can be seen in the Figure 7.2.



Source: Baseline Hydrogeological Study (BHS), 2023.

Figure 7.2 - Sedimentary sequence of the Cenozoic in the Lagoa Salgada project area

The Paleozoic formations recognized in the Lagoa Salgada deposit were primarily macroscopically defined and subsequently analysed through geochemical studies, and academic references. When contextualized in the regional geology, they belong to the Volcano-Sedimentary Complex with ages ranging from the Famennian to the middle Visean, with the deposit showing an NW-SE alignment that is part of a belt about 8 km in length and 1 km in width in the concession area.

In more detail, there is significant differences between the local geology from north and south sectors of the exploration area. The northern sector is characterized by the presence of a *cap rock* called dtV- dacitic tuffitic Volcanic Rock. In this formation, phenocrysts of quartz and feldspar occur, and their abundance can vary depending on the dacitic mass. This can be observed from the interpretation of the available geochemical analyses, which classify the abundance of phenocrysts as mdtV- muddy dacitic tuffitic Volcanic Rock or fdtV- feldspar dacitic tuffitic Volcanic rock. Recent geochemical studies confirm that hydrothermal alteration of a chloritic nature is directly related to the proximity of this formation to the mineralized massive sulphides. Altered rocks show a clear evolution in the system from a Na+K rich environment to a Fe+Mg enriched environment as shown in Figure 7.3. This lithology is generally found over massive sulphide mineralization in the northern sector, but it can also be in direct contact with the gossan. This last type of mineralization was formed due to a paleo-weathering of the massive sulphides that oxidized, also promoting supergene enrichment with potentially economic levels of Au, Ag, Sn and Pb.

Often intercalated with the gossan are deca-centimetric levels of grey-red cherts and jaspers at the top of the massive mineralization. The massive sulphides in the Lagoa Salgada deposit are polymetallic in nature and show evidence of metal mobilization at their top contact, resulting in the enrichment in metals such as Pb, Ag, and Cu and the depletion of the zinc in the transition zone (also referred as transition massive sulphides – tMS). At portions of its base contact and in other cross-cutting structures, there are alunitic levels that may derive from white smoker-type structures or epigenetic hydrothermal alterations.

At the base of the massive sulphides, there is acid (felsic) to intermediate volcanic rocks that have been classified according to their geochemistry as rhyodacites, dacites, and andesites. It is also important to note that the geochemistry of these lithologies indicates bimodal volcanism, and that these felsic lithologies are the main hosts of stringer-type mineralization with a stockwork texture. This volcanic bimodal character can be assessed by the REE spider plot normalized to the Boynton (1984) chondrite. The spectrum indicates that despite the different intensities, the acid volcanic rocks (aV) such as rhyodacites and the intermediate volcanic rocks (iV) such as andesites and dacites, show parallel spectre demonstrating a similar origin, as can be observed in the Figure 7.4.

A longitudinal view of all these lithologies can be seen in Figure 7.5.

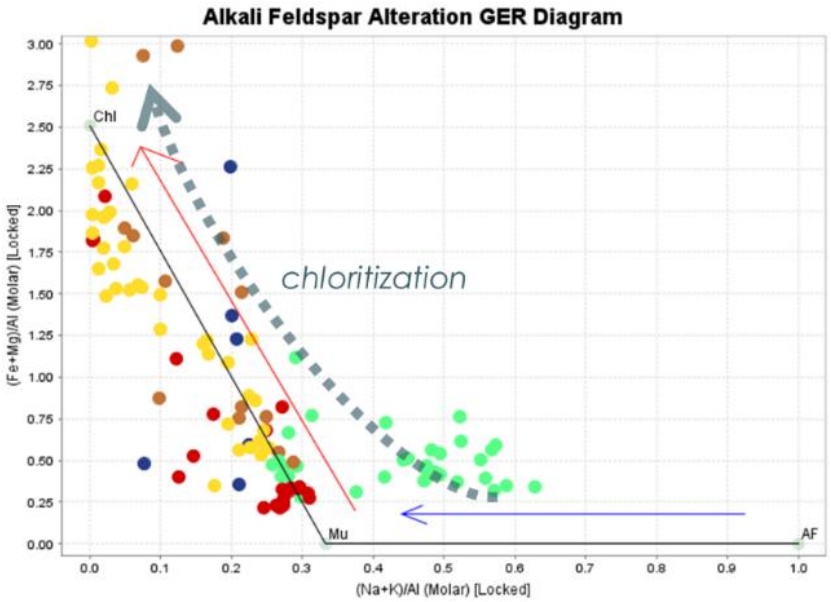


Figure 7.3 - Alkali Feldspar Alteration diagram according with Stanley 2017 applied to the dtV rock samples showing the chloritization alteration trend

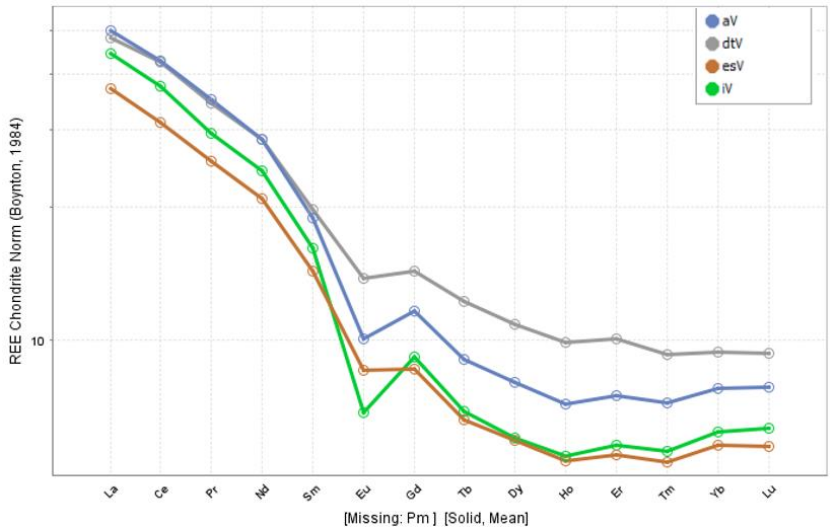


Figure 7.4 - REE spider plot normalized to the Boynton (1984) chondrite with the average values of the acid volcanic rocks (aV) and the intermediate volcanic rocks (iV)

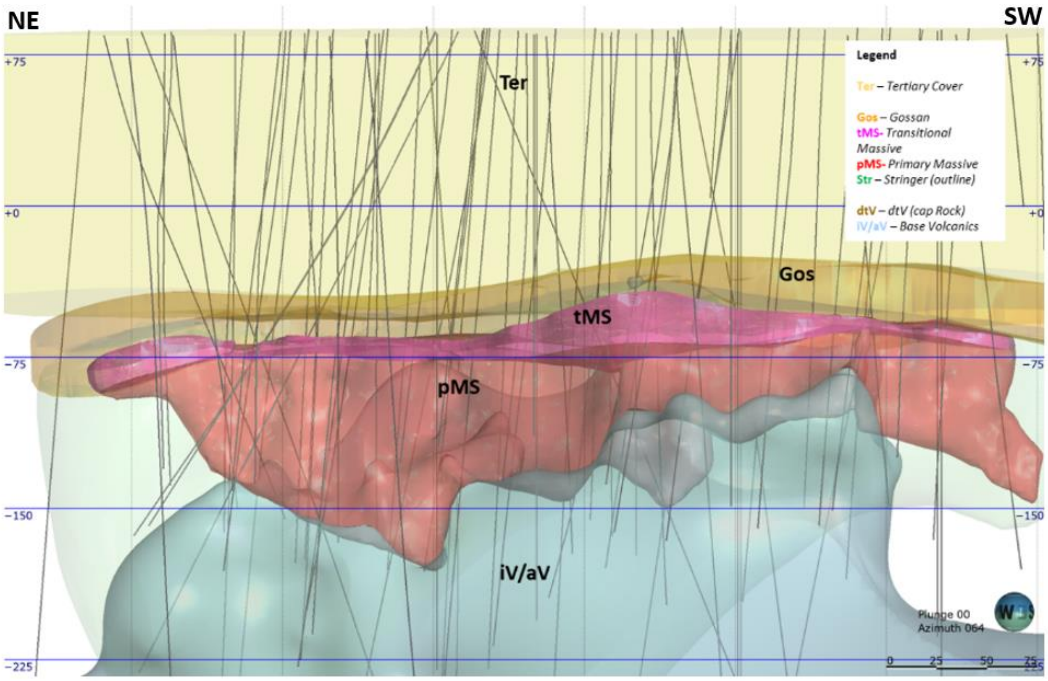


Figure 7.5 - Longitudinal view of the north sector deposit showing all the mineralized domains

The stratigraphy in the south deposit is different from the north sector. The dtV unit is the cover rock of the entire mineralized system in the northeast areas. Stratigraphically below this is the esV unit i.e, an exhalative sedimentary Volcanic rock, which displays an alternation of pyritic levels with quartz-feldspar levels, with some presence of amphiboles in the latter which might be a lithostratigraphic equivalent of the massive sulphides in the south sector. The pyritic levels of this formation regularly exhibit cubic crystals that often appear subrounded, suggesting some sedimentary transport before the deposition. Another characteristic of esV is its hydrothermal alteration of a sericitic nature that occurs pervasively. Some 50 cm to 4 m thick veinlets of massive sulphides occur in esV horizons and contain copper and zinc as sulphides with minor gold. At the base of the esV formation, intermediate to basic rocks such as dacites and andesites occur, designated in this project as iV - intermediate Volcanic Rocks. The iV is the main hosts unit of the fissural remobilization mineralization type (FR), which is also of a polymetallic nature. This unit presents two types of hydrothermal alteration, sericitic and chloritic. The chloritic altered levels (quantified by the CCPI index, Chlorite Carbonate Pyrite Index proposed by Gemmel (2007)) measures the intensity of chloritization. Figure 7.6 shows a scatter plot with the CCPI vs the copper equivalent (CuEq) identifying the samples that belong to the distinct FR corridors and the ones that belong to the wall rock of these corridors. Clay content differs according to host rocks as demonstrated in the ternary plot of Figure 7.7; esV lithologies a contain higher content of clays than the iV lithologies. This fact is of high importance due to metallurgy concerns. Examples of all described lithologies in both the north and south sectors are shown in Figure 7.8.

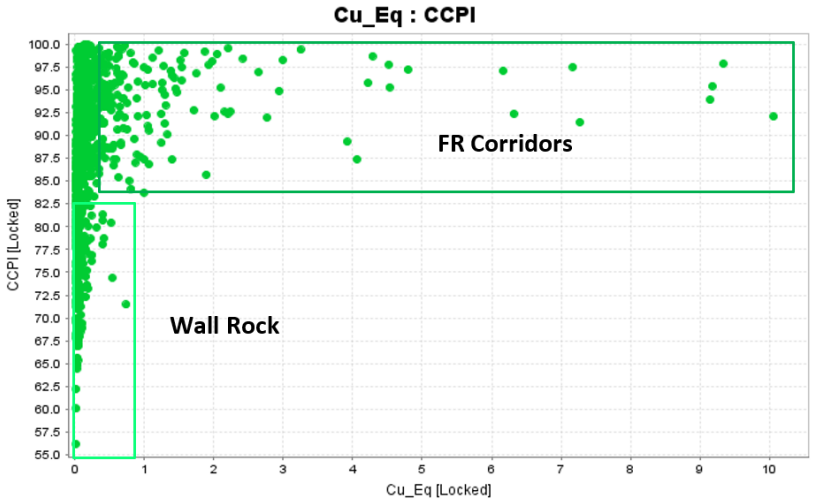


Figure 7.6 - Plot of the iV rocks belonging to the south sector with the CCPI ratio from Gemmel (2007) vs the Cu_Eq (%).

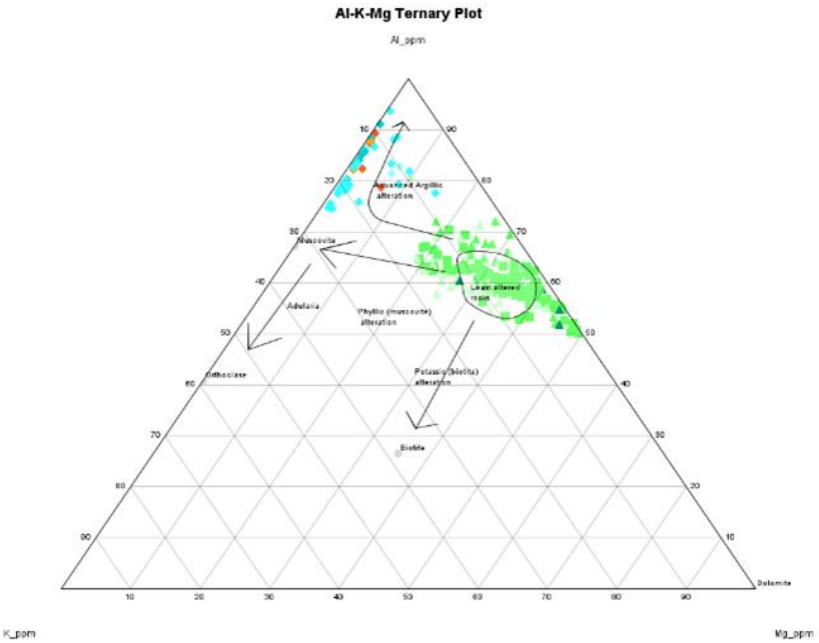


Figure 7.7 - Al-K-Mg ternary plot showing the esV and iV (dacites and andesites) samples clay alterations for epithermal systems by Scott Haley 2016.

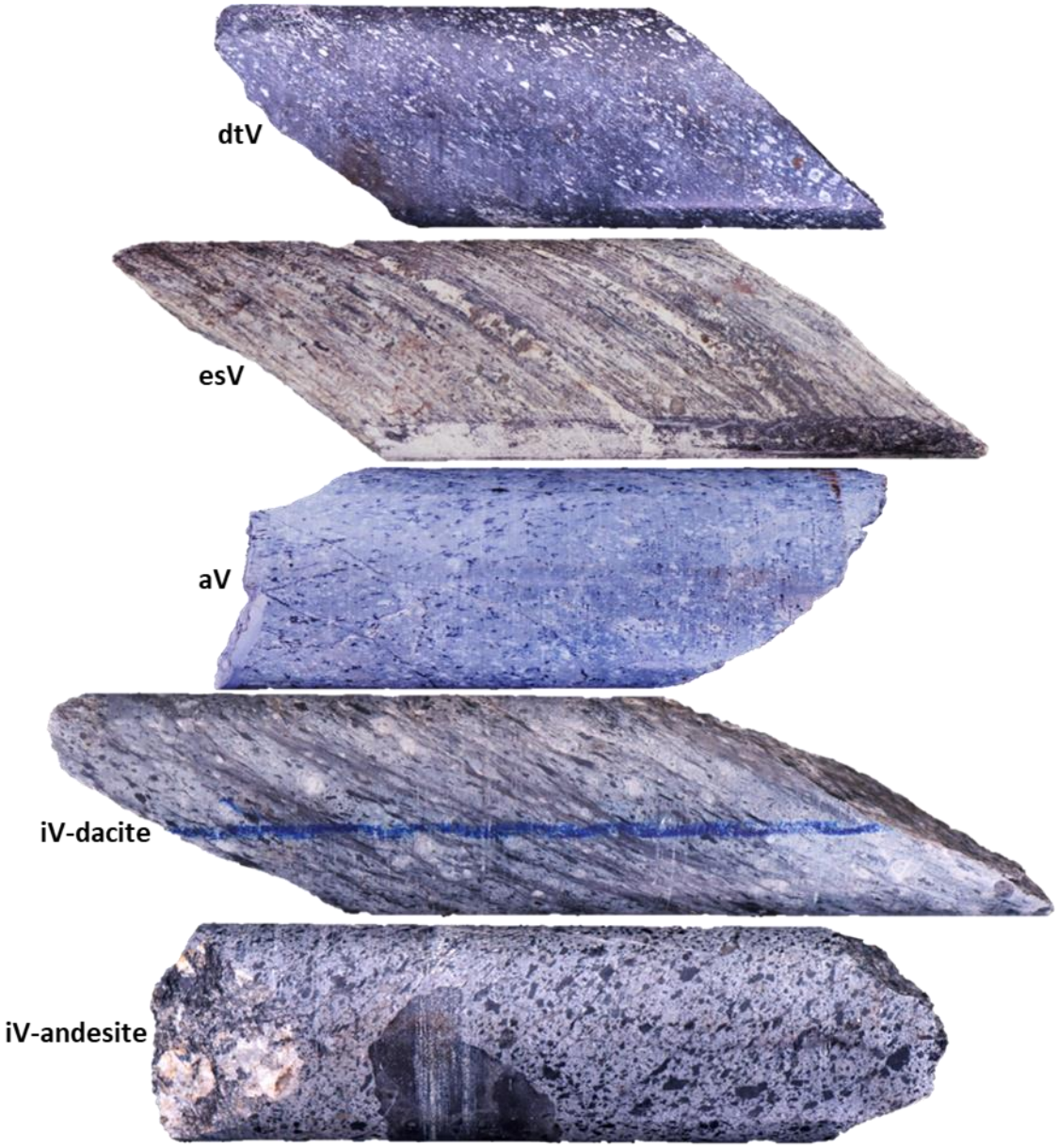


Figure 7.8 - Example of the distinct lithologies present in the Lagoa Salgada deposit

8 DEPOSIT TYPES

From a genetic point of view the Lagoa Salgada deposit is considered a polymetallic Volcanogenic Massive Sulfide (VMS). These deposits arise from the association between volcanic events and hydrothermal activity in a submarine environment. From a geotectonic point of view, most VMS-type deposits are deposited on the seafloor near mid-ocean ridges in volcanic arc environments. In this environment, seawater infiltrates the crust and is heated in deeper zones by the heat of magma chambers, creating convective cells that allows the extraction of metals from deeper rocks by dissolution in the aqueous medium through the interaction between water and rock. Subsequently, with the flow generated by the convective cell, the mineralized solution is cooled due to its ascension and contact with seawater, causing the precipitation of metals such as copper, lead, and zinc. The mineralizing aqueous fluids usually use weaknesses in the host rocks for their circulation, namely, fault planes and other anisotropies. Thus, and due to their circulation, large areas of hydrothermal alteration events are expected to occur in the host rocks, such as chloritization and sericitization. This type of mineralization generally deposits in second or third-order volcanic arc basins, which allows them to be quickly covered, ensuring their preservation.

VMS-type deposits tend to occur in clusters as sets of lenses along a well-defined lithostratigraphic layer. In this layer or stacked as paralleled lenses in more than one stratigraphic level, the hydrothermal alteration zone allows for the identification of the extension of the VMS deposit since, despite the separate occurrence of the various massive sulfide lenses, the alteration halo is continuous. However, in some rare cases, it is possible to observe the deposition of sulfides within sub-surface weaknesses through non-aerial means.

In the Lagoa Salgada deposit, sub seafloor alteration affected rhyodacite with phenocrysts of quartz and feldspar and a rhyodacite with quartz phenocrysts intercalated with a sandstone (Matos et al. 2000). The true thickness of the Lagoa Salgada lithostratigraphic sequence is difficult to determine, as there are disruptions and repetitions of the volcaniclastic units due to the presence of inverse faults.

8.1 THE LAGOA SALGADA TYPES OF MINERALIZATION

The mineralization types are different for each sector of the Lagoa Salgada deposit. They all show different typologies and modes of occurrence although all the mineralization domains are polymetallic.

8.1.1 NORTH SECTOR

The north sector of this deposit shows four distinct mineralization domains, namely the gossan, the transition massive sulfides, the primary massive sulfides, and the stringer zone. The gossan (Gos) mineralization results from the supergene alteration of the primary massive sulfides after intense oxidation and weathering when the primary mineralization

was exposed to the paleosurface. The net effect of the supergene effect is the enrichment of less mobile elements such as Au Ag, Sn and Pb.

The transitional massive sulfides (tMS) zone is also the result of a supergene process. Some of the metals leached in the relatively shallower oxidizing levels and precipitated immediately below more reductive conditions (below the paleo water table). This domain is particularly rich in Cu, Pb and Ag. This can be observed in Figure 8.1, where a group of samples (green circle) shows a low Zn/Zn (Cu+Pb) ratio and a high S/S+Fe ratio.

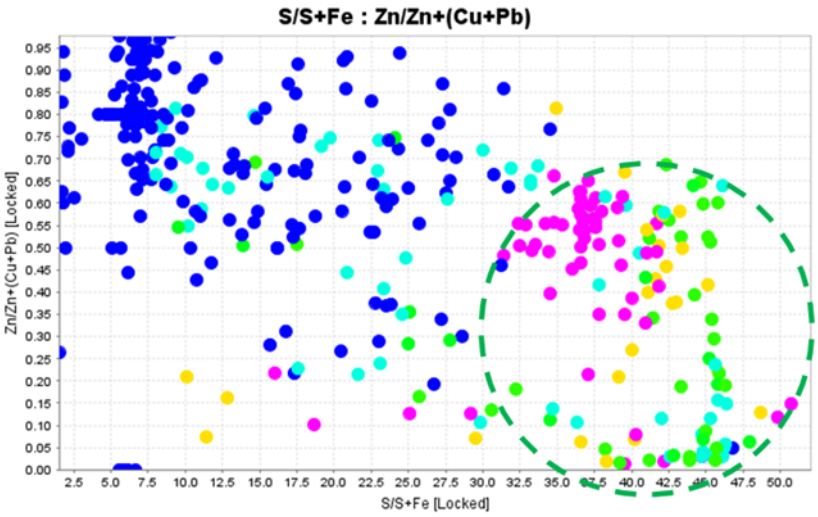


Figure 8.1 - Plot showing the massive sulphide samples according in a Zn/Zn (Cu+Pb) ratio vs S/S+Fe ratio

The primary massive sulphides (pMS) result from the extraction of metals from deeper volcanic rocks that, by dissolution in the aqueous solutions through the interaction between water and rock. These were transported and subsequently cooled, causing the precipitation of metals such as Cu, Pb, Zn and Sn in the case of the Lagoa Salgada and creating massive sulphide lenses due to the circulation of hydrothermal fluids.

The stringer (Str) domain is characterized by stockwork polymetallic mineralization occurring in intermediate-basic volcanic rocks at the bottom of the primary massive sulphides. From a metallogenetic point of view, these were the corridors that fed the massive sulphide lens. At the Figure 8.2 shows examples of mineralization types from the north sector.

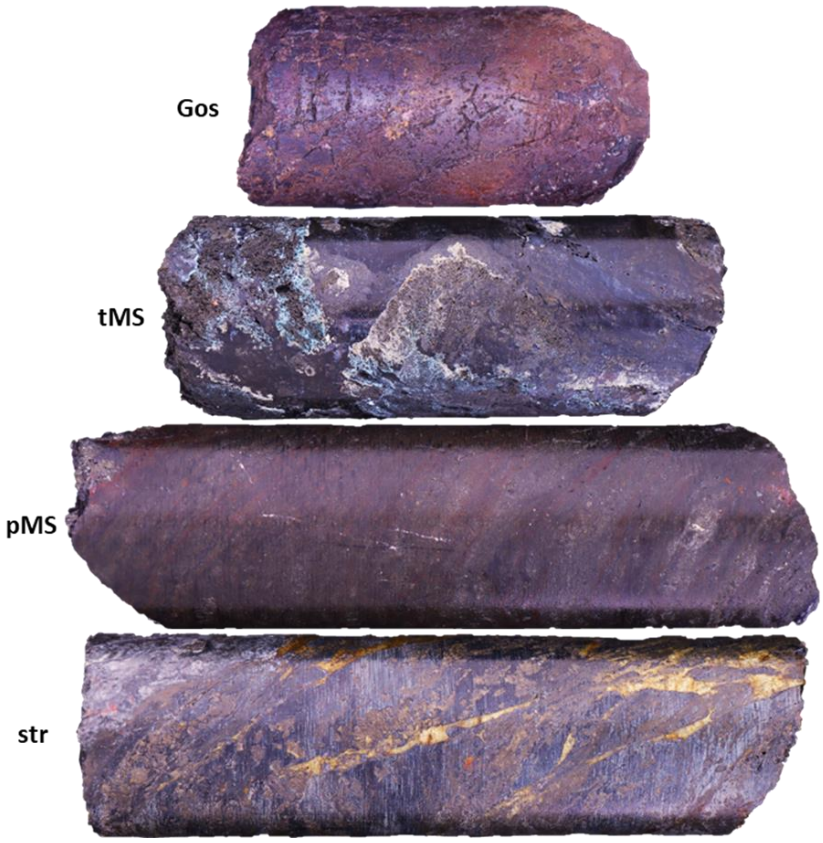


Figure 8.2 - Drill core samples showing the different types of mineralization of the north sector (GOS-gossan, tMS-transition massive sulphides, pMS- primary massive sulphides, str-stringer)

8.1.2 SOUTH SECTOR

The mineralization style at the south sector of the Lagoa Salgada deposit is different from the north showing two distinct domains, the semi-massive sulphides (sMS) zone, and the fissure remobilization (FR) mineralization zone. An image of the south sector mineralization domains is shown on Figure 8.3.

The first sMS mineralization which is only present in the esV lithologies is characterized by semi massive pyritic intercalated with volcanic rocks. It is believed that the esV lithology is the lateral equivalent of the north sectors massive sulphides, representing a distal volcanogenic massive sulphide related facies. Both units have been tectonically packed into sub-vertical lenses following the latest stage of deformation.

The second mineralization style of the southern sector is fissural remobilization (FR) mineralization. This polymetallic occurrence is hosted in the base southwestern iV rocks with the mineralization corridors following a main trend with an azimuth that varies between 140˚ to 160˚ and dipping between 70˚ to 85˚ NE. Three main corridors have been defined in this domain, all dipping to the NE and are open along strike to the north (Figure 8.4).

An important fact to retain about the FR domain is that the hosting rocks between the three corridors still carry some lower grade but potentially economic mineralization. The 3 corridors represent some higher-grade zones within a broader envelope of mineralized iV rocks. The FR domain is polymetallic with Cu, Zn, Pb, and Ag; however, Cu (chalcopyrite) and Zn (in sphalerite) are predominant. From a genetic point of view, the occurrence of this mineralization occurred through a post-orogenic percolation of the mineralized fluids induced by an epigenetic hydrothermal alteration in the iV rocks. This is illustrated in Figure 7.6 above (under Section 7). This analysis (seen in Figure 7.6) was one of the facts that helped to define the main corridors of mineralization as all the samples of the corridors have a CCPI index above 85.

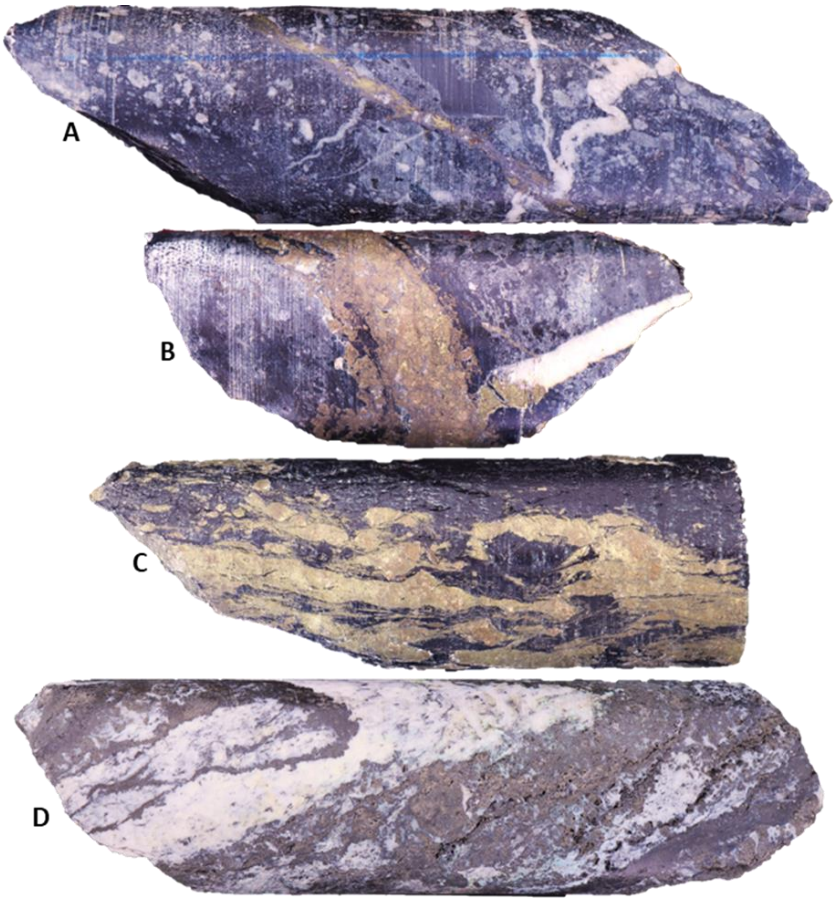


Figure 8.3 - Drill core samples showing the different types of mineralization of the south sector. A and B FR mineralization cross cutting post orogenic quartz veins, C -FR mineralization cross cutting the foliation from IV rocks, D- Example of sMS corridor with h

8.1.3 LAGOA SALGADA MINERALIZATION CONCEPTUAL MODEL

A conceptual model showing the distinct types of mineralization present in the Lagoa Salgada Deposit and their spatial relationship is presented in Figure 8.5.

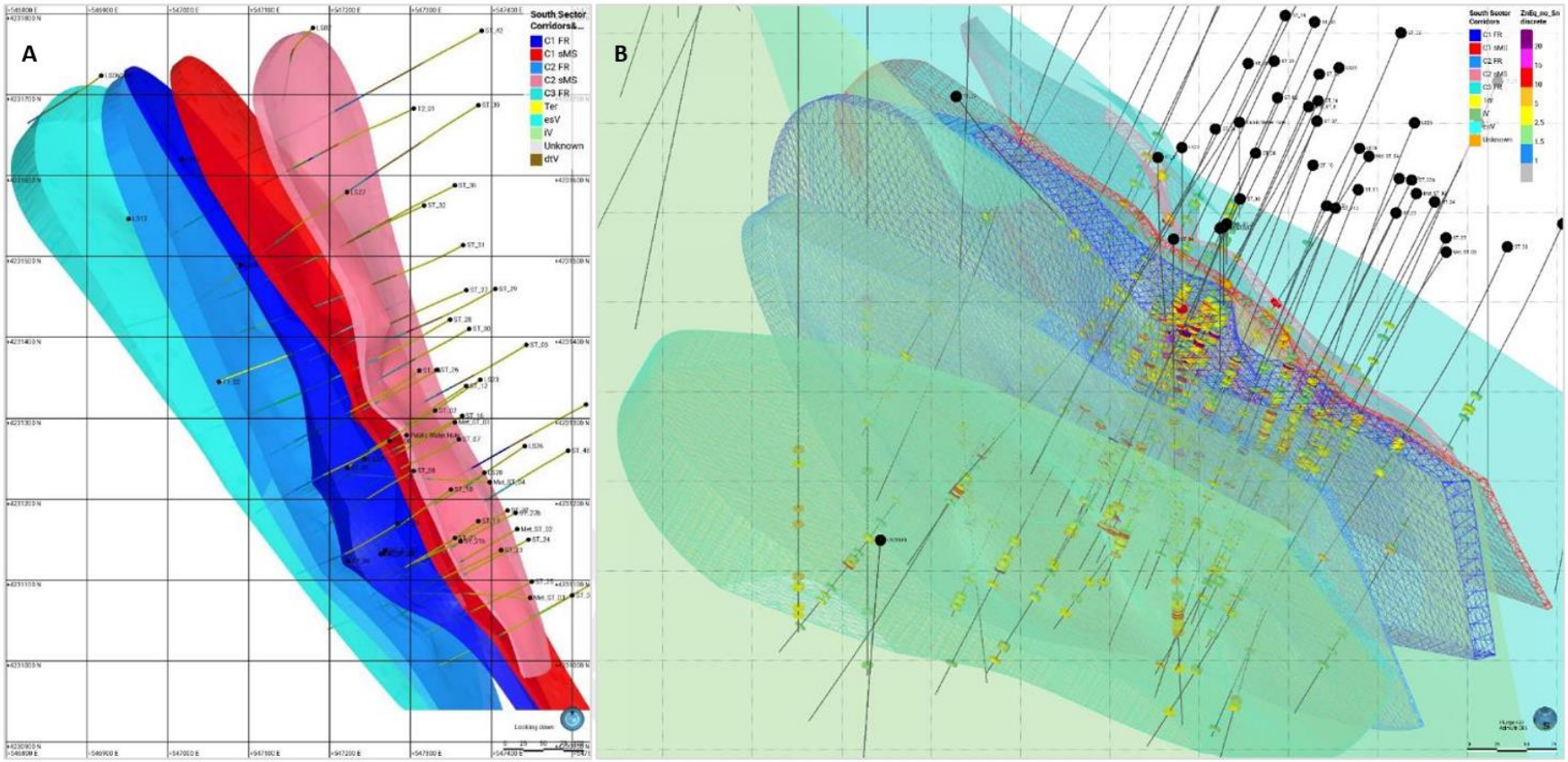


Figure 8.4 - A- Plan view from the southern sector of the Lagoa Salgada with all the corridors of mineralization, B-South view of the south sector corridors within the host esV and iV lithologies of the Lagoa Salgada deposit

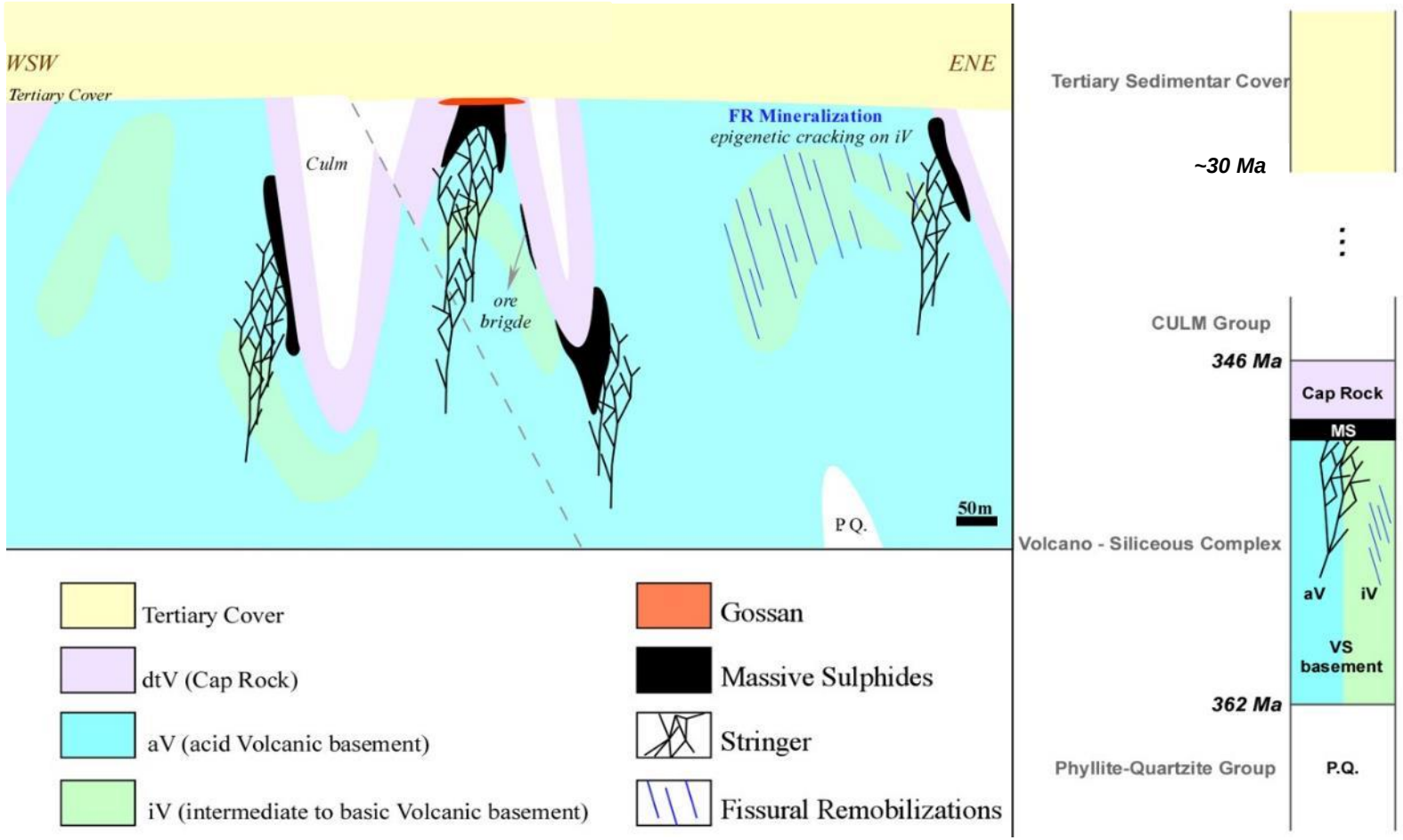


Figure 8.5 - Conceptual model showing the distinct types of mineralization present in the Lagoa Salgada Deposit

9 EXPLORATION

Exploration activities and results prior to 2015 are summarized in the History Section of this report. Historical results prompted the latest exploration programs developed after the acquisition of Redcorp-Empreendimentos Mineiros, Lda by Mineral & Financial Investments and the subsequent Ascendant Resources entry in 2018. Since 2015, the main exploration goals have been to increase and upgrade the mineral resources to the Measured/Indicated levels, and to establish mineralogical/metallurgical characteristics to pave the way for economic studies. Notable progress has been achieved on all fronts. The main exploration activities have included petrographic studies, geophysical investigations (complemented by core drilling) and metallurgical tests.

9.1 PETROGRAPHIC STUDY

In 2016, the Faculty of Sciences of the University of Porto conducted a petrographic study of about 20 samples from various stratigraphic horizons, namely the Gossan, Massive Sulfides, Chert/Jasper and Stockwork. The objective was to identify the mineralogical phases of interest, as well as their modes of occurrence. This study led to the conclusion that for the massive sulfides, sphalerite and galena mineralization occur together with arsenopyrite and some chalcopyrite. Sphalerite and galena can occur with various grain sizes, and larger crystals of sphalerite contain inclusions of chalcopyrite. Galena typically occurs intergrown with chalcopyrite. The latter rarely exceeds 25 µm and occurs at the edges of pyrite grains or as filling in small fractures. Fine cassiterite crystals intergrown with sphalerite were also found.

In 2017 and 2019 EPDM conducted a series of mineralogical deportment and liberation analysis using a Mineral Liberation Analyzer (MLA). This technique was applied to 35 pulps received from the geochemistry lab, and to previously selected intervals of 36 core samples. In both cases, a wide variety of samples was selected from all the mineralized domains. In this analysis, modal mineralogy together with the distribution of the elements in the different minerals was performed (Figure 9.1 and Figure 9.2). The aim of applying this technique was to understand the principal carriers of the elements of interest together with their mineralogical properties such as size, shape, and intergrowth. The main conclusions of this study are that except for the FR, sphalerite is the main zinc-bearing mineral, whilst in the FR some of the zinc can be in the form of carbonate in smithsonite. The Zn from the Fr and Stringer domains are quite similar having a different composition from the Massive Sulfides, mainly because they are free of Hg. The principal Pb bearing mineral except for the Gossan domain is galena which is well correlated with sphalerite. Gossan includes cassiterite already liberated. The Ag is mainly distributed in two minerals: Freibergite and Tetrahedrite, and Cu is mainly in the form of chalcopyrite with some of it in chalcocite.

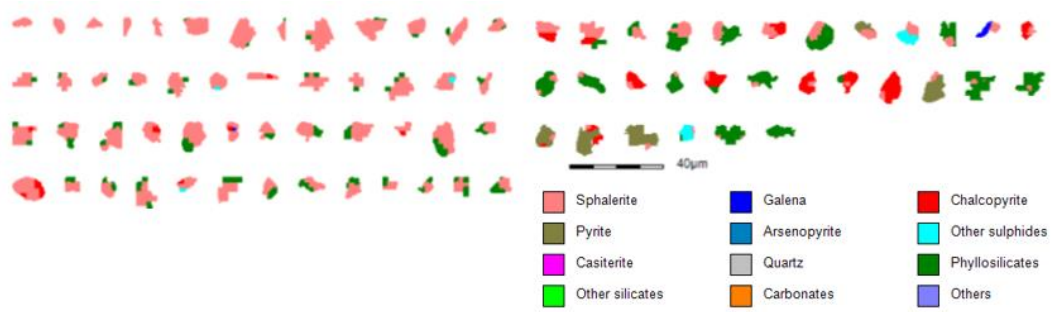


Figure 9.1 - Example of the mineral distribution in the different particles between 7-15 µm

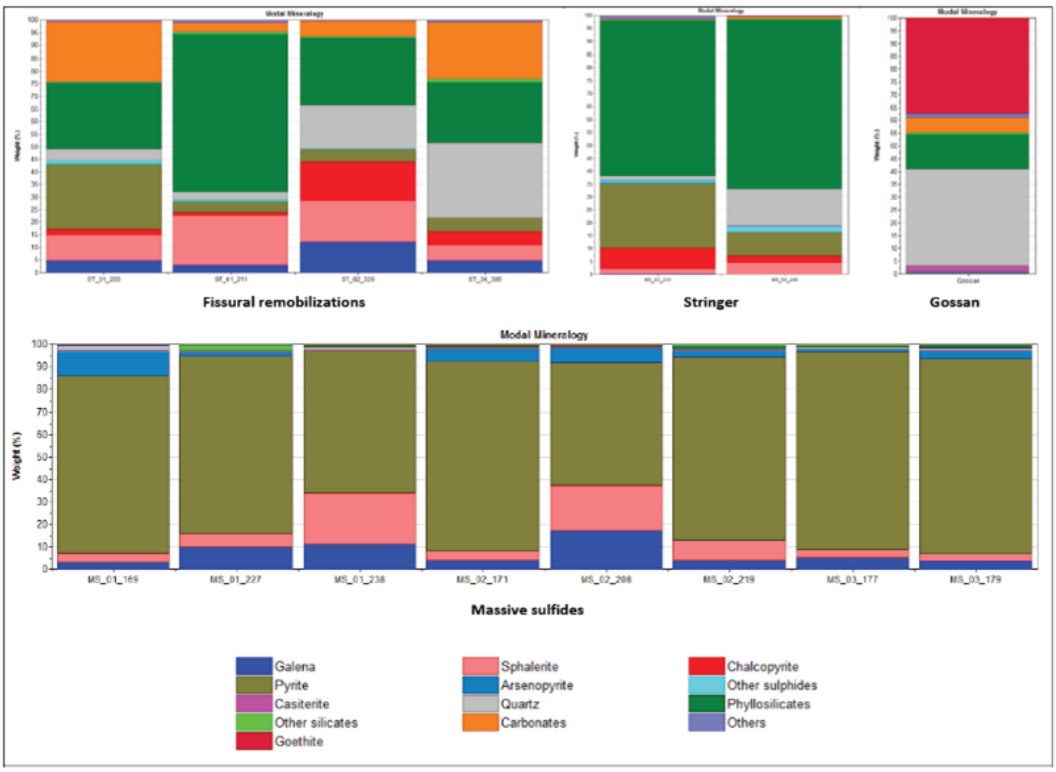


Figure 9.2 - Modal mineralogy for the different domains

In 2022 PetroLab performed a mineralogical study over 14 samples from all the different domains (Gos, tMS, pMS, str and FR). Three size fractions from each bulk sample were analyzed using a Zeiss EVO MA microscope coupled with two Bruker xFlash 6|60 x-ray detectors for energy-dispersive X-ray spectroscopy (EDX) analysis. From this study, the main conclusions are that the pMS samples are characterized by abundant pyrite, with varying proportions of other sulfides with a greater abundance of sphalerite and galena. In the gossan sphalerite is almost absent with minor amounts of galena. As for the distribution of the main elements, Cu is primarily hosted in chalcopyrite and in 2nd Cu phase, the Zn is primarily hosted in sphalerite with some of it in descloizite together with Pb which is mainly hosted in galena, and the Sn is exclusively hosted by cassiterite. The likely penalty elements

are arsenic (As) and mercury (Hg). As is hosted in the form of arsenopyrite with some enargite in the stringer although this is not relevant. The principal Hg hosting phase is cinnabar, but it can be in small portions in other Hg bearing phases in the Gossan and Stringer samples.

9.2 GEOPHYSICAL STUDIES

In 2017, a downhole survey campaign was conducted on three boreholes (LS_MS_01, LS_MS_03, and LS_MS_06) to investigate the existence of a conductive mineralized body between them. This body has an orientation of N160E and is approximately 500 meters long.

Subsequently, in 2018-2019, a more intensive geophysical campaign was carried out, including a study of the physical characteristics of the Lagoa Salgada deposit samples, a study of the deposit's gravity anomalies, and an induced polarization (IP) study of electrical resistivity, both on the surface and inside boreholes.

The study of the lithotypes at Lagoa Salgada allowed the definition of the physical characteristics of each one, with the goal of better understanding the obtained results in the later stages of the geophysical study. This way, the massive sulfides have a high density (specific gravity of ~4.6), high electrical conductivity, and electrical chargeability. The host volcanic rocks of the mineralization are moderately dense (specific gravity of ~2.8), with lower conductivity and chargeability. However, they have lower resistivity when altered to clay. When these volcanic rocks are mineralized with stringers or subjected to fissure remobilization, they are intermediate relative to the two types of rocks previously described, with an average specific gravity of 3.0 and a conductivity dependent on how metallic minerals are texturally arranged in the sample, although all samples with sulfides show a good response to changeability. Lastly, the tertiary cover of the deposit was tested, specifically, the Vale do Guizo formation, which showed the lowest density (specific gravity of ~2.2) and lower resistivity near the contact, which increases with decreasing depth.

The more detailed study and reinterpretation of the historical gravimetric data allowed for the understanding that the length of the deposit was not limited to just 400 meters from the massive sulfide deposit, extending southward. From the borehole records, it was possible to see that there is a good correspondence between the gravity anomaly with massive sulfides and stringer/stockwork, where the residual Bouguer anomaly is high (see Figure 9.3).

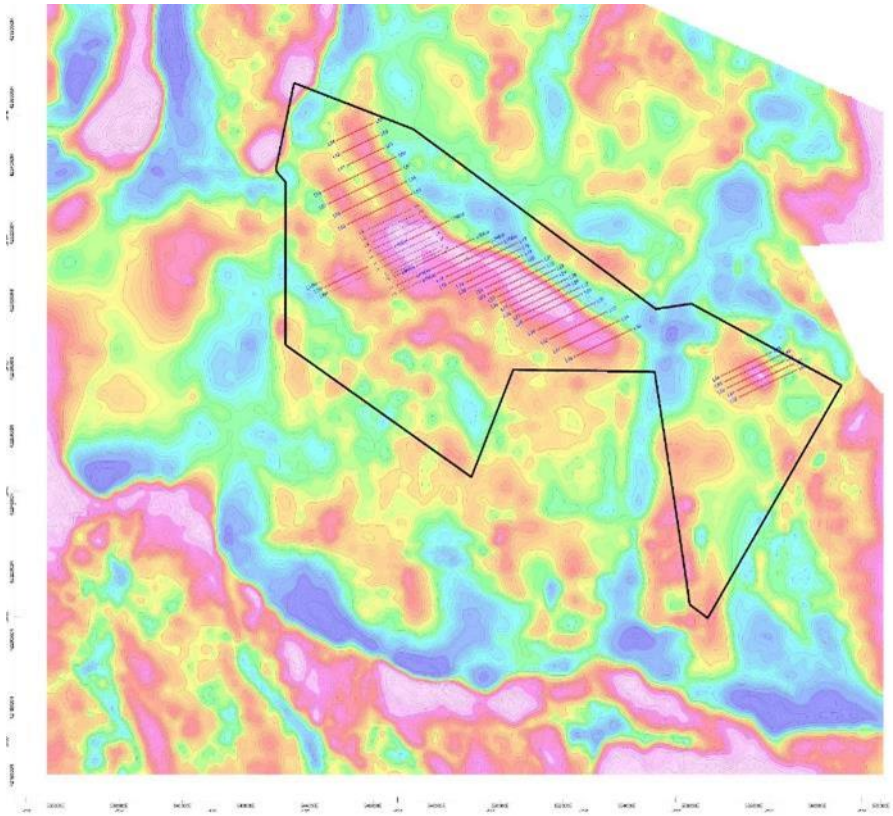


Figure 9.3 - 2019 reinterpretation of historical gravimetric data

In 2019 an IP/electrical resistivity study was conducted, with two phases: one on the surface and one inside of the boreholes. The surface study was carried out over 74.4 km^2, indicating high chargeability near the contact between the Volcanic-Silicious Complex (CVS) rocks and the tertiary cover, with the maximum at the northern sector of the Lagoa Salgada deposit and a peak of chargeability associated with the southern sector. Upon closer observation of the results, it was found that this peak is shifted approximately 100 m to the east, which could be an indicator of the lithology at the top of the iV rocks in the southern sector, where esV may occur due to their pyritic levels. In order to increase the resolution of the chargeability and resistivity models in depth, tests were conducted inside previously drilled boreholes. This method spatially correlated the chargeability model with the known mineralization for the northern resource and it's in agreement with the surface models that suggest that both the northern and southern sectors are part of the same anomalous zone. A map of the Chargeability from this IP survey can be seen in the Figure 9.4.

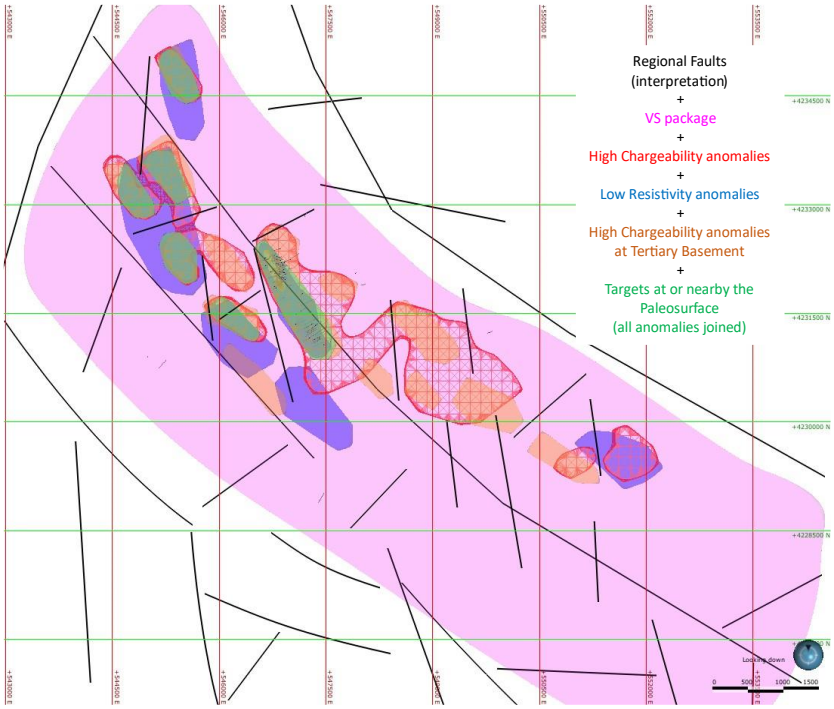


Figure 9.4 - Plan with the IP anomalies identified

In 2022 a ground Time Domain EM survey which covered the North and South Zones of the Lagoa Salgada VMS Deposit as well as the area lying to the northeast was conducted. A total of 43 kilometers of survey were completed revealing several EM anomalies not previously identified by other geophysical methods. Conductivity contour maps of the entire survey coverage illustrate distinctive conductivity highs. The survey lines that passed directly over the North Zone exhibited a unique data profile which can be seen in Figure 9.5. Additional to the anomaly over the North Zone, two new anomalies were identified. Anomaly B is shown in Figure 9.6.

Anomalies B and D prompted a step out drilling exploration program that commenced in 2023.

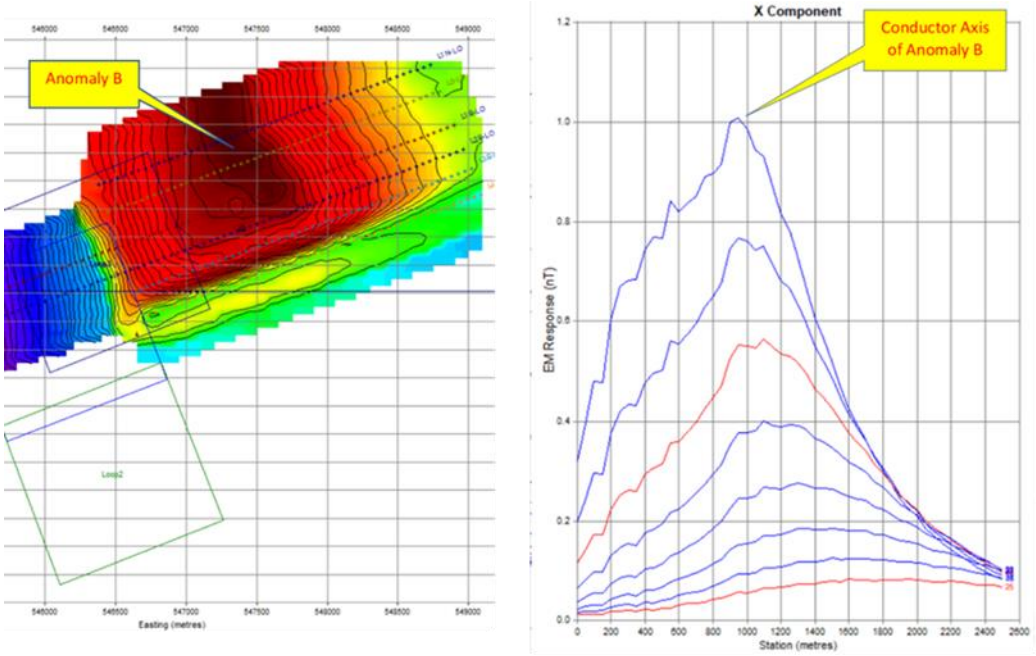


Figure 9.5 - Conductivity contours of Surface EM X Component highlighting Anomaly B in red. B- Surface EM Line 1-N X Component Profiles of Channels 18-25

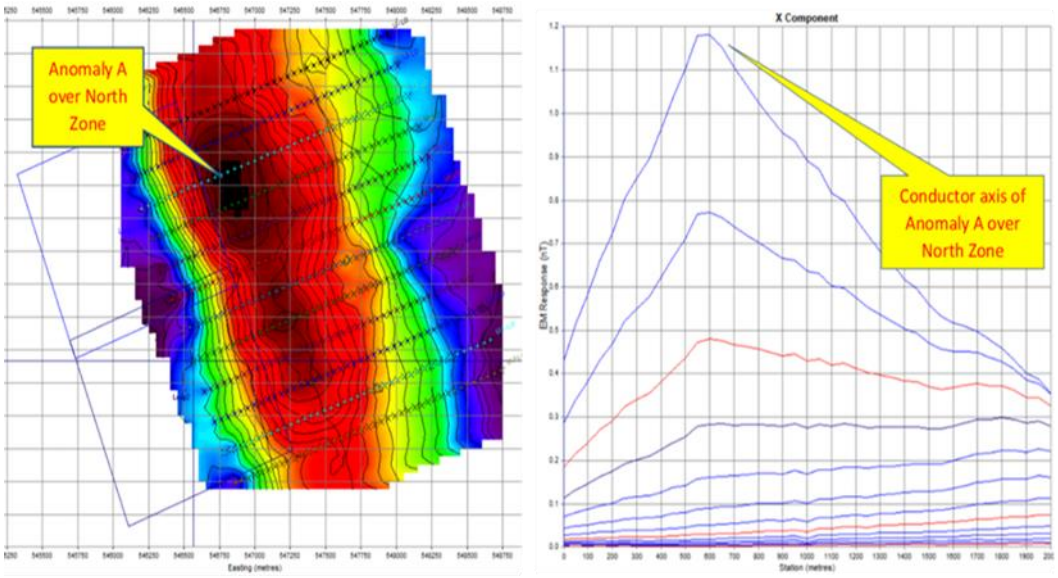


Figure 9.6 - A- Conductivity contours of Surface EM X Component highlighting Anomaly A in red. B- Surface EM Line 1-N X Component Profiles of Channels 18-30

10 DRILLING

The execution of core drilling is one of the main vectors for the development of the Lagoa Salgada project, having been explored mainly by drilling since 1995 to the present. Due to the total absence of Paleozoic outcrops within the area of interest, the geological cores obtained through drilling are the best source of information for the study of the various disciplines inherent in the project's progress. Accordingly, Redcorp has conducted drilling programs at the LS Property since 2005, including 2005-2009 under the direction of Redcorp Ventures, and 2011 – 2012 under the direction of Portex. The focus of drilling since 2007 has been on the North deposit and the South deposit areas. Since 2017 with the entrance of Ascendant, a total of 98 drill-holes totalling 35,795.7 m have been drilled by Drillcon Iberia. The average recovery of the drilling in the project, since 2017, is 94.1 %. Drilling was done in two phases until the end of 2020. In the first phase, a drilling rig with a tri-cone drill bit was used to drill the sedimentary units in a destructive mode, and then the drill hole was cased with a metal pipe for protection. Later, in the same location, a second drilling rig with HQ-sized diamond drill bits was installed to start recovering the core when drilling through the Paleozoic units. The drill bit size could be changed when the drill rods showed signs of stress, being reduced to NQ, but most of the drillings were carried out with the HQ size. From the end of 2020, a new working method was initiated, where in most of the drillings, the entire hole was cored, and the core was collected from the beginning. This method was used to obtain the necessary samples for the study of the sedimentary components for geotechnical, hydrogeological, and lithological purposes. Once the drill hole was completed, an NQ-size PVC pipe was inserted down the entire length of the drill hole. This was conducted to prevent the walls of the drill hole from collapsing prior to carrying out downhole geophysical surveys such as the Mise-a-la-Masse surveys.

Of the total drillings carried out, 11 drill-holes were done for metallurgical studies with a total of 4,359 m drilled. These studies were carried out at the Grinding Solutions Laboratories (GSL) in the UK. Seven of these holes were collared in the northern sector of the deposit and four in the southern sector. The objective of these studies was to fine tune the best techniques for processing the metals of economic interest.

Six of the 98 holes were completed for hydrogeological studies, with a total of 542 m drilled. These holes were completed with a tri-cone drill bit to drill the sedimentary units destructively. The holes were carried out in each sector of the deposit. Hole depths varied to study the different aquifers. Holes were fully lined with PVC pipes, with some areas perforated to allow water percolation. A summary of the drilling performed at the Lagoa Salgada is presented in Table 10.1. Figure 10.1 illustrates the locations of the drillholes on the property.

Table 10.1 - Total of drilling performed in the Lagoa Salgada Project to May 2023

Company	Period	Total Holes	Total Length (m)	Core Diameter
RTZ / EDM	1995	38	17,992	HQ
Redcorp Ventures	2005 to 2008	24	11,220	HQ
Portex	2011 to 2012	7	1,602	HQ

Company	Period	Total Holes	Total Length (m)	Core Diameter
Redcorp	2015 to 2017	10	3,464	HQ
Ascendant/Redcorp	2017 to 2018	20	7,077	HQ
Ascendant/Redcorp	2019	26	8,164	HQ
Ascendant/Redcorp	2020	3	1,284	HQ
Ascendant/Redcorp	2021	3	1,171	HQ
Ascendant/Redcorp	2022	31	12,378	HQ

10.1 ASCENDANT / REDCORP CORE LOGGING PROCEDURES

The ASCENDANT / REDCORP core logging procedures start from the moment the core is retrieved from inside the borehole. Listed below are the procedures that are adopted in the LS project:

- The drilling equipment operators retrieve the core from inside the borehole.
- The drilling equipment operator uses a black marker to write down the name of the borehole and box number. The depth of the borehole is also recorded on small wooden plates.
- Upon arrival at the core shed, the drill core goes through the following steps:
    - Core is reassembled in the box and if necessary, cleaned.
    - Core is photographed wet and dry with the results stored in an online database (IMAGO).
    - The following information is recorded in a digital database:
        - Core recovery.
        - Rock quality designation (RQD).
- The geotechnical log is carried out by collecting information about the various previously defined intervals according to their geotechnical characteristics. All relevant aspects for their characterization are observed and recorded, and all fractures are marked on the box according to their geotechnical family/class. Orientation is captured with IQ Logger (IMDEX) laser devise; all the collected information is stored in a digital database (Figure 10.2).
- The geological log is then carried out by recording the various lithological types, textural characteristics, mineralization, faults, and structures. All logs are stored in a MX Deposit digital database (Figure 10.3).
- Sampling was previously conducted in 1.0 m intervals respecting the contacts between different lithologies and since 2022 in 2.0m intervals.
- Blanks, High and Low multielement standards were inserted roughly every 20 samples.

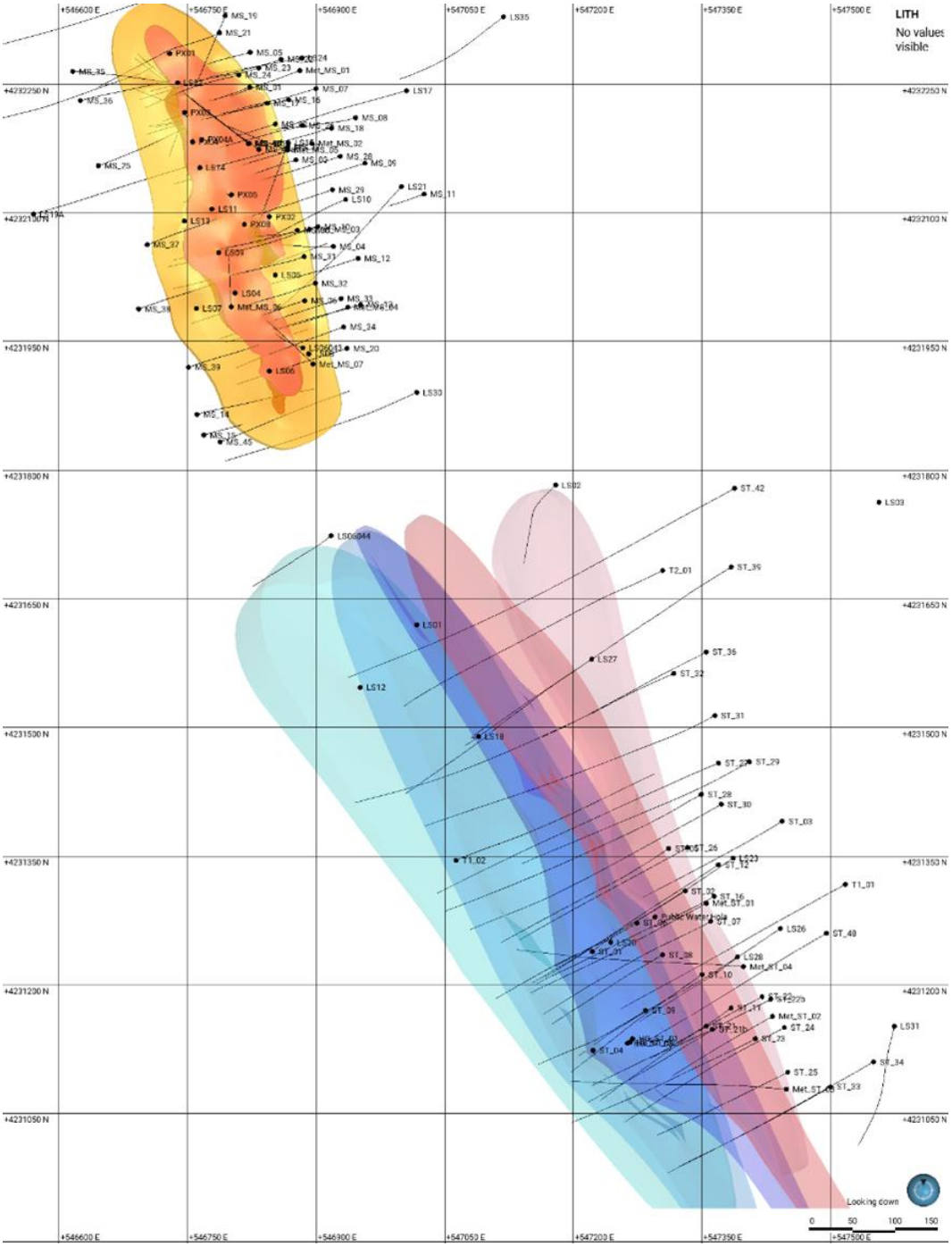


Figure 10.1 - Plan View of all drilling performed in the Lagoa Salgada property

Figure 10.2 - A- example of spreadsheet to collect geotechnical data; B- Orientation capture with IQ Logger (IMDEX) laser device

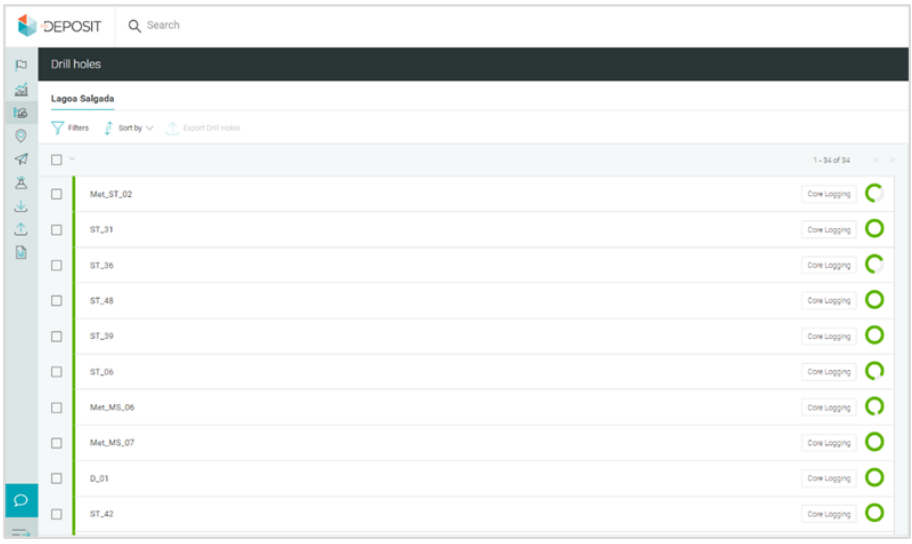


Figure 10.3 - Example of the drill holes in the MX Deposit database

10.1.1 2017-2018 Summary of Drilling Results

The 2017-2018 drilling focused on the North and South deposits. Analytical results confirmed the presence of tin mineralization in the MS zone of the North deposit in addition to zinc, lead, copper, silver, and gold. The South deposit appears to be enriched in copper at the expense of zinc and lead; however, the massive zone of the North deposit contains higher grade copper as compared to the stockwork zone of the South deposit.

Drillhole LS-ST-12 intersected MS on the eastern part of the South deposit. This intersection correlates well with the MS previously intersected in drillhole LS 23. Thus, there appears to be a MS zone associated with the South deposit which implies that the VMS system at the LS Property likely has more than one vent.

10.1.2 2019/2020 Summary of Drilling Results

The 2019/2020 drill program objectives were to upgrade the resources from the Inferred category to the Indicated / Measured categories and to expand the tonnages. Sectional interpretation of the drill intersections shows that the objectives were met.

10.1.3 2021 Summary of Drilling Results

Step-out drilling combined with geological/structural reinterpretation has culminated in the merging of the former Central deposit with the southern mineralized envelope to form one continuous deposit, the South deposit. Drill hole LS 27 confirmed the linkage.

10.1.4 2022 Summary of Drilling Results

The 2022 drilling main goal was to perform a series of step-outs with the aim of extending and infilling the northern area of the south sector; in addition, another goal was to collect samples for metallurgical testwork. The first goal was accomplished with success resulting in the definition of the FR corridors in the South deposit. For the second goal, metallurgical tests of the samples collected are still in progress.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 SAMPLE PREPARATION AND ANALYSES

11.1.1 SAMPLE PREPARATION AT SITE

Drill core in core boxes is transported from the drilling site to Redcorp coreshed by a designated Redcorp employee, by truck, and protected with straps and metal cover and then inspected to ensure that depth markers are in place, photographed, measured for core loss and RQD, then logged and marked for sampling. The sampling aspect involves cutting / splitting the drill core longitudinally into symmetrical halves followed by sampling. The cut process uses fresh water and is oversaw by Redcorp coreshed supervisor and the saw is cleaned with pressure water machine each 1m. The samples were taken at 1 m intervals until 2021 and then at 2 m intervals as from 2022. In both cases, the sample ends at lithological or alteration contacts within the mineralized zones, and, sometimes at longer intervals outside the mineralized zones, as determined by the project geologist. The entire length of Paleozoic rocks of the drill hole is sampled plus the last 10 m of the Conglomerates Tertiary basement from the Vale do Guizo unit. The sample is placed in a sample bag and then it is sealed. The position of the sample on the remaining half-core in the core box is marked with a corresponding ID tag for reference.

Sample reference sheets summarizing all the samples taken from each hole are provided during the core-cutting process. These sheets are used to identify where the quality control samples will be added to the sample stream and for preparing the requisition and shipment forms.

11.1.2 QUALITY CONTROL MEASURES

Redcorp has maintained well documented QA/QC measures since the inception of their drilling programs in 2014. Certified standard samples are inserted every 20th sample through the series and. Two blanks are also placed in every assay batch. A Redcorp Geologist is responsible of doing a visit to assess the conditions of the Lab at every campaign.

All standards and blanks are obtained from an independent third-party provider (CDN Resource Laboratories Ltd). Field duplicates consist of cutting the remaining half core into two with the diamond core saw, resulting in a quarter core being submitted to the laboratory as the field duplicate and a quarter core being retained for reference.

11.1.3 PACKAGING AND SECURITY

All activities pertaining to data collection, namely sampling, insertion of control samples, packaging, and transportation are conducted under the supervision of the project geologist.

Other than the insertion of control samples, there is no other action taken at site. Thus, no aspect of the sample preparation for analysis is conducted by an employee, officer, director, or associate of the issuer.

Samples are placed in sequence into rice bags which are labelled with company code and sample series included in the bag. Requisition forms are compiled using the sample reference sheets that were generated since the previous shipment. Sample bags are sealed and then stored in a locked sample dispatch room. When a shipment is ready, the sealed bags are dispatched to the ALS (Seville) laboratory via direct courier. Laboratory personnel check to ensure that no seal has been tampered with and acknowledge receipt of samples in good order via e-mail.

11.1.4 Laboratory Details

Redcorp uses the ALS (Seville) facility as their sample preparation laboratory and ALS (Sudbury) for the analytical work. The analysing laboratory (ALS) is ISO / IEC 17025:2005 accredited and both branches (ALS Seville and Sudbury) are independent of Redcorp/Ascendant. The ALS Laboratory is among several laboratories that regularly participate in the PTP-MAL (Proficiency Testing Program for Mineral Analysis Laboratories) round-robin laboratory program provided by Natural Resources, Canada, for minerals containing gold, platinum, palladium, silver, copper, lead, zinc, and cobalt.

11.1.5 Laboratory Sample Preparation And Analysis

Redcorp's samples were prepared by crushing the sample with up to 70% of the material passing a 2 mm screen, split to 250 g, and pulverized under hardened steel to 85% passing a 75 µ screen.

ALS (Seville) then sent the prepared sample to their sister laboratory in Sudbury, Ontario, for analysis. The remaining sample pulps and sample rejects are sent back to Redcorp.

The core samples are analysed for gold (ppm) by fire assay (Au-AA25), and for the other elements by multi-element analysis using optical emission spectrometry and the Varian Vista inductively coupled plasma spectrometer (ME-ICPORE). Samples from the North deposit MS zone are also assayed for tin (Sn) by ICP-AES after Sodium Peroxide Fusion (Sn-ICP81x). Besides that, since 2022 samples are also analysed by a four-acid digestion coupled with a mass spectrometer (ME-MS61) to analyse a wide range of elements.

11.1.6 Bulk Density

Bulk density measurements were collected on roughly alternate drillholes. The bulk density measurement used the instantaneous water immersion method which records the dry weight immediately followed by the weight in water which is used to calculate the bulk density. The results were entered into the database to correspond with the drillhole number, depth, grade, and rock and alteration types. Since 2021, certified specific gravity

measurements are being done for the samples sent to the laboratory at the rate of 1 SG determination in every 5 samples.

11.1.7 QUALITY CONTROL RESULTS

All assays are reported directly to Redcorp via e-mail to designated personnel. Signed assay certificates are sent via courier or post. The monitoring of the performance of the QA/QC samples is conducted immediately after the assay results are received. The assay results for control samples were plotted upon receipt of the initial assays. Certified reference materials (CRM) / standards were considered a failure if the assays were close to or outside 3 standard deviations and the whole batch would be re-analysed.

Three different types of standards were to test distinct aspects of the analyses. Standard GLG907-1 was used to test the Au assaying acting like a blank due to its low grade; standard CDN-ME-1501 was applied to test low values of the metals of interest namely the Au, Ag, Cu, Pb, and Zn; and standard CDN-ME-1804 was used to test the data quality at high values of the already mentioned metals. Blanks were considered a failure if they reported values three times above the detection limit. Overall, the performance of all control samples (blanks and standards) for analytical work has been satisfactory. The performance of distinct standards for the distinct metals can be seen in Figure 11.1, Figure 11.2, and Figure 11.3.

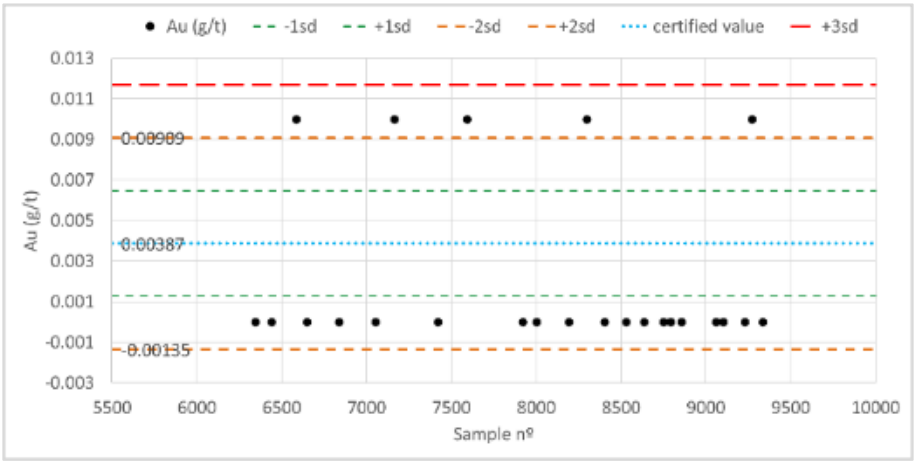


Figure 11.1 - Plot of the Au assay from the CDN-ME-GLG907-1 standard results with the several standard deviations

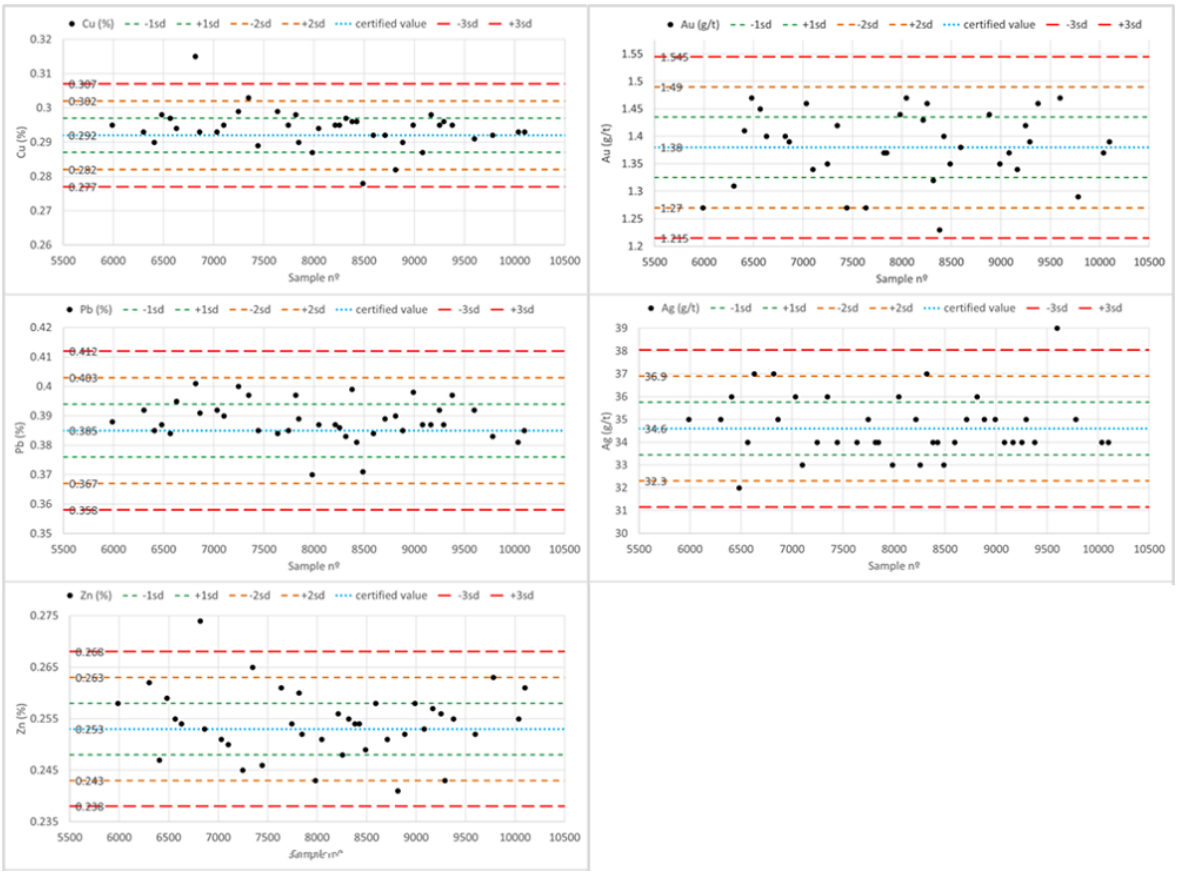


Figure 11.2 - Plot of the multielement assay from CDN-ME-1501 standard results with the several standard deviations

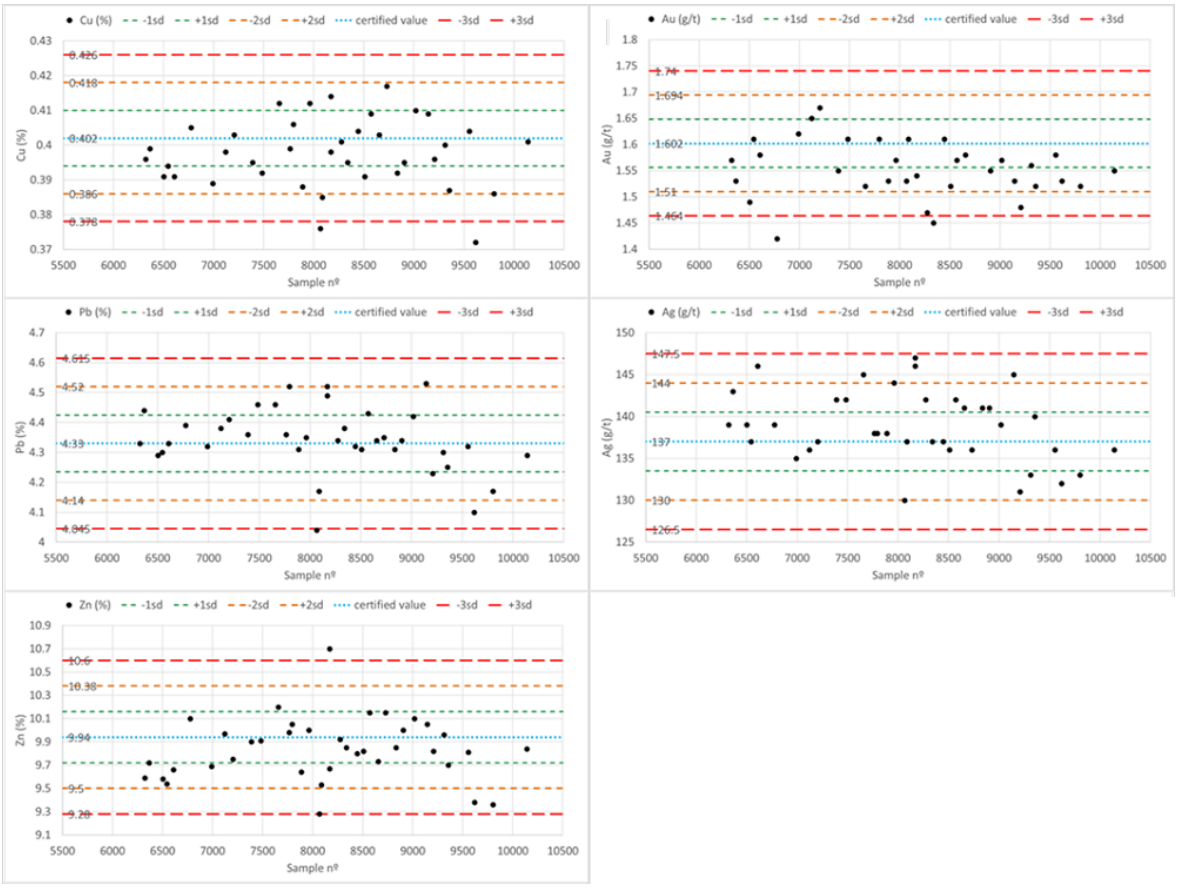


Figure 11.3 - Plot of the multielement assay from the CDN-ME-1804 standard results with the several standard deviations

11.2 COMMENTS

The QP considers the sample preparation, security, and analytical procedures to be adequate to ensure the credibility of the analytical results used for Mineral Resource estimation. The monitoring of the laboratory's performance on a real time basis ensures that corrective measures, if needed, are taken at the relevant time and gives confidence in the validity of the assay data.

12 DATA VERIFICATION

The steps undertaken by the QP's to verify the data in this Technical Report include several site visits to the Property, analysing monitoring reports on the performance of control samples and conducting a resource database validation.

No samples were collected to verify the mineralization at the LS Property during the site visits, as the mineralization is easily identified in drill cores with the unaided eye.

12.1 SITE VISIT

Quadrante Project Director, João Horta, MIMMM, visited the LS property several times during last 18 months. The Redcorp staff in attendance were João Barros (Redcorp Managing Director), Vítor Arezes (Senior Project Geologist) Luis Lima (Assistant Project Geologist).

Micon senior economic geologist, Charley Murahwi, P.Geo., FAusIMM, visited the LS Property from 16 to 19 October 2018, from 13 to 17 November 2018, from 28 to 31 May 2019, and from 24 to 27 May 2023. The Redcorp staff in attendance were Joao Barros (formerly Redcorp Managing Director and now President - Ascendant), Vitor Arezes (Senior Project Geologist) Luis Lima (Assistant Project Geologist). The site visits data verification activities and results achieved are summarized below.

12.1.1 VERIFICATION OF PHYSICAL FEATURES/DTM

Observations on the ground confirm a monotonous flat topography that conforms to the digital terrain model (DTM) provided by Ascendant/Redcorp with the database. Thick sequences of alluvium necessitate 4-wheel drive vehicles in wet conditions.

12.1.2 REVIEW OF DRILLING, LOGGING AND SAMPLING PROCEDURES

Drilling is conducted to industry standards with very minimal core losses. Downhole surveys are conducted using a Reflex Ez-Shot high precision magnetic and gravimetric instrument.

Micon reviewed the drill core logging procedures and sample collection methods and found them to be in line with the CIM best practice guidelines. Drill core is cut with a diamond saw to attain symmetrical halves. Wherever core is friable or heavily weathered, splitting is done manually.

Samples are dispatched to the laboratory in secure containers. This minimizes damage to sample bags during transportation that may result in contamination between samples.

12.1.3 Examination of Mineralization in Drill Core

Most of the drilling on the LS Project was conducted using HQ-size core, yielding good core recoveries, and in turn, representative samples. Micon examined diamond drill cores from 6 holes of the North deposit and three holes of the South deposit. All the major mineralization and alteration styles described in the geology section of this report were confirmed.

In several cases, it is difficult to identify the best mineralized zones visually but, overall, assay results generally match the mineralized intercepts observed in drill cores.

12.1.4 Review of QA/QC Protocols and Data Collection Techniques

Micon has reviewed Ascendant's/Redcorp's QA/QC protocols and found them to be well documented/executed and in line with the CIM Best Practice Guidelines. Recent/current drilling is being conducted using HQ-size core, yielding good core recoveries, and in turn, representative samples.

12.2 Review Of QA/QC Results

As observed and described in Section 11.0, the performance of control samples demonstrates that the analytical work to date is credible for Ag, Pb, Zn, Au, Cu and Sn. However, for future work, Micon recommends that between 5% and 10% of the sample assay rejects be analysed at an umpire laboratory. Also, the insertion rate of control samples for blanks should be increased to 5% to monitor contamination more closely.

12.3 Database Validation

Ascendant/Redcorp provided Micon with a complete updated Mineral Resource database comprising collar, survey, assay, lithology, alteration, and structure tables in csv and excel file formats. In addition, DTM and tertiary cover contacts were provided in DXF file format. The resource database review and validation were performed by Micon involved the following steps:

- Comparing the database assays and intervals against the original assay certificates and drill logs.
- Checking for any non-conforming assay information such as duplicate samples and missing sample numbers.
- Verifying the collar elevations to ensure a satisfactory match with the DTM / topo map.

No errors of significance were found.

12.4 DATA VERIFICATION CONCLUSIONS

The QP has not found any issues with Ascendant/Redcorp's data collection techniques and QA/QC protocols. Thus, the database of the LS Project is considered to have been generated in a credible manner and to be sufficiently error-free to support Mineral Resource estimates.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

Metallurgical testing of Lagoa Salgada samples began in 2019 with a series of characterisation tests completed by Grinding Solutions (GSL hereafter) in Cornwall, UK. The work was followed by additional metallurgical testing at the same laboratory in 2021, 2022 and 2023, and several mineralogical assessments by Petrolab Laboratory were completed in parallel, as well as reproducibility flotation testwork developed by Maelgwyn Mineral Services Africa.

The approach to flowsheet development was to prepare representative master composites for each ore type, then proceed through open circuit to identify and optimize flowsheet conditions and reagent schemes. Locked cycle tests were then conducted on some of the master composites to demonstrate the anticipated overall metallurgical performance within a continuous circuit.

Sub-samples of individual drill hole material used to create the master composites were subsequently tested to provide variability data for feed grade and feed grade metal ratios, allowing the assessment of flowsheet robustness and required reagent dosage ranges.

Tests were completed on blends of Primary Massive Sulphide (PMS), Stockwork (STW), Gossan (GO), Transition Massive Sulphide (TMS) and Stringer (STR) ores to allow comparison with individual composite results and to assess the viability of co-processing the ore types.

Mineralogical assessments were undertaken to provide information to refine the comminution/beneficiation process during optimization, and to provide reasonable expectations for metallurgical performance versus mineral liberation and association within each ore type. Samples from various open and locked cycle test products were used to characterise final concentrates and tailings.

Although the principal domains testing results are conclusive and well advanced, there is still work to be developed with all the domains to confirm the final grades, recoveries and process flowsheet being published in this document.

13.2 HISTORICAL METALLURGICAL TESTING

The following studies were undertaken with respect to Lagoa Salgada samples.

- Scoping Metallurgical Study on the Lagoa Salgada Deposit, Portugal (GSL, 2019).
- Lagoa Salgada Testwork Confirmation Programme (GSL, 2021).

13.2.1 2019 Scoping Metallurgical Study

Samples tested consisted of PMS and STW. The main objective of the testwork programme was to determine the achievable grades and recoveries of Cu, Pb, Zn, Sn, Ag and Au. A key focus of the programme was to understand if this material could produce similar results to other operating mines in the local area. The main testwork programme was carried out on a composite sample of fresh material, while the optimum flotation conditions from this programme were taken forward for comparative cleaner tests on the oxidised material.

13.2.1.1 Primary Massive Sulphide (PMS)

Flotation locked cycle results delivered Pb recoveries of 36.9% at a grade of 33.4% Pb (10.8% Zn) and delivered Zn recoveries of 60.7% at a grade of 42.4% Zn (5.3% Pb). Distributions of 93.8% Sn, 89.9% Au and 76.6% Ag reported to the final tailings.

Table 13.1 – 2019 LCT results for PMS (concentrate grades).

	Assay Calibrated XRF %					
Product	**% Cu**	**% Pb**	**% Zn**	**% Fe**	**ppm Au**	**ppm Ag**
Pb Conc	1.1	33.4	10.8	18.1	1.7	313.8
Zn Conc	1.1	5.3	8.6	8.6	0.8	144.1
Zn Cl Sc/Ro Tail	0.3	1.8	35.6	35.6	0.9	58.4
Total	0.3	2.9	2.8	32.7	0.9	68.0

Table 13.2 – 2019 LCT results for PMS (concentrate recoveries).

	Recovery					
Product	**% Cu**	**% Pb**	**% Zn**	**% Fe**	**% Au**	**% Ag**
Pb Conc	10.1	36.9	12.2	1.8	6.3	14.8
Zn Conc	13.1	7.3	60.7	1.1	3.9	8.6
Zn Cl Sc/Ro Tail	76.9	55.7	27.1	97.2	89.9	76.6
Total	100.0	100.0	100.0	100.0	100.0	100.0

The final Pb concentrate contained 313.8 g/t Ag and the final Zn concentrate contained 144.1 g/t Ag to recoveries of 14.8% and 8.6% respectively; both Ag concentrates are above payable levels.

Au distribution was poor with 6.3% recovered to the Pb concentrate at a grade of 1.7 g/t, and 3.9% recovered to the Zn concentrate at a grade of 0.8 g/t.

The Lagoa Salgada massive sulphide deposit exhibits an extremely fine-grained mineral matrix, similar to other Iberian Pyrite Belt deposits. The Cu contained within the deposit is extremely fine, beyond economic range for recovery, and thus the focus was on the Pb and Zn portion of the sample.

The galena is also very fine in nature with mediocre concentrate grades of 32% Pb and a very modest recovery of 38% achieved in the locked cycle test. The sphalerite is the coarsest of all the economic sulphide minerals of interest and generated locked cycle results of 42% Zn grade to a 61% recovery. Both the concentrates contained elevated levels of Hg, Sb and As that would require further investigation into their deportment to the respective concentrates. Ag concentrations in both the Pb and Zn concentrates should be of payable levels.

The Sn, present chiefly as cassiterite, reported almost exclusively (94%) to the Zn flotation tailings, suggesting that at a primary grind size of D80 of 29 µm, the cassiterite is well liberated and should be available for concentration. The mineralogical assessment suggests that Sn liberation is excellent below 20 µm.

The Au deportment to Pb and Zn concentrates was poor, with a combined 10% of the Au being recovered to the Pb and Zn concentrates. 90% of the Au reported to the tailings and investigations are required to ascertain if this is recoverable, particularly by gravity concentration within the comminution circuit.

13.2.1.2 STOCKWORK (STW)

Locked cycle results delivered Cu recoveries of 61.6% at a grade of 24.7% Cu (14.6% Pb, 8.0% Zn loss to Cu concentrate), Pb recoveries of 74.7% at a grade of 49.0% Pb (29.6% Cu, 18.0% Zn loss to Pb concentrate) and Zn recoveries of 60.5% at a grade of 53.2% Zn (4.3% Cu, 2.72% Pb loss to Zn concentrate).

Table 13.3 – 2019 LCT results for PMS (concentrate grades).

	Assay Calibrated XRF %					
Product	**% Cu**	**% Pb**	**% Zn**	**% Fe**	**ppm Au**	**ppm Ag**
Cu Conc	24.7	16.4	12.1	15.7	0.4	321.6
Pb Conc	6.9	49.0	15.8	5.8	0.2	460.9
Zn Conc	1.0	1.8	53.2	3.7	0.1	84.0
Zn Cl Sc Tail	0.3	1.2	3.5	17.6	0.2	27.6
Zn Cl Ro Tail	0.1	0.4	0.9	16.6	0.1	7.7
Total	1.8	5.1	6.8	14.8	0.2	64.4

Table 13.4 – 2019 LCT results for PMS (concentrate recoveries).

	Recovery					
Product	**% Cu**	**% Pb**	**% Zn**	**% Fe**	**% Au**	**% Ag**
Cu Conc	61.6	14.6	8.0	4.8	11.4	22.6
Pb Conc	29.6	74.7	18.0	3.1	10.0	55.7

Zn Conc	4.3	2.7	60.5	1.9	6.2	10.1
Zn Cl Sc Tail	1.1	1.5	3.3	7.6	8.9	2.7
Zn Cl Ro Tail	3.4	6.4	10.2	82.6	63.5	8.8
Total	100.0	100.0	100.0	100.0	100.0	100.0

The final Cu concentrate contained 321.6 g/t Ag, Pb concentrate contained 460.9 g/t Ag and the final Zn concentrate contained 84.0 g/t Ag to recoveries of 22.6%, 55.7% and 10.1% respectively. The Cu and Pb will contain payable amounts of Ag, but the content in the Zn concentrate is below payable levels.

Au distribution was poor with 11.4% recovered to the Cu concentrate at a grade of 0.38 g/t, 10.0% recovered to the Pb concentrate at a grade of 0.19 g/t and 6.2% recovered to the Zn concentrate at a grade of 0.12 g/t.

The stockwork component of the Lagoa Salgada deposit exhibits preactivated Pb and Zn which are difficult to depress. It is most likely that the solubilisation of secondary Cu minerals in the grinding process are the cause of the pre-activation.

This material is coarser than the massive sulphide with separate, although un-optimised, Cu, Pb and Zn concentrates produced at saleable grades. The Cu concentrate generated a concentrate grade of 24.7% Cu, but also contained higher levels of Pb and Zn. This highlights the increased levels of secondary Cu minerals, both through pre-activation and the fact that saleable concentrate could be made with such high levels of Pb and Zn.

The mineralogy suggests that the Cu is the coarsest of the minerals in this material, as opposed to the massive sulphide where it was the finest, so fine that it is uneconomic to recover.

Pb concentrate was primarily contaminated with left over Cu and entrainment of Zn. If the separation between the Cu and the Pb could be improved, as well as additional depression of the Zn, Pb grades of above 50% could be achievable.

The Zn concentrate was very clean, greater than 53% Zn. Additional recovery opportunities for the Zn lie with improving the Pb/Zn separation and reclaiming some extra recovery from the final tailings.

Ag recovery was very good with over 78% recovered to payable quantities in the Cu and Pb concentrates, a stark contrast to the MS where only 23.4% of the Ag was recovered.

13.2.2 2021 Testwork Confirmation Programme

The aim of the testing was to confirm results previously achieved on PMS and STW samples with head grades more representative of potential ROM material.

Overall, with both aspects of testing carried out, the metallurgical test work results support the use of a conventional polymetallic process flowsheet capable of recovering copper,

lead, zinc, gold, and silver in three saleable concentrates. In addition to this, there is potential for the for the pyrite tailings to be leached for additional gold and silver, with the remaining tailings used for tin extraction.

It should be noted that due to limited availability of fresh core, the testing material used was approximately 2 years old and partially oxidized which had a negative impact. limiting results at this time. GSL was confident that with further work on fresh core, recovery expectations will be in line with or better than the average seen at existing mines on the Iberian Pyrite Belt.

13.2.2.1 PRIMARY MASSIVE SULPHIDE (PMS)

Extended rougher scoping study showed that pH control in the mill largely controlled the mass pull and subsequently the grade of Pb and Zn concentrates. Evaluation of reagents and residence time resulted in Pb rougher concentrate of 13.5% Pb at 64% recovery. Zn rougher concentrate of 14.3% Zn at 68% recovery.

Cleaner tests showed that 5 stages of cleaning were required to maximise the grade of both the Pb and Zn. Pb concentrates grade of 29.8% Pb at 21% recovery was achieved. Zn concentrate grades of 46.6% Zn at 45% recovery was achieved.

The first of two MS LCT showed a reduction in Pb and Zn grades and recoveries during the test. The second MS LCT, using a modified flowsheet, resulted in Pb grade 22% at 43% recovery and Zn grade 35% at 66% recovery. Compared to previous testing it could be seen that the reduction in grade was countered by the increase in recovery. The increased recovery was a function of the oxidised state of the material reducing selectively as well as increased activation of pyrite.

13.2.2.2 STOCKWORK (STW)

STW LCT resulted in Cu grade 25% at 69% recovery, Pb grade 28% at 16% recovery and Zn grade 44% at 54% recovery. Large losses of Pb were seen to the Zn concentrate due to circuit configuration and losses to the Cu concentrate.

Blend LCT result in Cu grade 24% at 53% recovery, Pb grade 12% at 5% recovery Zn grade 28% at 64% recovery. The Cu concentrate achieved with a blend of the PMS and STW, demonstrated that the Cu in the STW could still be recovered at saleable grade if processed as a blend. Higher grades of Pb and Zn required adjustments to residence time and circuit configuration however flotation as a blend was possible. Extraction for Au through leaching of the bulk sulphide concentrate showed that poor recovery was achieved (maximum 25%). Extraction increased with decreasing particle size.

Metallurgical testing showed that despite the oxidized sample, comparable performance could be achieved and allowed for the development of a more robust and flexible circuit that could handle variability throughout the orebody and if encountered, oxidized fresh feed.

13.3 2022-2023 METALLURGICAL TESTING

In February 2022, a new testing programme was initiated at GSL to provide additional data to support the feasibility-level process design. Samples selected for this work include the following composites:

- Primary Massive Sulphide (PMS).
- Stockwork (STW).
- Gossan (GOS).
- Transition Massive Sulphide (TMS).
- Stringer (STR).

Testing initially looked to confirm previous testing responses seen with PMS and STW samples, and looked to evaluate a viable process route for the 3 newly identified domains which previously had either not been tested or included in the PMS domain. The testing programme also included variability composites for hardness and beneficiation response characterisation. Key elements of the testing programme include the following:

- Mineralogical investigation (for PMS, STW, GOS, TMS, STR).
- Comminution characterisation, including standardized Bond Ball Work Index, Bond Rod Work Index, SAG Mill Comminution and Abrasion Index testing (for PMS, STW, GOS, TMS, STR).
- Flotation testing (for PMS, STW, GOS, TMS, STR).
- Leaching testing (for STW, GOS).
- Gravity testing (for STW, GOS).

13.3.1 SAMPLE SELECTION

The samples used for the 2022-2023 testwork programme consisted of the zone composites for PMS, STW, GOS, TMS and STR that were obtained from the latest drilling programme. The test charges used had been freezer stored, and benchmark testing at the start of the programme confirmed that the metallurgical response for the samples was consistent with the earlier work.

The head assays of blended composites are shown in Table 13.5.

Table 13.5 – Composite samples head assays.

Domain	% Cu	% Pb	% Zn	% Sn	ppm Ag	ppm Au

PMS	0.3	3.6	4.6	0.46	73.3	0.8
STW	0.50	1.01	1.72	-	21.6	0.1
GOS	0.08	1.71	0.49	0.26	41.1	1.0
TMS	1.33	2.52	0.19	0.24	233.1	1.4
STR	0.40	0.18	0.66	0.14	19.8	0.1

Variability composites were also generated for grindability and beneficiation testwork. The composites organised by zone and downhole depth were prepared and submitted for testwork. The head analysis of these composites is presented in following tables.

Table 13.6 – PMS variability samples head assays.

Sample	% Cu	% Pb	% Zn	ppm Ag	ppm Au
PMS1	0.24	3.22	3.72	32.22	0.23
PMS2	0.29	1.90	3.79	44.67	0.38
PMS3	0.33	0.27	0.56	15.00	0.07
PMS4	0.17	7.01	6.29	111.33	1.47
PMS5	0.24	8.38	8.16	151.00	1.19
PMS6	0.24	1.20	1.92	47.00	0.19
PMS7	0.25	7.93	6.83	163.67	1.12
PMS8	0.25	2.10	4.54	39.50	0.30
PMS9	0.19	6.76	7.40	118.00	1.73
PMS10	0.30	1.99	2.69	33.50	0.25
PMS11	0.76	0.13	1.06	16.50	0.12
PMS12	0.25	7.65	8.91	129.00	1.39
PMS13	0.29	6.53	5.37	182.00	0.89
PMS14	0.30	2.55	1.90	37.00	0.62
PMS15	0.26	3.27	4.50	38.50	0.26

Table 13.7 – STW variability samples head assays.

Sample	% Cu	% Pb	% Zn	ppm Ag	ppm Au
STW 1	0.43	0.72	1.7	18.8	0.12
STW 2	0.31	0.044	0.11	9.5	0.08
STW 3	0.44	0.41	1.93	17.2	0.09
STW 4	0.39	0.92	1.47	20.2	0.2
STW 5	0.028	0.071	0.12	1.1	0.19
STW 6	0.38	1.21	2.35	22.1	0.11
STW 7	0.025	0.17	0.33	2.2	0.04
STW 8	0.15	0.36	0.6	4.5	0.03

STW 9	0.39	0.082	0.31	17.6	0.13
STW 10	0.12	0.018	0.069	4.9	0.03
STW 11	1.29	1.18	2.07	26.4	0.17
STW 12	0.19	0.31	0.62	5.1	0.11
STW 13	0.83	0.22	0.5	11.2	0.09
STW 15	0.012	0.056	0.15	1	0.03
STW 17	0.093	0.16	0.46	3.2	0.04
STW 18	2.72	0.69	1.69	91.6	0.06
STW 19	3.5	1.38	1.72	71.6	0.19
STW 20	0.28	0.11	0.48	9.7	0.08

Table 13.8 – GOS variability samples head assays.

Sample	% Cu	% Pb	% Zn	ppm Ag	ppm Au
GOS 1	0.06	2.11	0.46	22.81	0.64
GOS 2	0.12	1.69	0.63	13.75	0.45
GOS 3	0.03	1.83	0.31	42.50	1.33
GOS 4	0.03	2.81	0.42	12.17	0.13

Table 13.9 – TMS variability samples head assays.

Sample	% Cu	% Pb	% Zn	ppm Ag	ppm Au
TMS 1	2.06	3.33	0.28	44.50	0.90
TMS 2	1.07	0.67	0.06	98.00	1.51
TMS 3	1.76	2.95	0.06	241.50	0.70
TMS 4	2.71	1.51	0.04	250.00	1.12

Table 13.10 – STR variability samples head assays.

Sample	% Cu	% Pb	% Zn	ppm Ag	ppm Au
STR1	0.39	0.34	0.94	15.75	0.02
STR2	0.84	0.04	0.52	24.00	0.01
STR3	0.30	0.61	1.01	14.50	0.04
STR4	0.03	0.36	1.29	8.75	0.01
STR5	0.14	1.15	2.28	52.00	0.59

STR6	0.09	2.61	2.68	46.33	0.06
STR7	0.23	0.53	0.98	23.00	0.05
STR8	0.27	1.26	2.13	45.33	0.46
STR9	3.86	1.19	3.68	86.67	0.52

13.3.2 MINERALOGICAL INVESTIGATION

Representative samples of the master composites were submitted for detailed feed mineralogy to determine the major mineral species and occurrences. Quantitative mineralogical analysis using ZEISS Mineralogic Mining (a product for automated mineral characterisation based on scanning electron microscopy) was requested.

A phase classification scheme was developed using the Mineralogic Mining software. Delineating grains into different phase classes is achieved by matching criteria that compare the quantitative measurements of elemental composition, as determined from the ED spectrum, with standard mineral composition data. A mineral group name or a general phase name (after dominant elements) is used for a class where there is a range in the elemental composition data such that a specific mineral member cannot be separately identified.

Phase/mineral data is reported in terms of weight percent. However, all data acquired is from 2D sections of 3D particles. Mass values are derived from the measurement of particle/grain areas, with no correction for stereological error, and an assumed phase density.

The target phases are sphalerite, chalcopyrite, cassiterite and galena. Together, they represent the principal mineralogies of each reconstructed sample. Sphalerite is highly localised in the PMS and STW samples, with a consistent abundance across the three size fractions. Galena displays the same trend as sphalerite in PMS and STW, but significant quantities are also seen in TMS. Galena in the GOS and STR samples does not reach significant values. Generally, GOS contains the greatest cassiterite abundances. Chalcopyrite is almost absent from the GOS samples, with the highest quantities found in the STW and TMS samples.

Gangue phases are dominated by pyrite, a quartz-Fe oxide assemblage, or a mixture of the two. STW hosts pyrite, as well as iron oxides and micas. PMS is dominated by pyrite, with additional sulphides. The abundant pyrite may have been oxidised, providing the mineralogy of the TMS and particularly the GOS assemblage. GOS and STR also contain chlorite, clays, and mica. Additional notable sulphides are also found in PMS and TMS, including cinnabar and chalcostilbite. Further Pb, Ag, Zn and Hg bearing phases are concentrated in the GOS and STR samples, likely from the breakdown of sulphides such as sphalerite, galena, and cinnabar.

Mineral abundances are summarized in following Table 13.11, Table 13.2, Table 13.3, Table 13.4 and Table 13.5.

Table 13.11 – Mineral distribution for the PMS domain samples.

[Wt.%]	Drill hole 1	Drill hole 2	Drill hole 3	Drill hole 4
Sphalerite	14.4	3.2	0.2	15.2
Galena	6.4	2.2	0.7	7.4
Cassiterite	0.3	0.1	0.4	0.3
Chalcopyrite	0.2	0.2	0.3	0.2
2nd Cu Phases	0.1	0.1	0.0	0.1
Pyrite	77.7	93.1	94.2	75.5
Fe Oxides	0.4	0.6	0.4	0.5
Quartz	0.2	0.2	2.9	0.4
Chlorite Group	0.1	0.3	0.0	0.0
Carbonates	0.1	0.0	0.0	0.0
Accessory Phases	0.1	0.1	0.2	0.1
Cinnabar	0.1	0.1	0.1	0.1
Kaolinite Illite	0.0	0.5	0.2	0.1
Mica Group	0.0	0.1	0.0	0.0
Chalcostilbite	0.0	0.1	0.3	0.1

Table 13.12 – Mineral distribution for the STW domain samples (sample with high clay content).

[Wt.%]	Combined Drill Holes
Sphalerite	2.00
Zincian Siderite	1.39
Chalcopyrite	0.47
Galena	0.54
Plumbogummite	0.18
2nd Cu Phases	0.08
Quartz	39.60
Chlorite Group	26.45
Mica Group	7.89
Carbonates	8.85
Pyrite	7.21
Kaolinite_Illite	3.46
Accessory Phases	0.97
Fe Oxides	0.92

Table 13.13 – Mineral distribution for the GOS domain samples.

[Wt.%]	Drill hole 1	Drill hole 2	Drill hole 3	Drill hole 4
Sphalerite	0.1	0.2	0.2	0.1
Galena	0.9	1.8	1.2	0.2
Cassiterite	0.6	0.9	0.4	0.0
Chalcopyrite	0.1	0.0	0.0	0.0
2nd Cu Phases	0.0	0.0	0.0	0.0
Pyrite	0.6	1.0	1.5	0.2
Fe Oxides	58.2	90.0	43.7	33.8
Quartz	37.6	3.6	38.7	19.8
Chlorite Group	0.6	0.2	1.9	9.2
Carbonates	0.1	0.1	6.0	2.7
Accessory Phases	0.5	0.3	4.3	4.9
Cinnabar	0.0	0.0	0.0	0.0
Kaolinite_Illite	0.0	0.0	0.1	17.0
Mica Group	0.0	0.1	0.7	12.0
Hg bearing	0.1	0.1	0.2	0.0
Pb bearing	0.3	0.2	0.4	0.0
Descloizite	0.1	1.5	0.8	0.0

Table 13.14 – Mineral distribution for the TMS domain samples.

[Wt.%]	Drill hole 1	Drill hole 2	Drill hole 3	Drill hole 4
Sphalerite	1.0	0.7	0.1	0.6
Galena	7.8	1.5	6.1	3.7
Cassiterite	0.2	0.2	0.3	0.3
Chalcopyrite	0.9	1.1	0.8	2.2
2nd Cu Phases	0.1	0.2	0.2	0.1
Pyrite	87.5	94.6	54.4	65.8
Fe Oxides	0.5	0.3	24.1	19.4
Quartz	1.6	1.2	13.3	7.2
Chlorite Group	0.0	0.0	0.1	0.1
Carbonates	0.0	0.0	0.1	0.1
Accessory Phases	0.1	0.1	0.3	0.3
Cinnabar	0.1	0.1	0.1	0.2
Chalcostibite	0.1	0.0	0.0	0.0

Table 13.15 – Mineral distribution for the STR domain samples.

[Wt.%]	Drill hole 1	Drill hole 2
Sphalerite	1.3	1.7
Galena	0.4	0.3
Cassiterite	0.2	0.5
Chalcopyrite	0.6	0.6
2nd Cu Phases	0.3	0.4
Pyrite	13.3	30.9
Fe Oxides	4.9	5.7
Quartz	34.6	17.8
Chlorite Group	32.5	36.7
Carbonates	0.5	0.5
Accessory Phases	2.2	0.7
Kaolinite_Illite	1.8	1.2
Mica Group	7.2	2.5
Chalcostibite	0.0	0.1
Hg bearing	0.1	0.0
Pb bearing	0.0	0.1

13.3.3 COMMINUTION CHARACTERISATION

Comminution testing was carried out at GSL and consisted of JK Dropweight, SMC, SMC-SCSE, Bond Abrasion, French Crushability, French Abrasion, BRWi and BBWi measurements on composite samples. Table 13.16 presents a summary of the results of these tests.

STW zone samples are, on average, harder than those from the rest of domains. Results have shown average BRWI and BBWI values of 16.4 kWh/t and 17.7 kWh/t, respectively.

Table 13.16 – Summary of comminution testing results.

Comminution Test	PMS	STW	GOS	TMS	STR
JK Dropweight (kWh/t)	5.96	7.82	-	-	-
SMC (kWh/m3)	4.7	4.1	2.7	5.4	3.4
SMC - SCSE (kWh/t)	6.35	7.01	6.81	7.26	7.01
Bond Abrasion	0.72	0.15	-	-	-
French Crushability	51%	46%	66%	56%	58%
French Abrasion (g/t)	1,640	480	1,140	2,080	500
BRWi (kWh/t)	12.9	16.4	12.3	Not run	11.4
BBWi (kWh/t)	12.6	17.7	13.1	12.8	13.2

13.3.4 FLOTATION TESTING

The flotation programme consisted of open circuit rougher and cleaner tests for the 5 different domains (PMS, STW, GOS, TMS, STR), with locked cycle completed for PMS, STW and STR used toward the end of the programme to generate recovery predictions. Variability flotation tests were completed on PMS, STW and STR samples.

The flotation laboratory programme consisted of more than 100 laboratory scale flotation tests (excluding variability testwork), distributed as shown below:

- PMS: 17 rougher tests, 15 roughers + cleaner tests, and 5 locked cycle tests.
- STW: 23 rougher tests, 8 roughers + cleaner tests, and 5 locked cycle tests.
- GOS: 2 rougher tests.
- TMS: 13 rougher tests, and 4 roughers + cleaner tests.
- STR: 13 rougher tests, 5 roughers + cleaner tests, and 2 locked cycle tests.

The key finding for the different domains is explained in the following chapters.

13.3.4.1 PRIMARY MASSIVE SULPHIDE (PMS)

A total of 17 rougher tests were conducted to float separate Pb and Zn concentrates. Tests FT7, FT13, FT14 and FT15 achieved the best performance with accumulated recoveries of 70.9% Pb to a grade of 14.6% and 79.7% Zn to a grade of 15.3%. Recovery/grade curves shown in Figure 13.1 and Figure 13.2.

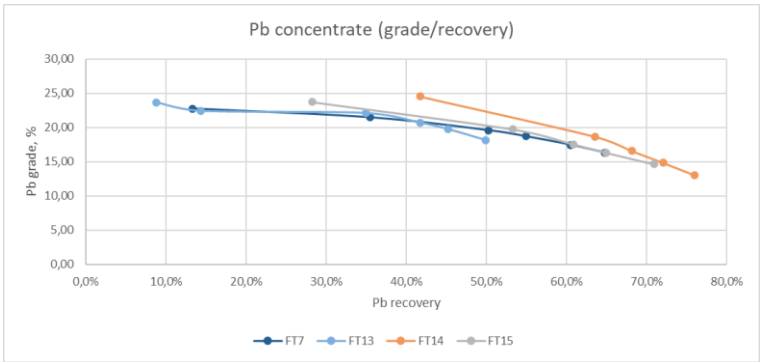


Figure 13.1 – Pb concentrate in rougher flotation tests for PMS composite sample.

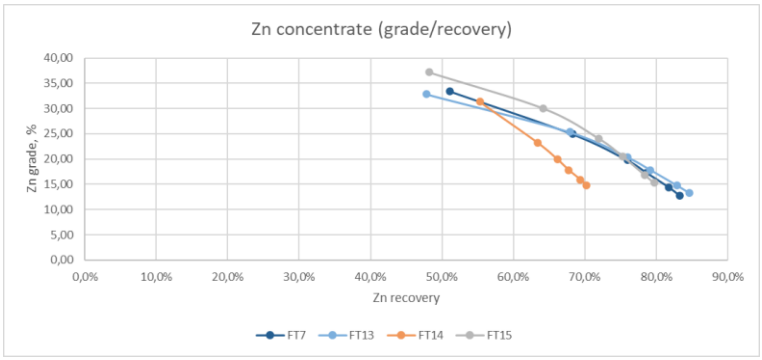


Figure 13.2 – Zn concentrate in rougher flotation tests for PMS composite sample.

A total of 15 open circuit tests (rougher and cleaner) were conducted to float separate Pb and Zn concentrates. Tests OCT1, OCT3, OCT10 and OCT11 performed the best, with the highest combined results showing a Pb recovery of 30.5% to a concentrate of 43.2% in the 5th concentrate, and a Zn recovery of 63.4% to a concentrate of 39.3%, also in the 5th concentrate. Recovery/grade curves shown in Figure 13.3 and Figure 13.4.

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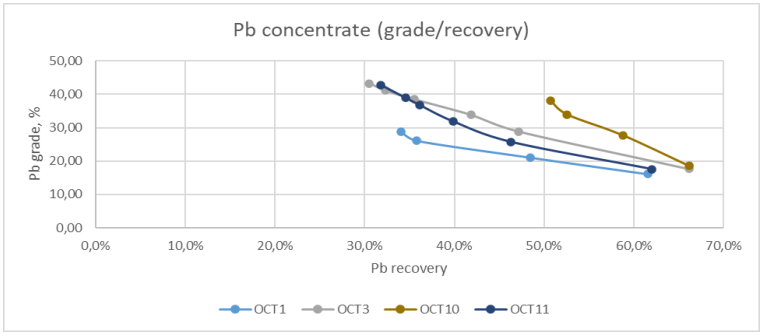


Figure 13.3 – Pb concentrate in cleaner flotation tests for PMS composite sample.

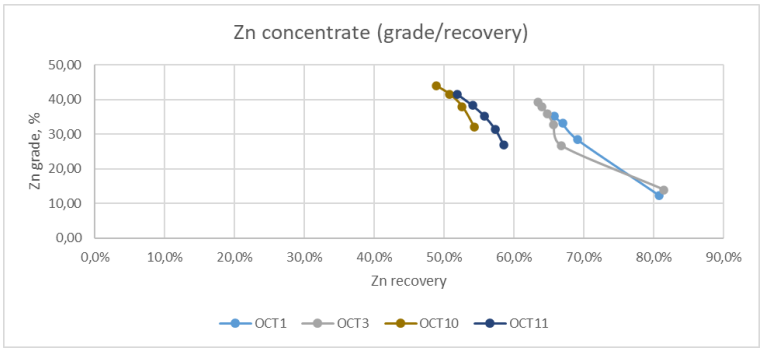


Figure 13.4 – Zn concentrate in cleaner flotation tests for PMS composite sample.

After the completion of open circuit testwork, 5 locked cycle tests were performed. The best results were achieved following the flowsheet illustrated in Figure 13.5.

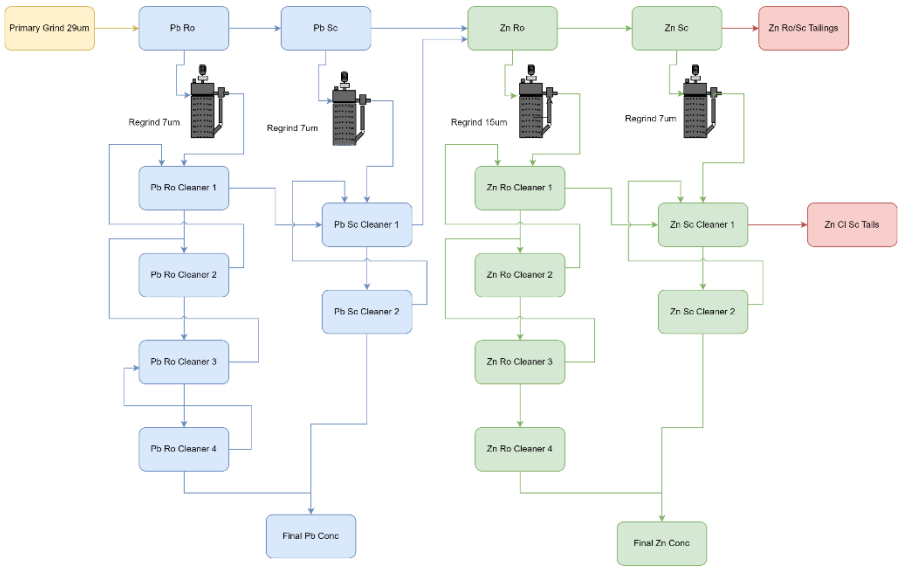


Figure 13.5 – PMS flotation locked cycle test flowsheet.

The results of the best performing locked cycle tests are summarized in Table 13.17 and Table 13.18.

Table 13.17 – LCT results for PMS composite sample (concentrate grades).

	Assay Calibrated XRF %					
Product	**% Cu**	**% Pb**	**% Zn**	**% Fe**	**ppm Au**	**ppm Ag**
Pb Cl4 Con	0.87	36.52	3.14	23.34	0.80	260.0
Pb Sc Cl2 Con	0.49	9.30	4.50	35.09	0.77	124.3
Zn Ro Cl4 Con	0.36	3.69	36.35	17.35	0.71	82.7
Zn Sc Cl2 Con	0.51	3.98	11.36	32.94	1.07	113.3
Zn Sc Cl1 Tails	0.25	1.63	1.92	39.68	0.93	61.7
Zn Ro/Sc Tails	0.13	0.95	0.37	41.86	0.55	45.7
Total	0.23	3.66	3.91	38.21	0.64	68.8

Table 13.18 – LCT results for PMS composite sample (concentrate recoveries).

	Recovery					
Product	**% Cu**	**% Pb**	**% Zn**	**% Fe**	**% Au**	**% Ag**
Pb Cl4 Con	20.7	54.5	4.4	3.3	6.8	20.6
Pb Sc Cl2 Con	8.5	10.1	4.6	3.6	4.8	7.2
Zn Ro Cl4 Con	10.0	6.6	60.5	3.0	7.2	7.8
Zn Sc Cl2 Con	15.7	7.7	20.6	6.1	11.8	11.7
Zn Sc Cl1 Tails	7.0	2.9	3.2	6.9	9.6	5.9
Zn Ro/Sc Tails	38.1	18.2	6.7	77.1	59.8	46.8
Total	100	100	100	100	100	100

Duplicate LCT was performed in Maelgwyn Mineral Services Africa with good reproducibility achieved for both Pb and Zn concentrates. Results shown in Table 13.19 and Table 13.20.

Table 13.19 – LCT1 duplicate for PMS composite sample.

	Grades		Recoveries	
Product	**% Pb**	**% Zn**	**% Pb**	**% Zn**
Pb Cl4 Con	37.2	3.1	57.7	4.3
Pb Sc Cl2 Con	11.8	3.8	12.4	3.6

ZnRo Cl4 Con	2.4	49.9	2.6	48.9
Zn Sc Cl2 Con	3.1	15.1	2.8	12.2
ZnSc Cl1 Tails	1.7	3.0	10.8	17.5
Zn Ro/Sc Tails	1.0	1.1	13.7	13.6
Total	4.04	4.50	100	100

Table 13.20 – LCT2 duplicate for PMS composite sample.

	Grades		Recoveries	
Product	**% Pb**	**% Zn**	**% Pb**	**% Zn**
Pb Cl4 Con	40.9	2.2	48.2	2.5
Pb Sc Cl2 Con	10.4	3.4	18.9	6.2
ZnRo Cl4 Con	2.8	46.5	3.7	60.2
Zn Sc Cl2 Con	3.9	13.6	5.7	19.9
ZnSc Cl1 Tails	2.0	1.7	10.2	8.5
Zn Ro/Sc Tails	1.3	0.3	13.3	2.7
Total	4.80	4.87	100	100

13.3.4.2 STOCKWORK (STW)

A total of 23 rougher tests were conducted to float separate Cu, Pb and Zn concentrates. Tests FT7, FT20, FT21 achieved the best performance accumulated recoveries of 66% Cu to a grade of 10.5%, 54% Pb to a grade of 23.8% and 74% Zn to a grade of 12.5%. Recovery/grade curves shown in Figure 13.6, Figure 13.7 and Figure 13.8.

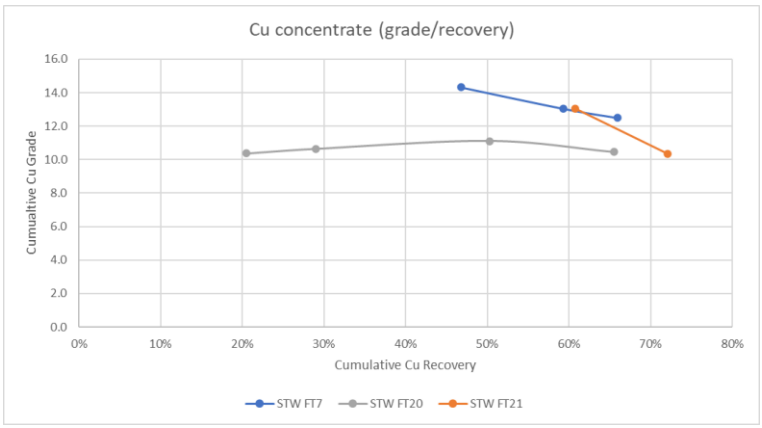


Figure 13.6 – Cu concentrate in rougher flotation tests for STW composite sample.

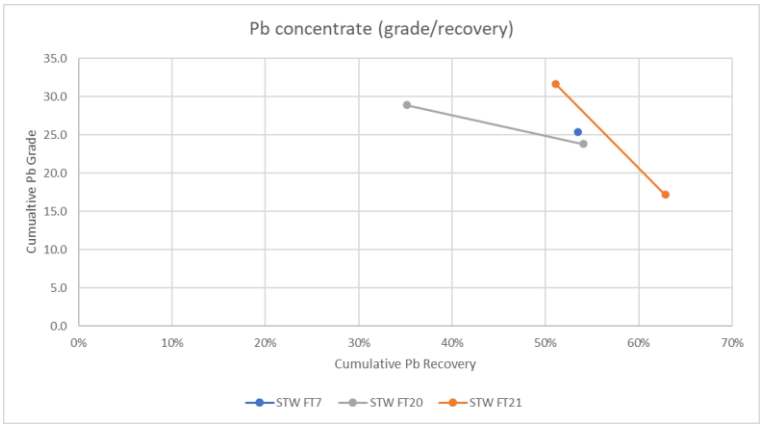


Figure 13.7 – Pb concentrate in rougher flotation tests for STW composite sample.

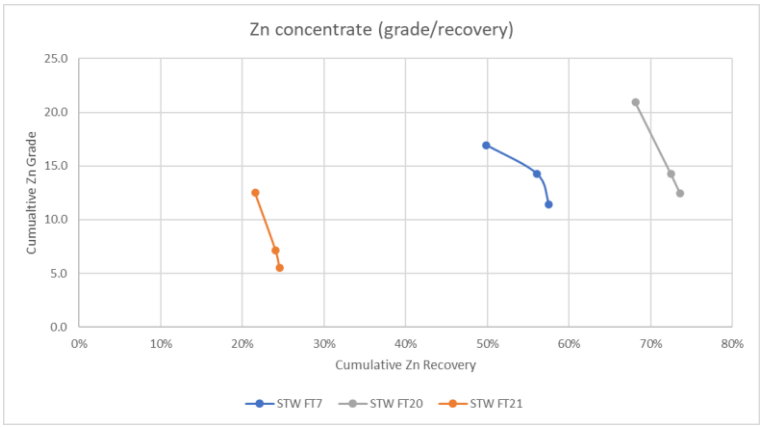


Figure 13.8 – Zn concentrate in rougher flotation tests for STW composite sample.

A total of 8 open circuit tests (rougher and cleaner) were conducted to float separate Cu, Pb and Zn concentrates. Tests OCT4, OCT7, OCT8 achieved the best performance showing a Cu recovery of 51.5% to a concentrate of 28.52% in the 1st concentrate, a Pb recovery of 40.7% to a concentrate of 61.3% in the 3rd concentrate and a Zn recovery of 65.6% to a concentrate of 42.4%, also in the 3rd concentrate. Recovery/grade curves shown in Figure 13.9, Figure 13.10 and Figure 13.11.

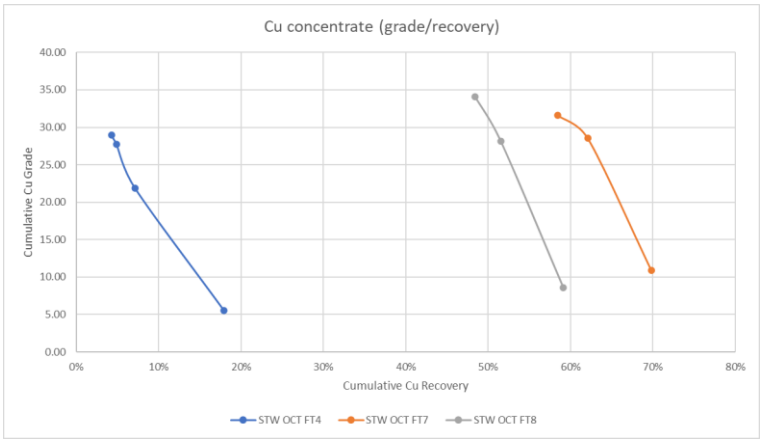


Figure 13.9 – Cu concentrate in cleaner flotation tests for STW composite sample.

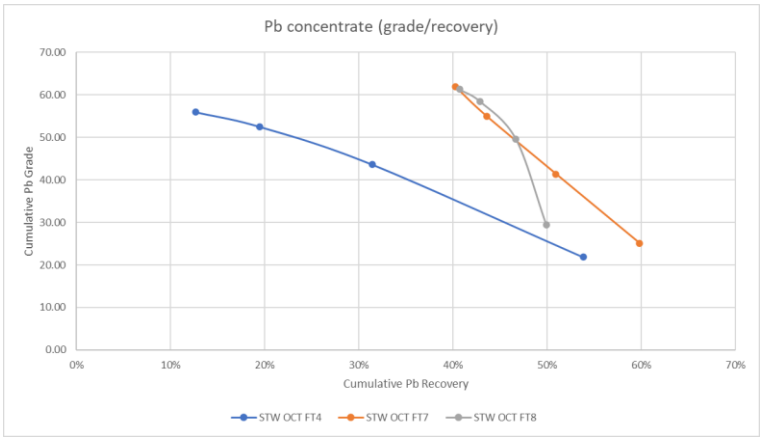


Figure 13.10 – Pb concentrate in cleaner flotation tests for STW composite sample.

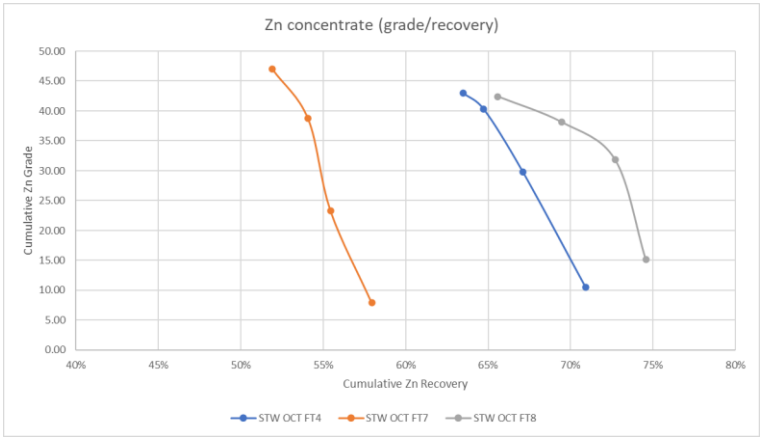


Figure 13.11 – Zn concentrate in cleaner flotation tests for STW composite sample.

After the completion of open circuit testwork, 5 locked cycle tests were performed. The best results were achieved following the flowsheet illustrated in Figure 13.12.

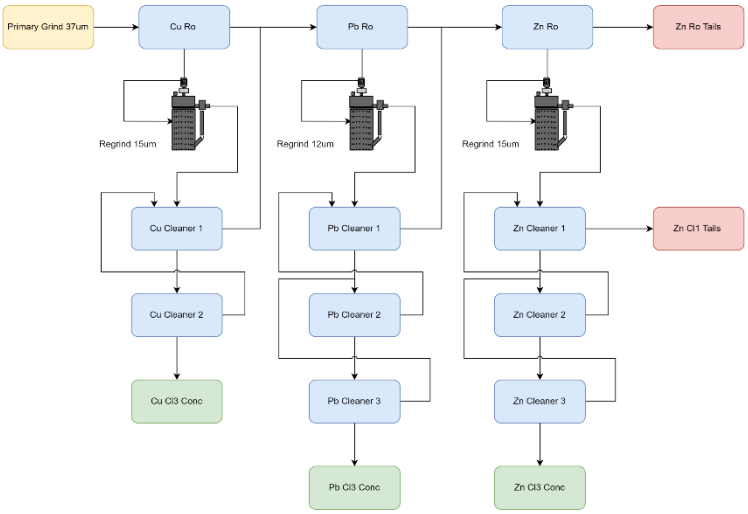


Figure 13.12 – STW flotation locked cycle test flowsheet

The results of the best performing locked cycle tests are summarised in Table 13.21 and Table 13.22.

Table 13.21 – LCT results for STW composite sample (concentrate grades).

	Assay Calibrated XRF %					
Product	**% Cu**	**% Pb**	**% Zn**	**% Fe**	**ppm Au**	**ppm Ag**
Cu Cl2 Con	28.9	8.1	15.2	16.3	0.3	395.5
Pb Cl3 Con	6.0	55.3	10.7	6.2	0.2	577.5
Zn Cl3 Con	2.7	7.0	49.5	4.4	0.2	157.2
Zn Cl1 Tails	0.3	1.0	1.0	15.5	0.1	20.8
Zn Ro Tails	0.1	0.2	0.3	13.2	0.1	2.4
Total	0.53	1.43	1.93	12.99	0.07	21.37

Table 13.22 – LCT results for STW composite sample (concentrate recoveries).

	Recovery					
Product	**% Cu**	**% Pb**	**% Zn**	**% Fe**	**ppm Au**	**ppm Ag**
Cu Cl2 Con	51.2	5.3	7.4	6.4	4.2	17.4
Pb Cl3 Con	19.9	68.3	9.8	8.4	4.7	47.9

Zn Cl3 Con	12.9	12.3	64.5	17.7	5.8	18.5
Zn Cl1 Tails	3.8	4.6	3.2	9.0	13.1	6.2
Zn Ro Tails	12.1	9.4	15.1	58.4	72.2	9.9
Total	100	100	100	100	100	100

13.3.4.3 GOSSAN (GOS)

Two rougher tests were conducted to evaluate the potential of preconcentrating Pb and Zn minerals and see the deportment of precious elements, as well as to try and concentrate the tin bearing minerals.

Concentration was unsuccessful with recoveries below 10% for Pb and Zn, a very poor upgrade to a sulphide concentrate. Tin flotation of sulphide tailings were also ineffective with no discernible concentration.

Leaching and gravity works were performed using this sample; the results are shown in the corresponding sections.

13.3.4.4 TRANSITION MASSIVE SULPHIDE (TMS)

A total of 13 rougher tests were conducted to float bulk a Cu/Pb concentrate. Tests FT5, FT6, FT7, FT8 and FT11 achieved the best performance with the highest combined results showing accumulated recoveries over 80% but poor upgrade ratios, resulting in 2.2% Cu and 4.5% Pb concentrate grades as shown in Figure 13.13 and Figure 13.14. Au and Ag have shown good concentration ratios compared to all rougher concentrates. Potential for leaching will be evaluated in the corresponding chapter.

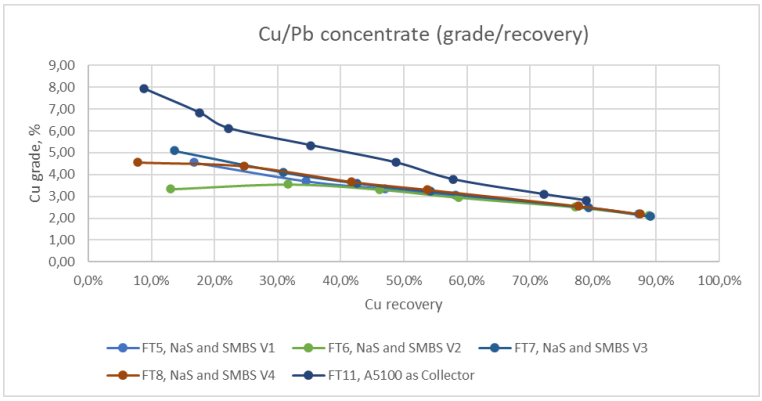


Figure 13.13 – Cu in Cu/Pb concentrate in rougher flotation tests for TMS composite sample.

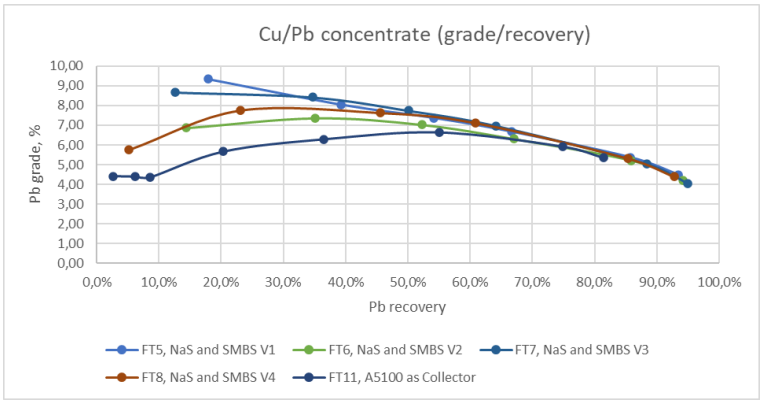


Figure 13.14 – Pb in Cu/Pb concentrate in rougher flotation tests for TMS composite sample.

3 open circuit tests (rougher and cleaner) were conducted to improve the concentrate upgrade ratios. Tests OCT2 and OCT3 achieved the best performance with the highest combined results showing recoveries of 13.0% for Cu and 19.4% for Pb to a concentrate graded 6.17% Cu and 17,7% Pb. Recovery/grade curves shown in Figure 13.15 and Figure 13.16.

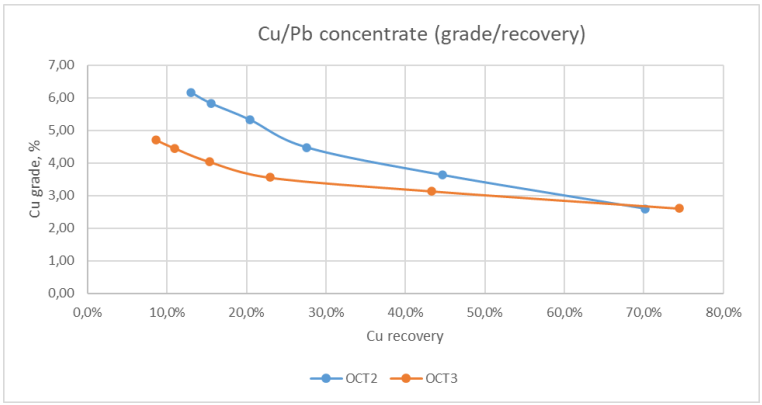


Figure 13.15 – Cu in Cu/Pb concentrate in cleaner flotation tests for TMS composite sample.

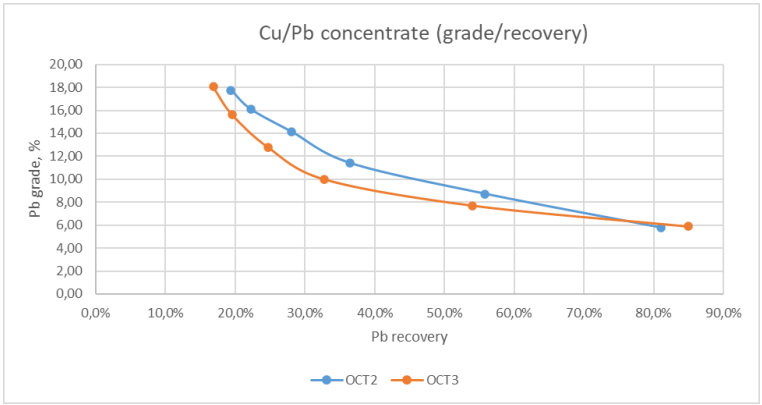


Figure 13.16 – Pb in Cu/Pb concentrate in cleaner flotation tests for TMS composite sample.

Flotation testwork has demonstrated a good potential to produce a bulk Cu/Pb concentrate, however this domain will require more work to improve both Cu and Pb grades before completing locked cycle tests.

13.3.4.5 STRINGER (STR)

A total of 13 rougher tests were conducted to float separate Cu and Zn concentrates. Tests FT11, FT12, and FT13 achieved the best performance with the highest combined results showing accumulated recoveries of 67.4% Cu to a grade of 5.9% and 54.9% Zn to a grade of 3.3%. Recovery/grade curves shown in Figure 13.17 and Figure 13.18.

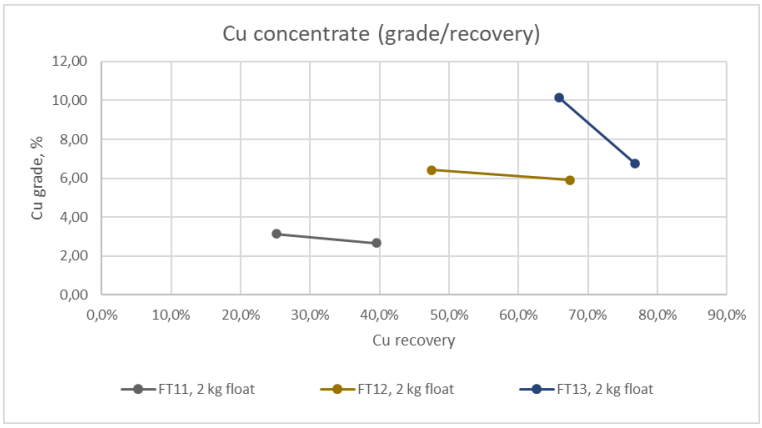


Figure 13.17 – Cu concentrate in rougher flotation tests for STR composite sample.

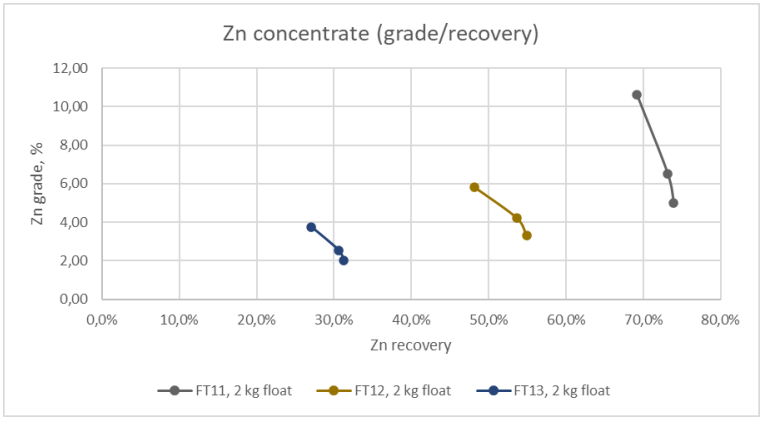


Figure 13.18 – Zn concentrate in rougher flotation tests for STR composite sample.

A total of 5 open circuit tests (rougher and cleaner) were conducted to float separate Cu and Zn concentrates. Tests OCT4 and OCT5 achieved the best performance with the highest combined results showing a Cu recovery of 31.3% to a concentrate of 24.7% in the 3rd concentrate and a Zn recovery of 53.2% to a concentrate of 27.9% in the 5th concentrate. Recovery/grade curves shown in Figure 13.19 and Figure 13.20.

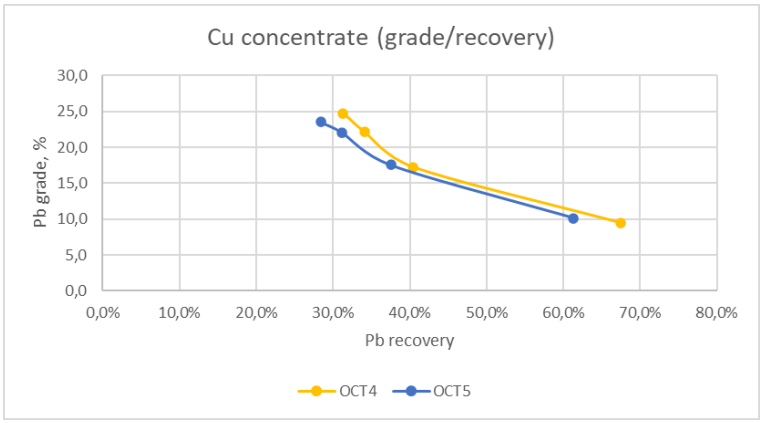


Figure 13.19 – Cu concentrate in cleaner flotation tests for STR composite sample.

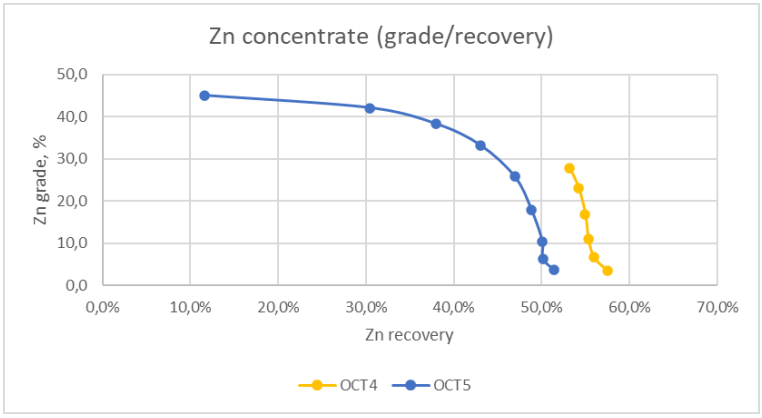


Figure 13.20 – Zn concentrate in cleaner flotation tests for STR composite sample.

After the completion of open circuit testwork, 2 locked cycle tests were performed. The best results were achieved following the flowsheet illustrated in Figure 13.21.

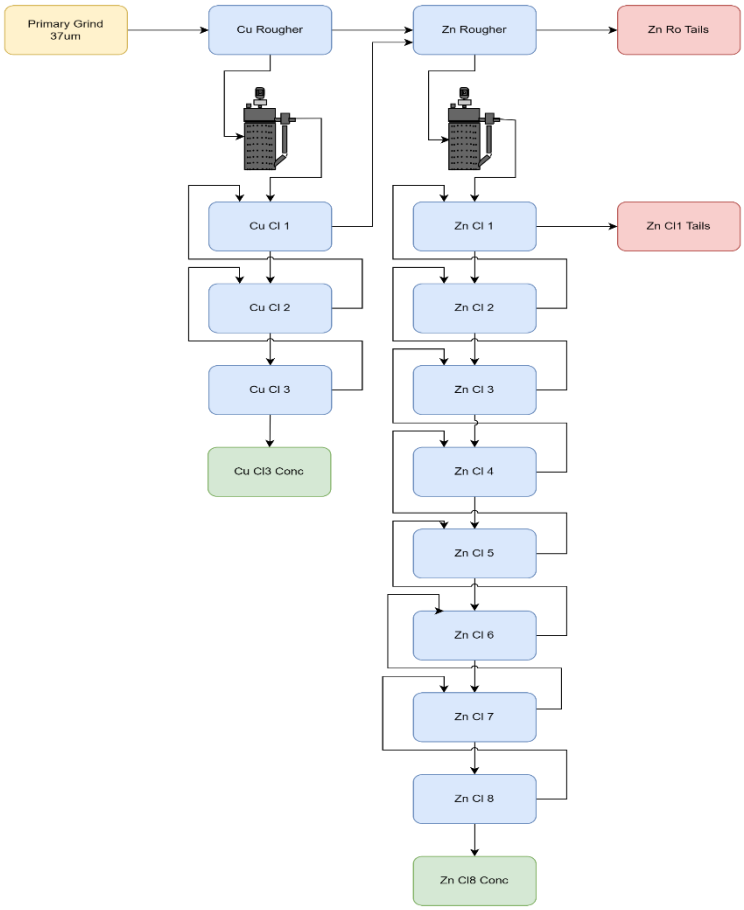


Figure 13.21 – STR flotation locked cycle test flowsheet.

The results of the best performing locked cycle test are summarized in Table 13.23 and Table 13.24.

Table 13.23 – LCT results for STR composite sample (concentrate grades).

	Assay Calibrated XRF %					
Product	**% Cu**	**% Pb**	**% Zn**	**% Fe**	**ppm Au**	**ppm Ag**
Cu Cl3 Con	28.4	3.4	9.9	15.0	0.4	685.5
Zn Cl8 Con	9.5	12.0	31.3	7.9	0.2	389.5
Zn Cl1 Tails	0.3	0.3	0.5	17.5	0.1	18.9
Zn Ro Tails	0.1	0.1	0.4	11.4	0.1	4.5
Total	0.31	0.30	0.81	11.71	0.10	13.18

Table 13.24 – LCT results for STR composite sample (concentrate recoveries).

	Recovery					
Product	**% Cu**	**% Pb**	**% Zn**	**% Fe**	**ppm Au**	**ppm Ag**
Cu Cl3 Con	36.2	4.6	4.9	0.5	1.6	20.8
Zn Cl8 Con	39.8	52.7	50.9	0.9	2.3	39.0
Zn Cl1 Tails	5.8	5.9	3.5	9.3	7.1	8.9
Zn Ro Tails	18.2	36.8	40.8	89.3	89.0	31.2
Total	100	100	100	100	100	100

13.3.4.6 CONCENTRATE QUALITY

Minor element ICP-OES scans were completed on the final copper and zinc concentrates produced from the locked cycle tests (PMS, STW and STR domains). Table 13.25,

Table 13.26 and

Table 13.27 summarises the various assays.

Table 13.25 – PMS concentrates detailed analysis.

Product	**Pb Cl4 Con**	**Pb Sc Cl2 Con**	**Zn Cl4 Con**	**Zn Sc Cl2 Con**
% Cu	0.9	0.5	0.4	0.5
% Pb	36.5	9.3	3.7	4.0
% Zn	3.1	4.5	36.4	11.4
% Fe	23.3	35.1	17.3	32.9
% S	34.2	44.6	37.8	42.9
Au (mg/kg)	0.8	0.8	0.7	1.1
Ag (mg/kg)	260.1	124.3	82.7	113.3
% As	1.7	1.9	1.6	2.4
% Sb	0.2	0.1	0.1	0.1
% Hg	0.1	0.1	0.3	0.1
% Bi	0.0	0.0	0.0	0.0
% Sn	0.1	0.1	0.1	0.1

Table 13.26 – STW concentrates detailed analysis.

Product	**Cu Cl2 Con**	**Pb Cl3 Con**	**Zn Cl3 Con**
% Cu	28.9	6.0	2.7

% Pb	8.1	55.3	7.0
% Zn	15.2	10.7	49.5
% Fe	16.3	6.2	4.4
% S	29.1	20.3	30.1
Au (mg/kg)	0.3	0.2	0.2
Ag (mg/kg)	395.5	577.5	157.2
% As	0.1	0.0	0.0
% Sb	0.0	0.0	0.0
% Hg	0.0	0.0	0.0
% Bi	0.0	0.1	0.0

Table 13.27 – STR concentrates detailed analysis.

Product	Cu Cl3 Con	Zn Cl8 Con
% Cu	28.4	9.5
% Pb	3.4	12.0
% Zn	9.9	31.3
% Fe	15.0	7.9
% S	33.5	30.2
Au (mg/kg)	0.4	0.2
Ag (mg/kg)	685.5	389.5
% As	3.4	2.1
% Sb	1.4	0.7
% Hg	0.0	0.2
% Bi	0.0	0.0

Arsenic content is high for both PMS Pb and Zn concentrates as well as STR Cu and Zn concentrates. The same goes for the Antimony and Mercury content in the PMS Pb and Zn concentrates as well as the STR Zn concentrate.

Pb + Zn content in both STW and STR concentrates is high and would require further investigation to ascertain the possibility of improving the selectivity on final concentrates.

Fe content in the STW concentrate, as well as rest of deleterious elements into this domain, are normally below the penalty limits. The good quality of the three Cu, Pb and Zn STW concentrates would make the products suitable for blending to soften the final deductions in the rest of the domains.

Silver content would represent a good credit for all the PMS, STW and STR concentrates produced. Gold is below the payable limits for all cases.

13.3.5 LEACHING TESTING

13.3.5.1 GOSSAN (GOS)

One single GOS bulk composite sample was tested for Au and Ag extraction using conventional cyanidation processes at different grind sizes (5 mm and 75 microns).

Table 13.28 and

Table 13.29 show the resulting solutions assay and cumulative distribution for different extraction time for the best performing grind size (75 microns).

Table 13.28 – 75 microns GOS sample resulting solutions for cyanide leaching.

Hours	ppm Cu	ppm Pb	ppm Zn	ppm Fe	ppm S	ppm Sn	ppm Au	ppm Ag
2	49.3	0.1	19.1	60.2	44.6	0.0	0.2	7.5
4	47.9	0.1	23.4	57.5	46.8	0.0	0.3	11.8
6	48.3	0	25.1	57.5	47.9	0.0	0.3	12.6
24	48.7	0.1	31.2	59.4	51.3	0.0	0.3	13.6
48	48.8	0	33.5	60	53.5	0.0	0.3	13.8
Residue	580	18,500	5,000	219,800	14,500	2,300	0.11	6.4

Table 13.29 – 75 microns GOS sample solutions cumulative distribution for cyanide leaching.

Hours	**% Cu**	**% Pb**	**% Zn**	**% Fe**	**% S**	**% Sn**	**% Au**	**% Ag**
2	23.3	0.0	1.3	0.1	1.1	0.0	64.4	47.2
4	23.0	0.0	1.7	0.1	1.2	0.0	85.2	75.8
6	23.4	0.0	1.8	0.1	1.2	0.0	88.7	81.3
24	23.5	0.0	2.2	0.1	1.3	0.0	91.2	87.3
48	23.6	0.0	2.4	0.1	1.3	0.0	91.4	88.7
Residue	76.4	100.0	97.6	99.9	98.7	100.0	8.6	11.3

According to the GOS sample tested, 91.4% Au and 88.7% Ag extraction can be achieved with a grind size of 75 microns and 48 hours of extraction. The NaCN addition was 2.6 kg/t with a consumption of 0.9 kg/t.

13.3.5.2 TRANSITION MASSIVE SULPHIDE (TMS)

Four feed samples and one rougher concentrate produced in rougher flotation test FT5 were compared to a conventional cyanide leaching process to evaluate the potential of Au

and Ag extraction. With 48 hours of testing and different grind sizes, a maximum Au extraction of 60.2% at 20 microns was achieved with erratic performance for Ag. Cyanide consumption was in the range of 25 kg/t for all tests. Results summarised in Table 13.30.

Table 13.30 – FT5 TMS rougher concentrate cyanide leaching for 48 hours and different grind sizes.

Test	Microns	Sample	Cu	Pb	Zn	Fe	S	Sn	Au	Ag
CN 1	100	Feed	65.4	2.6	0.4	0.0	1.0	0.0	48.5	47.0
CN 2	80	Feed	62.0	2.4	0.2	0.0	1.2	0.0	51.3	43.4
CN 3	60	Feed	62.7	1.4	0.5	0.0	1.3	0.0	54.2	55.5
CN 4	40	Feed	61.8	1.2	0.5	0.0	1.4	0.0	53.6	57.2
CN 5	20	Feed	60.1	1.2	0.6	0.0	1.5	0.0	57.0	60.8
CN 6	29	FT5 Ro Con	62.8	0.3	0.0	0.0	1.2	0.0	60.2	0.7
CN7	10	Feed	58.1	-	-	-	-	-	53.1	0.6

13.3.6 GRAVITY TESTING

13.3.6.1 PRIMARY MASSIVE SULPHIDE (PMS)

PMS flotation tailings were compared to a conventional Sn recovery circuit, starting with a bulk sulphide flotation for the removal of pyrite and remaining sulphide minerals, and gravity on bulk sulphide flotation tailings. Table 13.31 shows the gravity release product assay and distribution where interesting Sn grades and recoveries were achieved in concentrate number 3. The test shows good potential for Sn concentrate production in the PMS domain. However, more work is required to establish the final performance of the process.

Table 13.31 – PMS gravity release testwork with sulphide flotation tailings on primary flotation tailings.

Stream	Mass (%)	Sn Grade	Sn Rec
Sn Con 1	1	1.5	0%
Sn Con 2	3	10.9	11%
Sn Con 3	2	35.2	26%
Sn Con 4	1	11.0	2%
Sn Con 5	1	20.0	4%
Sn Tails	93	1.7	56%

13.3.6.2 GOSSAN (GOS)

Four Falcon tests at 185, 114, 75 and 50 microns grind sizes, as well as two shaking tables were performed with GOS composite samples. Pb, Sn and Ag have shown promising concentrations in all testwork. Table 13.32 and

Table 13.33 show the product assay and distribution for the best performing test (Falcon at 53 microns grind size), where combined concentrates of 18.8% Pb, 0.04% Ag and 1.92% Sn, as well as respective recoveries of 53.8% Pb, >99% Ag and 24.6% Sn were achieved.

Table 13.32 – GOS Falcon product assays at 53 microns grind size.

Product	% Cu	% Pb	% Zn	% Fe	% Ag	% Sn
Con 1	0.4	32.2	2.1	6.9	1.3	4.5
Con 2	0.8	38.2	5.4	6.0	0.1	4.1
Con 3	1.0	33.0	5.9	20.6	0.0	2.7
Con 4	0.6	11.2	2.5	54.6	0.0	1.2
Con 5	0.2	4.4	1.1	61.5	0.0	0.8
Tails	0.0	1.5	0.5	34.1	0.0	0.4
Head	0.1	2.5	0.7	34.4	0.0	0.4

Table 13.33 – GOS Falcon product distribution at 53 microns grind size.

Product	% Cu	% Pb	% Zn	% Fe	% Ag	% Sn
Con 1	0.8	1.5	0.4	0.0	64.4	1.2
Con 2	9.6	12.6	6.4	0.1	20.5	7.5
Con 3	23.1	19.1	12.3	0.9	15.1	8.6
Con 4	13.2	6.4	5.1	2.3	0.0	4.0
Con 5	7.0	3.3	3.1	3.4	0.0	3.3
Tails	46.2	57.1	72.8	93.3	0.0	75.4
Head	100.0	100.0	100.0	100.0	100.0	100.0

The tests completed show good potential for Pb/Sn/Ag concentrate production in the GOS domain. However, more work is required to establish the final performance of the process.

13.3.7 VARIABILITY TESTING

13.3.7.1 PRIMARY MASSIVE SULPHIDE (PMS)

Fifteen PMS variability samples representing different zones of the deposit were produced for comminution and flotation testing. Comminution was performed on 14 variability

samples using the Geopyroa, French Crushability and Abrasiveness tests, the results of which are summarised in Table 13.34.

Table 13.34 – Summary of Geopyroa, French Crushability and Abrasiveness testing results on PMS variability samples.

Sample	Solid Density (S.G)	Crushability	Abrasiveness	DWI (kWh/m3)	BWI (kWh/t)	PLT Is (MPa)	UCS (MPa)
PMS_2	4.83	40	1760	8.2	12.6	20.9	418.2
PMS_3	3.09	78	680	1.3	9.6	3.8	75.5
PMS_4	4.91	43	1940	6.7	13.5	16.4	327.7
PMS_5	4.9	42	1920	6.5	13	22.8	456.1
PMS_6	4.77	44	1560	6.3	12.3	16.4	327
PMS_7	4.8	41	2060	6.9	13.4	24.8	496
PMS_8	4.65	38	2000	9.8	14.7	24	479
PMS_9	4.89	44	1800	5.9	12.5	23.9	478.4
PMS_10	4.8	44	1720	6.7	13	23.6	471.6
PMS_11	4.37	61	1160	2.0	8.9	10.6	211.4
PMS_12	4.8	43	1740	5.7	11.3	16.7	333.7
PMS_13	4.75	48	1580	6.1	11.7	18	359.1
PMS_14	4.92	33	2320	10.2	15	23.4	467
PMS_15	4.85	42	1580	6.3	11.4	23	460.9

Pb and Zn flotation testwork, using rougher + one cleaner open circuit configuration, was tested. Head grade vs. recovery to concentrates is shown in Figure 13.22 and Figure 13.23.

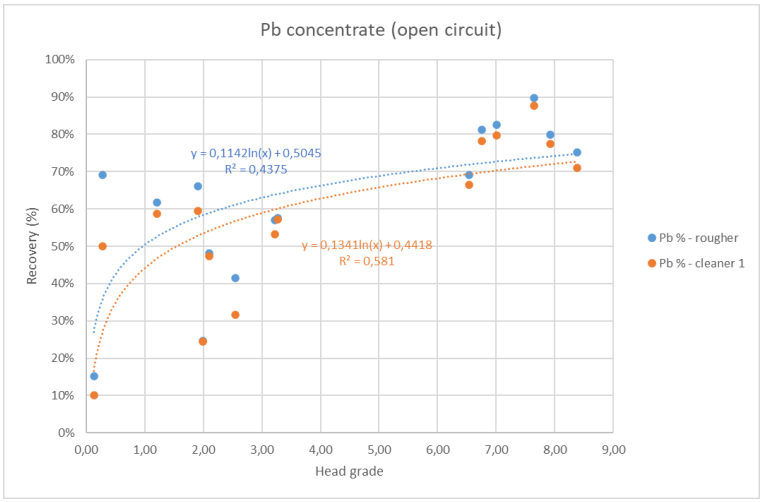


Figure 13.22 – Pb head grade vs. Pb concentrate recovery for the PMS variability samples in open circuit (rougher + 1 cleaner) flotation testwork.

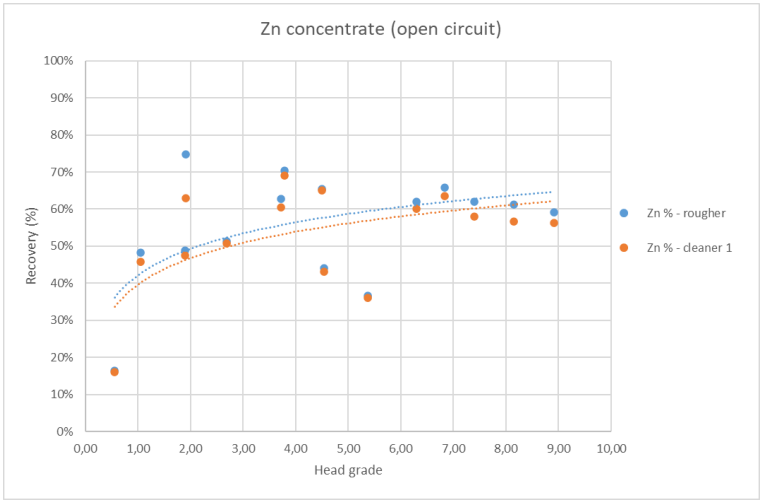


Figure 13.23 – Zn head grade vs. Zn concentrate recovery for the PMS variability samples in open circuit (rougher + 1 cleaner) flotation testwork.

Head grade vs. concentrate grades is shown in Figure 13.24 and Figure 13.25.

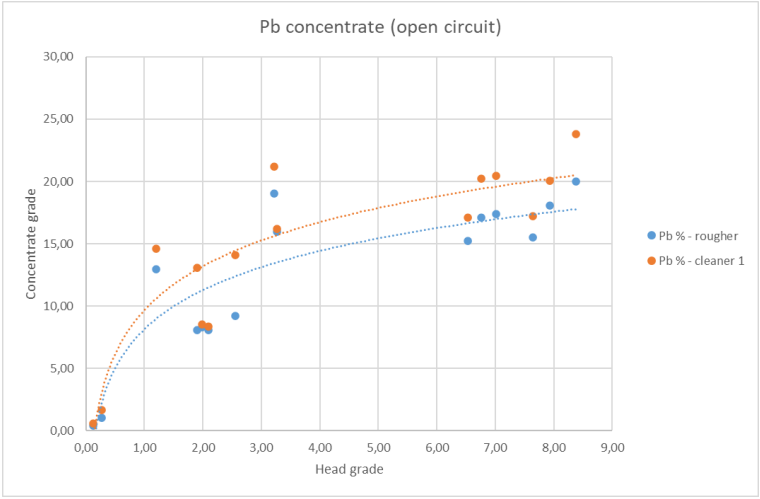


Figure 13.24 – Pb head grade vs. Pb concentrate grade for the PMS variability samples in open circuit (rougher + 1 cleaner) flotation testwork.

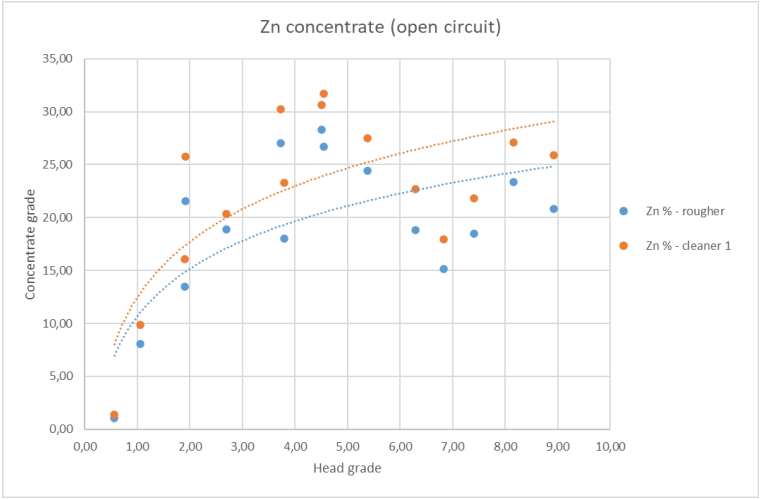


Figure 13.25 – Zn head grade vs. Zn concentrate grade for the PMS variability samples in open circuit (rougher + 1 cleaner) flotation testwork.

13.3.7.2 STOCKWORK (STW)

Twenty STW variability samples representing different zones of the deposit were produced for comminution and flotation testing. Comminution was performed on 17 variability samples using the Geopyroa , French Crushability and Abrasiveness tests, the results of which are summarised in Table 13.35.

Table 13.35 – Summary of Geopyroa, French Crushability and Abrasiveness testing results on STW variability samples.

Sample	Solid Density (S.G)	Crushability	Abrasiveness	DWI (kWh/m3)	BWI (kWh/t)	PLT Is (MPa)	UCS (MPa)
STW_2	3.23	55	1200	1.1	8.7	6.1	121.3
STW_3	3.34	41	280	4.3	12.5	14	279.1
STW_4	3.02	36	240	5.6	15.1	19.5	389.8
STW_5	2.72	43	400	3.3	12.2	11	220.4
STW_6	2.82	41	880	5.2	15.2	13.6	272.8
STW_7	2.75	38	500	4.2	13.3	17.4	348.1
STW_8	2.78	36	520	5.3	15.6	19.9	397.2
STW_9	3.61	41	1060	3.4	10.6	14.5	289.4
STW_10	3.01	46	840	3.0	11.1	12.2	244
STW_11	2.99	36	240	5.1	14	15.3	306
STW_12	2.76	41	520	3.5	11.8	14.2	284
STW_13	2.96	37	380	4.6	13.8	16.4	329.1
STW_15	2.77	39	380	6.0	15.2	24.6	492.8
STW_17	2.84	37	340	4.3	13.2	15	300
STW_18	2.52	43	400	2.9	10.8	16.4	327.7
STW_19	3.39	36	200	4.6	12.6	17	340.8

Cu, Pb and Zn flotation testwork, using rougher + one cleaner open circuit configuration, was tested. Head grade vs. recovery to concentrates is shown in

Figure 13.26, Figure 13.27 and Figure 13.28.

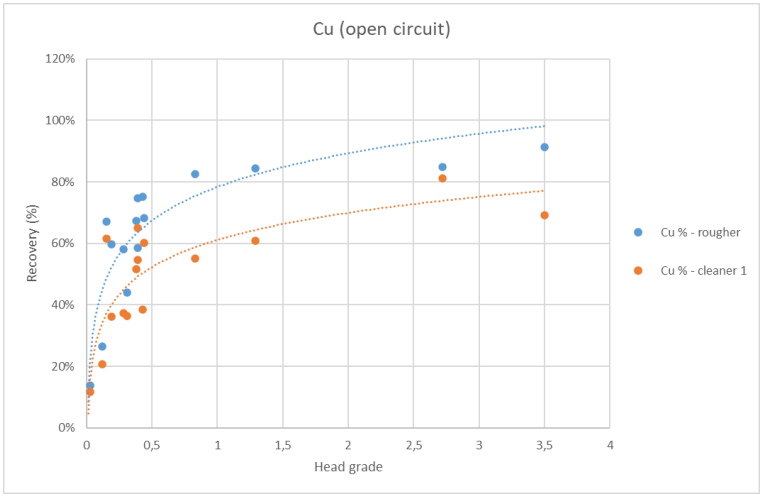


Figure 13.26. – Cu head grade vs. Cu concentrate recovery for the STW variability samples in open circuit (rougher + 1 cleaner) flotation testwork

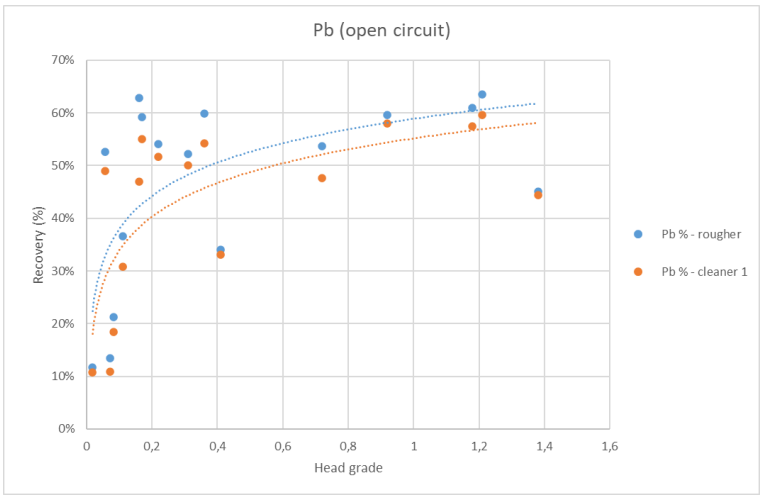


Figure 13.27. – Pb head grade vs. Pb concentrate recovery for the STW variability samples in open circuit (rougher + 1 cleaner) flotation testwork

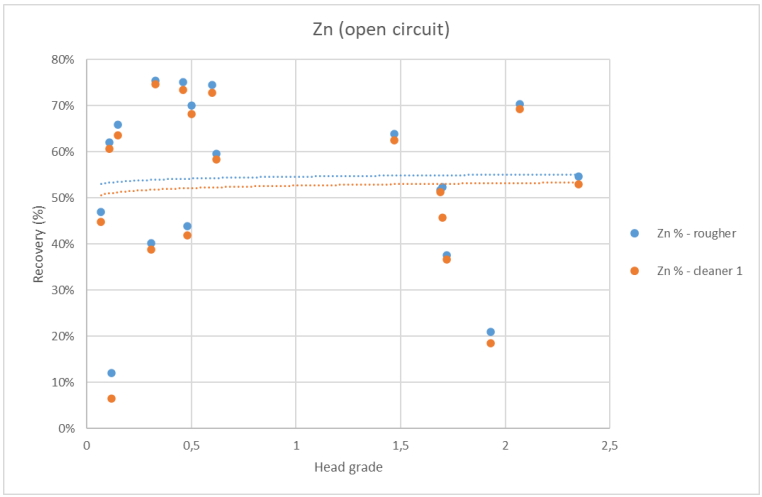


Figure 13.28. – Zn head grade vs. Zn concentrate recovery for the STW variability samples in open circuit (rougher + 1 cleaner) flotation testwork

Head grade vs. concentrate grades is shown in Figure 13.29, Figure 13.30 and Figure 13.31.

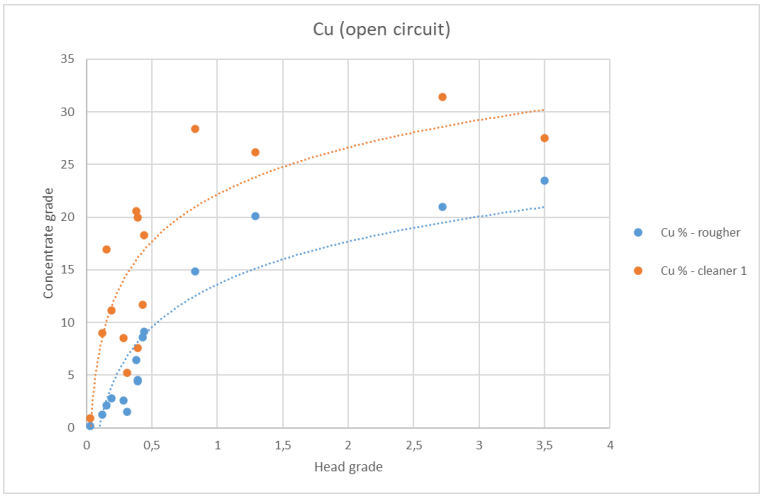


Figure 13.29 – Cu head grade vs. Cu concentrate recovery for the STW variability samples in open circuit (rougher + 1 cleaner) flotation testwork

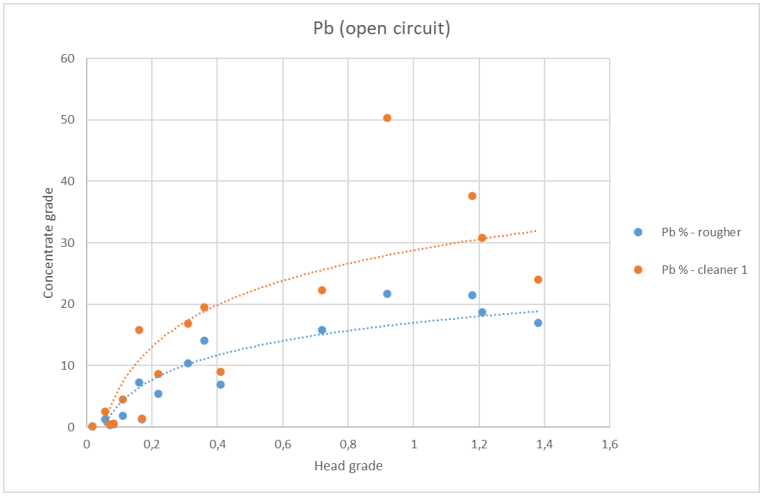


Figure 13.30 – Pb head grade vs. Pb concentrate recovery for the STW variability samples in open circuit (rougher + 1 cleaner) flotation testwork

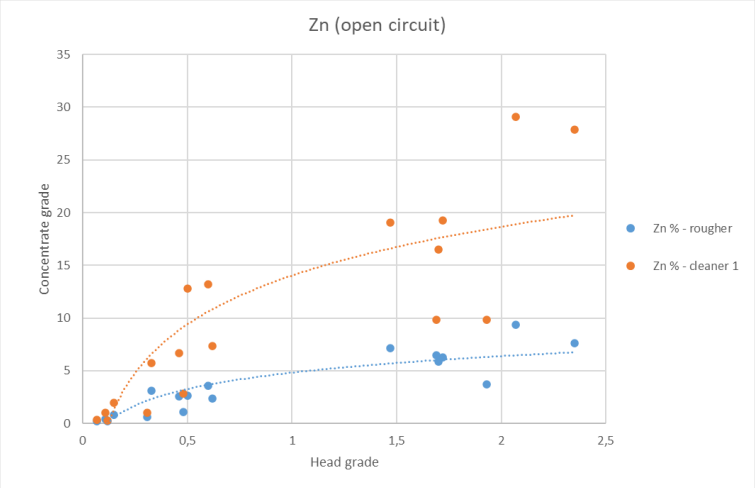


Figure 13.31 – Zn head grade vs. Zn concentrate recovery for the STW variability samples in open circuit (rougher + 1 cleaner) flotation testwork

13.3.7.3 GOSSAN (GOS)

Four GOS variability samples representing different zones of the deposit were produced for comminution and leaching testing. Comminution was performed on 3 variability samples using the Geopyroa, French Crushability and Abrasiveness tests, the results of which are summarised in

Table 13.36.

Table 13.36 – Summary of Geopyroa, French Crushability and Abrasiveness testing results on GOS variability samples.

Sample	Solid Density (S.G)	Crushability	Abrasiveness	DWI (kWh/m3)	BWI (kWh/t)	PLT Is (MPa)	UCS (Mpa)
GOS_2	3.51	59	1360	4.4	14.1	15.8	315.1
GOS_3	2.51	50	2200	3.9	19.1	10.5	209.8
GOS_4	3.63	67	760	2.9	16.8	9	179.9

Each GOS sample ground to P80 75 µm and subjected to a 48-hour bottle roll leach with conditions equivalent to the bulk sample. Poor extraction was observed in comparison to the bulk sample with a maximum of 66% Ag extraction seen with GOS 4 and 51% Au extraction with GOS 2. Reagent consumption was equivalent to that of the bulk sample.

Table 13.37 – GOS variabaility leach extractions and reagent consumption.

Sample	Extraction	Consumption

	Cu	Au	Ag	NaCN (kg/t)	CaO (kg/t)
GOS 1	2%	21%	32%	0.26	0.37
GOS 2	16%	51%	60%	0.61	0.43
GOS 3	4%	26%	55%	0.14	0.54
GOS 4	4%	29%	66%	0.68	0.50

13.3.7.4 TRANSITION MASSIVE SULPHIDE (TMS)

Four TMS variability samples representing different zones of the deposit were produced for comminution using the Geopyroa test (results summarized in Table 13.38). Further flotation testwork in composite samples is required for a better definition of the optimum process before advancing with variability samples.

Table 13.38 – Summary of Geopyroa, French Crushability and Abrasiveness testing results on TMS variability samples.

Sample	Solid Density (S.G)	Crushability	Abrasiveness	DWI (kWh/m3)	BWI (kWh/t)	PLT Is (MPa)	UCS (MPa)
TMS_1	4.81	72	940	4.0	10.4	11.4	227.4
TMS_2	3.62	43	1780	6.6	12.5	15.1	301.5
TMS_3	4.09	67	860	2.4	8.7	9.8	196
TMS_4	3.28	40	1900	4.1	12.2	13.8	276.3

Each TMS sample was ground to P¬80 ¬20 µm replicating the best performance from the bulk sample leach testing. Bottle roller testing, with a pre-aeration stage added with O2 for 16 hours, resulted in the reduction of lime in the process. Extraction of Au was lower than the bulk with a maximum of 54% seen for TMS 4. Ag extraction, as before, proved to be erratic.

Table 13.39 – TMS variability leach extractions and reagent consumption.

Sample	Extraction			Consumption	
	Cu	Au	Ag	NaCN (kg/t)	CaO (kg/t)
TMS 1	44%	31%	0%	19.70	0.14
TMS 2	69%	42%	11%	16.92	0.31
TMS 3	63%	30%	0%	17.61	0.20
TMS 4	43%	54%	0%	16.25	0.27

13.3.7.5 STRINGER (STR)

Nine STR variability samples representing different zones of the deposit were produced for comminution and flotation testing. Comminution was performed on 8 variability samples using the Geopyroa, French Crushability and Abrasiveness tests, the results of which are summarized in Table 13.40.

Table 13.40 – Summary of Geopyroa, French Crushability and Abrasiveness testing results on STR variability samples.

Sample	Solid Density (S.G)	Crushability	Abrasiveness	DWI (kWh/m3)	BWI (kWh/t)	PLT Is (MPa)	UCS (MPa)
STR_2	2.89	60	340	3.1	11.2	19.7	393.2
STR_3	3.17	46	180	2.1	9.6	12.1	241.7
STR_4	2.92	49	348.84	3.6	12.3	15.9	318.7
STR_5	4.13	44	1100	5.2	11.6	15.4	308.1
STR_6	3.29	41	360	4.6	12.4	16.7	334
STR_7	2.81	44	540	4.4	14.2	14.2	283.4
STR_8	4.36	48	760	3.3	9.3	9.6	192.5
STR_9	4.03	46	100	3.6	10.7	18.7	374.8

Cu/Pb (depending on head grade) and Zn flotation testwork, using rougher + one cleaner open circuit configuration, was tested. Head grade vs. concentrate recovery is shown in Figure 13.32, Figure 13.33 and Figure 13.34.

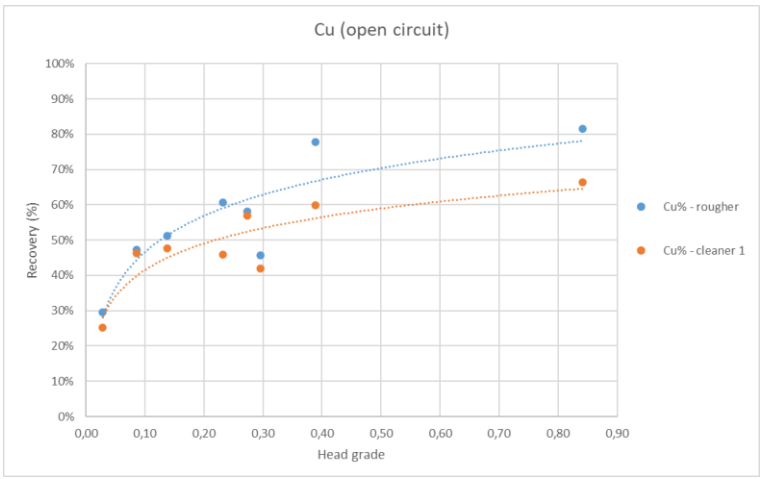


Figure 13.32 – Cu head grade vs. Cu/Pb concentrate recovery for the STR variability samples in open circuit (rougher + 1 cleaner) flotation testwork

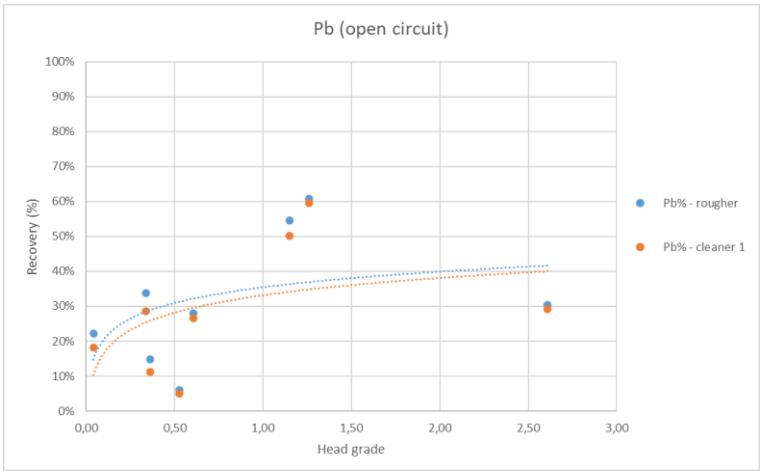


Figure 13.33 – Pb head grade vs. Cu/Pb concentrate recovery for the STR variability samples in open circuit (rougher + 1 cleaner) flotation testwork

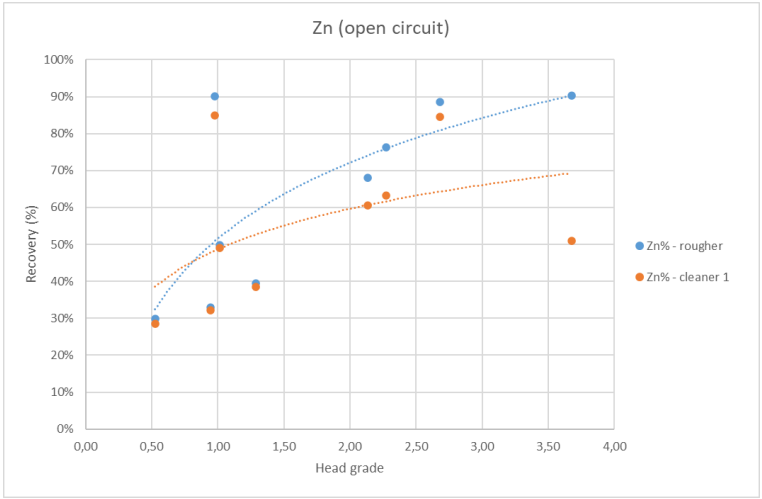


Figure 13.34 – Zn head grade vs. Zn concentrate recovery for the STR variability samples in open circuit (rougher + 1 cleaner) flotation testwork

Head grade vs. concentrate grades is shown in Figure 13.35, Figure 13.36 and Figure 13.37.

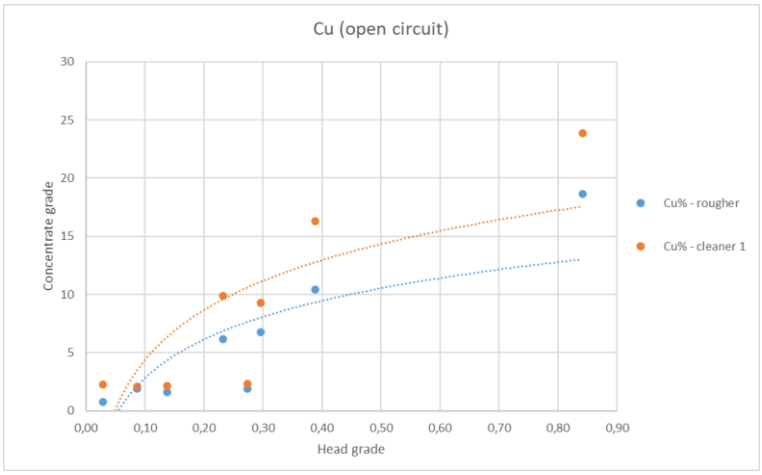


Figure 13.35 – Cu head grade vs. Cu/Pb concentrate recovery for the STR variability samples in open circuit (rougher + 1 cleaner) flotation testwork

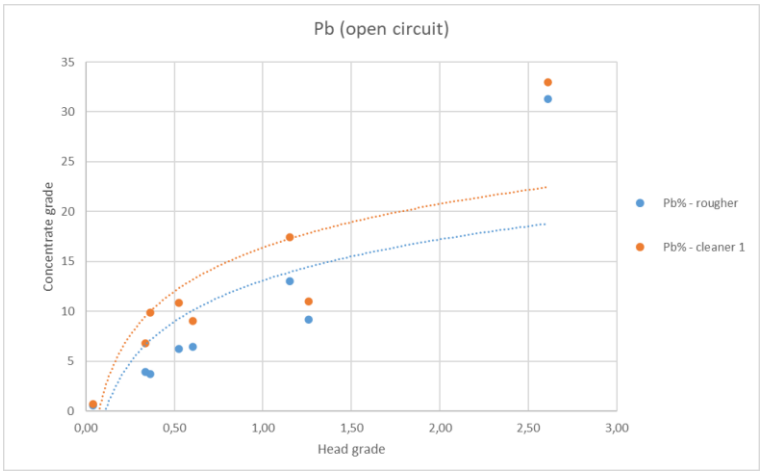


Figure 13.36 – Pb head grade vs. Cu/Pb concentrate recovery for the STR variability samples in open circuit (rougher + 1 cleaner) flotation testwork

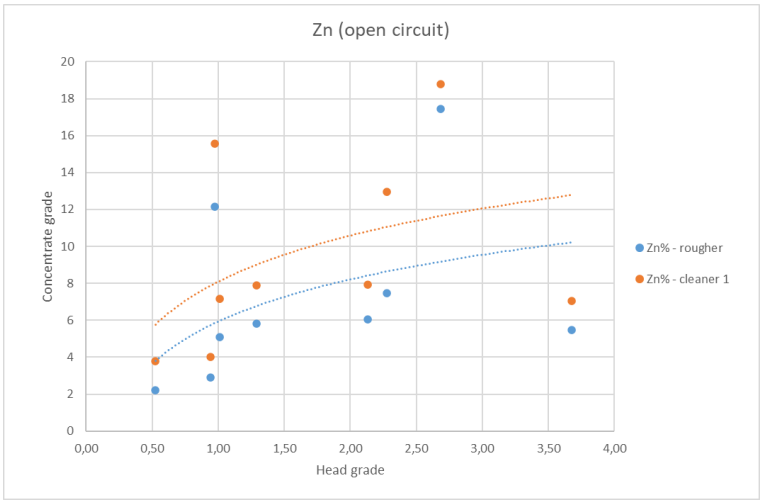


Figure 13.37 – Zn head grade vs. Zn concentrate recovery for the STR variability samples in open circuit (rougher + 1 cleaner) flotation testwork

13.4 METALLURGICAL PERFORMANCE PROJECTIONS AND RECOMMENDATIONS

All metallurgical and mineralogical tests results described herein were used to develop the best estimates of metallurgical projections described in this section.

13.4.1.1 PRIMARY MASSIVE SULPHIDE (PMS)

- Lead concentrate: 30% Pb grade and 60% recovery (30% Ag and 10% Au associated recoveries), supported by LCTs completed by GSL and Maelgwyn laboratories.
- Zn concentrate: 35% Zn grade and 70% recovery (15% Ag recovery and 20% Au), supported by LCTs completed by GSL and Maelgwyn laboratories.
- Sn concentrate: 50% Sn grade and 40% recovery, estimated with the theoretical recovery/grade curve built with the mineralogical analysis completed by Petrolab. Open circuit test works with sulphide flotation, tin flotation and tin gravity completed by GSL, anticipating good tin mineral concentration. Additional work is required to confirm final tin grade and recovery, as well as auxiliary elements in the final tin concentrate.

13.4.1.2 STOCKWORK (STW)

- Cu concentrate: 20% Cu grade at 65% recovery (15% Ag and 5% Au associated recovery), supported by LCTs completed by GSL laboratory.
- Pb concentrate: 35% Pb grade at 65% recovery (50% Ag and 5% Au associated recovery), supported by LCTs completed by GSL laboratory.

- Zn concentrate: 45% Zn grade at 75% recovery (20% Ag and 5% Au associated recovery), estimate based on open circuit tests developed by GSL laboratory.

13.4.1.3 GOSSAN (GOS)

- Au/Ag bullion: 91% Au and 89% Ag extraction, supported by one bulk sample performance leaching test work. Very erratic behaviour of variability samples, where performances varied from 90% to 25% extraction for the two metals. The proposed numbers come from bulk sample but, further test work is required to confirm the data.

- Pb/Sn concentrate: 20% Pb and 40% Sn recoveries to a 30% Pb and 10% Sn grade concentrate. Estimate based on the gravity test work completed by GSL, demonstrating that both metals can be easily concentrated and could potentially bring some additional Ag and Au credits. Further test work to simulate the complete gravity circuit and determine the final grades and recoveries.

13.4.1.4 TRANSITION MASSIVE SULPHIDE (TMS)

- Pb concentrate: 35% Pb grade at 50% recovery (70% Ag and 50% Au associated recovery), estimated with open circuit tests developed by GSL and estimated with mineralogical analysis completed by Petrolab. Additional work is required to confirm the Pb upgrade levels, as well as Pb, Au and Ag recoveries.

  Alternatively, leaching could be applied to this domain with 45% Au and 3% Ag extraction, according to the cyanidation testing completed in GSL. With the Cu head grade and speciation, it cannot be estimated, anticipated, or justified, at this stage, that better performance of a leaching process will be achieved.

- Zn concentrate: 40% Zn grade at 50% recovery (5% Ag and 10% Au associated), estimated with mineralogical analysis completed by Petrolab. Additional work required.

- Sn concentrate: 50% grade and 40% recovery, estimated with the theoretical recovery/grade curve built with the mineralogical analysis completed by Petrolab. Metallurgical test works required to demonstrate the possibility of recovering this metal.

13.4.1.5 STRINGER (STR)

- Cu concentrate: 20% grade at 60% recovery (10% Ag associated recovery), supported by LCTs completed by GSL laboratory.

- Pb concentrate: 30% grade at 50% recovery (15% Ag associated recovery), estimate based on open circuit tests developed by GSL and supported by the mineralogical

analysis completed by Petrolab. Additional work is required to confirm the Pb upgrade level and recovery.

- Zn concentrate: 40% grade at 70% recovery (15% Ag associated recovery), supported by open circuits completed by GSL laboratories. Additional works required to confirm the proposed numbers.

13.5 RECOMMENDATIONS FOR FURTHER INVESTIGATION

Some of the metallurgical performances described in the previous chapter will require additional work in the laboratory. Key investigation works to be performed are described below.

13.5.1.1 PRIMARY MASSIVE SULPHIDE (PMS)

Grades and recoveries are well supported by LCT performed in the laboratory, however additional works to improve the Zn concentrate grade should be completed. Sn concentration has shown a good potential but will require more work.

13.5.1.2 STOCKWORK (STW)

Grades and recoveries are well supported by LCT performed in the laboratory, except for the Zn recovery that would require more testing to achieve the target values determined by open circuit test works.

13.5.1.3 GOSSAN (GOS)

Leaching efficiencies come from one individual sample and the performance of variability samples were not consistent with these values, so further works is required to determine the difference between materials and the optimal route to achieve the desirable performances. Pb and Sn concentration have shown a good potential but will require more investigation.

13.5.1.4 TRANSITION MASSIVE SULPHIDE (TMS)

The performance of open circuit testwork has shown a good potential to produce Pb and Zn concentrates, however the performance of cleaners will require more investigation, as well as LCTs to confirm the final values. Sn concentration looks feasible based on mineralogical analysis but will require laboratory testing.

13.5.1.5 STRINGER (STR)

Open circuit and LCT completed have shown a good potential for the three Cu, Pb and Zn concentrates but additional works are required to confirm target values stablished with open circuit tests.

14 MINERAL RESOURCE ESTIMATES

During 2021 and 2022, Ascendant/Redcorp drilled 26 infill and step-out diamond drill holes on the North and South deposits. Micon developed and updated the LS project mineral resources incorporating the recent 2021/22 drilling campaign and provided a final estimate of the LS mineral resource in May 2023 which forms the foundation of the DFS described in this report. The estimation protocols/methodologies used in 2019 for the north deposit and in 2021 for the south deposit were used as base, but the wireframing had to be refined considering the new drilling information. In essence, the additional drilling (i) enabled the subdivision of the North deposit massive sulphide domain into an upper transitional zone immediately beneath the gossan domain, and a lower primary (un-weathered) primary massive zone underlain by the stockwork zone, (ii) culminated in the merging of the Central zone with the South zone to form a continuous South sector, and (iii) enabled the subdivision of the South sector into 5 different corridors.

14.1 EXPLORATORY DATA ANALYSIS

14.1.1 DATABASE DESCRIPTION

The LS property deposits have been tested by diamond drilling over a cumulative strike length of approximately 1.7 km and down to a maximum vertical depth of about 800 m. The resource database is derived from 132 surface diamond drillholes, all of which were utilized in the resource estimation. Original assay certificates from the laboratory were provided as csv and pdf documents. A detailed DTM and overburden depth model were provided as dxf surfaces.

The average drillhole spacing in the densest drilled areas of the project is about 20 m; the spacing in the less densely drilled areas is between 40 and 150 m.

14.1.2 DEPOSIT COMPONENTS

The LS property is comprised of multi-metal deposits whose main components are zinc, lead, copper, gold, silver, and tin. The correlation between metals varies within each domain, reflecting primary and secondary processes such as metamorphism and remobilization which affected the metals differently.

14.2 OVERVIEW OF ESTIMATION METHODOLOGY

Following the completion of the database validation as outlined in Chapter 12 above, Micon estimated the LS property mineral resources following a logical sequence involving:

- Geological interpretation.
- Determination and modelling of estimation domains.

- Compositing and grade capping.
- Statistics within domains.
- Variography.
- Definition of resource parameters and block model.
- Grade interpolation and resource definition.
- Mineral resource classification.
- Estimation checks to verify the results.
- Application of an NSR for valuing the deposits economics.

The mineral resource block model was developed using coordinate projection to UTM WGS 84 Zone 29N.

14.3 GEOLOGICAL INTERPRETATION

Geophysics (Figure 14.1) suggests the LS mineralization extends continuously beneath Quaternary and Tertiary cover sedimentary lithologies over the entire drilled strike length of about 1.7 km and remains open to the northwest and southeast. However, two deposits are recognized: North deposit and the South deposit. The respective locations are shown in Figure 14.1. The separation between the two deposits appears to be an artifact of the drilling coverage, i.e., the zone separating the two deposits has not been tested by drilling.

The North deposit is complex, being comprised of 4 types of polymetallic mineralization: gossan (Gos), transition massive sulphide (tMS), primary massive sulphide (pMS), and stringer (str) mineralization beneath, and on the periphery of the primary sulphide zone. The Gos mineralization resulted from the weathering of the underlying primary sulphide mineralization and the tMS represents the transition from Gos to pMS.

The South deposit comprises polymetallic fissural remobilizations, stringer and isolated gold-rich silicified zones which appear to be structurally controlled. It is mostly hosted in a unit locally known as the iV. The iV is heterogeneous, composed of intricately interleaved/interlayered dacitic-andesitic-basaltic volcanic rocks. The sulphide mineralization has been remobilized into fissures/cracks arising from brittle failure during deformation, hence the name ‘fissural mineralization’. Although the entire iV unit is mineralized to some extent, potentially economic grade mineralization occurs in steeply dipping corridors; the steep dip is attributed to post mineralization east-west compressional deformation.

Appreciable tin mineralization is restricted to the North deposit whereas zinc, lead, copper, gold, and silver are common to all the deposits. However, copper and zinc are apparently dominant over lead, gold, and silver in the south deposit.

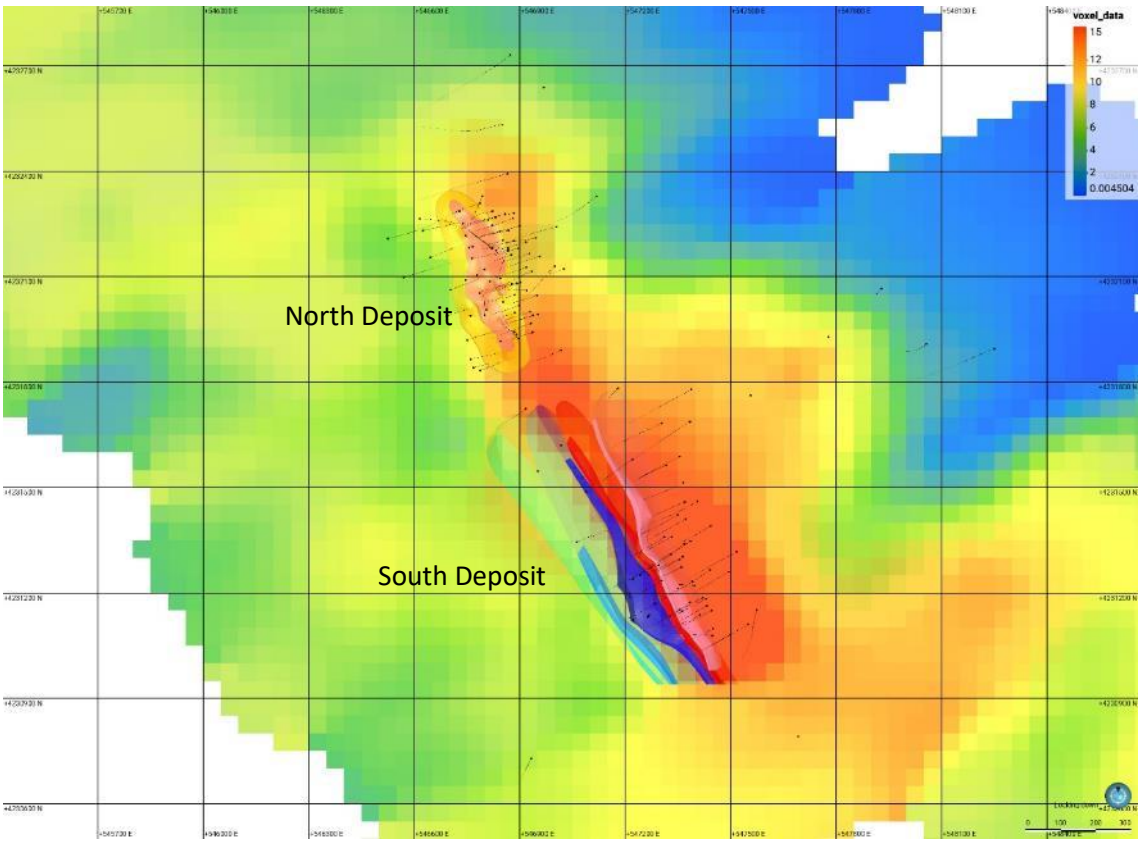


Figure 14.1 - IP chargeability slice at -100 m Z (corresponding to around 190 m depth) showing the location of the currently known deposits on the LS property

14.4 SELECTION AND MODELLING OF ESTIMATION DOMAINS

Software Leapfrog Edge was used for the modelling of estimation domains.

14.4.1 NORTH DEPOSIT

Micon's estimation domain selection criterion is based purely on geology for the Gos, tMS and pMS domains and on the zinc equivalent (ZnEq) threshold value for the str (stringer) domain. The ZnEq% threshold value for modelling was obtained via the following formula:

$$ZnEq\% = \frac{Zn\% \times 26.46 + Pb\% \times 22.05 + Cu\% \times 77.16 + Au\% \times 54.66 + Ag\% \times 0.71}{26.46}$$

Note: In the above equation, the multiplying factor for each metal grade is the relevant metal price per %unit. From the subsequent probability plot, the ZnEq threshold value selected for modelling the str wireframe is 1.5%.

As already mentioned in Sections 7 and 8, the main change from previous modelling updates was the definition of the new domain tMS (transition massive sulphides). It

derivates from leached massive sulphides between the paleo-gossan and the primary massive sulphides (pMS). This domain shows some mineralogical and paragenetic particularities with some secondary epigenetic minerals but is mostly recognized by the geochemical ratios. Zn is mostly depleted while Cu, Ag, Au and REEs are enriched relative to the primary massive sulphides (pMS). Figure 14.2 shows a comparison of Zn and Cu values of the massive sulphides tMS (pink) and pMS (red), where a distinct opposite geochemical tendence can be observed between tMS and pMS. Lately, this has a direct impact in the recoveries assumptions, as this domain must be processed separately from the primary massive sulphides. Accordingly, it was estimated as a separate entity.

The wireframing of the tMS was initially created using an isotropic interpolation from the logged massive sulphides and then refined with manually defined polylines and control points to honour the geological interpretation and avoid non-coherent forms (see green polylines and points in Figure 14.3). Once the MS base wireframe was created, a normalized Cu/Zn+Pb ratio was used with the complementary help of S values to define the paleo leaching/enriching table and the contact between the transition and primary massive sulphides (shown as dark purple points in Figure 14.3). It is important to mention that the same logic was followed in the definition of the metallurgical samples. Both petrographic and mineralogical studies delivered results that confirm the correctness of this split.

The str (stringer) zone requires a different modelling approach as it lacks a natural boundary and straddles across different lithologies (acidic to intermediate volcanics) from the lower volcanic formation to the top dacitic tuffites. The str is clearly more developed within the lower volcanic rocks but still shows some influence in the *cap rock*. To delineate the stringer envelope, a series of sections with polylines and control points (shown as green polylines and points in Figure 14.3) were systematically interpreted using the 1.5% ZnEq as reference at a low threshold allowing high internal dilution for continuity and subsequently doing a bulk estimation to highlight the higher grades areas.

In summary, the selected North deposit domains as shown in Figure 14.3 are as follows:

- Gos - Gossan (orange).
- tMS – transition massive sulphides (purple).
- pMS – primary massive sulphides (red).
- str - stringer (green).

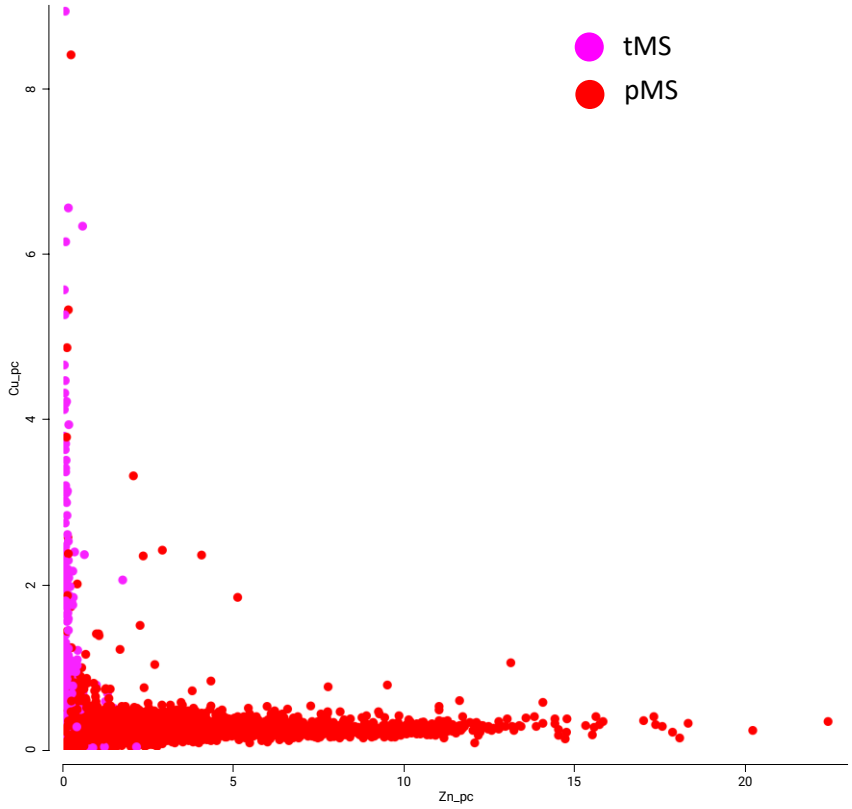


Figure 14.2 - Scatter graph of Cu % vs Zn % tMS and pMS samples

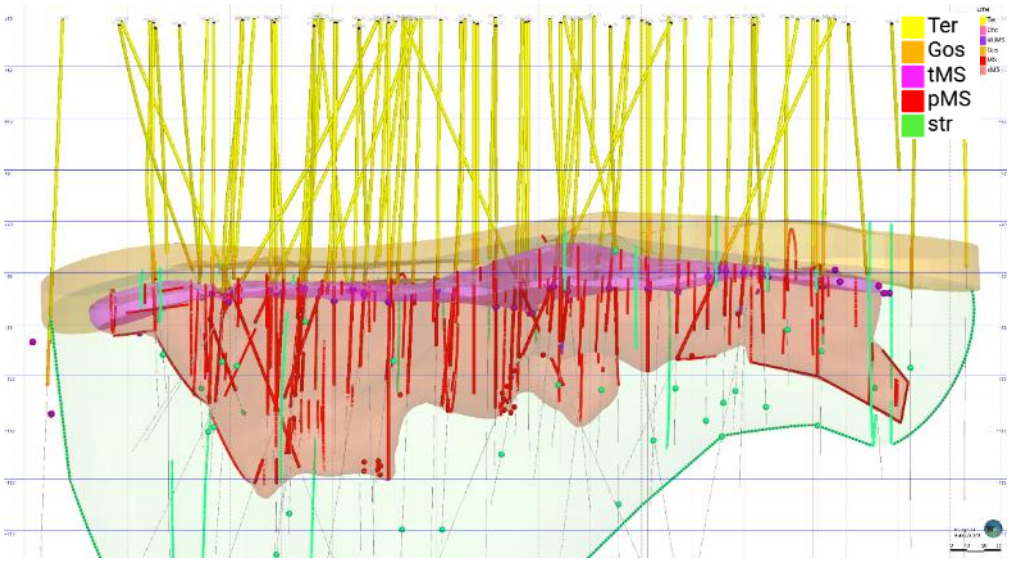


Figure 14.3 - 3D perspective of the LS project North deposit domains

14.4.2 SOUTH DEPOSIT

The infill and step-out drilling enabled a detailed section by section interpretation of CuEq% grades within the main host-rocks of mineralization comprising esV and iV rocks. A generalized 3D perspective of the host rocks enclosing the higher-grade corridors is shown in Figure 14.4. A more detailed perspective of the corridors (higher-grade zones) which excludes the broader outer envelope is shown in Figure 14.5. As stated in Section 8, the areas in-between the corridors also contain potentially economic mineralization but with lower metal grades than the 5 corridors.

The modelling involved the following steps:

- Defining the boundary/outer limit of the iV and esV units using geological logs to split sMS and FR corridors sub-domains. FR corridors are exclusively enclosed in iV lithology and whilst sMS corridors are in the hanging-wall esV rock.
- Assigning each sample, a copper equivalent value (CuEq%) calculated with the formula below (where the multiplier is the metal price per %unit):

$$CuEq\% = \frac{Zn\% \times 26.46 + Pb\% \times 22.05 + Cu\% \times 77.16 + Au\% \times 54.66 + Ag\% \times 0.71}{77.16}$$

- Wireframing/modelling the main corridors of mineralization using the interpretation of each section *grade-shells* (average of 20m *step*).

Statistical analysis appears to suggest that that the boundary between the geological envelopes and the higher-grade corridors are hard as shown in Figure 14.6. However, as stated earlier, the enclosing envelope is not barren but also contains potententially economic lower-grade mineralization. Thus, the bigger volumes/wall rocks enclosing the corridiors were also estimated. It is interesting to note that metallurgical variability tests showed similar recoveries between wall rocks and the high-grade corridors.

A geological cross section (Figure 14.7) shows the average CCPI (Chlorite Carbonate Pyrite Index) of the iV domain against the higher-grade sub-domains of C1 FR, C2 FR and C3 FR using the CuEq % assays. This substantiates the initial *grade-shell* corridor domaining as is explained in Sections 7 and 8.

In summary, the selected South deposit sub-domains are as follow:

- C1 FR – Fissural Remobilizations Corridor 1.
- C2 FR – Fissural Remobilizations Corridor 2.
- C3 FR – Fissural Remobilizations Corridor 3.
- C1 sMS – semi-massive sulphides Corridor 1.
- C2 sMS – semi-massive sulphides Corridor 2.

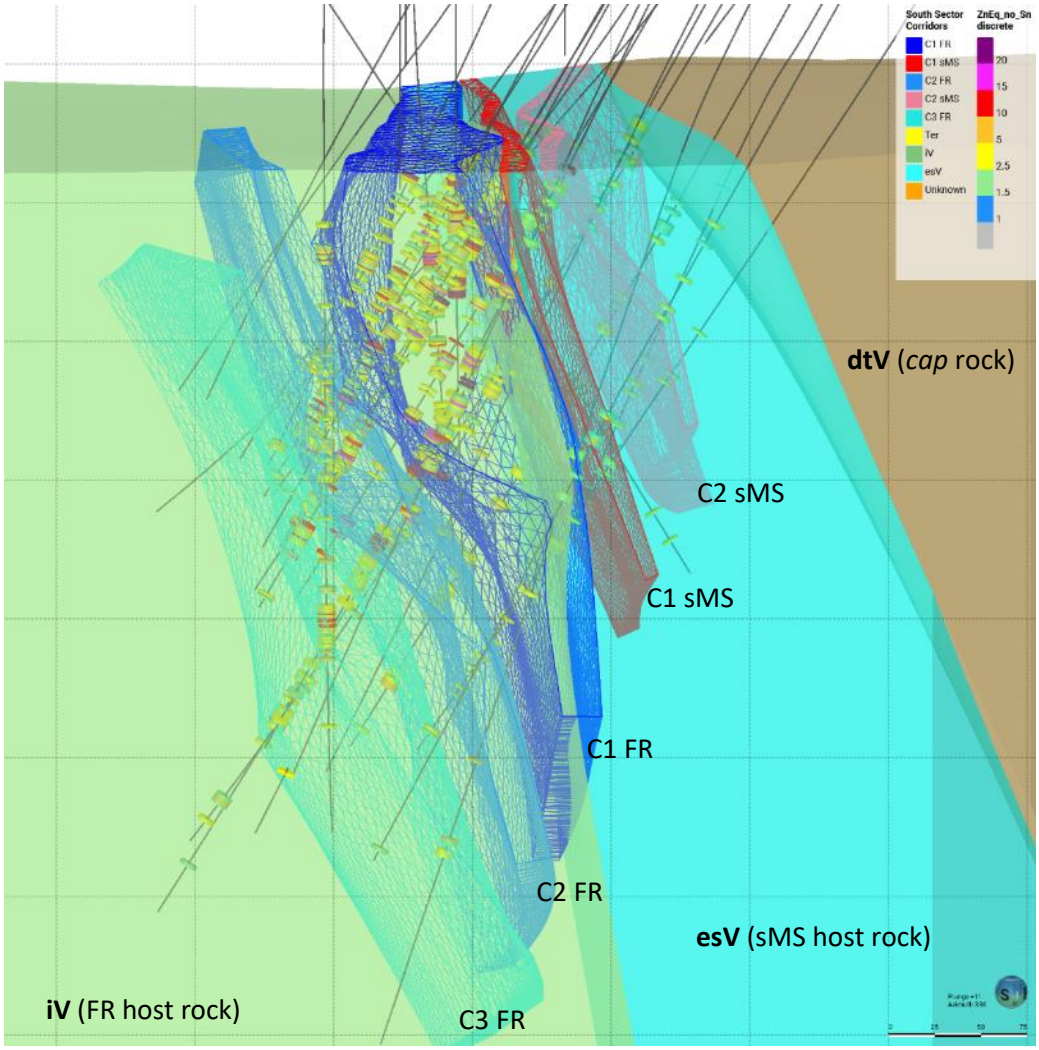


Figure 14.4 – Generalized 3D Perspective of the South Deposit

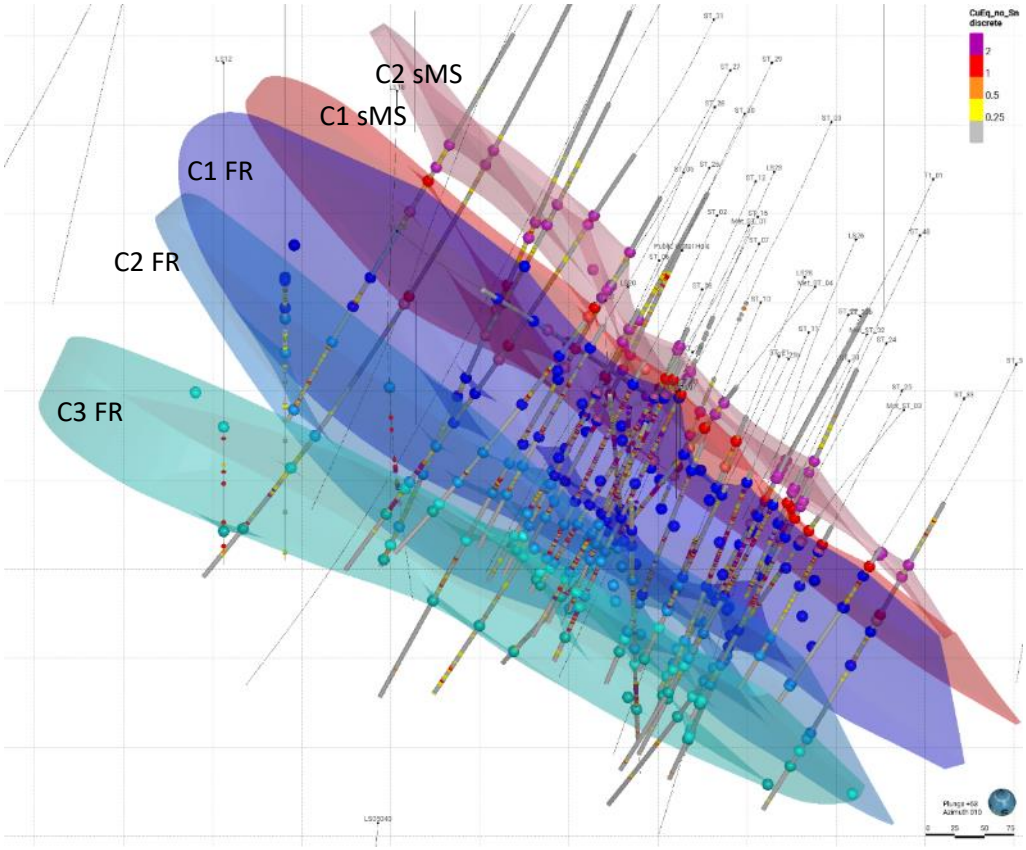


Figure 14.5 - 3D perspective of the South deposit with all corridors contact points (azimuth 10º/plunge +53º)

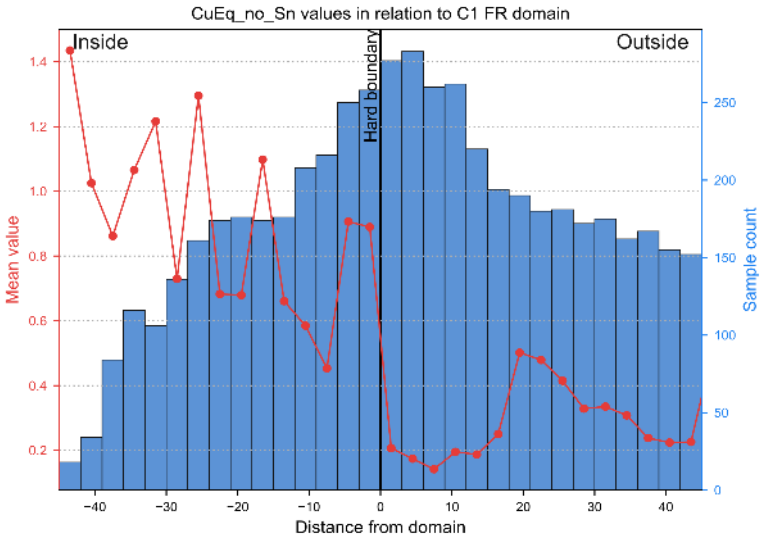


Figure 14.6 - South deposit boundary analysis between C1 FR mineralized corridor and geological model

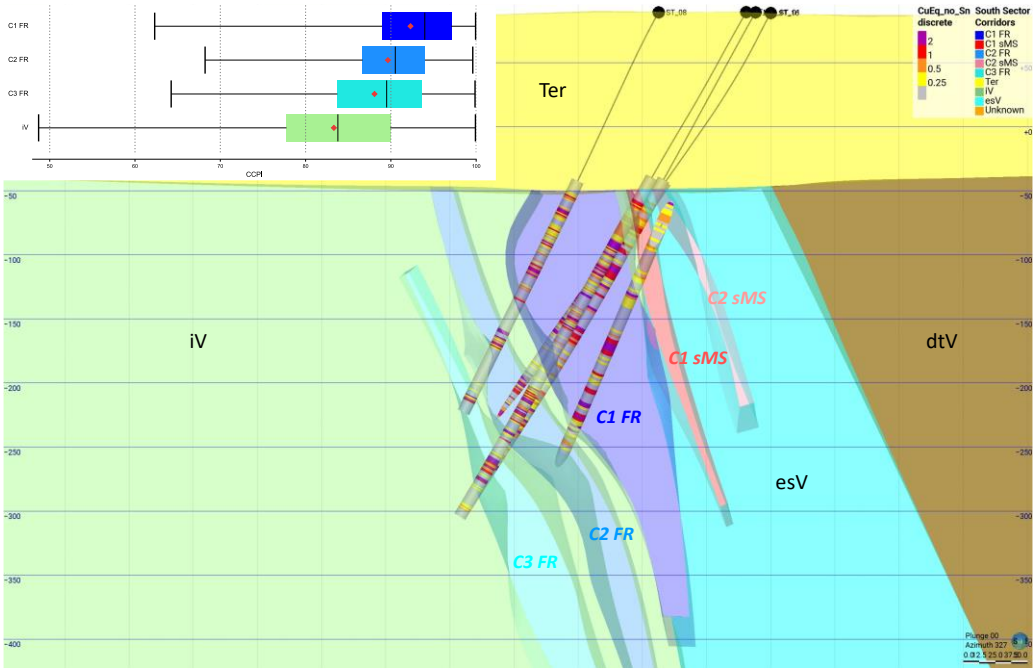


Figure 14.7 - South sector cross-section with CCPI index of iV domain versus their mineralized sub-domains

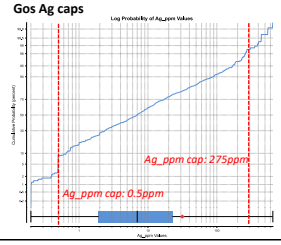
14.5 GRADE CAPPING, COMPOSITING, STATISTICS AND VARIOGRAPHY

14.5.1 GRADE CAPPING, COMPOSITING AND STATISTICS

In previous estimations, Micon examined the relationship between sample length and grade and established that a considerable number of high grades were associated with lengths greater than the mode of the sample lengths of 1 m, therefore a 2 m composite was used. Following that, all the recent drilling used a sampling length of 2m. All domains were composited into 2 m for the determination of grade capping threshold values, using population histograms and probability / log probability plots. The summary statistics and log-probability plots for the different main domains (i.e., the best/highest grade mineralized domain) are shown in Figure 14.8 for the key elements. Grade capping values are indicated in red in the plots for each element/metal. Some older assays under the detection limit were inputted in the database as “0” (zero) and in other cases, during the composites creation, some of the lower values (half of the detection limit) were broken into lower values at the residual lengths at the edges of the wireframes. For that reason, a lower cap equivalent to half of the detection limit of each metal was added. The capping was applied at the 2 m composites prior to variography. The summary statistics of the capped and un-capped composites are shown in Table 14.1 and Table 14.2.

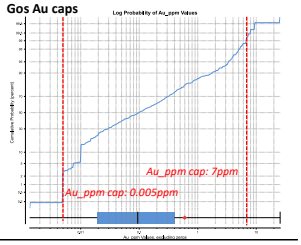
Gos Ag caps

	Composited	Uncomposited
Count	570	970
Length	1139.4	1127.0
Mean	31.57	32.12
SD	70.32	98.38
CV	2.23	3.06
Variance	4945.5	9678.5
Minimum	0.20	0.20
Q1	1.93	1.00
Q2	6.99	5.70
Q3	22.64	19.00
Maximum	650.09	1917.00



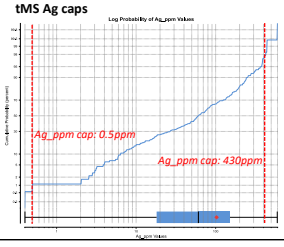
Gos Au caps

	Composited	Uncomposited
Count	570	970
Length	1139.4	1127.0
Mean	0.60	0.63
SD	1.62	2.41
CV	2.70	3.81
Variance	2.6	5.8
Minimum	0.00	0.00
Q1	0.02	0.01
Q2	0.09	0.07
Q3	0.40	0.35
Maximum	25.59	54.20



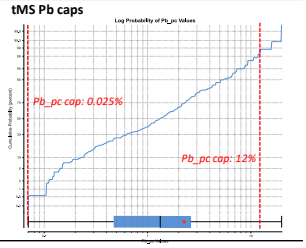
tMS Ag caps

	Composited	Uncomposited
Count	179	309
Length	357.6	338.8
Mean	102.81	99.57
SD	112.41	119.24
CV	1.09	1.20
Variance	12635.5	14217.9
Minimum	0.40	0.50
Q1	18.11	20.00
Q2	60.76	59.00
Q3	151.50	134.00
Maximum	596.00	736.00



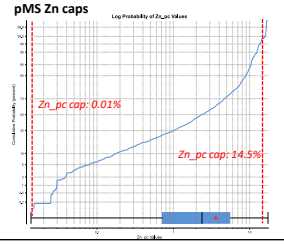
tMS Pb caps

	Composited	Uncomposited
Count	179	309
Length	357.6	338.8
Mean	2.26	2.10
SD	2.88	2.78
CV	1.27	1.33
Variance	8.3	7.7
Minimum	0.07	0.06
Q1	0.47	0.44
Q2	1.33	1.20
Q3	2.63	2.45
Maximum	19.87	22.10



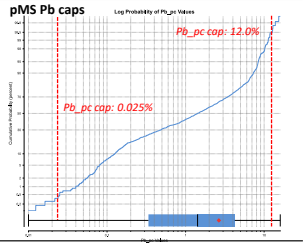
pMS Zn caps

	Composited	Uncomposited
Count	1147	2001
Length	2293.3	2270.1
Mean	3.59	3.62
SD	3.47	3.61
CV	0.97	1.00
Variance	12.0	13.0
Minimum	0.01	0.01
Q1	0.74	0.68
Q2	2.38	2.38
Q3	5.58	5.63
Maximum	17.43	22.40



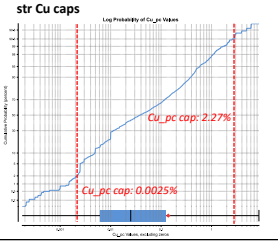
pMS Pb caps

	Composited	Uncomposited
Count	1147	2001
Length	2293.3	2270.1
Mean	2.65	2.70
SD	2.92	3.05
CV	1.10	1.13
Variance	8.5	9.3
Minimum	0.01	0.01
Q1	0.34	0.32
Q2	1.42	1.38
Q3	4.22	4.21
Maximum	16.04	19.29



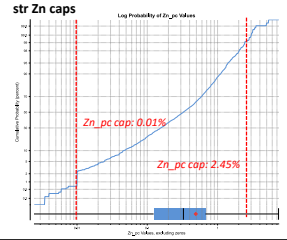
str Cu caps

	Composited	Uncomposited
Count	2089	3583
Length	4174.5	4069.6
Mean	0.14	0.13
SD	0.40	0.48
CV	2.91	3.64
Variance	0.2	0.2
Minimum	0.00	0.00
Q1	0.01	0.00
Q2	0.03	0.02
Q3	0.13	0.11
Maximum	8.69	16.60



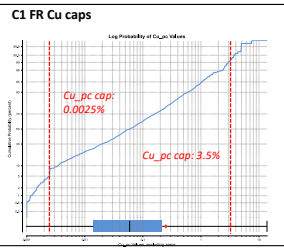
str Zn caps

	Composited	Uncomposited
Count	1147	2001
Length	2293.3	2270.1
Mean	2.65	2.70
SD	2.92	3.05
CV	1.10	1.13
Variance	8.5	9.3
Minimum	0.01	0.01
Q1	0.34	0.32
Q2	1.42	1.38
Q3	4.22	4.21
Maximum	16.04	19.29



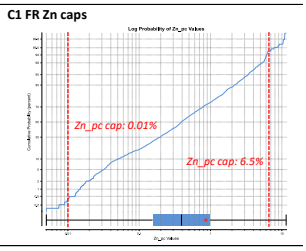
C1 FR Cu caps

	Composited	Uncomposited
Count	1415	2287
Length	2830.1	2790.5
Mean	0.25	0.25
SD	0.66	0.80
CV	2.64	3.20
Variance	0.4	0.6
Minimum	0.00	0.00
Q1	0.02	0.01
Q2	0.06	0.04
Q3	0.22	0.16
Maximum	13.74	15.54



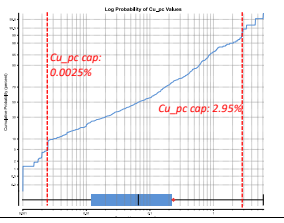
C1 FR Zn caps

	Composited	Uncomposited
Count	1415	2287
Length	2830.1	2790.5
Mean	0.87	0.87
SD	1.31	1.59
CV	1.51	1.82
Variance	1.7	2.5
Minimum	0.01	0.00
Q1	0.16	0.11
Q2	0.40	0.33
Q3	1.01	0.87
Maximum	11.49	16.55



C2 FR Cu caps

	Composited	Uncomposited
Count	549	851
Length	1097.8	1050.2
Mean	0.23	0.24
SD	0.52	0.61
CV	2.21	2.59
Variance	0.3	0.4
Minimum	0.00	0.00
Q1	0.01	0.01
Q2	0.07	0.05
Q3	0.22	0.20
Maximum	6.11	7.20



C2 FR Zn caps

	Composited	Uncomposited
Count	549	851
Length	1097.8	1050.2
Mean	0.65	0.67
SD	1.15	1.39
CV	1.78	2.07
Variance	1.3	1.9
Minimum	0.00	0.00
Q1	0.06	0.04
Q2	0.21	0.16
Q3	0.59	0.56
Maximum	11.19	14.00

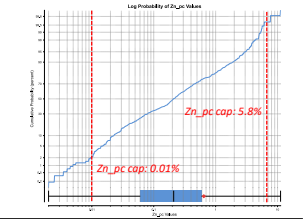


Figure 14.8 - Summary statistics of composited and uncomposited samples and log-probability plot for the cardinal elements per domain

Table 14.1 - Summary statistics of the Capped and Un-capped composites from the north sector domains

Gos

	Ag_ppm		Au_ppm		Cu_pc		Pb_pc		Sn_pc		Zn_pc	
	UnCap	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*
Top Cap		*275*		*7*		*1.2*		*14*		*1.40*		*2*
Base Cap		*0.5*		*0.005*		*0.0025*		*0.01*		*0.005*		*0.05*
Nº of Top caps		*8*		*4*		*4*		*6*		*5*		*5*
Nº of Base caps		*15*		*17*		*9*		*25*		*48*		*1*
Count	570	570	570	570	570	570	570	570	409	409	570	570
Mean	31.56	29.20	0.60	0.56	0.092	0.086	1.81	1.74	0.161	0.156	0.48	0.47
Variance	4942.8	3227.0	2.6	1.5	0.045	0.023	10.8	8.0	0.09	0.08	0.16	0.13
Minimum	0.20	0.50	0.000	0.005	0.000	0.005	0.000	0.010	0.005	0.005	0.03	0.05
Maximum	650.09	275.00	25.59	7.00	3.31	1.20	34.75	14.00	2.38	1.40	4.32	2.00

tMS

	Ag_ppm		Au_ppm		Cu_pc		Pb_pc		Sn_pc		Zn_pc	
	UnCap	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*
Top Cap		*430*		*5*		*4*		*12*		*0.65*		*1.95*
Base Cap		*0.5*		*0.005*		*0.0025*		*0.05*		*0.005*		*0.01*
Nº of Top caps		*3*		*2*		*1*		*3*		*4*		*2*
Nº of Base caps		*1*		*12*		*4*		*0*		*0*		*0*
Count	179	179	179	179	179	179	179	179	117	117	179	179
Mean	102.75	101.68	1.14	1.05	0.98	0.97	2.25	2.19	0.186	0.175	0.28	0.26
Variance	12632.8	11768.8	2.98	1.55	1.04	0.98	8.26	6.49	0.05	0.03	0.23	0.15
Minimum	0.400	0.500	0.000	0.005	0.000	0.005	0.070	0.070	0.010	0.010	0.010	0.010
Maximum	596.00	430.00	15.99	5.00	5.32	4.00	19.87	12.00	1.55	0.65	4.09	1.95

pMS

	Ag_ppm		Au_ppm		Cu_pc		Pb_pc		Sn_pc		Zn_pc	
	UnCap	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*
Top Cap		*280*		*3.6*		*1.45*		*12*		*0.43*		*14.5*
Base Cap		*0.5*		*0.005*		*0.0025*		*0.0025*		*0.005*		*0.01*
Nº of Top caps		*16*		*14*		*6*		*6*		*11*		*7*
Nº of Base caps		-		*68*		*6*		-		-		-
Count	1147	1147	1147	1147	1147	1147	1147	1147	853	853	1147	1147
Mean	62.16	61.61	0.65	0.65	0.30	0.29	2.65	2.64	0.137	0.135	3.59	3.58
Variance	3502.7	3231.6	0.59	0.54	0.09	0.03	8.50	8.28	0.011	0.009	12.05	11.76
Minimum	0.50	0.50	0.000	0.005	0.000	0.003	0.010	0.010	0.006	0.006	0.01	0.01
Maximum	406.26	280.00	5.70	3.60	6.86	1.45	16.04	12.00	0.83	0.43	17.43	14.50

Str

	Ag_ppm		Au_ppm		Cu_pc		Pb_pc		Sn_pc		Zn_pc	
	UnCap	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*
Top Cap		*85*		*0.65*		*2.27*		*1.37*		*0.42*		*2.45*
Base Cap		*0.5*		*0.005*		*0.0025*		*0.0025*		*0.005*		*0.001*
Nº of Top caps		*10*		*15*		*12*		*27*		*16*		*21*
Nº of Base caps		*13*		*104*		*322*		*68*		-		*1*
Count	2172	2172	2172	2172	2172	2172	2172	2172	1579	1579	2172	2172
Mean	8.50	8.12	0.06	0.05	0.14	0.14	0.14	0.13	0.034	0.030	0.49	0.48
Variance	266.4	129.4	0.029	0.008	0.164	0.164	0.151	0.048	0.010	0.003	0.296	0.229
Minimum	0.20	0.50	0.000	0.000	0.000	0.003	0.000	0.003	0.005	0.005	0.000	0.001
Maximum	442.00	85.00	5.00	0.65	8.69	8.69	12.41	1.37	1.96	0.42	7.38	2.45

Table 14.2 - Summary Statistics of the Capped and Un-capped composites from the south sector domains

C1 FR

	Ag_ppm		Au_ppm		Cu_pc		Pb_pc		Zn_pc	
	UnCap	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*
Top Cap		*93*		*0.44*		*3.5*		*5.8*		*6.5*
Base Cap		*0.5*		*0.005*		*0.0025*		*0.0025*		*0.01*
Nº of Top caps		*11*		*12*		*9*		*6*		*10*
Nº of Base caps		*2*		*18*		*76*				*5*
Count	1418	1418	1418	1418	1415	1415	1415	1415	1415	1415
Mean	8.96	8.61	0.06	0.06	0.251	0.237	0.49	0.48	0.87	0.85
Variance	294.4	207.1	0.0	0.0	0.438	0.253	0.9	0.7	1.72	1.41
Minimum	0.10	0.50	0.000	0.000	0.000	0.003	0.003	0.003	0.01	0.01
Maximum	242.66	93.00	2.70	1.94	13.74	3.50	12.51	5.80	11.49	6.50

C2 FR

	Ag_ppm		Au_ppm		Cu_pc		Pb_pc		Zn_pc	
	UnCap	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*
Top Cap		*88*		*2.2*		*2.95*		*3.7*		*5.8*
Base Cap		*0.5*		*0.005*		*0.0025*		*0.0025*		*0.01*
Nº of Top caps		*5*		*6*		*3*		*3*		*3*
Nº of Base caps		*1*		*4*		*29*		-		*11*
Count	537	537	549	549	549	549	549	549	549	549
Mean	8.57	8.31	0.11	0.08	0.23	0.23	0.32	0.31	0.65	0.63
Variance	252.3	201.1	0.30	0.07	0.27	0.20	0.37	0.35	1.31	1.11
Minimum	0.400	0.500	0.000	0.005	0.000	0.003	0.003	0.003	0.002	0.010
Maximum	144.53	88.00	7.68	2.20	6.11	2.95	4.75	3.70	11.19	5.80

C3 FR

	Ag_ppm		Au_ppm		Cu_pc		Pb_pc		Zn_pc	
	UnCap	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*
Top Cap		*105*		*0.5*		*2.4*		*2.6*		*4.8*
Base Cap		*0.5*		*0.005*		*0.0025*		*0.0025*		*0.01*
Nº of Top caps		*3*		*2*		*2*		*2*		*5*
Nº of Base caps		-		*9*		*43*		-		*12*
Count	451	451	451	451	451	451	451	451	451	451
Mean	8.25	8.11	0.03	0.03	0.22	0.21	0.23	0.23	0.50	0.49
Variance	263.0	232.0	0.01	0.00	0.28	0.16	0.17	0.16	0.82	0.71
Minimum	0.50	0.50	0.000	0.005	0.001	0.003	0.003	0.003	0.01	0.01
Maximum	133.50	105.00	1.80	0.50	7.78	2.40	3.45	2.60	6.82	4.80

C1 sMS

	Ag_ppm		Au_ppm		Cu_pc		Pb_pc		Zn_pc	
	UnCap	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*
Top Cap		*41*		*0.7*		*1.42*		*0.62*		*1.5*
Base Cap		*0.5*		*0.005*		*0.0025*		*0.0025*		*0.01*
Nº of Top caps		*2*		*5*		*3*		*3*		*2*
Nº of Base caps						*3*		*2*		*8*
Count	265	265	265	265	265	265	265	265	265	265
Mean	7.40	7.33	0.10	0.10	0.20	0.20	0.08	0.08	0.20	0.19
Variance	64.3	58.7	0.021	0.018	0.077	0.065	0.032	0.015	0.110	0.081
Minimum	0.50	0.50	0.005	0.005	0.000	0.003	0.000	0.003	0.000	0.010
Maximum	54.69	41.00	0.94	0.70	2.31	1.42	1.98	0.62	2.70	1.50

C2 sMS

	Ag_ppm		Au_ppm		Cu_pc		Pb_pc		Zn_pc	
	UnCap	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*	*UnCap*	*Cap*
Top Cap		*32.5*		*0.76*		*1.95*		*0.4*		*1.5*
Base Cap		*0.5*		*0.005*		*0.0025*		*0.0025*		*0.01*
Nº of Top caps		*1*		*3*		*2*		*2*		*2*
Nº of Base caps		-				*3*		*3*		*13*
Count	149	149	149	149	149	149	149	149	149	149
Mean	6.47	6.43	0.09	0.08	0.25	0.24	0.06	0.05	0.25	0.25
Variance	36.5	34.7	0.035	0.025	0.137	0.106	0.012	0.007	0.116	0.107
Minimum	0.50	0.50	0.005	0.005	0.000	0.003	0.000	0.003	0.000	0.010
Maximum	37.10	32.50	1.17	0.76	2.60	1.95	0.83	0.40	1.75	1.50

14.5.2 VARIOGRAPHY

Micon completed a geostatistical analysis of all domains to determine the optimum grade interpolation parameters. Conventional variograms were applied at Gos, tMS and pMS and correlograms were computed for the str domain and all the south sector domains. Examples of the variograms and correlograms are shown in Figure 14.9 to Figure 14.13, which clearly demonstrate how the search ellipse ranges where determined. Note that the more economically important metals per domain were the drivers of the chosen ranges. Nugget effects were calculated from the downhole variogram, and the sill locked at 1 for variogram and 0 for correlogram.

The similar ranges of influence along the major and semi-major directions confirm the isotropic nature of the tMS, pMS and str domains.

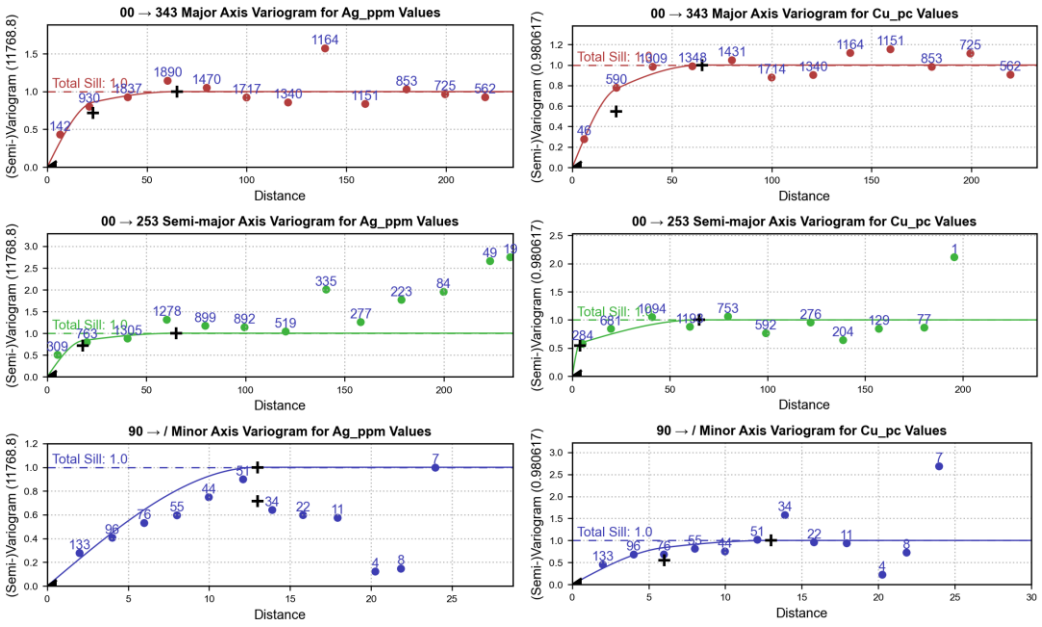


Figure 14.9 - North deposit Gos domain Ag and Au variograms

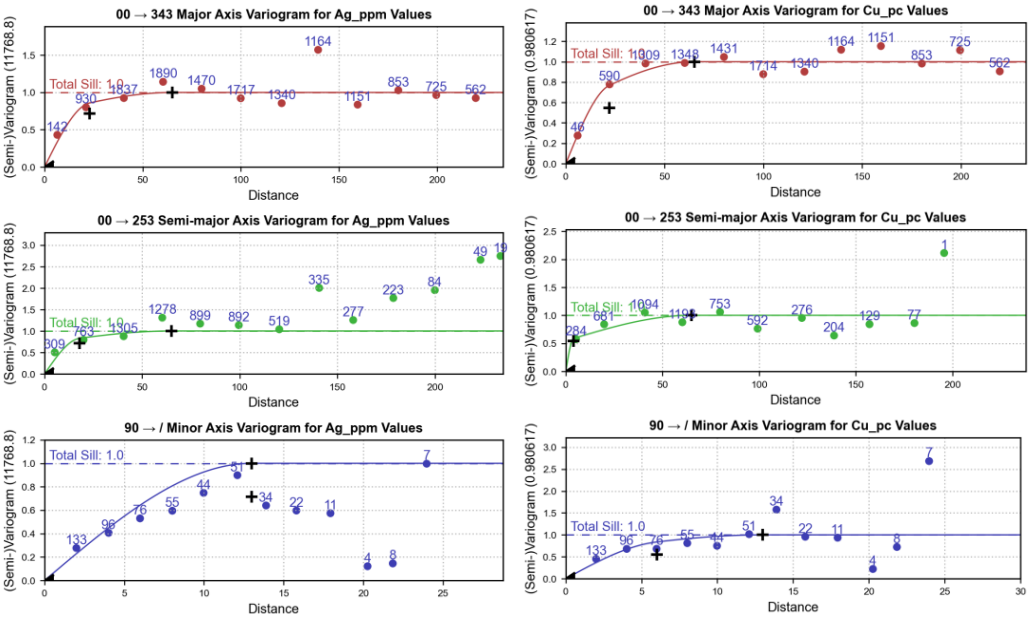


Figure 14.10 - North deposit tMS domain Ag and Cu variograms

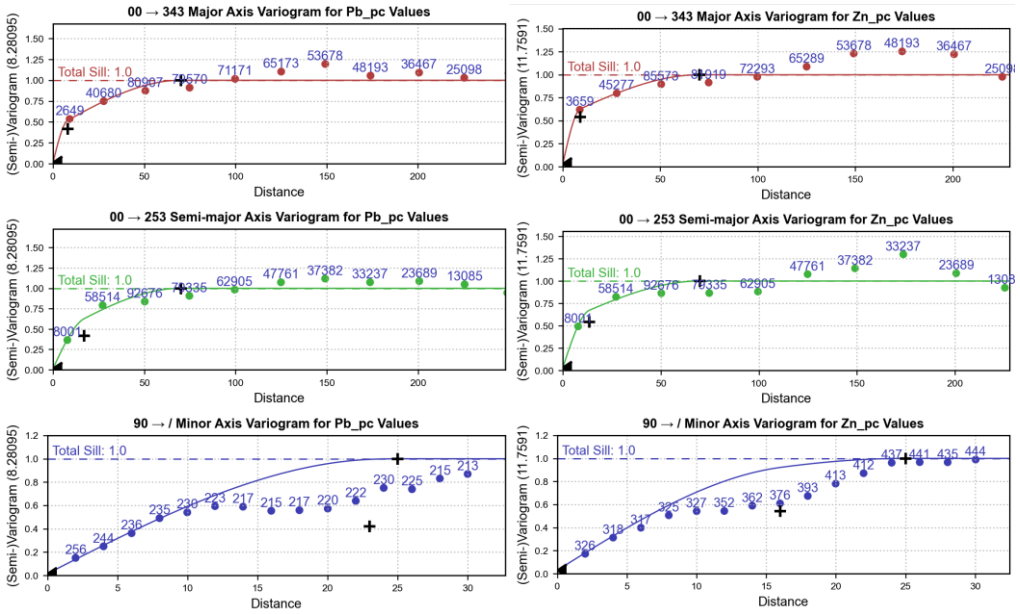


Figure 14.11 - North deposit pMS domain Pb and Zn variograms

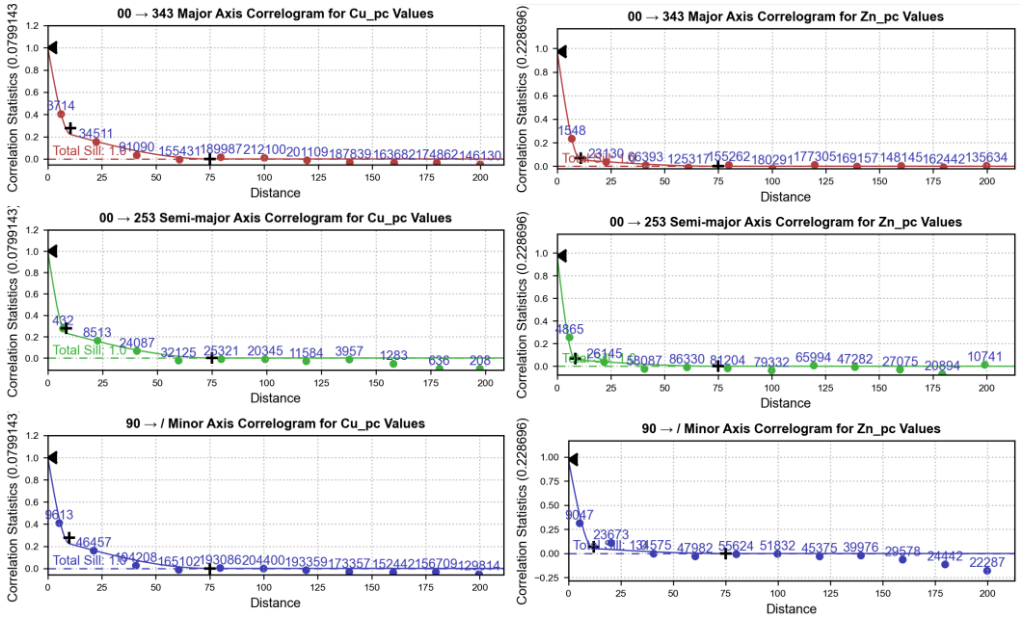


Figure 14.12 - North deposit str domain Cu and Zn correlograms

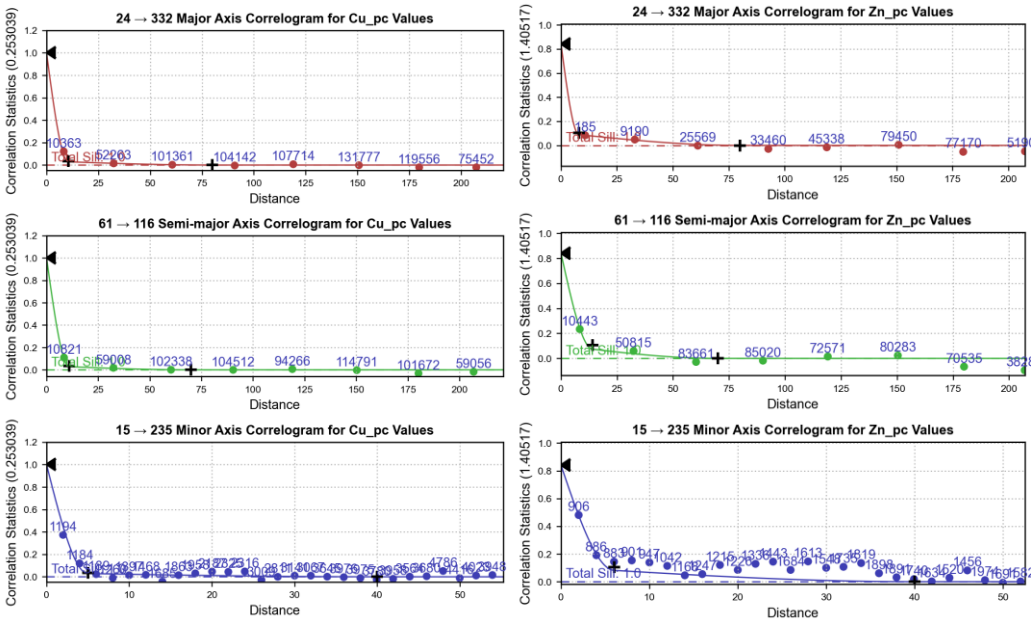


Figure 14.13 - South deposit C1 FR domain Zn and Cu correlograms

14.6 ESTIMATION

14.6.1 BLOCK SIZE ANALYSIS

Sensitivity analysis on block size was performed to select the most appropriate block size for the estimation. The method involved running multiple ordinary kriging (OK) estimations on zinc in the pMS Domain with different block sizes and then comparing kriging efficiency and slope of regression. The optimal size is the one which shows the highest kriging efficiency coupled with the highest slope of regression. Zinc was chosen because it is the primary component of the deposit whilst the pMS is the most significant domain on the LS property. The results indicate an optimum block size of 5 x 10 x 5 m, or 10x20x10 as demonstrated in Figure 14.14 below. The 5 x 10 x 5 m block size was selected as it honours better the local variability of the composites and mining shapes coherency.

14.6.2 RESOURCE BLOCK MODEL DEFINITION

The north deposit block model attributes are presented in Table 14.3. The upper limit (Z) is approximately at the Surface. The block size and definition are based on the sensitivity analysis (Figure 14.14), drillhole spacing, the envisaged selective mining unit (SMU) and the geometry of the deposit. The Azimuth of the Block Model follows the main geometry of the mineralized domains.

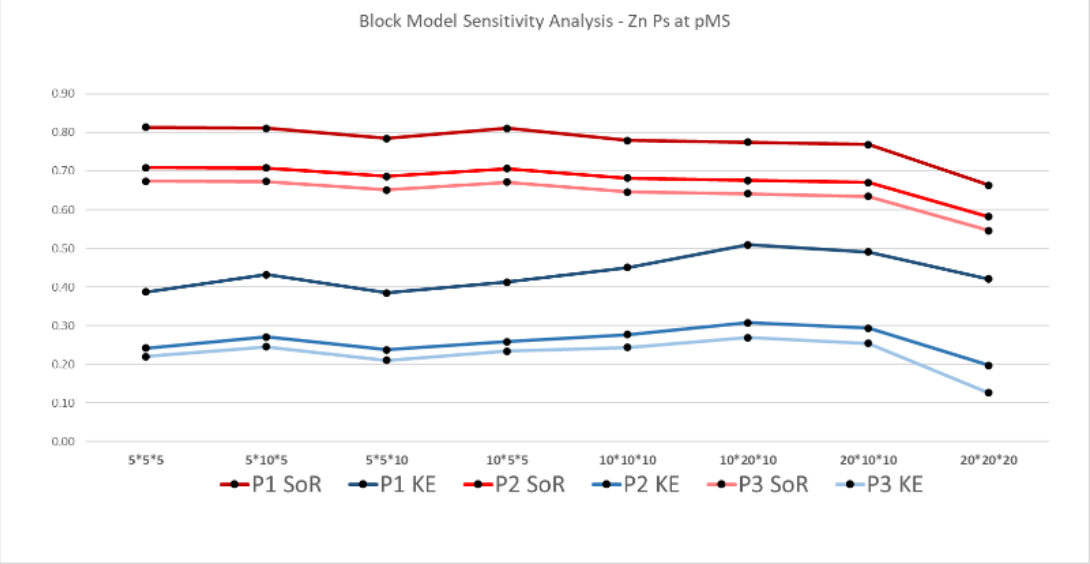


Figure 14.14 - North deposit pMS domain block model sensitivity analysis

Table 14.3 - LS north deposit block model definition

Item	X	Y	Z
Base point	546907	4232486	110
Extents	425	690	600
Block Size (m)	5	10	5
Sub-block minimum size (m)	2.5	5	1
Azimuth		163	

The south deposit block model attributes are presented in Table 14.4 - LS South Deposit Block Model Definition. The block size is based on the sensitivity analysis shown in Figure 14.14. The Azimuth of the Block Model follows the main geometry of the mineralized domains.

Table 14.4 - LS South deposit block model definition

Item	X	Y	Z
Base point	547310	4231975	110
Extents	690	1000	755
Block Size (m)	5	10	5
Sub-block minimum size (m)	2.5	5	2.5
Azimuth		150	

14.6.3 DENSITY

Bulk density measurements were conducted regularly by the company on site in previous campaigns. However, during the 2021/2022 drilling campaign, systematic certified S.G. (specific gravity) determinations were carried out at the ALS Lab involving a total of 358 samples from the North Sector and 914 samples from the South Sector. Figure 14.15 shows the spatial location of all the ALS Lab certified S.G. measurements that were used to populate the blocks with density data.

The density was estimated using ID3 for all the domains except for the Gossan (Gos) domain where a R-sq of 89.5% regression was achieved only using Fe. Figure 14.16 displays the *four-in-one* residual plots for the Gos regression.

$$Gos\ SG\ Regression = 2.0831 + (0.03003 \times Fe\%)$$

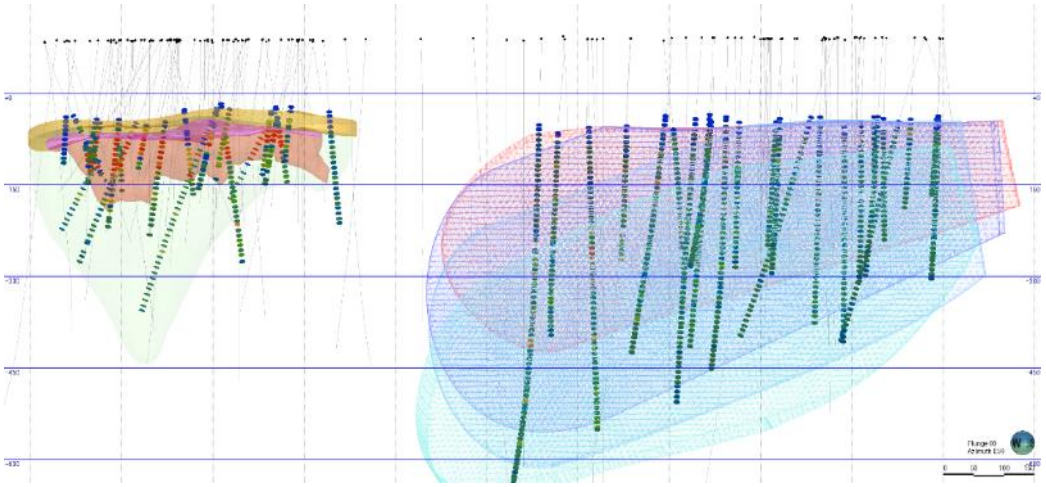


Figure 14.15 - North deposit Gos domain residual plots of specific gravity regression

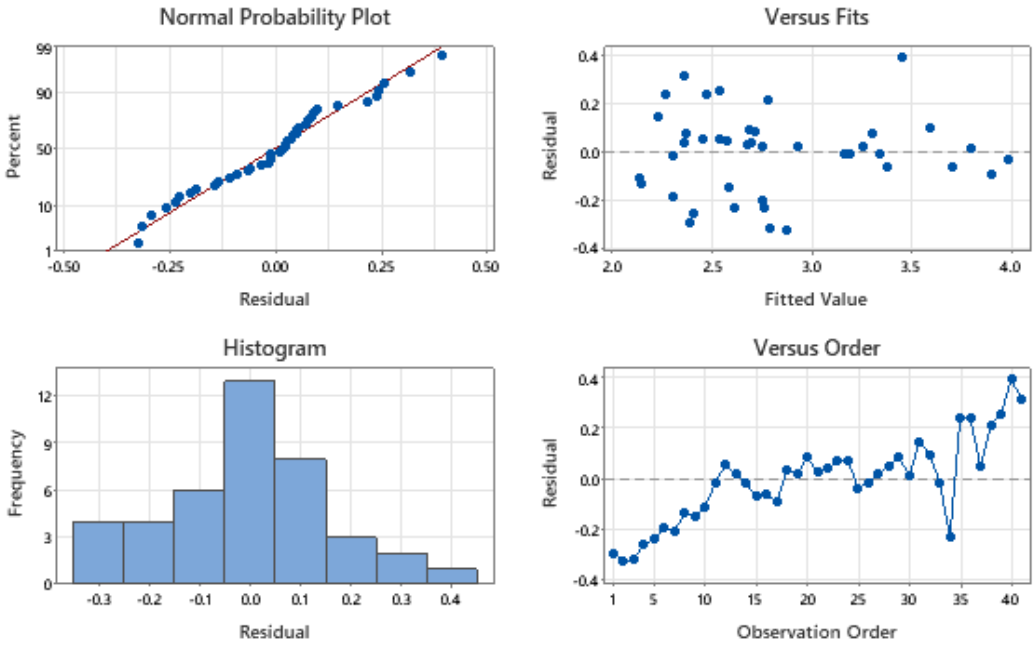


Figure 14.16 - North deposit Gos domain residual plots of specific gravity regression

14.6.4 SEARCH PARAMETERS

14.6.4.1 NORTH DEPOSIT

The search ellipse configurations were defined using variography as a guide, combined with the geometry of the deposit and average drillhole spacing. A 3-pass estimation procedure for all domains was used for the interpolation. For the passes, the maximum and minimum number of samples per drillhole was set to control the number of drillholes in the interpolation. The search parameters adopted for grade interpolation are summarized in Table 14.5. The variogram ranges of the key metals of each domain were the guidelines of the resource category. Pass 1 equals 40% of the variogram range; Pass 2 equals variogram range; Pass 3 equals variogram range x 1.5. The same parameters and ranges were used for the different interpolation methods (OK, ID3 and NN).

Table 14.5 - Summary of search parameters for the north deposit

	Pass	Dip (º)	Dip Az. (º)	Pitch (º)	Major (m)	Semi (m)	Minor (m)	Minimum samples	Maximum samples	Samples p/ hole	Minimum Holes
Gos	1	0	163	90	34	20	8	12	18	3	4
	2	0	163	90	85	50	20	9	15	3	3
	3	0	163	90	127.5	75	30	6	12	3	2
tMS	1	0	163	90	26	26	5.2	12	18	3	4
	2	0	163	90	65	65	13	9	15	3	3
	3	0	163	90	97.5	97.5	19.5	6	12	3	2
pMS	1	0	163	90	28	28	10	12	18	3	4
	2	0	163	90	70	70	25	9	15	3	3
	3	0	163	90	105.0	105	37.5	6	12	3	2
str	1	0	163	90	30	30	30	12	18	3	4
	2	0	163	90	75	75	75	9	15	3	3
	3	0	163	90	112.5	112.5	112.5	6	12	3	2

14.6.4.2 SOUTH DEPOSIT

The search ellipse configurations were defined using variography as a guide, combined with the geometry of the deposit and average drillhole spacing. A 4-pass estimation procedure for all domains was used for the interpolation. For the passes, the maximum and minimum number of samples per drillhole was set to control the number of drillholes in the interpolation. The search parameters are summarized in Table 14.6. Pass 1 equals 40% of variogram range, P2 equals the variogram range, P3 equals 1.5 x variogram range (150% of variogram range) and Pass 4 equals 2 x variogram range (200% variogram range). The same parameters and ranges were used for the different interpolation methods (OK, ID3 and NN).

Table 14.6 - Summary of search parameters for the south deposit

	Pass	Dip (º)	Dip Az. (º)	Pitch (º)	Major (m)	Semi (m)	Minor (m)	Minimum samples	Maximum samples	Samples p/ hole	Minimum Holes
All Corridors	1	75	55	25	32	28	16	12	15	3	4
	2	75	55	25	80	70	40	9	12	3	3
	3	75	55	25	120	105	60	6	9	3	2
	4	75	55	25	160	140	80	6	9	3	2

The variogram ranges chosen for the passes were obtained from Cu and Zn correlograms which are the key potentially economic elements in this deposit. Because of the irregular geometry, variable orientations were used in the search ellipsoids.

14.6.5 Grade Interpolation And Validation

Ordinary kriging (OK) was used for grade interpolation for both the North and South deposit domains. Parallelly, inverse distance cubed (ID3) and nearest neighbour (NN) interpolation techniques were used as validation for the OK block grades.

14.6.5.1 Visual Validation of the OK Block Grades

The model blocks and the drillhole intercepts were reviewed interactively in 3D mode to ensure that the blocks were honouring the drillhole data. The agreement between the block grades and the drill intercepts of the deposits was found to be satisfactory. An example of the North deposit is shown in Figure 14.17 and an example of the South deposit is shown in Figure 14.18.

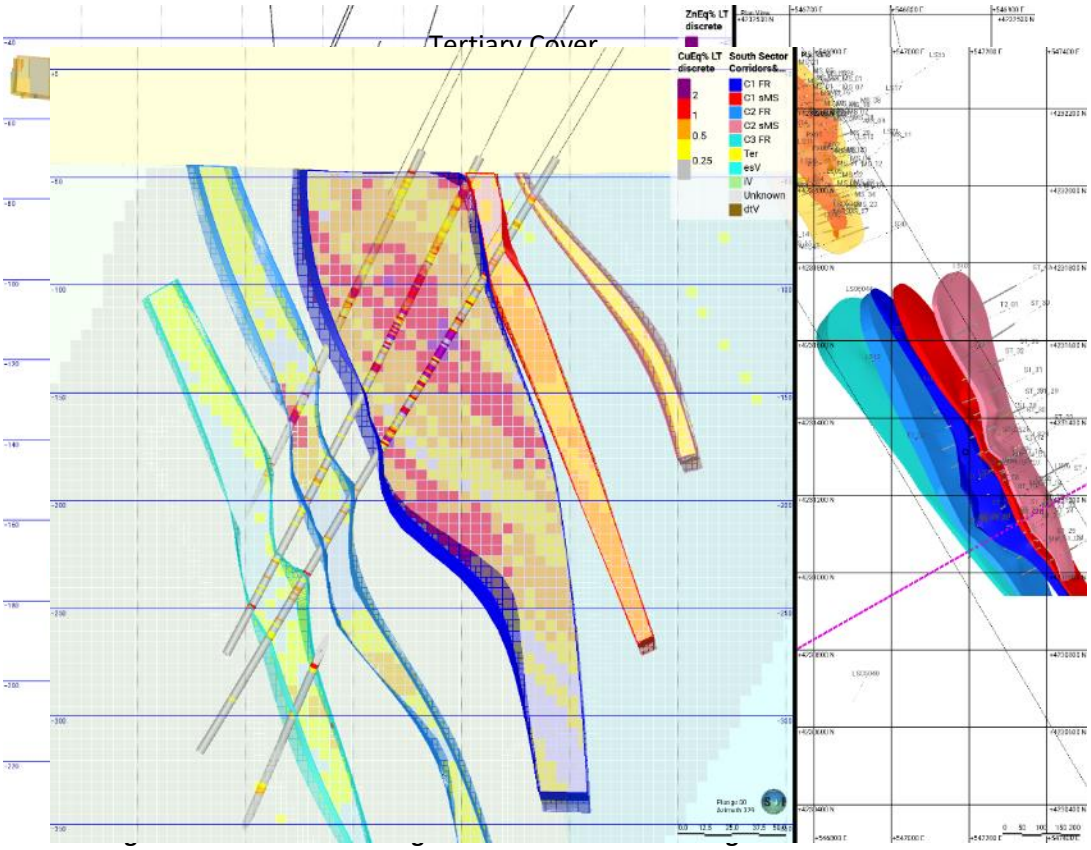


Figure 14.18 - Section through the south deposit sowing the match between block and assays grade

14.6.5.2 VALIDATION OF OK BLOCK GRADES BY SWATH PLOTS

Validation using swath plots produced satisfactory results. In all cases, a satisfactory overall match is reflected between block grades of OK, ID3, NN and composites. For the swath plots shown in Figure 14.19 to Figure 14.24 below, the blocks volume from OK are represented with blue bars, NN average values are represented in Red, ID3 in Orange, and composites in Black.

14.6.5.3 VALIDATION BY GLOBAL STATISTICS

Complementary validation was conducted by comparing global statistics of the block grades utilizing the ordinary kriging (OK) versus the capped composites. The South deposit still has the bulk of its blocks within the Inferred resources; thus, to have a proper comparison, Micon used the P1 and P2 blocks and composites from C1 FR, which contains most of the resource from the South deposit. The results for all the potential economic metals shown in Table 14.7 indicate a positive match with the original OK method used for the estimate.

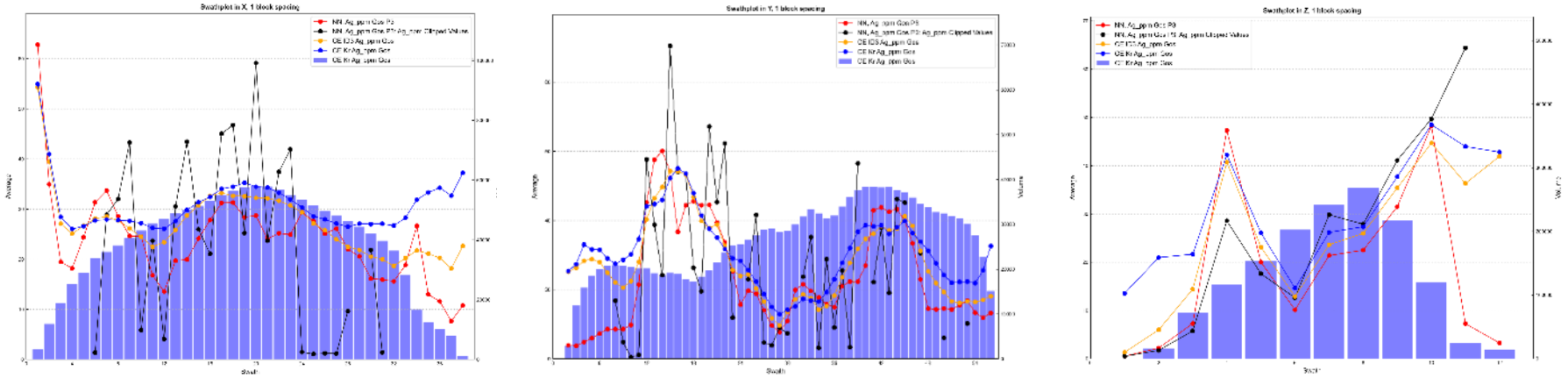


Figure 14.19 - North deposit GO domain Ag swath plots

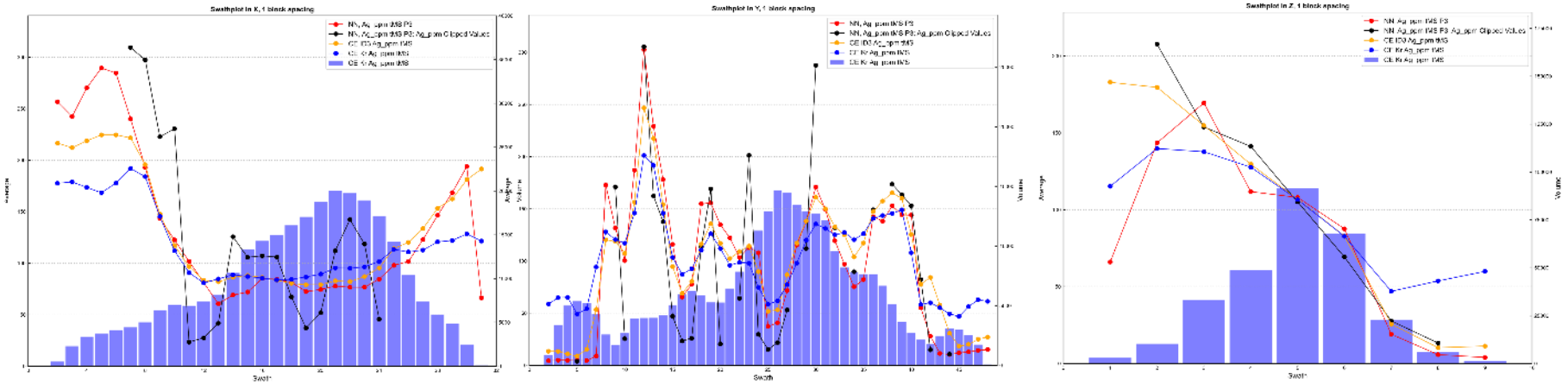


Figure 14.20 - North deposit tMS domain Au swath plots

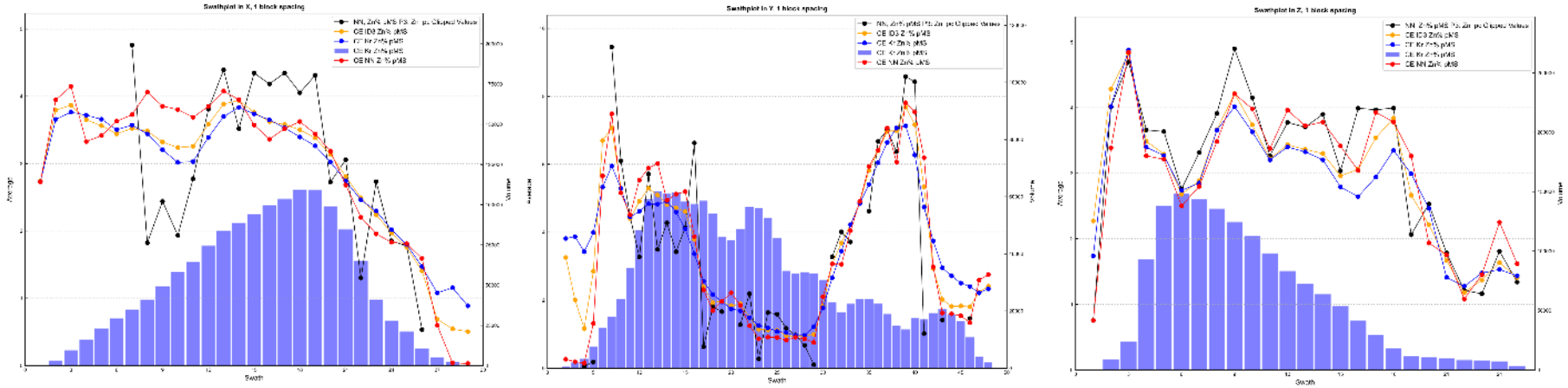


Figure 14.21 - North deposit pMS domain Zn swath plots

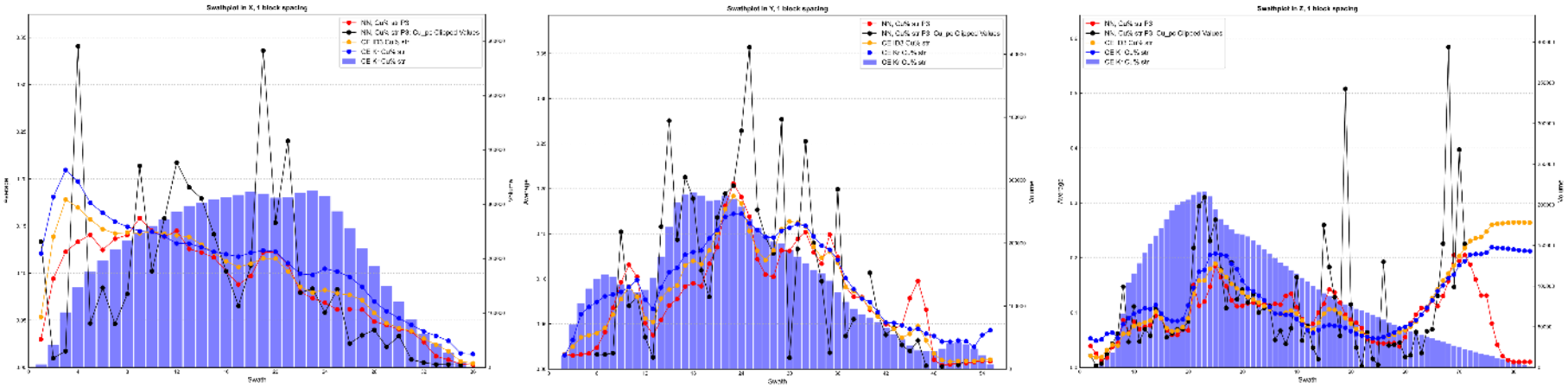


Figure 14.22 - North deposit str domain Cu swath plots

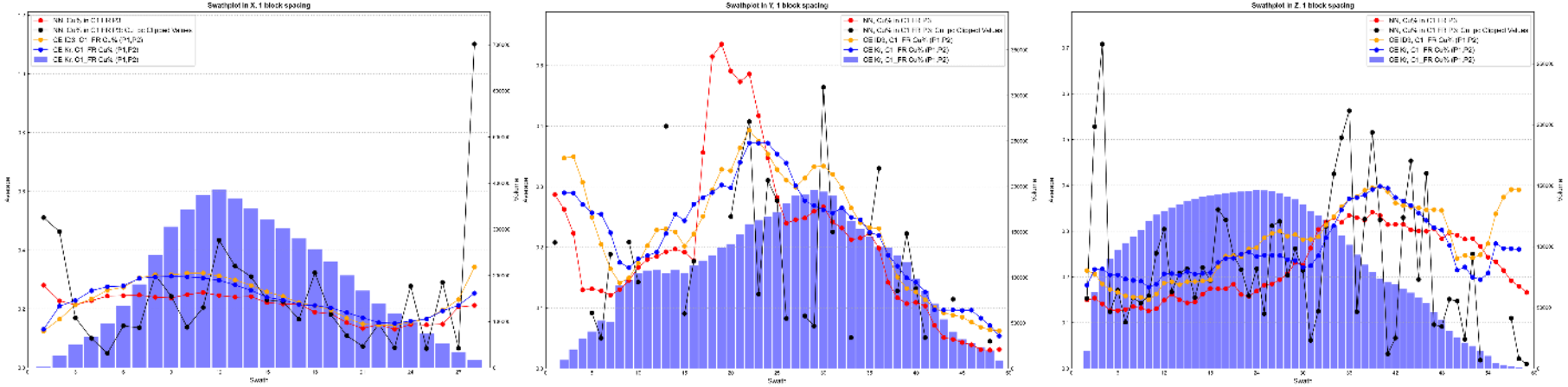


Figure 14.23 - South deposit C1 FR (P1, P2) domain Cu swath plots

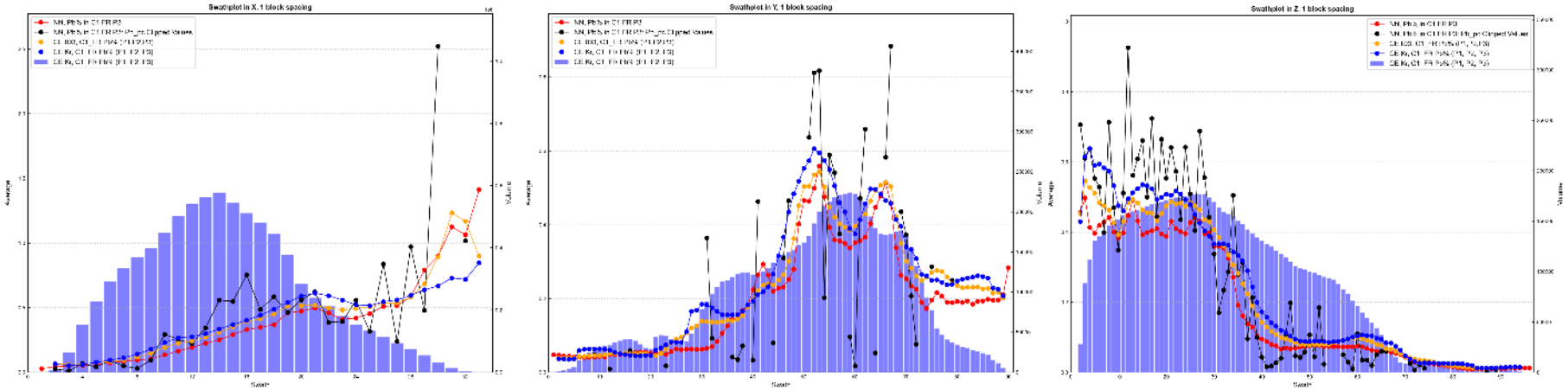


Figure 14.24 - South deposit C1 FR domain Pb swath plots

Table 14.7 - Comparison between ID3, NN, and OK estimation results for Zn in the MS domain

Gos		
Ag	*Cap Composites*	*Block Model (OK)*
Count	570	15734
Mean	29.2	30.0
Variance	3227.0	785.1
Minimum	0.50	0.70
Maximum	275.00	199.94

Au	*Cap Composites*	*Block Model (OK)*
Count	570	15734
Mean	0.56	0.51
Variance	1.48	0.30
Minimum	0.01	0.01
Maximum	7.00	5.14

Cu	*Cap Composites*	*Block Model (OK)*
Count	570	15734
Mean	0.09	0.09
Variance	0.02	0.00
Minimum	0.01	0.01
Maximum	1.20	0.55

Pb	*Cap Composites*	*Block Model (OK)*
Count	570	15734
Mean	1.74	1.60
Variance	8.0	1.9
Minimum	0.01	0.03
Maximum	14.00	9.38

Sn	*Cap Composites*	*Block Model (OK)*
Count	409	15646
Mean	0.16	0.14
Variance	0.1	0.0
Minimum	0.01	0.01
Maximum	1.40	0.99

Zn	*Cap Composites*	*Block Model (OK)*
Count	570	15734
Mean	0.47	0.45
Variance	0.1	0.0
Minimum	0.05	0.12
Maximum	2.00	1.55

tMS		
Ag	*Cap Composites*	*Block Model (OK)*
Count	179	4995
Mean	101.7	103.4
Variance	11768.8	3183.6
Minimum	0.50	11.09
Maximum	430.00	359.43

Au	*Cap Composites*	*Block Model (OK)*
Count	179	4995
Mean	1.05	1.12
Variance	1.55	0.47
Minimum	0.01	0.11
Maximum	5.00	3.80

Cu	*Cap Composites*	*Block Model (OK)*
Count	179	4995
Mean	0.97	0.99
Variance	0.98	0.18
Minimum	0.01	0.12
Maximum	4.00	2.74

Pb	*Cap Composites*	*Block Model (OK)*
Count	179	4995
Mean	2.19	2.16
Variance	6.5	1.5
Minimum	0.07	0.18
Maximum	12.00	8.23

Sn	*Cap Composites*	*Block Model (OK)*
Count	117	4949
Mean	0.17	0.17
Variance	0.0	0.0
Minimum	0.01	0.00
Maximum	0.65	0.60

Zn	*Cap Composites*	*Block Model (OK)*
Count	179	4995
Mean	0.26	0.26
Variance	0.1	0.0
Minimum	0.01	0.00
Maximum	1.95	1.59

pMS		
Ag	*Cap Composites*	*Block Model (OK)*
Count	1147	11603
Mean	61.6	58.9
Variance	3231.6	1248.3
Minimum	0.50	5.61
Maximum	280.00	268.30

Au	*Cap Composites*	*Block Model (OK)*
Count	1147	11603
Mean	0.65	0.61
Variance	0.54	0.20
Minimum	0.01	0.01
Maximum	3.60	3.14

Cu	*Cap Composites*	*Block Model (OK)*
Count	1147	11603
Mean	0.29	0.29
Variance	0.03	0.01
Minimum	0.00	0.04
Maximum	1.45	1.07

Pb	*Cap Composites*	*Block Model (OK)*
Count	1147	11603
Mean	2.64	2.43
Variance	8.3	3.4
Minimum	0.01	0.07
Maximum	12.00	8.75

Sn	*Cap Composites*	*Block Model (OK)*
Count	853	11524
Mean	0.14	0.12
Variance	0.0	0.0
Minimum	0.01	0.02
Maximum	0.43	0.33

Zn	*Cap Composites*	*Block Model (OK)*
Count	1147	11603
Mean	3.58	3.25
Variance	11.8	4.6
Minimum	0.01	0.09
Maximum	14.50	10.81

str		
Ag	*Cap Composites*	*Block Model (OK)*
Count	2172	55127
Mean	8.1	7.5
Variance	129.4	23.3
Minimum	0.50	0.64
Maximum	85.00	57.27

Au	*Cap Composites*	*Block Model (OK)*
Count	2172	55127
Mean	0.05	0.04
Variance	0.01	0.00
Minimum	0.00	0.01
Maximum	0.65	0.46

Cu	*Cap Composites*	*Block Model (OK)*
Count	2172	55127
Mean	0.14	0.12
Variance	0.16	0.01
Minimum	0.00	0.00
Maximum	8.69	1.68

Pb	*Cap Composites*	*Block Model (OK)*
Count	2172	55127
Mean	0.13	0.12
Variance	0.0	0.0
Minimum	0.00	0.00
Maximum	1.37	1.12

Sn	*Cap Composites*	*Block Model (OK)*
Count	1579	51934
Mean	0.03	0.03
Variance	0.0	0.0
Minimum	0.01	0.00
Maximum	0.42	0.38

Zn	*Cap Composites*	*Block Model (OK)*
Count	2172	55127
Mean	0.48	0.46
Variance	0.2	0.1
Minimum	0.00	0.02
Maximum	2.45	1.77

C1 FR (P1, P2)		
Ag	*Cap Composites*	*Block Model (OK)*
Count	1318	30995
Mean	8.9	8.9
Variance	220.1	31.2
Minimum	0.50	0.85
Maximum	93.00	47.42

Au	*Cap Composites*	*Block Model (OK)*
Count	1318	30995
Mean	0.05	0.05
Variance	0.00	0.00
Minimum	0.01	0.01
Maximum	0.44	0.24

Cu	*Cap Composites*	*Block Model (OK)*
Count	1318	30995
Mean	0.24	0.25
Variance	0.27	0.04
Minimum	0.00	0.01
Maximum	3.50	1.82

Pb	*Cap Composites*	*Block Model (OK)*
Count	1318	30995
Mean	0.51	0.46
Variance	0.7	0.1
Minimum	0.00	0.01
Maximum	5.80	2.81

Zn	*Cap Composites*	*Block Model (OK)*
Count	1318	30995
Mean	0.90	0.82
Variance	1.5	0.3
Minimum	0.01	0.03
Maximum	6.50	4.20

14.6.5.4 OVERALL COMMENTS AND SENSITIVITY TABLES

All the three methods used to validate block grade estimation supported the estimation results. Figure 14.25 presents the North and South deposit mineral inventory's sensitivity to cut-off grade for ZnEq% and CuEq%, respectively.

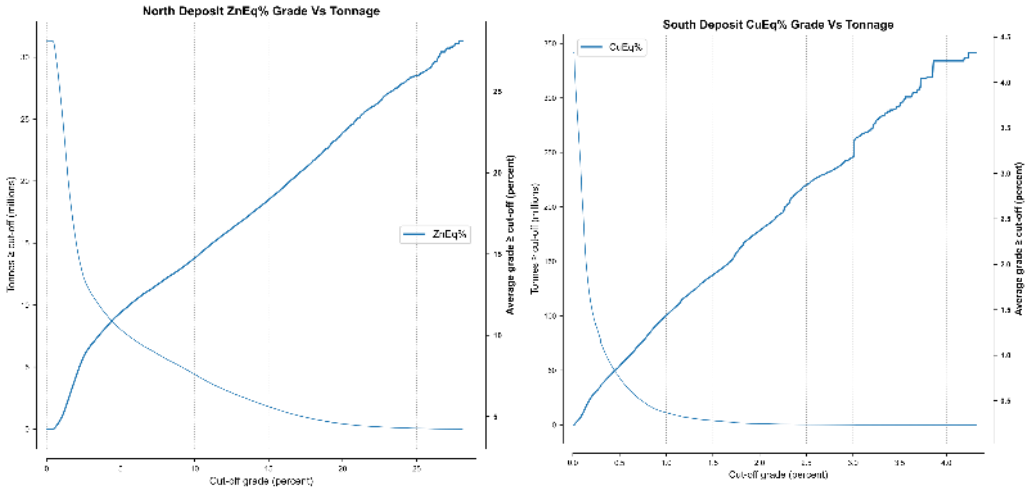


Figure 14.25 - LS Project – North and South deposit *Grade Tonnage* graphs

14.7 MINERAL RESOURCE PARAMETERS AND REPORT

14.7.1 PROSPECTS FOR ECONOMIC EXTRACTION

The CIM Definition Standards (2014) require that a Mineral Resource must have reasonable prospects for eventual economic extraction.

The forecasted long term metal commodity prices are zinc = $2,646/t, lead = $2,205/t, copper = $7,716/t, gold = $1,700/oz, silver = $22.00/oz, and tin = $26,455/t (shown in Table 14.8). While for the *grade-shells* definition and estimations overall validation, the metal equivalency was the guideline, for the resource report a NSR based on the metallurgical recoveries and metal quotations assumptions was used instead: NSR-MRE (mineral resource estimate NSR). These recovery assumptions are summarized in Table 14-9 and the economic assumptions in Table 14.10.

Based on the Table 14.9, economic and recovery assumptions obtained by the other technical disciplines, a differential NSR formula was developed per each domain, where the overall recovery of each pay metal was considered. For the mineral resource estimate reporting, the treatment costs, refining costs, penalties, and deductions were not accounted for.

The different NSR-MRE formulas applied per block for each domain are summarized bellow:

$$\boldsymbol{Gos} = (Sn\% \times 0.40) \times Sn\$ + (Pb\% \times 0.20) \times Pb\$ + (Au\ ppm \times 0.91) \times Au\$ + (Ag\ ppm \times 0.89) \times Ag\$$$

$$\boldsymbol{tMS} = (Sn\% \times 0.40) \times Sn\$ + (Pb\% \times 0.50) \times Pb\$ + (Ag\ ppm \times 0.65) \times Ag\$ + (Zn\% \times 0.50) \times Zn\$$$

$$\boldsymbol{pMS} = (Sn\% \times 0.40) \times Sn\$ + (Pb\% \times 0.60) \times Pb\$ + (Ag\ ppm \times 0.45) \times Ag\$ + (Zn\% \times 0.70) \times Zn\$$$

$$\boldsymbol{str} = (Sn\% \times 0.60) \times Sn\$ + (Pb\% \times 0.50) \times Pb\$ + (Ag\ ppm \times 0.40) \times Ag\$ + (Zn\% \times 0.70) \times Zn\$$$

$$\boldsymbol{South\ Deposit} = (Cu\% \times 0.65) \times Cu\$ + (Zn\% \times 0.75) \times Zn\$ + (Pb\% \times 0.65) \times Pb\$ + (Au\ ppm \times 0.05) \times Au\$ + (Ag\ ppm \times 0.80) \times Ag\$$$

A formula of NSR-MRE was applied per each block of each domain and the specific economic assumptions per domain were applied as resource cut-off from the sum of G&A, Mining and Processing costs.

Table 14.8 - Summary of the metal prices assumptions at the LS project

Metal	Value used	Unit
Zn %	2,646	$/t
Cu %	7,716	$/t
Pb %	2,205	$/t
Sn %	26,455	$/t
Ag %	22,0	$/oz
Au %	1,700	$/oz

Table 14.9 - Summary of recovery assumptions per domain at the LS project

	Cu %	Pb %	Zn %	Ag %	Au %	Sn %
Gossan		20		89	91	40
Transition MS		50	50	65		40
Fresh MS		60	70	45		40
Stringer	60	50	70	40		
South Deposit	65	65	75	80	5	

Table 14.10 - Summary of economic assumptions for the underground mine at the LS project

	Gos	tMS	pMS	str	South Deposit
G&A	$2.50	$2.50	$2.50	$2.50	$2.50
Mining	$29.98	$29.98	$19.50	$19.50	$19.50
Processing	$15.76	$14.06	$19.06	$15.83	$15.83
Total	**$48.24**	**$46.54**	**$41.06**	**$37.83**	**$37.83**

14.7.2 CLASSIFICATION OF THE MINERAL RESOURCE

14.7.2.1 NORTH DEPOSIT

The geological and grade continuity of about 27% of the North deposit is good enough to deserve being categorized into the Measured class, about 67.5% into the Indicated category and the balance into the Inferred. A plan view of the resource categorization and the search ellipse pass is shown in Figure 14.26.

Measured resources are informed by the P1; Indicated resources are informed by P2 and Inferred resources by P3/P4.

The gossan, transition and primary sulphide domains in the north deposit show notable continuity and coherence. Stope optimization carried out for the reserve determination (Next Chapter) shows that the geometry shapes under the constraining economic factors can be mostly integrated into stopes thus demonstrating reasonable prospects for eventual economic extraction for the bulk of these domains. For the stringer domain the geometry of the blocks above COG was assessed visually determining that they show the minimum size suitable for stopes and proximity to planned UG development. A few str isolated blocks are recognized as being nonmaterial for this estimate.

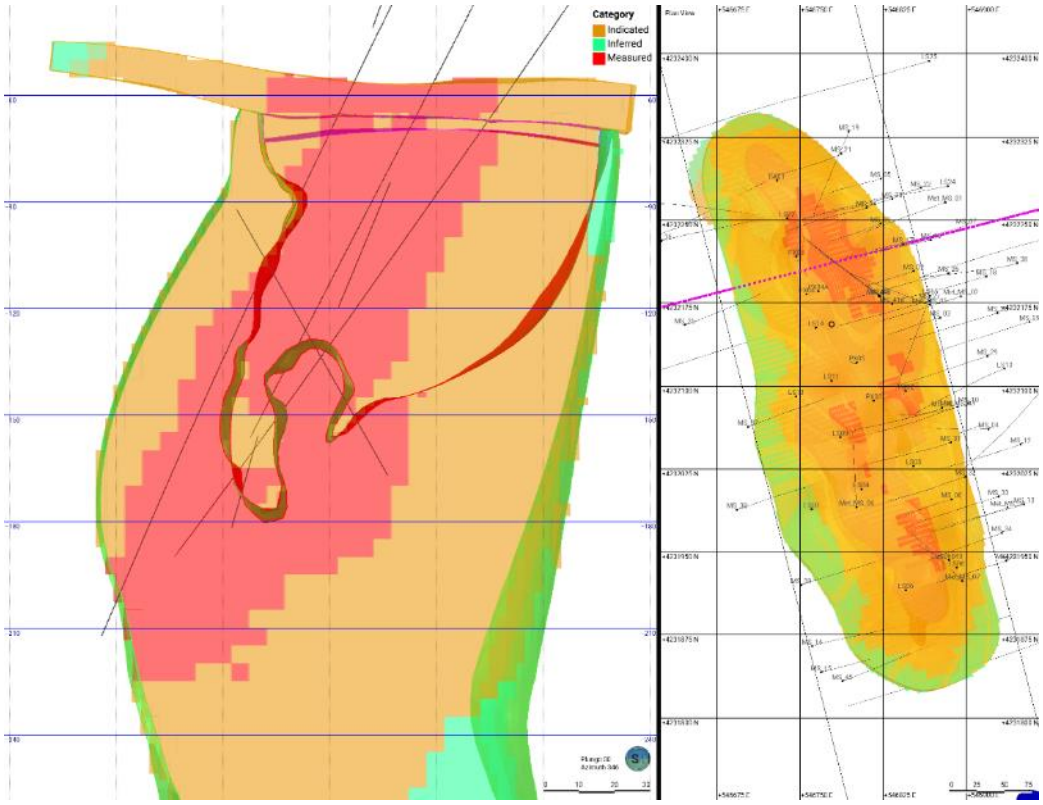


Figure 14.26 - Plan view of the north deposit showing mineral resource classification

14.7.2.2 SOUTH DEPOSIT

Micon has classified the south deposit mineral resource estimate in the Indicated and Inferred categories. Both conclusive metallurgical tests and infill drilling are required to upgrade the resource into the Measured class. A 3D perspective view of the resource categorization is shown in Figure 14.27. The approach used to categorize the Indicated Resource was to select those blocks in P1 and P2 interpolation and P3 plus P4 for the Inferred resources.

The results were then smoothed to remove isolated small blocks and produce coherent shapes of reasonable volume, eliminating the spotted dog effect. All other blocks (Passes 3 and 4) were classified in the Inferred category. The contribution from Pass 4 is minor (<5%).

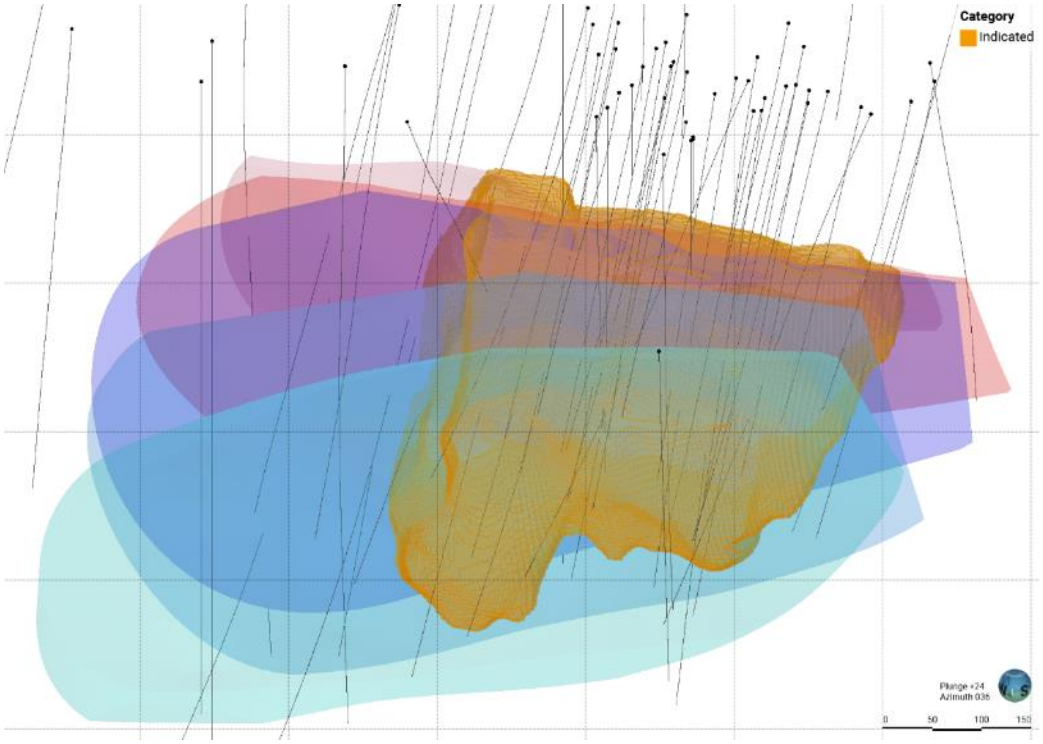


Figure 14.27 - 3D perspective of the south deposit resource classification

In order to guarantee that the resource displayed coherent and contiguous mining shapes backing reasonable prospect for economic extraction, a numerical model of the blocks above the costs of mining, processing and G&A (37.83$) was developed to create meshes to harmonize the report of the NSR Resources. All meshes under the minimum stope size of 3000m^3 were discarded. At the Indicated portion of the model, the constrained shapes created with the numerical model mimic the shapes of the LOM stopes. The sMS domains were not considered for the Resource report as the volume above NSR-MRE is minor and marginal within the reasonable prospect for economic extraction.

14.7.3 MINERAL RESOURCE STATEMENT

The mineral resources for the LS project North and South deposits are summarized in Table 14.11 and Table 14.12, respectively. The overall LS project resource estimate is displayed at Table 14.13. The QP considers that the resource estimate for the LS project has been reasonably prepared and conforms to the current CIM standards and definitions for estimating Mineral Resources.

Table 14.11 - LS Property Mineral Resource estimate of the North Deposit as of May 23, 2023

Deposit	Domain	Category	Density	Mass	NSR-MRE	Zn	Pb	Cu	Sn	Ag	Au	ZnEq	NSR-MRE	Zn	Pb	Cu	Sn	Ag	Au
			g/cm³	Mt	$/t	%	%	%	%	g/t	g/t	%	M$	kt	kt	kt	kt	koz	koz
North	Gos	Measured	3.15	0.39	123.6	0.53	2.96	0.12	0.29	45.0	1.03	9.60	48.8	2.1	11.7	0.5	1.2	571.2	13.0
		Indicated	2.92	1.50	105.2	0.45	2.39	0.10	0.24	44.3	0.82	8.05	157.4	6.7	35.7	1.5	3.7	2 128.9	39.4
		Measured + Indicated	**2.96**	**1.89**	**109.0**	**0.46**	**2.51**	**0.10**	**0.26**	**44.4**	**0.86**	**8.38**	**206.2**	**8.8**	**47.4**	**1.9**	**4.8**	**2 700.1**	**52.5**
		Inferred	2.56	0.35	81.1	0.42	1.67	0.10	0.10	36.9	0.79	5.77	28.6	1.5	5.9	0.4	0.4	418.2	9.0
	tMS	Measured	4.22	0.06	110.5	0.21	1.99	1.09	0.21	138.2	1.09	13.07	6.2	0.1	1.1	0.6	0.1	249.2	2.0
		Indicated	4.05	0.86	107.6	0.26	2.50	1.08	0.20	120.5	1.25	13.28	92.7	2.2	21.5	9.3	1.7	3 341.9	34.6
		Measured + Indicated	**4.06**	**0.92**	**107.7**	**0.25**	**2.47**	**1.08**	**0.20**	**121.6**	**1.24**	**13.27**	**98.9**	**2.3**	**22.7**	**9.9**	**1.8**	**3 591.2**	**36.5**
		Inferred	3.74	0.07	75.6	0.27	2.12	0.84	0.14	74.3	1.41	10.74	5.6	0.2	1.6	0.6	0.1	177.7	3.4
	pMS	Measured	4.47	1.85	164.4	4.37	3.38	0.28	0.14	73.4	0.82	13.13	303.4	80.6	62.4	5.2	2.7	4 353.3	48.9
		Indicated	4.43	3.61	133.1	3.49	2.65	0.27	0.12	63.1	0.66	10.79	480.7	126.2	95.8	9.8	4.5	7 323.8	76.7
		Measured + Indicated	**4.44**	**5.46**	**143.7**	**3.79**	**2.90**	**0.28**	**0.13**	**66.5**	**0.72**	**11.58**	**784.1**	**206.8**	**158.3**	**15.0**	**7.1**	**11 677.1**	**125.6**
		Inferred	4.81	0.06	88.7	2.78	1.47	0.11	0.07	33.6	0.40	6.73	5.2	1.6	0.9	0.1	0.0	62.9	0.7
	str	Measured	2.90	0.26	48.6	0.77	0.22	0.54	0.09	24.2	0.06	4.24	12.7	2.0	0.6	1.4	0.2	203.3	0.5
		Indicated	2.92	0.37	42.5	0.74	0.25	0.43	0.06	20.7	0.06	3.52	15.8	2.8	0.9	1.6	0.2	247.5	0.7
		Measured + Indicated	**2.91**	**0.63**	**45.0**	**0.75**	**0.24**	**0.48**	**0.08**	**22.1**	**0.06**	**3.82**	**28.5**	**4.8**	**1.5**	**3.0**	**0.5**	**450.8**	**1.2**
		Inferred	2.83	0.01	44.8	0.58	0.38	0.46	0.13	29.4	0.20	4.75	0.3	0.0	0.0	0.0	0.0	7.2	0.0
North	**Total**	Measured	3.98	2.56	145.1	3.32	2.96	0.30	0.16	65.4	0.78	11.68	371.1	84.8	75.8	7.7	4.2	5 377.1	64.3
		Indicated	3.80	6.34	117.7	2.17	2.43	0.35	0.16	63.9	0.74	10.06	746.7	137.9	154.0	22.2	10.1	13 042.2	151.4
		Measured + Indicated	**3.85**	**8.90**	**125.6**	**2.50**	**2.58**	**0.34**	**0.16**	**64.4**	**0.75**	**10.52**	**1 117.7**	**222.7**	**229.8**	**29.9**	**14.3**	**18 419.3**	**215.8**
		Inferred	2.86	0.49	80.6	0.68	1.70	0.22	0.11	42.0	0.83	6.62	39.7	3.3	8.4	1.1	0.5	666.0	13.1

Differences may occur in totals due to rounding.

1. Mineral resources unlike mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
2. The mineral resources have been estimated in accordance with the CIM Best Practice Guidelines (2019) and the CIM Definition Standards (2014). Coherent and contiguous mining shapes backing reasonable prospect for economic extraction were considered.
3. Mineralized Zones: Gos=Gossan, tMS= transition massive sulphides pMS= primary massive sulphides, Str=Stringer, FR= -Fissural Remobilizations.
4. ZnEq% = ((Zn Grade*26.46)+(Pb Grade*22.05)+(Cu Grade * 77.16)+(Au Grade*54.66)+(Ag Grade*0.71)+(Sn Grade * 264.55))/26.46
5. Metal Prices: Cu $7,716/t, Zn $2,646/t, Pb $2,205/t, Au $1,700/oz, Ag $22.00/oz, Sn $26,455/t
6. NSR-MRE: Gos= ((Sn%*0.40)*Sn$)+((Pb%*0.20)*Pb$)+((Au ppm*0.91)*Au$)+((Ag ppm*0.89)* Ag$)
   tMS= ((Sn%*0.40)*Sn$)+((Pb%*0.50)*Pb$)+((Ag ppm*0.65)* Ag$)+((Zn% *0.5)*Zn$)
   pMS= ((Sn%*0.40)*Sn$)+((Pb%*0.60)*Pb$)+((Ag ppm*0.45)* Ag$)+((Zn%*0.70)*Zn$)
   str= ((Cu%*0.60)*Cu$)+((Pb%*0.5)*Pb$)+((Ag ppm*0.4)* Ag$)+((Zn%*0.70)*Zn$)
7. NSR-MRE cut-off: Gos= $48.24, tMS= $46.54, pMS= $41.06, str= $37.83

Table 14.12 - LS Property South Deposit Resources as of May 23, 2023

Deposit	Domain	Category	Density g/cm³	Mass Mt	NSR-MRE $/t	Cu %	Zn %	Pb %	Ag g/t	Au g/t	CuEq %	NSR-MRE M$	Cu kt	Zn kt	Pb kt	Ag koz	Au koz
South	C1 FR	**Indicated**	**2.89**	**6.43**	**63.3**	**0.40**	**1.26**	**0.75**	**13.13**	**0.07**	**1.21**	**407.0**	**25.5**	**81.2**	**48.2**	**2 715.4**	**13.7**
		Inferred	3.06	0.83	50.4	0.61	0.47	0.23	12.26	0.13	1.04	42.1	5.1	3.9	1.9	329.0	3.4
	C2 FR	**Indicated**	**2.86**	**2.50**	**67.5**	**0.50**	**1.23**	**0.64**	**15.58**	**0.04**	**1.28**	**168.5**	**12.4**	**30.8**	**16.0**	**1 251.4**	**3.6**
		Inferred	3.01	3.66	62.0	0.54	0.96	0.42	17.32	0.08	1.20	226.6	19.6	35.0	15.5	2 036.7	9.6
	C3 FR	**Indicated**	**2.85**	**1.11**	**62.5**	**0.39**	**1.17**	**0.54**	**20.77**	**0.04**	**1.16**	**69.2**	**4.3**	**13.0**	**6.0**	**740.0**	**1.3**
		Inferred	3.07	3.64	60.1	0.67	0.69	0.22	16.62	0.04	1.16	218.7	24.5	25.1	7.9	1 943.9	5.3
South	**Total**	**Indicated**	**2.88**	**10.04**	**64.2**	**0.42**	**1.24**	**0.70**	**14.58**	**0.06**	**1.22**	**644.7**	**42.2**	**124.9**	**70.2**	**4 706.8**	**18.6**
		Inferred	3.04	8.13	59.9	0.60	0.79	0.31	16.49	0.07	1.16	487.4	49.1	64.0	25.3	4 309.5	18.3

Differences may occur in totals due to rounding.

1. Mineral resources unlike mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
2. The mineral resources have been estimated in accordance with the CIM Best Practice Guidelines (2019) and the CIM Definition Standards (2014). Coherent and contiguous mining shapes backing reasonable prospect for economic extraction were considered.
3. Mineralized Zones: Gos=Gossan, tMS= transition massive sulphides pMS= primary massive sulphides, Str=Stringer, FR= -Fissural Remobilizations.
4. CuEq% = ((Zn Grade*26.46)+(Pb Grade*22.05)+(Cu Grade * 77.16)+(Au Grade*54.66)+(Ag Grade*0.71)+(Sn Grade * 264.55))/77.16
5. Metal Prices: Cu $7,716/t, Zn $2,646/t, Pb $2,205/t, Au $1,700/oz, Ag $22.00/oz, Sn $26,455/t
6. NSR-MRE: South Deposit= ((Cu%*0.65)*Cu$)+((Zn%*0.75)*Zn$)+((Pb%*0.65)*Pb$)+((Au ppm*0.05)*Au$)+((Ag ppm*0.80)*Ag$)
7. NSR-MRE cut-off: South Deposit= $37.83

Table 14.13 - LS Property Overall North and South Deposit Resources as of May 23, 2023

Deposit	Domain	Category	Density g/cm³	Mass Mt	NSR-MRE $/t	Zn %	Pb %	Cu %	Sn %	Ag g/t	Au g/t	ZnEq %	NSR-MRE M$	Zn kt	Pb kt	Cu kt	Sn kt	Ag koz	Au koz
North	Gos	**Measured + Indicated**	**2.96**	**1.89**	**109.0**	**0.46**	**2.51**	**0.10**	**0.26**	**44.4**	**0.86**	**8.38**	**206.2**	**8.8**	**47.4**	**1.9**	**4.8**	**2 700.1**	**52.5**
		Inferred	2.56	0.35	81.1	0.42	1.67	0.10	0.10	36.9	0.79	5.77	28.6	1.5	5.9	0.4	0.4	418.2	9.0
	tMS	**Measured + Indicated**	**4.06**	**0.92**	**107.7**	**0.25**	**2.47**	**1.08**	**0.20**	**121.6**	**1.24**	**13.27**	**98.9**	**2.3**	**22.7**	**9.9**	**1.8**	**3 591.2**	**36.5**
		Inferred	3.74	0.07	75.6	0.27	2.12	0.84	0.14	74.3	1.41	10.74	5.6	0.2	1.6	0.6	0.1	177.7	3.4
	pMS	**Measured + Indicated**	**4.44**	**5.46**	**143.7**	**3.79**	**2.90**	**0.28**	**0.13**	**66.5**	**0.72**	**11.58**	**784.1**	**206.8**	**158.3**	**15.0**	**7.1**	**11 677.1**	**125.6**
		Inferred	4.81	0.06	88.7	2.78	1.47	0.11	0.07	33.6	0.40	6.73	5.2	1.6	0.9	0.1	0.0	62.9	0.7
	str	**Measured + Indicated**	**2.91**	**0.63**	**45.0**	**0.75**	**0.24**	**0.48**	**0.08**	**22.1**	**0.06**	**3.82**	**28.5**	**4.8**	**1.5**	**3.0**	**0.5**	**450.8**	**1.2**
		Inferred	2.83	0.01	44.8	0.58	0.38	0.46	0.13	29.4	0.20	4.75	0.3	0.0	0.0	0.0	0.0	7.2	0.0
North	**Total**	**Measured + Indicated**	**3.85**	**8.90**	**125.6**	**2.50**	**2.58**	**0.34**	**0.16**	**64.4**	**0.75**	**10.52**	**1 117.7**	**222.7**	**229.8**	**29.9**	**14.3**	**18 419.3**	**215.8**
		Inferred	2.86	0.49	80.6	0.68	1.70	0.22	0.11	42.0	0.83	6.62	39.7	3.3	8.4	1.1	0.5	666.0	13.1

Deposit	Domain	Category	Density g/cm³	Mass Mt	NSR-MRE $/t	Cu %	Zn %	Pb %	Ag g/t	Au g/t	CuEq %	NSR-MRE M$	Cu kt	Zn kt	Pb kt	Ag koz	Au koz
South	C1 FR	**Indicated**	**2.89**	**6.43**	**63.3**	**0.40**	**1.26**	**0.75**	**13.13**	**0.07**	**1.21**	**407.0**	**25.5**	**81.2**	**48.2**	**2 715.4**	**13.7**
		Inferred	3.06	0.83	50.4	0.61	0.47	0.23	12.26	0.13	1.04	42.1	5.1	3.9	1.9	329.0	3.4
	C2 FR	**Indicated**	**2.86**	**2.50**	**67.5**	**0.50**	**1.23**	**0.64**	**15.58**	**0.04**	**1.28**	**168.5**	**12.4**	**30.8**	**16.0**	**1 251.4**	**3.6**
		Inferred	3.01	3.66	62.0	0.54	0.96	0.42	17.32	0.08	1.20	226.6	19.6	35.0	15.5	2 036.7	9.6
	C3 FR	**Indicated**	**2.85**	**1.11**	**62.5**	**0.39**	**1.17**	**0.54**	**20.77**	**0.04**	**1.16**	**69.2**	**4.3**	**13.0**	**6.0**	**740.0**	**1.3**
		Inferred	3.07	3.64	60.1	0.67	0.69	0.22	16.62	0.04	1.16	218.7	24.5	25.1	7.9	1 943.9	5.3
South	**Total**	**Indicated**	**2.88**	**10.04**	**64.2**	**0.42**	**1.24**	**0.70**	**14.58**	**0.06**	**1.22**	**644.7**	**42.2**	**124.9**	**70.2**	**4 706.8**	**18.6**
		Inferred	3.04	8.13	59.9	0.60	0.79	0.31	16.49	0.07	1.16	487.4	49.1	64.0	25.3	4 309.5	18.3

Differences may occur in totals due to rounding.

1. Mineral resources unlike mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
2. The mineral resources have been estimated in accordance with the CIM Best Practice Guidelines (2019) and the CIM Definition Standards (2014). Coherent and contiguous mining shapes backing reasonable prospect for economic extraction were considered.
3. Mineralized Zones: Gos=Gossan, tMS= transition massive sulphides pMS= primary massive sulphides, Str=Stringer, FR= -Fissural Remobilizations.
4. ZnEq% = ((Zn Grade*26.46)+(Pb Grade*22.05)+(Cu Grade * 77.16)+(Au Grade*54.66)+(Ag Grade*0.71)+(Sn Grade * 264.55))/26.46
5. CuEq% = ((Zn Grade*26.46)+(Pb Grade*22.05)+(Cu Grade * 77.16)+(Au Grade*54.66)+(Ag Grade*0.71)+(Sn Grade * 264.55))/77.16
6. Metal Prices: Cu $7,716/t, Zn $2,646/t, Pb $2,205/t, Au $1,700/oz, Ag $22.00/oz, Sn $26,455/t.
7. NSR-MRE: Gos= ((Sn%*0.40)*Sn$)+((Pb%*0.20)*Pb$)+((Au ppm*0.91)*Au$)+((Ag ppm*0.89)* Ag$)
   tMS= ((Sn%*0.40)*Sn$)+((Pb%*0.50)*Pb$)+((Ag ppm*0.65)* Ag$)+((Zn% *0.5)*Zn$)
   pMS= ((Sn%*0.40)*Sn$)+((Pb%*0.60)*Pb$)+((Ag ppm*0.45)* Ag$)+((Zn%*0.70)*Zn$)
   str= ((Cu%*0.60)*Cu$)+((Pb%*0.5)*Pb$)+((Ag ppm*0.4)* Ag$)+((Zn%*0.70)*Zn$)
   South Deposit= ((Cu%*0.65)*Cu$)+((Zn%*0.75)*Zn$)+((Pb%*0.65)*Pb$)+((Au ppm*0.05)*Au$)+((Ag ppm*0.80)*Ag$)
8. NSR-MRE cut-off: Gos= $48.24, tMS= $46.54, pMS= $41.06, str= $37.83, South Deposit= $37.83

14.7.3.1 RISK ASSESSMENT

At present there are no known environmental, permitting, legal, title, taxation, socio-economic, marketing, or political issues which would adversely affect the mineral resources estimated above. However, mineral resources, which are not mineral reserves, do not have demonstrated economic viability. There are no mineral reserves on the LS property. There is no assurance that any or all the requisite consents, permits or approvals, regulatory or otherwise, will be obtained for the project. Other hindrances may include interference with ability to work on the property and lack of efficient infrastructure. There is no assurance that the project will be placed into production.

15 MINERAL RESERVE ESTIMATES

15.1 GENERAL

The mine design and Mineral Reserve estimation have been completed to a level of appropriate for feasibility studies. The Mineral Reserve estimate stated herein is consistent with the National Instrument 43-101: Standards of Disclosure for Mineral Projects and is suitable for public reporting. As such, the Mineral Reserves are based on Measured and Indicated Resources, and do not include any Inferred Resources.

The Mineral Reserves are developed from the Mineral Resource block models, "ocBM_1Q2023_SS (CM_VA) 5_10_5 [REPORT]1Mar.dm", for the South area (dated September 2022) and "sBM_1Q2023_NS (CM_VA) 5_10_5 [REPORT]28Feb.dm" for the North area (May 2022), both provided by Ascendant.

15.2 NET SMELTER RETURN MODEL

The Net Smelter Return model intends to simulate the geo-metallurgical behavior of each mineable shape in isolation, that is, without any blending with other shapes. Effectively, each shape ought to produce within specification products on its own.

IGAN coded the NSR values for all the mineralisation defined in the Mineral Resource block models and with consideration to the effects of deleterious elements in the concentrate net value. To do so, costing, pricing, processing recoveries and concentrate commercial terms provided by both Ascendant and Mine Pro were used.

The provided metal prices and currency exchange rate are summarized in Table 15.1.

Table 15.1 – Metal prices

Item	Price	Unit
Zn price	2,646	USD/t
Pb price	2,205	USD/t
Cu price	7,716	USD/t
Ag price	22.0	USD/oz
Au price	1,700	USD/oz
Sn price	26,455	USD/lb
Exchange rate	1.06815	USD/EUR

The NSR value of every block in the resource model is estimated as the sum of the net value of the relevant concentrate product masses that may be obtained from the material tonnage, expressed on a USD/t basis. A generalised formula is shown below:

$$NSR\left(\frac{USD}{t}\right) = NSR_{Cu} + NSR_{Zn} + NSR_{Sn} + NSR_{Bu}$$

Where:

- NSR_{Cu}: NSR value of copper concentrate.
- NSR_{Zn}: NSR value of zinc concentrate.
- NSR_{Sn}: NSR value of tin concentrate.
- NSR_{Bu}: NSR value of silver/gold bullion.

For each relevant product, its net value is estimated as the sum of the revenue streams generated by its contributing metal recovered masses (i.e., zinc, lead, copper, silver, gold, and tin) minus all costs of metal refining, concentrate treatment, element penalties, concentrate freight, and other relevant deductions and charges.

$$NSR\left(\frac{USD}{t}\right) = \frac{\sum_{i=1}^{n}((Met_i \times \%Pay_i \times \%Rec_i) - TC - FC - \sum_{i=1}^{n}(RC_i) - \sum_{j=1}^{k}(Ded_j)}{Mass}$$

Where:

- i: Saleable Payable metal (i.e., Zn, Pb, Cu, Ag, Au, Sn) in product.
- j: Penalty metal (i.e., Zn, Pb, Cu, Ag, Au, Sn) in j-eth product.
- Met: In-situ metal mass.
- %Pay: Percentage of payable metal mass.
- %Rec: Percentage of recovered metal in product.
- TC: Treatment or smelting charge per dry tonne of product.
- FC: Freight charge per dry tonne of product.
- RC: Refining and shipment cost of payable metal.
- Ded: Deleterious metal deduction.

Table 15.2 presents the commercial terms that have been used to estimate the NSR.

Table 15.2 – Sellable product commercial terms

Product	Item	Value	Unit	Comment
Pb	Ag metal payable	95	%	50 g minimum deduction
Pb	Pb metal payable	95	%	3% minimum deduction

Product	Item	Value	Unit	Comment
Pb	TC	80	USD	
Pb	FC	35	USD	
Pb	Ag refining and delivery	0.8	USD/oz	
Pb	As penalty charge	1.5	USD/0.1%	Up to 0.5% Free. 0.7% max
		3.0	USD/0.1%	Grade exceeding 0.7%
Pb	Hg penalty charge	1.5	USD/100 ppm	100 ppm free, 300 ppm max
		3.0	USD/100 ppm	Grade exceeding 300 ppm
Zn	Ag metal payable	65	%	
Zn	Ag metal deduction	93.3	g	Fixed deduction
Zn	Ag payable of metal price	95	%	
Zn	Zn metal payable	85	%	80 kg minimum deduction
Zn	Zn payable of metal price	95	%	
Zn	TC	175	USD	
Zn	FC	70	USD	
Zn	Fe penalty charge	3.00	USD/1%	Free up to 7.5%
Cu	Cu metal payable	96.5	%	<18%Cu. Minimum deduction: 1.2%
		96.5	%	18-20%Cu. Minimum deduction: 1.1%
		96.5	%	20-30%Cu. Minimum deduction: 1.0%
		96.5	%	>30% Cu.
Cu	Au metal payable	90	%	1-3 ppm
		92	%	3-5 ppm
		93	%	5-8 ppm
		94	%	8-10 ppm
		95	%	>10 ppm
Cu	Ag metal payable	90	%	Minimum deduction 30 ppm
Cu	TC	30	USD	
Cu	FC	35	USD	
Cu	Cu refining and delivery	0.03	USD/lb	
Cu	Au refining and delivery	5	USD/ozt	
Cu	Ag refining and delivery	0.4	USD/ozt	
Cu	As penalty charge	2.5	USD/100 ppm	Free up to 2,000 ppm
Cu	Pb+Zn penalty charge	2.5	USD/1%	Free up to 4% Zn+Pb

Product	Item	Value	Unit	Comment
Cu	Bi penalty charge	2.0	USD/100 ppm	Free up to 300 ppm
Cu	Sb penalty charge	2.5	USD/1,000 ppm	Free up to 1,000 ppm
Cu	Hg penalty rate	1	USD/10 ppm	Free up to 20 ppm
Sn	Sn metal payable	95	%	
Sn	Pb metal payable	75	%	
Sn	TC	300	USD	
Sn	FC	70	USD	
Sn	RC	50	USD	
Sn	As penalty charge	20	USD	
Sn	Pb+Zn+Ag+Ni+Co penalty charge	300	USD	
Sn	Bi penalty charge	300	USD	
Sn	Cu penalty charge	300	USD	
Sn	Sb penalty charge	300	USD	
Sn	Fe, Mg, WO3 penalty charge	300	USD	
Sn	F, Cl penalty charge	300	USD	
Au/Ag	Ag metal payable	90	%	
Au/Ag	Au metal payable	99.5	%	
Au/Ag	TC	30	USD	Per Kg of bullion
Au/Ag	RC	20	USD	Per Kg of bullion

Table 15.3 presents the product recoveries and fixed grades considered to estimate the NSR value of each sellable product.

Table 15.3 – Sellable product recoveries and grades

Product	Item	Material	Value	Unit
Pb	Pb grade	PMS	30	%
		TMS	30	
		STR	30	
		STWK	35	
	Pb recovery	PMS	60	%
		TMS	50	
		STR	50	
		STWK	65	

Product	Item	Material	Value	Unit
Zn	Ag recovery	PMS	30	%
		TMS	65	
		STR	15	
		STWK	50	
	As grade	PMS	1.5	ppm
		TMS	10,000	
		STR	10,000	
		STWK	5,000	
	Hg grade	PMS	600	ppm
		TMS	1,500	
		STR	50	
		STWK	10	
	Zn grade	PMS	35	%
		TMS	40	
		STR	40	
		STWK	45	
	Zn recovery	PMS	70	%
		TMS	50	
		STR	70	
		STWK	75	
	Ag recovery	PMS	15	%
		TMS	1	
		STR	15	
		STWK	15	
	Fe grade	PMS	10	%
		TMS	10	
		STR	10	
		STWK	10	
Cu	Cu grade	STR	20	%
		STWK	20	
	Cu recovery	STR	60	%
		STWK	65	
	Au recovery	STR	0.5	%
		STWK	5	
	Ag recovery	STR	10	%
		STWK	15	
	As grade	STR	5,000	ppm
		STWK	1,000	
	Pb grade	STR	0.5	%
		STWK	1.5	
	Zn grade	STR	1.5	%
		STWK	1.5	
	Bi grade	STR	150	ppm
		STWK	100	

Product	Item	Material	Value	Unit
	Sb grade	STR	0.1	%
		STWK	0.1	
	Hg grade	STR	15	ppm
		STWK	10	
Sn	Sn grade	PMS	50	%
		TMS	50	
		GOS	10	
	Sn recovery	PMS	40	%
		TMS	40	
		GOS	40	
	Pb recovery	PMS	0.05	%
		TMS	0.5	
		GOS	20	
Ag/Au	Ag grade	GOS	300 000	ppm
	Ag recovery	GOS	89	%
	Au recovery	GOS	91	%

15.3 CUT-OFF CALCULATION

NSR cut-off grades were established following Mortimer's approach to cut-off grades (Hall, 2014), which identifies rock as economically mineable when the following criteria are met:

- The lowest grade of rock must pay for itself.
- The average grade of rock must provide a minimum average profit per ton.

The former, also known as "boundary cut-off", guarantees that no losses are incurred by mining and treating a volume of rock. The latter or "volume cut-off" assures that the rock volume yields a certain average level of profit per treated ton, so shared costs such as general & administration are covered by the average production grade.

Table 15.4 details the parameters and values of both cut-off values for each of the considered mining methods.

Table 15.4 – NSR cut-off values per mining method and material

Mining Method	Material	Mining Cost	Processing Cost	Boundary cut-off	G&A	Volume cut-off
LHOS	STWK	19.5	16.31	**35.81**	2.5	**38.31**
	PMS	19.5	19.62	**39.12**	2.5	**41.62**
	TMS	19.5	14.48	**33.98**	2.5	**36.48**

Mining Method	Material	Mining Cost	Processing Cost	Boundary cut-off	G&A	Volume cut-off
	GOS	19.5	16.22	**35.72**	2.5	**38.22**
	STR	19.5	16.31	**35.81**	2.5	**38.31**
Cut&Fill	STWK	29.98	16.31	**46.29**	2.5	**48.79**
	PMS	29.98	19.62	**49.60**	2.5	**52.10**
	TMS	29.98	14.48	**44.46**	2.5	**46.96**
	GOS	29.98	16.22	**46.20**	2.5	**48.70**
	STR	29.98	16.31	**46.29**	2.5	**48.79**

15.4 MODIFYING FACTORS

15.4.1 DILUTION AND RECOVERY ESTIMATES

In the estimate of the Mineral Reserves, modifying factors were applied to the tonnages and grade of all mining shapes (both stoping and cut & fill) to account for unplanned dilution and ore losses that are anticipated in this project.

Ore dilution includes overbreak into the design hanging wall, footwall, and adjacent backfilled stopes. Diluting materials are assumed to carry no metal values in the estimation of inventory grades. Ore losses (recovery factor) are related to the practicalities of extracting ore under varying conditions, including difficult mining geometry, problematic rock conditions, blasting issues, and other aspects.

The factors used in the estimate of the Mineral Reserves are presented in Table 15.5. The dilution has been estimated by IGAN considering its own experience and benchmarking against similar long-hole open-stope operations in the Iberian Pyritic Belt. The recovery has been estimated based on experience in operations within the region with similar mining methods, equipment, and operational drilling practices to those anticipated to be used in this project. As no paste strength tests are available at the time of the preparation of this report, an average value of dilution and recoveries has been assumed for both primary and secondary stopes.

Table 15.5 – Dilution and recovery factors

Mining method	Dilution factor (%)	Recovery Factor (%)
LHOS	10	85
Cut&Fill	10	95

15.5 MINING SHAPES

IGAN employed Deswik.SO ("Stope Optimizer") optimisation software to generate mineable stope and cut & fill shapes for each orebody meeting meet both cut-off grade and operational design criteria. These preliminary shapes were then individually refined to minimize the amount of sub-economic material within the shape volume that cannot be selectively mined.

The design criteria constrain the geometry of all planned excavations to what is achievable through the planned mining methods. Chapter 16 Section 4.1.3 provides further detail on mining shapes and design parameters. For each mine zone and on a level-by-level basis, a design combining both mining methods that maximises the ore tonnage was developed.

15.6 OREBODY DESCRIPTION

The Mineral Reserves delineated at the cut-off summarized in Table 15.4 define an orebody consisting of three contiguous volumes in the North Zone and several grade corridors in the South Zone. Both orebodies are separated by 250 meters.

15.6.1 NORTH ZONE

The North orebody approximately extends over 540- and 175 meters along and across strike, respectively. It also extends 420 meters vertically.

It is comprised of three adjacent zones (massive sulphides, gossan, and secondary sulphides/transition) as shown in Figure 15.1.

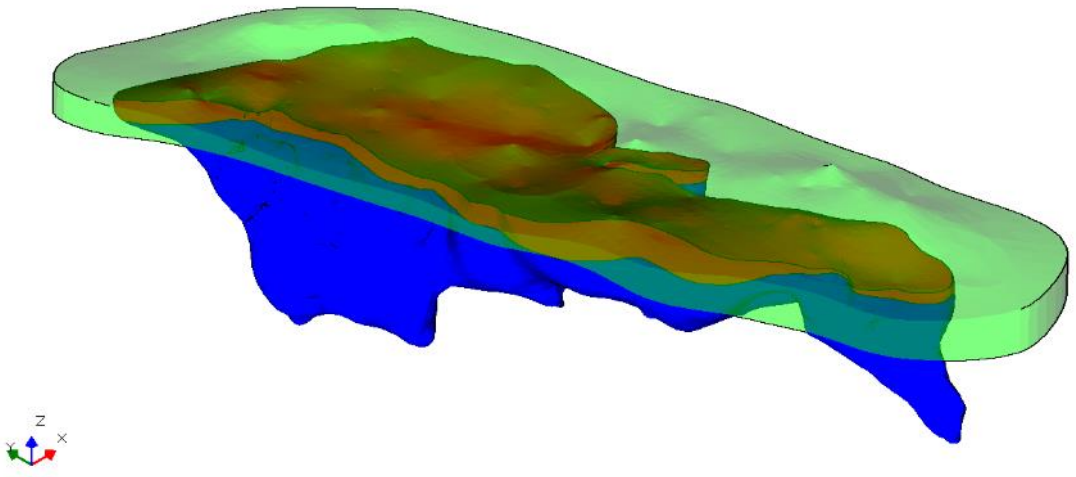


Figure 15.1 North orebody zones. Isometric view (oriented to NE)

The maximum dimensions of each individual zone (length along strike, width, average thickness dimensions are shown in Table 15.6.

Table 15.6 – North orebody zone average dimensions

Lithology	Length	Width	Thickness
Gossan	550	175	17
Transition zone	460	140	8
Massive sulphides	450	140	32
North	550	175	25

From an extraction point of view, given the Gossan and transition bodies are flat-lying and tabular, they can be extracted as a single body. This is shown in Figure 15.2, where the zones are coloured accordingly (Gossan: green, Transition: red, Primary sulphides: blue, Stringer: orange).

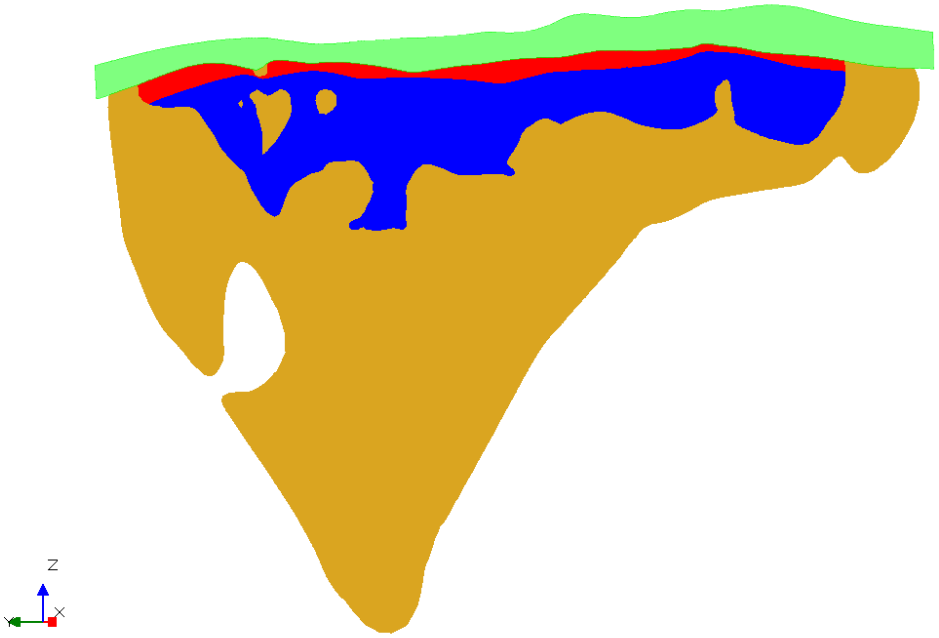


Figure 15.2 North orebody zones. Long section

15.6.2 SOUTH ZONE

The South orebody approximately extends over 900 and 220 meters along and across strike, respectively. It extends 600 meters vertically. It is comprised of five main mineralised corridors (Figure 15.3), enclosed within a low-grade envelope. All the corridors are interpreted to be of stringer lithology. From an extraction point of view, the corridors can be extracted as a single orebody, provided that the excavation is stable.

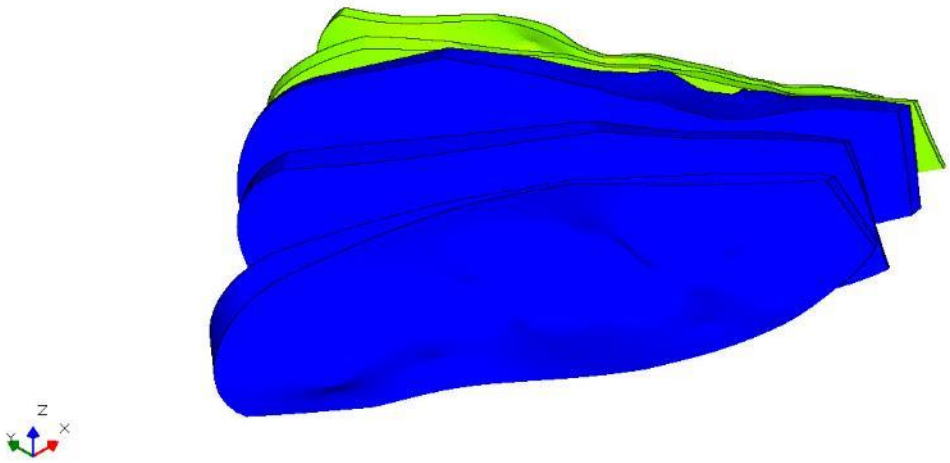


Figure 15.3 South orebody grade corridors. Isometric view (oriented to NE)

15.7 MINERAL RESERVE ESTIMATE

Table 15.7 presents the Mineral Reserve estimate, broken down by mine zone and reserve category. The values presented herein are reported in a diluted and recovered mass and grade basis and have been rounded to an appropriate number of significant figures that reflect the uncertainty in the estimate; thus, totals may reflect small rounding errors.

Table 15.7 – Mineral Reserve estimate per mine area

Zone	Reserve category	Mass (kt)	Cu (%)	Pb (%)	Zn (%)	Sn (%)	Ag (ppm)	Au (ppm)
North	Proven	2 100	0.26	2.8	3.2	0.15	62	0.75
	Probable	4 900	0.34	2.3	2.1	0.14	63	0.74
	Total							
South	Proven	-	-	-	-	-	-	-
	Probable	7 600	0.41	0.72	1.3	0	15	0.05
	Total					-		
Total	Proven	2 100	0.26	2.8	3.2	0.15	62	0.75
	Probable	12 500	0.38	1.3	1.6	0.06	33	0.32
	Total	14 600	0.37	1.5	1.8	0.07	37	0.38

Figure 15.4 presents the mineable shapes forming the bulk of the Mineral Reserves for the North (top) and South (bottom) mine zones.

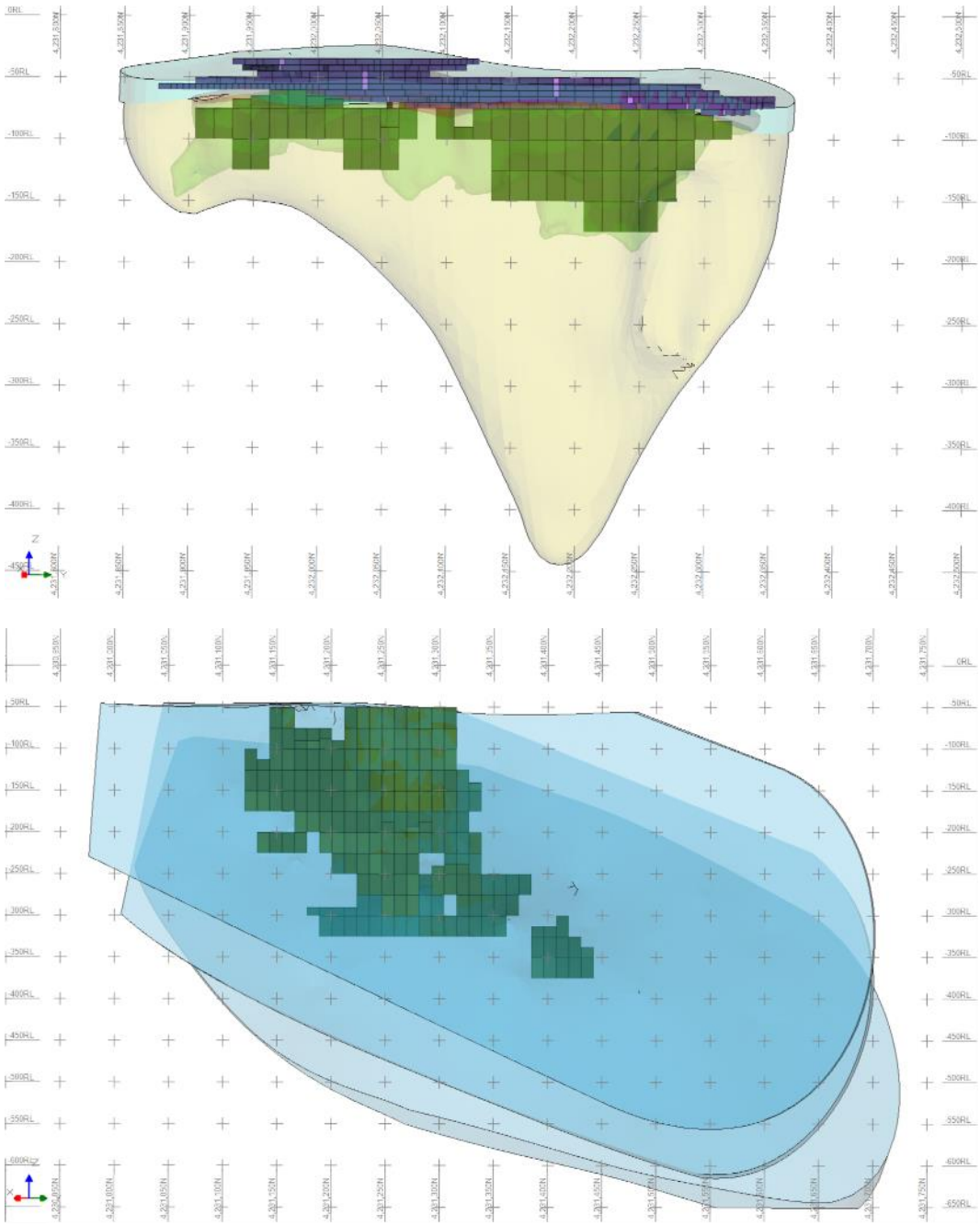


Figure 15.4 - Minable resource shapes and geology shells (long section)

The Mineral Reserve estimate per mine zone and lithology are detailed in Table 15.8. As with Table 15.7, the values are reported to a relevant number of significant figures that reflect the uncertainty in the estimate. The totals may reflect rounding errors as well.

Table 15.8 – Mineral Reserve estimate per lithology

Zone	Lithology	Mass (kt)	Cu (%)	Pb (%)	Zn (%)	Sn (%)	Ag (ppm)	Au (ppm)
North	Gossan	1 200	0.1	2.4	0.41	0.25	52	1.0
	Transition	880	0.98	2.2	0.23	0.18	110	1.1
	Massive sulphides	4 400	0.25	2.7	3.6	0.12	62	0.68
	Stringer	550	0.23	0.18	0.54	0.06	13	0.06
	Total	7 100	0.31	2.4	2.4	0.15	63	0.75
South	Stockwork	7 800	0.41	0.71	1.3	-	15	0.06
	Total	7 800	0.41	0.71	1.3	-	15	0.06

16 MINING METHODS

16.1 GENERAL

The proposed underground mine design supports the annual extraction of 1.2 million ore tonnes ("Mtpa") through a combination of transverse sublevel stoping and cut&fill. Cemented paste is ought to be employed as backfill material of the mining voids, so ore recovery and productivity are maximized.

A main decline, starting from the surface portal, placed close to the processing plant, will be used to access the mine. This main decline splits into two ramps, each providing access to one of the mine zones (North and South orebodies).

A fleet of LHDs ("Load-Haul-Dump") units will be used for both material loading and tramming between production areas and dedicated level stockpiles. From the level stockpiles, ore will be loaded into 65-tonne mine trucks and hauled to a Run of Mine ("ROM") pad, located on the surface. Waste is also transported to the surface with 65-tonne mine trucks.

The ventilation system is designed as a two parallel branch system. Fresh airflow ingresses through the main ramp and a central ventilation shaft. It is then distributed into two branches, each serving a mine area (i.e., North and South). Polluted air is exhausted through two exhaust shafts, located at the far end of each area. Two main ventilation fans are installed on the surface, one per ventilation exhaust shaft.

A pre-production development program will be required to provide access to the initial stoping levels in the North zone during the first twenty-four months. Production will start in the second year, reaching the nominal plant feed in the fourth year.

16.2 ANALYSIS OF MINING METHOD ALTERNATIVES

The selection of the mining method was based on the University of British Columbia ("UBC") Mining Method Selection, which is a modification of the Nicholas methodology.

The modified Nicholas method gives more importance to the mineralized hanging wall and less to the footwall, whereas Nicholas considers that both have the same relevance. Given the geometry and the available geotechnical parameter information, IGAN considered that the modified Nicholas method would better suit the Project needs. With the available information as well as its own experience at other Iberian Pyritic Belt mines, a ranking of different methods per lithology was produced by IGAN.

16.2.1 GOSSAN

The following assumptions were considered for this material:

- The hanging wall has a moderate geomechanical quality.

- Tabular geometry.
- Uniform grade distribution.
- Narrow thickness.
- Flat inclination.

Table 16.1 summarizes the geotechnical and geometrical parameters for the Gossan.

Table 16.1 – Gossan: Parameters

Geometry	
General form	Tabular
Thickness	Very thick >100mm
Dip	Flat <20°
Grade distribution	Uniform
Strength	
Hangingwall	Strong >15
Ore body	Weak <8
Footwall	Weak < 8
Fracture frequency	
Hangingwall	Wide: RQD 40–70
Ore body	Very close: RQD 0–20
Footwall	Very close: RQD 0–20
Fracture strength	
Hangingwall	Moderate
Ore body	Weak
Footwall	Weak

The results for all the considered mining methods are presented in Table 16.2.

Table 16.2 – Gossan: Modified Nicholas ranking

Method	**Ranking**
Open pit mining	35
Cut&fill stoping	31
Square-set stoping	35

Method	Ranking
Block caving	34
Longwall mining	-18
Room and pillar mining	-30
Top slicing	28
Shrinkage stoping	20
Sublevel caving	18
Sublevel stoping	-48

Although square-set stoping would be the most suitable underground mining method, it is a very old method with low productivities and not amenable to mechanization. These reasons led to its rejection. The second option, block caving, was discarded as well. As well as requiring a complex and extensive permitting process to be approved by the regulatory authorities, it would not be applicable to an orebody of this size. Cut and fill, ranked third in the list was selected as the method for this ore type. It is a method amenable to mechanisation, used within the region and well suited to the extraction of small ore volumes.

16.2.2 TRANSITION ZONE

The following assumptions were considered for this material:

- The hanging wall has poor geomechanical quality.
- Tabular geometry.
- Gradual grade distribution.
- Narrow thickness.
- Flat inclination.

Table 16.3 summarizes the geotechnical and geometrical parameters for the transition zone.

Table 16.3 – Transition zone: Parameters

Geometry	
General form	Tabular
Thickness	Thick 30–100m

Geometry	
Dip	Flat <20°
Grade distribution	Gradational
Strength	
Hangingwall	Weak <8
Ore body	Weak <8
Footwall	Strong >15
Fracture frequency	
Hangingwall	Wide: RQD 40–70
Ore body	Very Close: RQD 0–20
Footwall	Wide: RQD 40–70
Fracture strength	
Hangingwall	Weak
Ore body	Weak
Footwall	Moderate

The results for all the considered mining methods are presented in Table 16.4.

Table 16.4 – Transition zone: Modified Nicholas ranking

Method	Ranking
Open pit mining	36
Cut&fill stoping	31
Square-set stoping	35
Block caving	32
Longwall mining	-19
Room and pillar mining	-29
Top slicing	26
Shrinkage stoping	20
Sublevel caving	17
Sublevel stoping	-47

As can be seen, the method ranking obtained for the Transition ore does replicate that of the Gossan ore. Again, cut and fill was selected as the mining method of choice for this ore type as well.

16.2.3 MASSIVE SULPHIDES

The following assumptions were considered for this material:

- Moderate quality hangingwall.
- Massive geometry.
- Gradual grade distribution.
- Intermediate thickness (30 – 100 m).
- 20°-55° inclination.

Table 16.5 summarizes the geotechnical and geometrical parameters for the massive sulphides.

Table 16.5 – Massive sulphides: Parameters

Geometry	
General form	Massive
Thickness	Thick 30–100m
Dip	Intermediate 20°–55°
Grade distribution	Uniform
Strength	
Hangingwall	Moderate 8–15
Ore body	Strong >15
Footwall	Moderate 8–15
Fracture frequency	
Hangingwall	Wide: RQD 40–70
Ore body	Very Wide: RQD 70–100
Footwall	Wide: RQD 40–70
Fracture strength	
Hangingwall	Weak
Ore body	Strong
Footwall	Moderate

The results for all the considered mining methods are presented in Table 16.6.

Table 16.6 – Massive sulphides: Modified Nicholas ranking

Method	Ranking
Open pit mining	43
Cut&fill stoping	25
Square-set stoping	24
Block caving	28
Longwall mining	-79
Room and pillar mining	-22
Top slicing	36
Shrinkage stoping	36
Sublevel caving	33
Sublevel stoping	31

For this lithology, shrinkage stoping and top slicing are tied in first place as suitable underground methods. They however are old methods yielding low productivities and with little to no mechanisation potential, thus being discarded by IGAN. Sublevel caving is ranked second but, as with the block caving method, its implementation would require an extensive and complex environmental and social permitting process that would yield no additional benefits and thus granting its rejection. Sublevel stoping, ranked fourth, was selected as method for the area. The method is well-known in the region, has good productivity and it can be fully mechanized.

16.2.4 STOCKWORK

The following assumptions were considered for this material:

- Good quality hangingwall.
- Tabular geometry.
- Gradual grade distribution.
- Thickness 30–100m.
- >55° inclination.

Table 16.7 summarizes the geotechnical and geometrical parameters for the stockwork material.

Table 16.7 – Stockwork: Parameters

Geometry	
General form	Tabular
Thickness	Thick 30–100m
Dip	Steep >55°
Grade distribution	Gradational
Strength	
Hangingwall	Strong >15
Ore body	Moderate 8–15
Footwall	Moderate 8–15
Fracture frequency	
Hangingwall	Very Wide: RQD 70–100
Ore body	Wide: RQD 40–70
Footwall	Wide: RQD 40–70
Fracture strength	
Hangingwall	Moderate
Ore body	Moderate
Footwall	Weak

The results for all the considered mining methods are presented in Table 16.8.

Table 16.8 – Stockwork: Modified Nicholas ranking

Method	**Ranking**
Open pit mining	42
Cut&fill stoping	30
Square-set stoping	27
Block caving	25
Longwall mining	-80
Room and pillar mining	-21
Top slicing	26
Shrinkage stoping	29
Sublevel caving	30
Sublevel stoping	32

For this ore type, sublevel stoping is ranked first among the underground mining methods, being selected by IGAN as the method of choice for this ore type.

16.3 MINE DESIGN

16.3.1 ACCESS AND RAMP INFRASTRUCTURE

The mine portal will be located close to the processing plant. Figure 16.1 presents a plan view of the mine layout, being possible to locate the decline portal at its easternmost end.

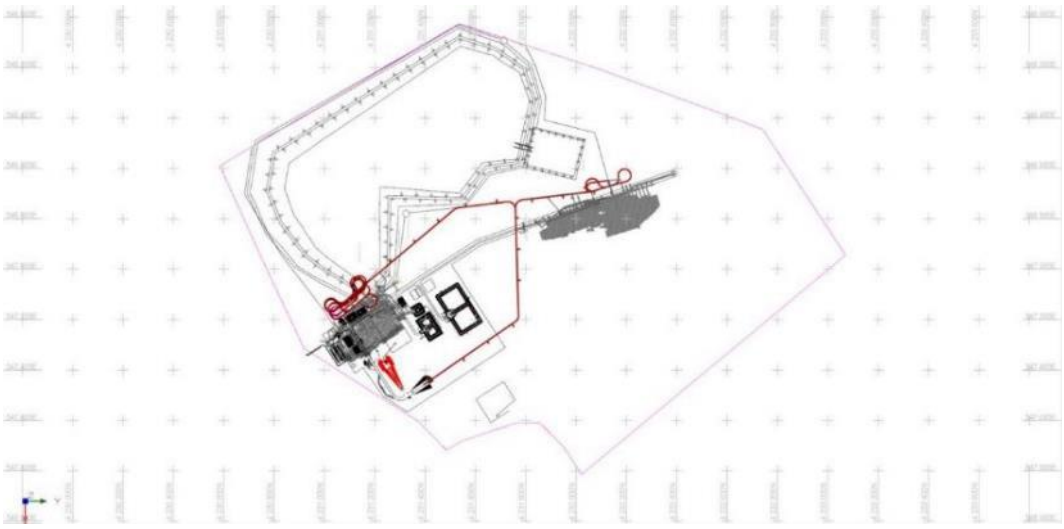


Figure 16.1 – Mine portal and declines (plan view)

The main decline bifurcates into two independent ramps, one per mine area, at the 15 mRL. This is shown in Figure 16.2 (declines in red). The zone ramps connect all the mine levels within.

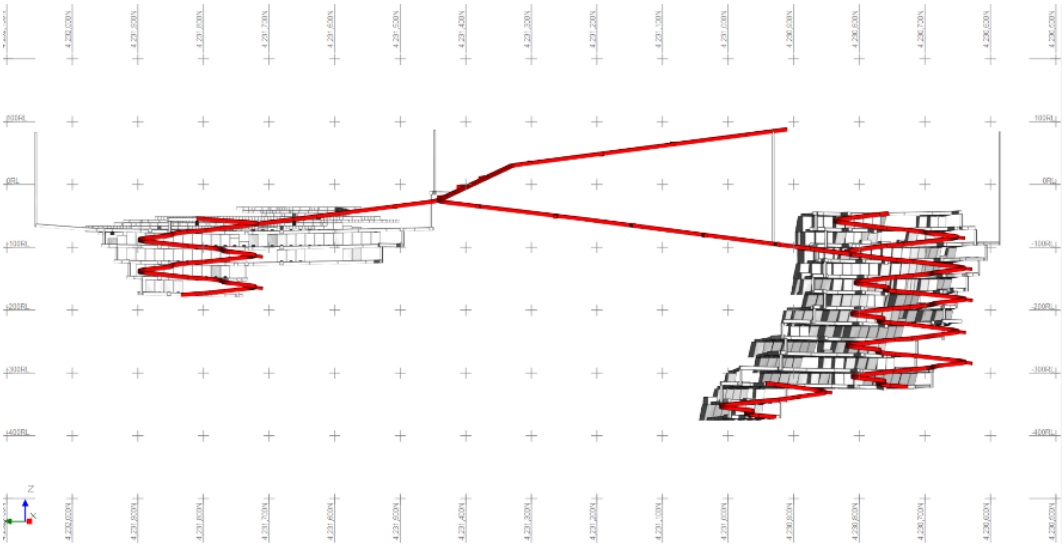


Figure 16.2 – Declines (long section)

The use of an independent ramp per zone instead of one providing access to both zones was preferred as it reduces the initial Capital Cost ("CAPEX"). Production starts earlier in the North zone and the mine economics favours the ramp design option providing faster access to it, which in this case is achieved through a dedicated zone ramp.

All ramps and main decline were designed with a 25 m turning radius and a maximum 12.5% gradient. As a rule, declines and ramps are designed with a minimum 75 m standoff distance from the ore contacts.

Passing, electrical and paste cuddies, and safety bays were not included in the design both due to their size and the typically operations-driven positioning. Paste cuddies are not included in the design as no information on the paste reticulation system was available at the time of preparing the mine plan.

Nonetheless, an allowance of a 25 m long passing bay, a 3m long electrical bay and 6 safety bays per 250 m of ramp has been considered in the creation of both mine plans and cost estimates. An allowance of 88 metres for pastefill cuddies has been also considered in the creation of both mine plans and cost estimates.

16.3.2 LEVEL DEVELOPMENT

Sublevel stoping extraction levels in both zones were designed with a 25 m vertical spacing, as determined by the stope height; all of them are accessed through the zone ramp. Footwall drives, as a rule, are designed with a minimum 35 m standoff distance from the ore contact. A typical stoping level layout is shown in Figure 16.3. Fresh and return airways are located at each of the sublevel ends, with the level access located at the approximate centre of gravity of the footwall drive. Ore drives, spaced 15 m from each other, provide access to stopes.

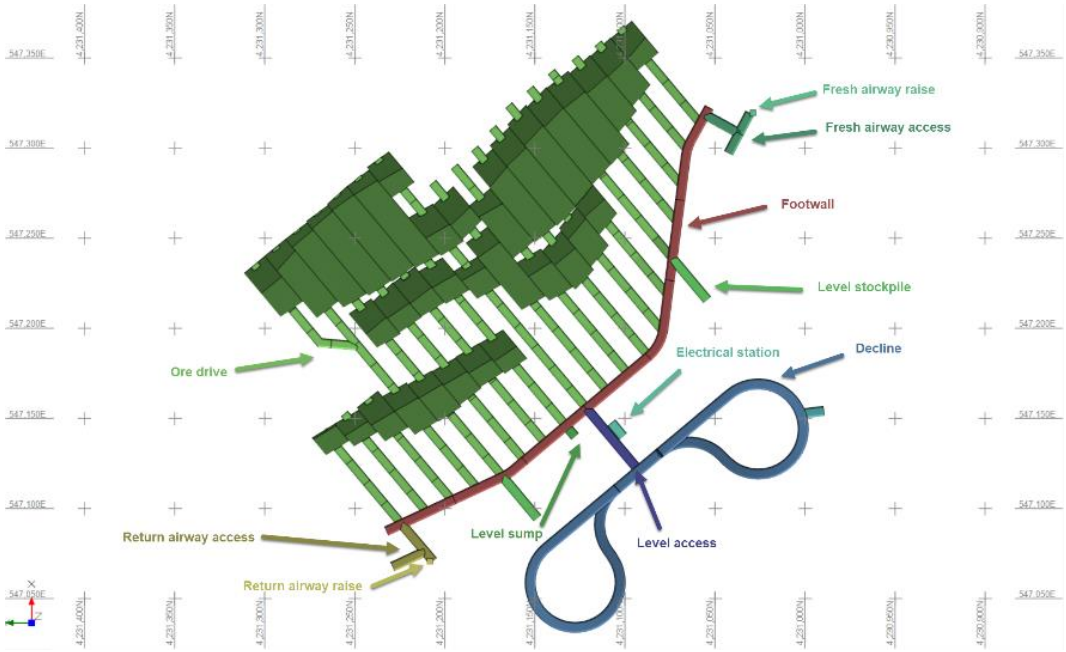


Figure 16.3 – Typical sublevel stoping level layout (plan view)

Cut&fill is the mining method employed for the extraction of the Gossan and Transition lithologies in the North orebody. These are in the upper levels of the North area (42, 50 and 75). Figure 16.4 presents a plan view of a typical cut&fill level layout.

Figure 16.4 – Typical cut&fill level layout (plan view)

Figure 16.5 shows the access infrastructure design for the South (top) and North (bottom) zones. Footwall drives include excavations for level sumps and level stockpiles. Although paste cuddies are not designed, an allowance for extra development metres was included in both mine plan and financial model. Electrical stations are located at the level accesses. The main ramp also includes excavation for decline sumps and stockpiles. Decline sumps are placed between every two levels of the internal ramps.

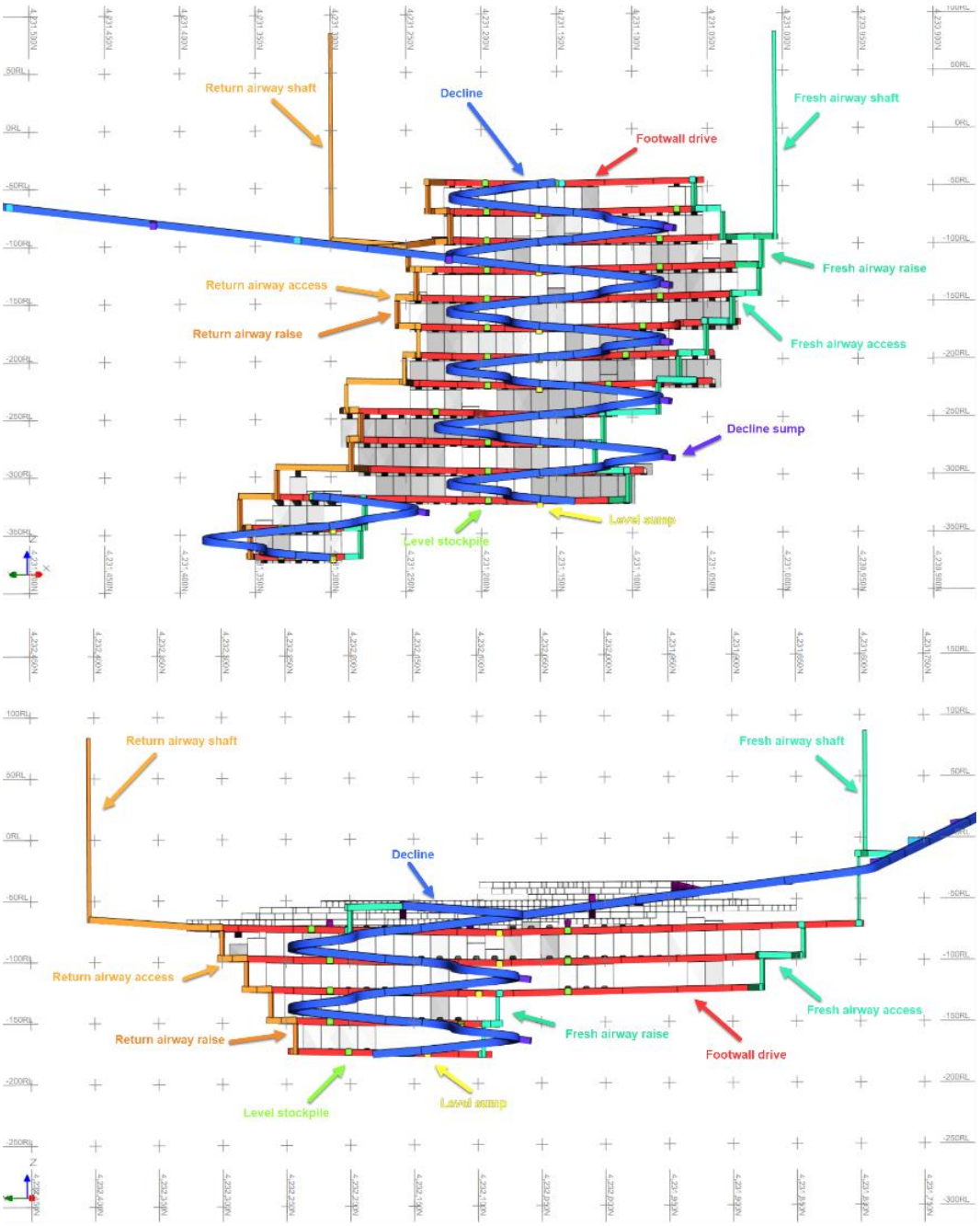


Figure 16.5 – Infrastructure design (long section)

Table 16.9 lists the sections used in the development activity types.

Table 16.9 – Development sections

Type	Section (m x m)
Ore access drive	4.5x5.0
C&F access	5.5x5.5
C&F central lift drive	5.0x5.0
C&F ore drive	5.0x5.0
Footwall drive	5.5x5.5
Level access	5.5x5.5
Decline	6.5x6.0
Decline stockpiles	6.5x6.0
Decline sumps	6.5x6.0 5.5x5.5 (North ramp)
Level stockpiles	5.5x5.5
Level sumps	5.5x5.5
Electrical station	5.5x5.5
Pumping station	5.5x5.5
Ventilation raise access	5.5x5.5
Ventilation shaft (Raise borer development)	Ø 3.5 Ø 2.4
Ventilation raise (Drill & Blast development)	3.5x3.5

Table 16.10 summarizes the design parameters and criteria used in the CAPEX (e.g., ramps, level accesses and footwall/hanging wall drives) and OPEX (e.g., ore drives) development. In accordance to generally accepted guidelines for a feasibility level study, gradients have been applied to all designed development, as needed.

Table 16.10 – Development design parameters

Type	Parameter	Value	Units	Comments
Decline	Gradient	-12.5 to +12.5	%	Minimum and maximum values
	Curve radius	25	m	Minimum value
Decline sump	Gradient	-2.5	%	Maximum value
	Length	12.5	m	Minimum value
Decline stockpile	Gradient	+2.0	%	Maximum value
	Length	15	m	Minimum value
C&F access	Gradient	+0.5 to +3.0	%	Minimum and maximum values
	Length	20	m	Minimum value

Type	Parameter	Value	Units	Comments
C&F central lift drive	Gradient	-15.0 to +18.5	%	Minimum and maximum values
Level access	Gradient	-2.0	%	Maximum value
	Length	40	m	Minimum value
Electrical station	Gradient	-2.0	%	Maximum value
	Length	10	m	Minimum value
	Spacing	1	unit	Number per level
Footwall drive	Gradient	-2.0 to +2.0	%	Minimum and maximum values
Level sump	Gradient	-7.5	%	Maximum value
	Length	7.2	m	Minimum value
	Spacing	1	unit	One per level
Level stockpile	Gradient	+2.0	%	Maximum value
	Length	30	m	Minimum value
	Spacing	100	m	Minimum value
Ventilation raise access	Gradient	+2.0	%	Maximum value
	Length	15	m	Minimum value
Pumping station	Gradient	-	%	Maximum value
		20	m	Minimum value
Ore drives	Length	0	%	Maximum value
		50	m	Minimum value

The prepared mine design lateral and vertical development distances are included in Table 16.11 and Table 16.12, respectively.

Table 16.11 – Total mine horizontal development distance per type

Development type	Distance (m)
Decline	6 297
Decline sump	150
Decline stockpile	96
C&F access	123
C&F central lift drive	1 534
C&F ore drive	25 283
Level access	890
Electrical station	220
Footwall drive	5 428
Level sump	266
Level stockpile	796
Ventilation raise access:	
Fresh airway	1 052
Return airway	1 152

Development type	Distance (m)
Pumping station	35
Ore drive	31 454
Total	**74 777**

Table 16.12 – Total mine vertical development distance per type

Development type	Distance (m)
Ventilation shaft	
Fresh airway	328
Return airway	329
Ventilation raise	
Fresh airway	415
Return airway	700
Total	**1 442**

16.3.3 STOPE DESIGN

Table 16.13 presents the geometric parameter values considered in the design of both LHOS stopes and cut&fill zones, with Figure 16.6 and Figure 16.7 showing them in context of a stope design.

Stope wall angles were selected with consideration to operational parameters, namely the maximum angle drilling equipment dump angle and blasted material angle of repose. The minimum inclination of the front stope face wall was established considering the typical angle that any production drilling equipment is able of drilling.

To guarantee an adequate flow of blasted material in the stopes, a minimum angle for relevant stope face walls (i.e., back wall) of 55° was selected. This value is typically large enough to ensure an adequate material flow.

Table 16.13 – Stope geometric design parameters

Parameter	Value	Unit	Comment
Maximum width	15	m	
Minimum length	8	m	
Maximum length	50	m	
Minimum height	7	m	
Maximum height	25	m	
Minimum angle:			
Front	60	°	Measured from the horizontal
Back	55		

Parameter	Value	Unit	Comment
Minimum angle: Front Back	 90 90	 °	 Measured from the horizontal
C&F height	5	m	
C&F width	5	m	
Minimum C&F length	5	m	
Maximum C&F length	600	m	

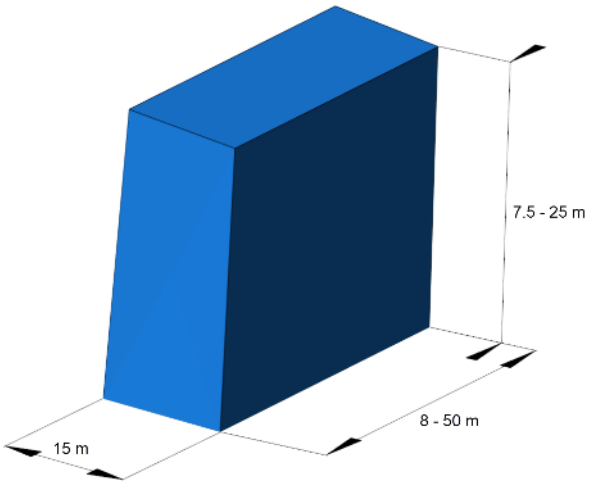


Figure 16.6 – Stope dimensions

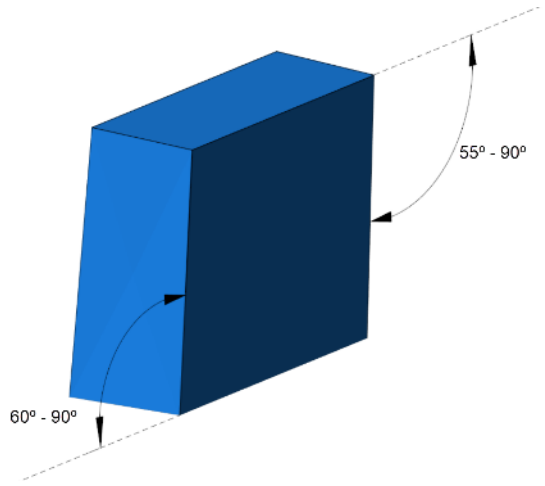


Figure 16.7 – Stope angular limits

16.4 MINING METHOD AND SEQUENCE

16.4.1 BLOCK DEFINITION

The deposit is divided into four mining blocks, defined by mining method, level and mine area. The number and geometry were determined in a way so they sustain the nominal annual target production of 1.2 Mt and with consideration of the following factors, as described as follows:

- Pre-production development requirements: the first mine area to be scheduled for production is the North orebody. This mine area is closest to the main decline access and thus, it can better support the mine production ramp up and nominal production rate sustainment.

- Grade profile: block delineation influences high-grade mineralisation accessibility over time. The mining blocks were defined so higher grades can be mined during the LOM early years. Inside the North block, priority has been given to high grade stopes that will be mined first and backfilled with high strength paste fill so they can be mined ahead of their neighbouring stopes.

- Nominal production rate: The South orebody was divided into two mining blocks, so the nominal mine production rate can be sustained over time. Level 175 in the South acts as a sill pillar to create the blocks. The stopes on that level will be backfilled with high strength paste so they can support mining of the stopes on level 150.

All mine blocks are represented in Figure 16.8.

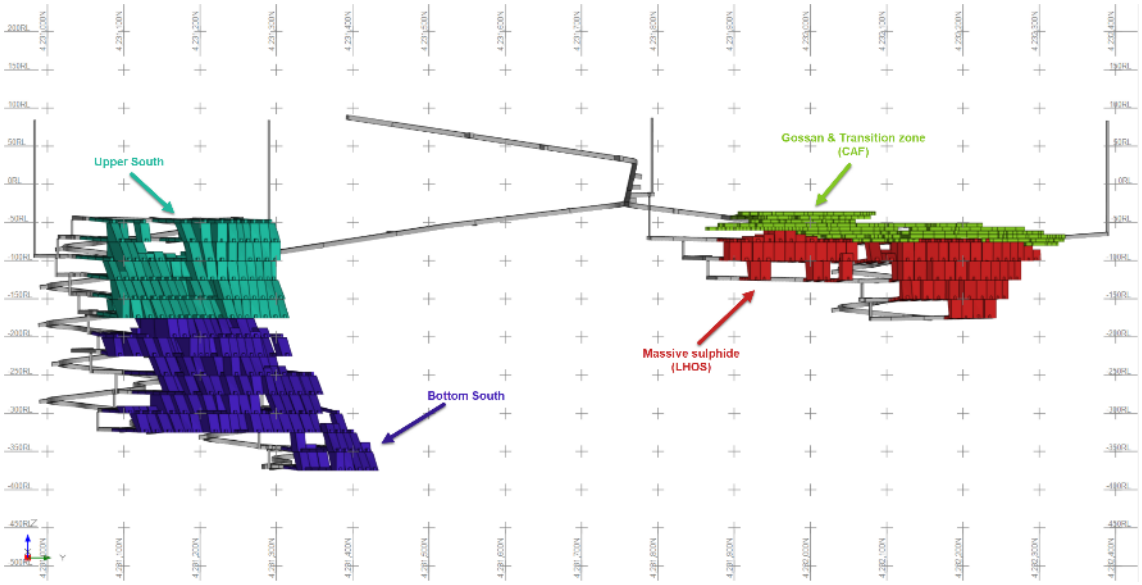


Figure 16.8 – Mine blocks

16.4.2 MINE PRODUCTION SEQUENCING

16.4.2.1 STOPE AND CUT&FILL PRODUCTION SEQUENCE

The typical stope production cycle is as follows:

- Cablebolt support starts seven days after the stope access development is completed. This provides time for stope preparation work (e.g., cablebolt layout design, geological mapping, survey, services installation, etc.).
- Production drilling in stopes starts a minimum of two days after the cablebolt support is installed.
- Mucking starts a minimum of two days after production drilling is completed.
- Paste backfilling starts two days after the production is completed. This delay allows the construction of plugs, shotcrete barricades, removal of ancillary equipment such as watering sprays, etc.
- A three-week curing period is maintained for the paste fill. This ensures that the paste backfill can develop adequate strength to ensure top, bottom and sidewall stability.

The cut&fill production cycle considers these aspects:

- Production drilling and mucking in ore drifts are to be started after their waste access development is completed.
- Paste fill starts three days after the end of production. This delay allows the construction of plugs, shotcrete barricades, removal of ancillary equipment such as watering sprays, etc.
- A four-week curing period is maintained for the paste fill.

16.4.2.2 GEOTECHNICAL SEQUENCE

In developing the mine plan, a set of geotechnical sequencing rules are considered for the stopes:

- An inverse pyramidal, primary-secondary ascending sublevel retreat sequence is followed in stoping zones. Figure 16.9 shows a plan view of the sequence in a typical level.

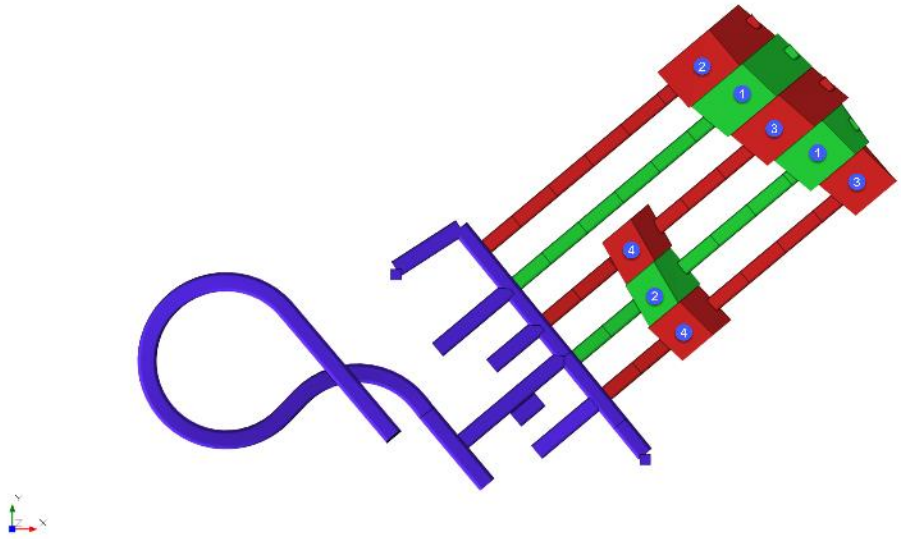


Figure 16.9 – LHOS Primary-secondary retreat sequence

- Secondary stope access drifts within a level can only be started after their neighbouring primary stopes are mined, backfilled and a curing period of at least three weeks is elapsed.

- The retreat sequence in the stoping drifts is achieved by starting cablebolting in the next stope to be mined after the previous stope was backfilled and a curing period of at least one week is elapsed.

- Primary stopes in a column can have a maximum lead of two levels relative to its adjacent secondary stope columns.

- A twenty-eight-day curing period is maintained to ensure stability before starting the production drilling of the stope above.

- The South orebody is divided into two mining blocks by a pillar at level 175. This pillar allows start production earlier in the mine plan, as the infrastructure development should only reach level 175 to begin mining.

For the cut&fill, the following aspects were considered:

- A primary-secondary retreat, overhand sequence is followed.

- Secondary cut&fill production starts after their neighbouring primary drifts are backfilled, and a curing period of at least four weeks is elapsed. Figure 16.10 shows the primary-secondary sequence.

- A four-week curing period is maintained to ensure stability before open a new level or sublevel.

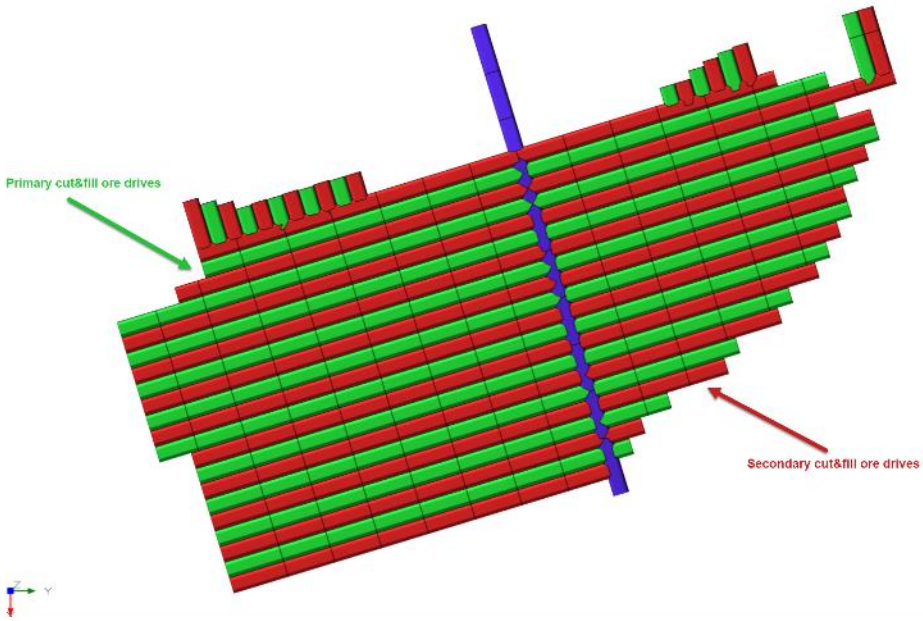


Figure 16.10 – Primary-secondary sequence in cut&fill

16.4.2.3 OTHER CONSIDERATIONS

The following restrictions were also considered by IGAN as part of the mine sequencing rules:

- Once the ventilation raises are completed, a thirty-day delay period is honoured prior to start developing ore drives. This provides time for the installation of ventilation circuit equipment (e.g., regulators, secondary fans etc) in the level, so mining is only allowed to happen once the level is included in the primary ventilation circuit.

- A primary ventilation circuit, with a ventilation raise reaching the level, must be established in a level prior to the start of any stope access drift.

- Footwall drives within a level can be developed once their level electrical station is completed.

- Ramps and lateral development can be advanced one level below the deepest level with an established primary ventilation circuit. Effectively this limits the ramp advance rate to a point where good ventilation conditions can be sustained.

16.4.3 BACKFILLING

According to the work completed by RMS, the primary means of backfilling at the Property will be cemented pastefill, generated from thickened mill tailings and mixed with a blend of sand and binding agents to a defined recipe.

Two paste fill mixes, one for stockwork lithology and other for the North (i.e., massive sulphide, transition zone, gossan and stringer lithologies), will be used in the backfill of mining voids.

The so-called stockwork mix will contain 5% binder and is anticipated to be used in the South sector; thus, its strength ought to be enough so the stability of voids is guaranteed. The North mix will contain 2.5% binder will be used in stopes as well as in cut&fill drifts. This mix will contain sufficient binder to remove any risk of future liquefaction.

An extra high strength paste fill will be required in bottom cut&fill lifts and that will be undercut by sill extraction from below as well as for the 175 mRL sill pillar and the high priority stopes in the North sector.

16.4.3.1 WASTE MANAGEMENT AND STOPE FILLING

Waste rock will need to be disposed of on an ongoing basis throughout the mine life. Waste will be hauled to surface for disposal at a designated waste dump. Some of the waste generated during the pre-production phase of the Project is anticipated to be used as construction material for the TSF dam and the RoM pad ramp.

The disposal of waste rock at the surface has the effect of increasing the total void volume requiring paste backfill, hence increasing the percentage of mill tailings that can be returned to underground and thus, reducing the size of the TSF.

Table 16.14 – Waste rock mining and paste fill requirements

Year	Paste backfill (000 m^3)	Waste to surface (kt)
Y1	0	12
Y2	85	320
Y3	210	300
Y4	500	320
Y5	650	320
Y6	610	260
Y7	610	170
Y8	650	78
Y9	450	65
Y10	360	39

Year	Paste backfill (000 m^3)	Waste to surface (kt)
Y11	250	13
Y12	240	19

16.5 DEVELOPMENT AND PRODUCTION SCHEDULE

16.5.1 STOPING RESOURCE SCHEDULING RATES

Stoping resource scheduling rates considered in the creation of the mine plans were calculated using the typical stope size and equipment productivity obtained from the financial model. The values are shown in Table 16.15.

Table 16.15 – Stoping activity resource scheduling rates

Resource	Unit	Rate
Stope production: Blast load + Mucking	t/d	1 100
Paste backfilling	m^3/d	2 000
Production drilling – Simba	m/d	210
Cabling	m/d	220

16.5.1.1 STOPE PRODUCTION

The considered stope production rate is inclusive of both mucking and explosive load and blast activity length. An allowance of four extra days to cater for production delays and unplanned issues is also included. The values are shown in Table 16.16.

Table 16.16 – Stope cycle productivity parameters

Resource	Unit	Rate
Stope size	36 925	tonne
Effective mucking rate	1 635	t/d
Effective load and blast rate	5 400	t/d
Other delays	4	days

The effective mucking rate is calculated with consideration of a typical cycle for a 21t unit working a 250 m tramming distance. It is assumed that the unit is operated partially by an onboard and remote operator. A 20% onboard and 80% remote time split is assumed in the estimate. The considered parameters are shown in Table 16.17.

Table 16.17 – LHD productivity parameters

Resource	Operated	Remote	Unit
Bucket nominal capacity	21	21	tonne
Fill factor	90	80	%
Tramming distance	250	250	m
Tramming speed	5	5	Km/h
Load and dump time	3	3	Minutes
Effective hours per shift	9.4	9.4	Hour
Shift	10.5	10.5	
Shift start meeting	0.25	0.25	
Travel	0.5	0.5	
Other breaks	0.4	0.4	
Shifts per day	2	2	Shifts

The effective load and blast rate is calculated with consideration of a 2 x 10.5-hour shift day and an hourly productivity of 288 t ore per effective hour. This is obtained from the parameters shown in Table 16.18.

Table 16.18 – Explosive loading productivity parameters

Parameter	Operated	Unit
Productivity	288	Loaded ore tonne/hour
Effective hours per shift	9.4	Hour
Shift	10.5	
Shift start meeting	0.25	
Travel	0.5	
Other breaks	0.4	
Shifts per day	2	Shifts

16.5.1.2 PRODUCTION DRILLING

A production rate of 220 metres per day is assumed, which is estimated from the parameters shown in Table 16.19.

Table 16.19 – Production drilling productivity parameters

Parameter	Operated	Unit
Productivity	11.7	m/operating hour
Effective hours per shift	9.4	Hour
Shift	10.5	
Shift start meeting	0.25	
Travel	0.5	
Other breaks	0.4	

Parameter	Operated	Unit
Shifts per day	2	Shifts

16.5.2 DEVELOPMENT RESOURCE SCHEDULING RATES

Horizontal development scheduling rates were established considering the geotechnical domain and section of the drift. A standard round length of 3.8 metres is considered in all sections. A 1x1 drilling pattern with 13 cut holes (4 reamed holes in a square with one blast hole in centre plus 4 helpers) is also assumed in all sections for the drilling and explosive loading equipment. Table 16.20 details the rates on a domain and section basis.

Table 16.20 – Horizontal development scheduling rates

Parameter	Section	Unit	Rate
DG2	5.0x5.0	m/d	1.90
DG3	5.0x4.5	m/d	1.90
DG3	5.5x5.5	m/d	2.53
DG4	5.5x5.5	m/d	3.80
DG4	6.5x6.0	m/d	3.80

Table 16.21 shows the vertical development scheduling rates.

Table 16.21 – Vertical development scheduling rates

Parameter	Unit	Rate
Ventilation raise-bored shaft	m/d	4
Ventilation drill and blast raise	m/d	4

16.5.3 RESOURCE AND PRODUCTION LIMITS

Following the strategy defined by Ascendant, an annual nominal production rate of 1.2 Mtpa was assumed. No ramp-up rate has been established, with the mine being considered as the system bottleneck. The horizontal development capacity is initially assumed to be 220 m per month per jumbo unit. An additional annual limit of 5 520 m of cut&fill development has also been considered.

16.5.3.1 PRE-PRODUCTION DEVELOPMENT

Pre-production development will occur over an approximately 12 months before the first cut&fill sublevel pass is started. The main development strategy priority zone is the North orebody, followed by the upper block within the South.

The first stopes will be mined in 100 Level of the North zone. Critical path pre-productions activities include:

- Decline development to access the production levels in the North.
- Development of the 15 and 75 North levels and the exhaust air ventilation shaft.

A total development requirement of 1,701 horizontal metres (CAPEX and OPEX development) and 98 vertical metres are planned to be completed during the first 12 months.

16.5.3.2 LOM DEVELOPMENT SCHEDULE

The North zone alone can sustain the nominal ore production rate (i.e., 1.2 Mtpa) over a period; therefore, it is required to develop the South zone main infrastructure in time, so once production in the North with the higher NSR values cannot sustain 1.2 Mtpa, production can be ramped up in the South to fill the gap. Table 16.22 presents the results of the development strategy.

Table 16.22 – LOM development plan results

Year	CAPEX		OPEX		Total	
	Horizontal	Vertical	Horizontal	Vertical	Horizontal	Vertical
Y1	1585	98	116		1701	98
Y2	2964	258	4507		7471	258
Y3	1099	128	5626		6725	128
Y4			4857		4857	
Y5	1923		4475		6398	
Y6	2603	511	3713		6315	511
Y7	2761	135	4905		7666	135
Y8	2135	141	5032		7167	141
Y9	1436	171	7151		8586	171
Y10			5652		5652	
Y11			4092		4092	
Y12			4892		4892	
Y13			1027		1027	
Y14			1501		1501	
Y15			725		725	

Chart 16-1 shows the evolution of the CAPEX and OPEX horizontal development through the LOM

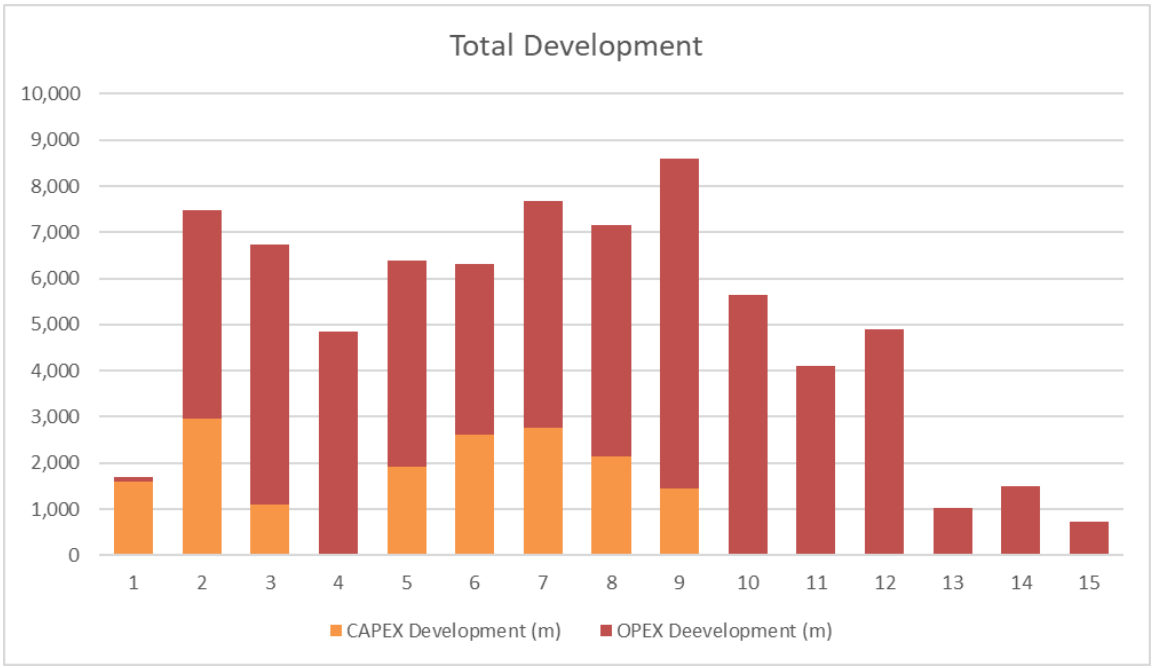


Chart 16-1 – Annual horizontal development per type

16.5.3.3 LOM PRODUCTION SCHEDULE

Table 16.23 and Chart -16-2 show the results obtained from the mine scheduling: the annual plant feed and the average NSR. Note that the values have been rounded to an adequate number of significant figures and thus, totals may reflect small rounding errors.

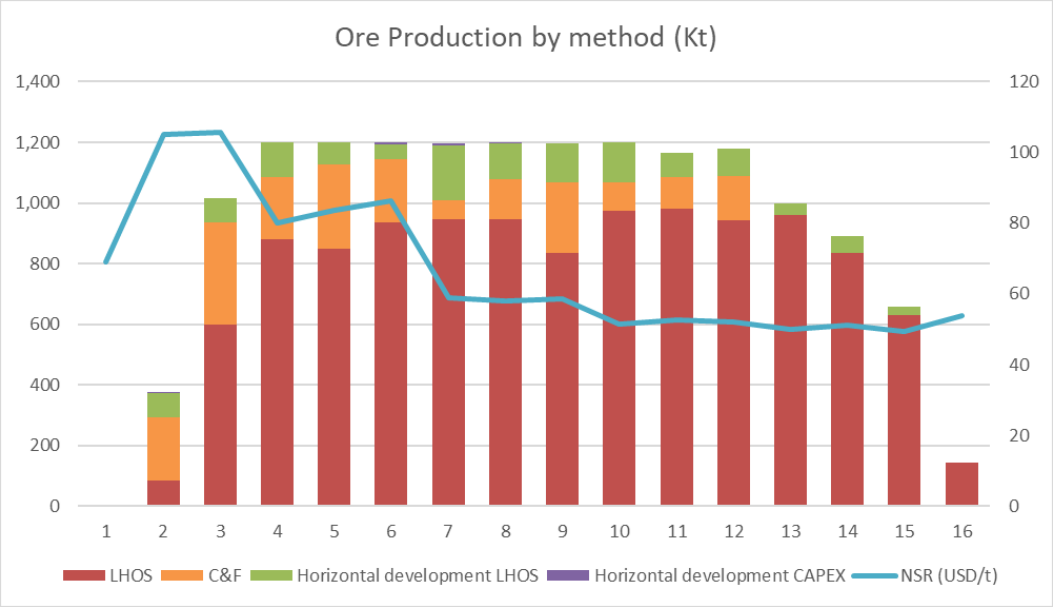


Chart -16-2 Ore production by method

Table 16.23 – LOM annual plant feed and average NSR values

Year	Mineral production (kt)	NSR (USD/t)
Y1	2	69
Y2	372	105
Y3	1016	106
Y4	1200	80
Y5	1200	84
Y6	1200	86
Y7	1199	59
Y8	1197	58
Y9	1197	59
Y10	1202	51
Y11	1165	53
Y12	1178	52
Y13	998	50
Y14	891	51
Y15	657	49
Y16	143	54

Table 16.24 and

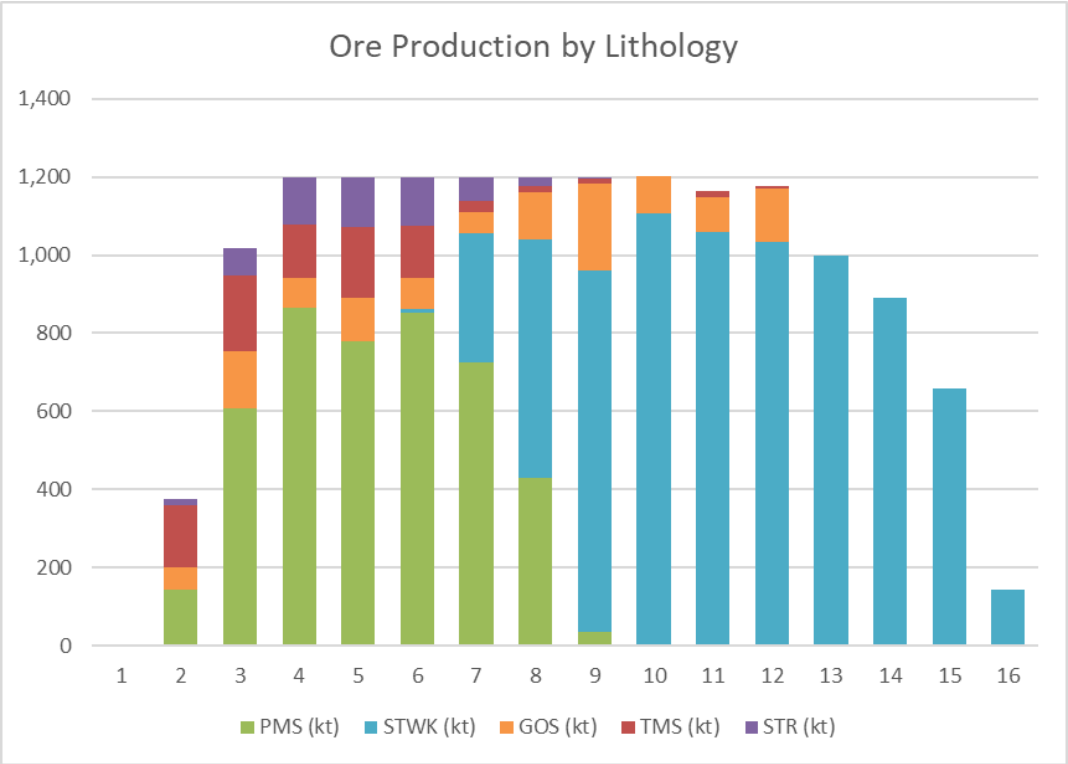


Chart 16-3 LOM annual plant feed by lithology

Table 16.25 illustrate the production and NSR values during the life of mine per mining method and ore type, respectively. The nominal plant feed is reached in year 4 and maintained until the twelfth year when the ramp-down starts. As it can be apprecitted, the South sector, with lower NSR values stays inactive until year 6. From that year onwards, its production grows to compensate the rampdown of the North sector.

Table 16.24 – LOM annual plant feed per mining method

Year	Mineral production LHOS (kt)	Mineral production C&F (kt)
Y1		2
Y2	83	209
Y3	600	335
Y4	880	204
Y5	849	277
Y6	934	209
Y7	948	62
Y8	948	132

Year	Mineral production LHOS (kt)	Mineral production C&F (kt)
Y9	834	236
Y10	974	96
Y11	982	105
Y12	944	144
Y13	960	
Y14	834	
Y15	629	

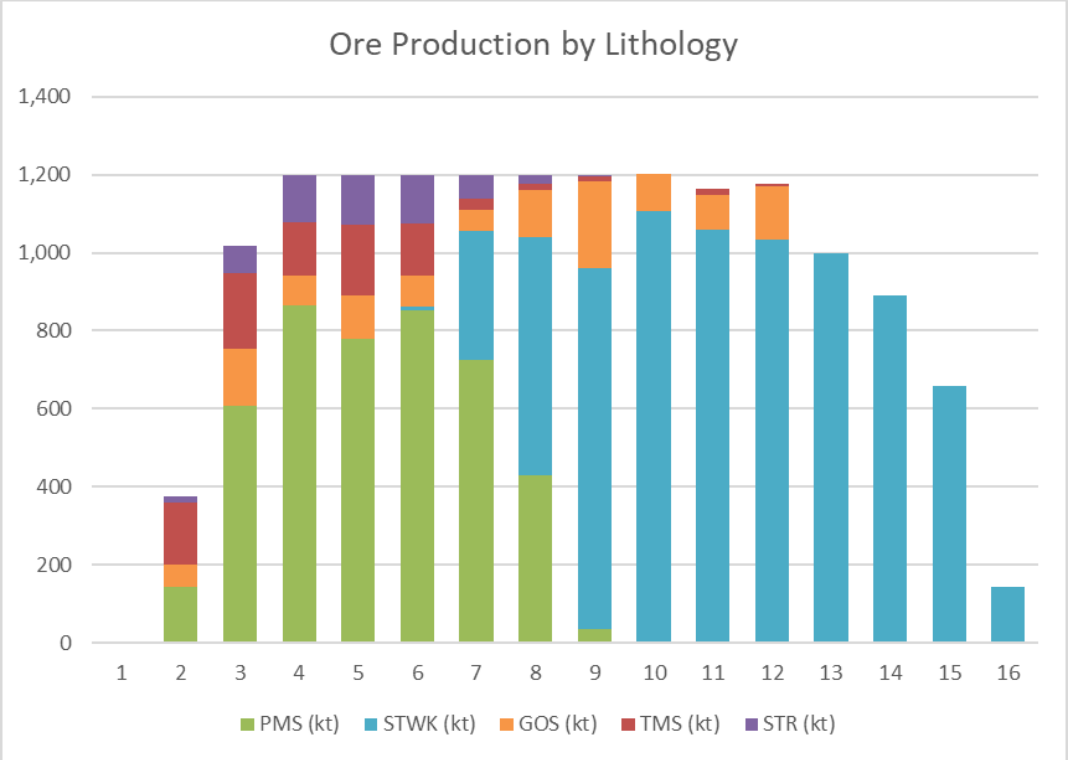


Chart 16-3 LOM annual plant feed by lithology

Table 16.25 – LOM annual plant feed per lithology

Year	GOS (kt)	TMS (kt)	PMS (kt)	STR (kt)	STWK (kt)
Y1	2	0	0	0	0
Y2	57	160	143	16	0
Y3	148	192	606	72	0

Year	GOS (kt)	TMS (kt)	PMS (kt)	STR (kt)	STWK (kt)
Y4	78	136	864	122	0
Y5	110	182	779	130	0
Y6	82	132	853	126	7
Y7	55	28	726	60	330
Y8	122	16	428	20	611
Y9	222	14	34	1	926
Y10	96	0	0	0	1,106
Y11	88	17	0	0	1,060
Y12	137	7	0	0	1,034
Y13	0	0	0	0	998
Y14	0	0	0	0	891
Y15	0	0	0	0	657

16.6 GEOTECHNICAL CHARACTERISATION

16.6.1 OVERVIEW

Geotechnical designs and recommendations are based on the results of available data and geotechnical assessments, which included intact rock assays, rock mass characterisation, structural geology, excavation and pillar stability analyses, and ground support design.

Geotechnical site investigations completed to support the underground rock mechanics assessments included: geotechnical drilling and logging, oriented drill core measurements, borehole televiewer surveys, laboratory testing of rock core samples.

The rock mass of the Lagoa Salgada deposits was divided into five geotechnical lithotypes, as shown in Table 16.26.

Table 16.26 – Geotechnical units

Zone	Unit	Code
North	Dacitic tuffite/Acid volcanics	dtV/aV
North	Gossan	Gos
North	Tertiary sediments	Ter
South	Intermediate to basic/acid volcanics	iV/aV
South	Exhalative, sedimentary, and volcanic	esV

16.6.2 DRILLING CAMPAIGN

The geotechnical survey campaign includes nine geotechnical boreholes totalling 3 310 drilled metres. To intersect the orebodies perpendicularly, a 60º angle from the horizontal was used on the boreholes. Figure 16.11 shows the North and South orebodies and the borehole distribution.

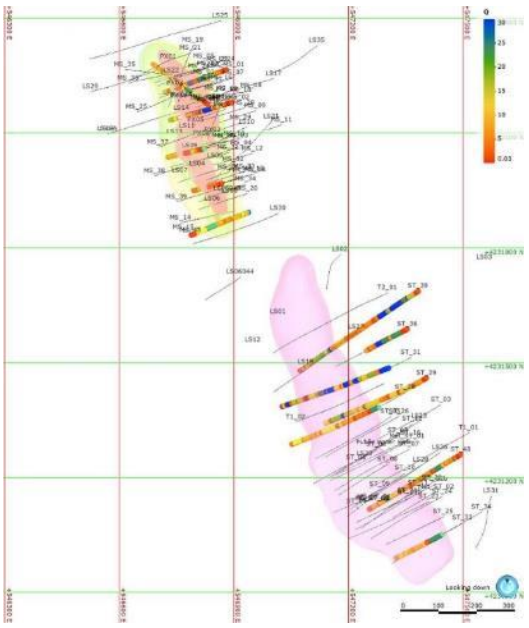


Figure 16.11 – Borehole distribution within orebodies

16.6.3 TEST WORK

Samples obtained from the geotechnical drilling campaign were sent to the Centro de Recursos Naturais e Ambiente ("Cerena") geotechnical laboratory in Lisbon to perform several assays; the type and number of assays are listed in Table 16.27.

Table 16.27 – Intact rock assay list per lithology

Assay	dtV/Av	MS	iV/av	esV	Gos	Conglomerates & sandstones
Apparent density	4	3	5	2	2	0
Moisture	4	3	5	2	2	0
Uniaxial compressive strength	4	5	7	3	5	2
UCS with Young modulus and Poisson Ratio	4	4	6	3	4	2
Brazilian test	4	4	6	3	4	2
3-Stage Triaxial Strength	2+4	3+6	4+8	2+4	3+6	3
Direct shear	9					0

Assay	dtV/Av	MS	iV/av	esV	Gos	Conglomerates & sandstones
Durability index	0	3	3	0	3	3
Sonic velocity test	6	3	3	3	3	3
Punctual load test (PLT)	40	30	50	20	25	20

16.6.4 INTACT ROCK PROPERTIES

Although all the units are defined as a homogeneous rock massif, they may be the result of a combination of several lithologies. Assay values are listed in Table 16.28.

Table 16.28 – Geotechnical properties

Unit	σ_C (MPa)	E (MPa)	v	E/σ_C	C (MPa)	Friction (º)
MS	129.74	118 883	0.22	1 244.01	19.93	49.17
dtV/aV	20.85	43 360	0.36	1 955.2	3.31	37.39
Gos	69.11	77 575	0.36	0.36	8.91	53.34
Ter	33.06	16 900	0.26	555.94	6.99	33.04
iV/aV	33.6	83 900	0.23	2 750.81	10.12	25.96

16.6.4.1 MASSIVE SULPHIDES

Young's modulus for intact rock has an average value of 118 883 MPa which is a high value for this type of material.

The modulus ratio was obtained from the assays and an average value of 1 244 has been obtained. Chart 16-4 shows the results obtained from the modulus ratio.

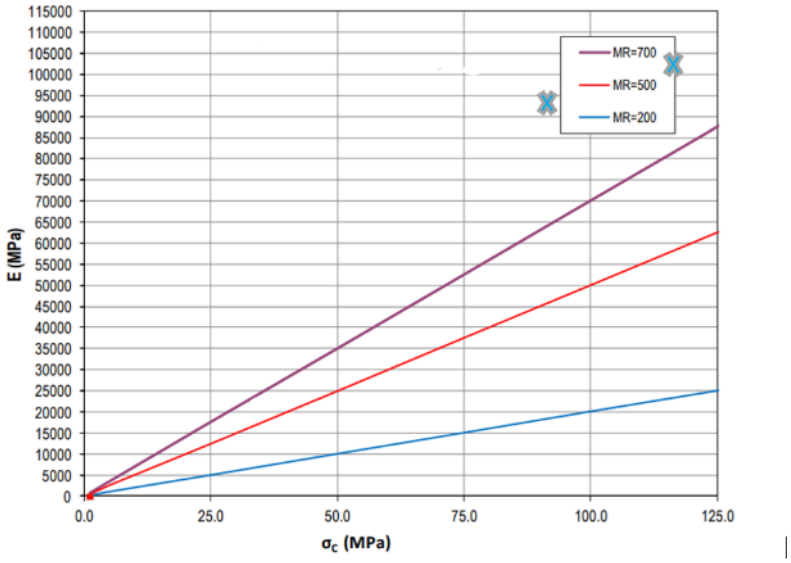


Chart 16-4 – Modulus ratio for the massive sulphide

Tensile strength has an average value of 15.23 MPa which is also an adequate value for this type of lithology.

The relation between tensile strength and uniaxial compressive strength was also represented to get the toughness value. As it is shown in Chart 16-5, the obtained toughness value is 10.

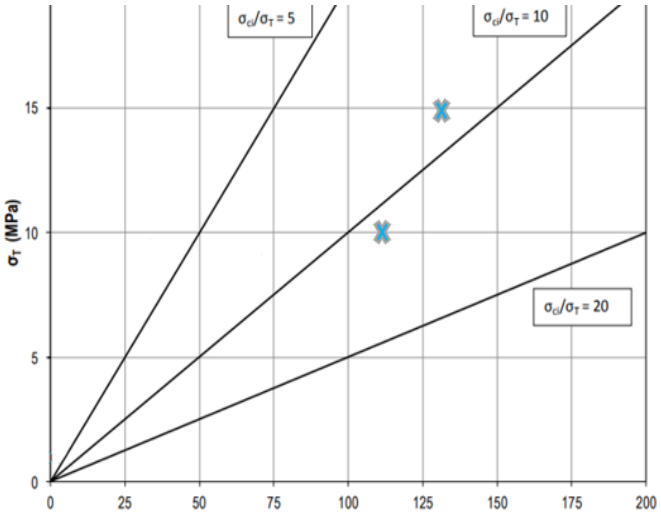


Chart 16-5 – Ratio between tensile strength and UCS for the massive sulphide

The Mohr circles for the massive sulphide are shown in Figure 16.12

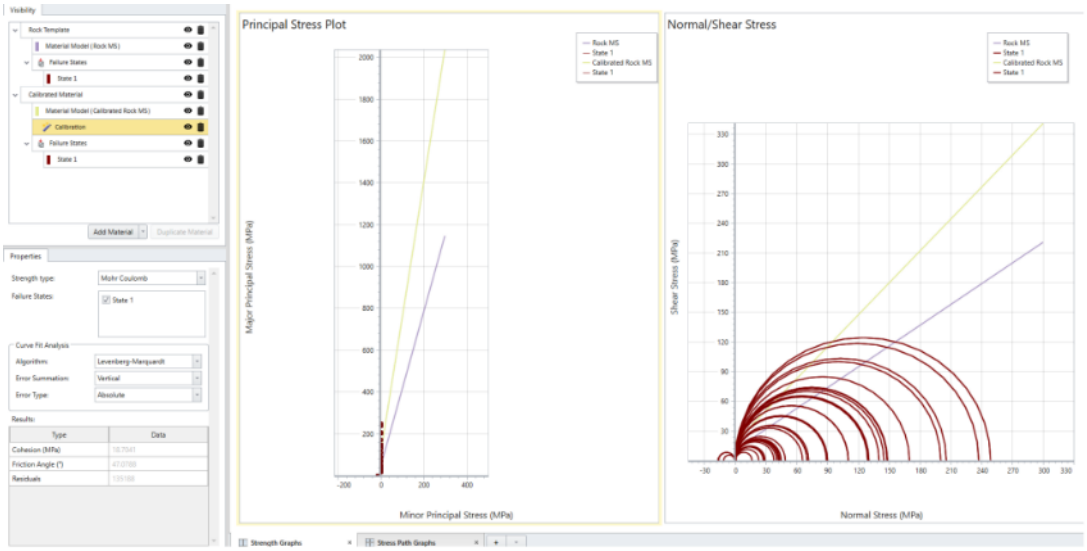


Figure 16.12 – Mohr circles for the massive sulphide (Quadrante)

Table 16.29 presents the intact rock parameters obtained from the characterisation of the massive sulphide.

Table 16.29 – Massive sulphide intact rock properties

Parameter	Unit	Value
γ_b	t/m^3	4.64
σ_c	MPa	129.74
E	MPa	118 883
Poisson's ratio (ν)	-	0.22
σ_T	MPa	15.23

16.6.4.2 GOSSAN

Young's modulus for the intact rock has an average value of 77 575 MPa, which is a high value for this type of material. The calculated modulus ratio has a value of 935.55 which is considered adequate for this sort of lithology. Chart 16-6 shows the results obtained from the modulus ratio.

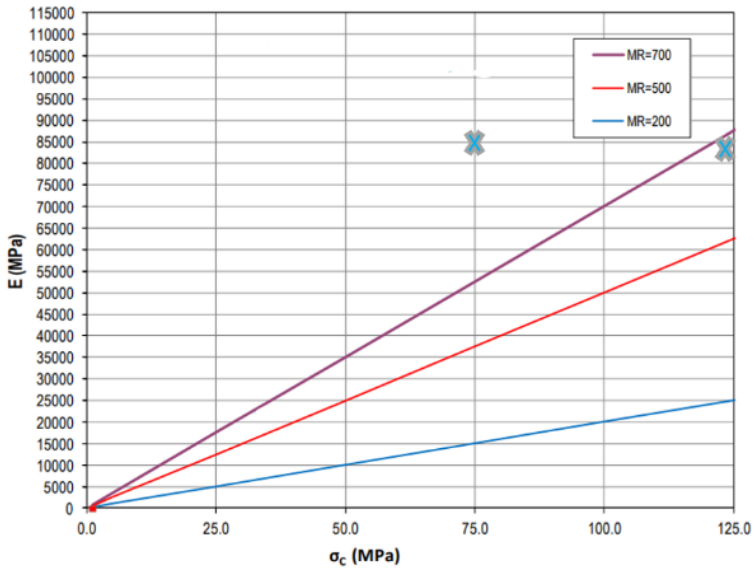


Chart 16-6 – Modulus ratio for the gossan

Assayed tensile strength has an average value of 4.21 MPa, which is also a high value for this type of lithology. The relation between tensile strength and uniaxial compressive strength was also plotted and used to calculate the toughness value. As shown in Chart 16-7, the obtained toughness value is 9.19.

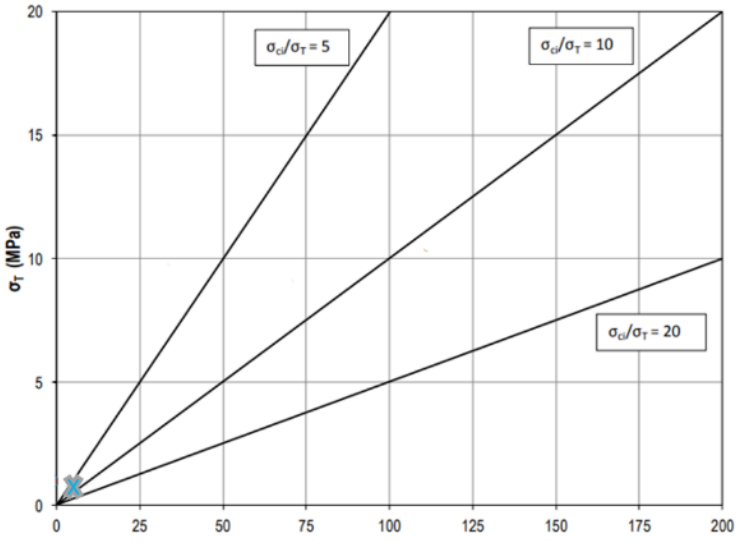


Chart 16-7 – Ratio between tensile strength and UCS for the gossan

Figure 16.13 presents the obtained Mohr circles for the gossan, as provided by Quadrante.

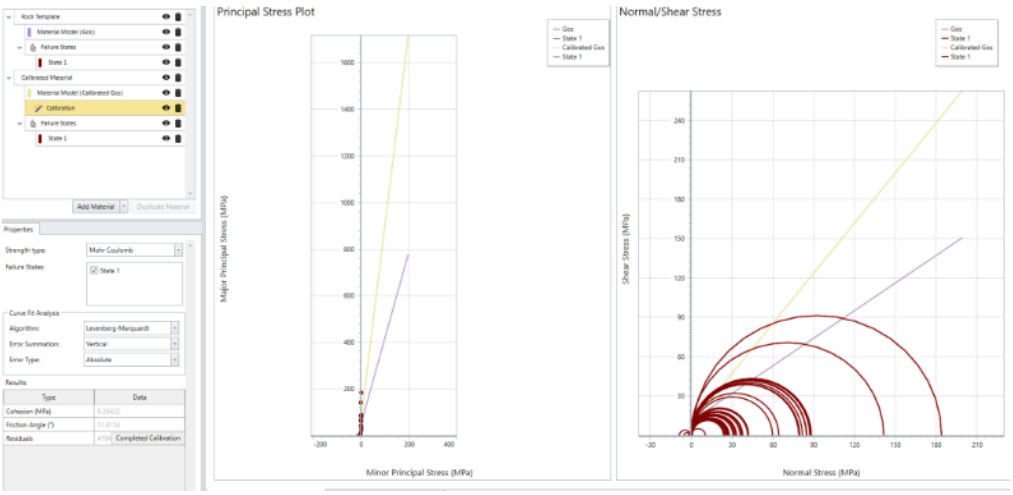


Figure 16.13 – Mohr circles for the gossan (Quadrante)

The intact rock parameters obtained for the gossan are shown in Table 16.30.

Table 16.30 – Gossan intact rock properties

Parameter	Unit	Value
γ_b	t/m^3	2.60
σ_c	MPa	69.11
E	MPa	77 575
Poisson's ratio (ν)	-	0.36
σ_T	MPa	4.21

16.6.4.3 DTV/AV

Young's modulus for intact rock has an average value of 43 360 MPa, which is a usual value for this type of material. The modulus ratio value is 1 950, with Chart 16-8 plotting the assayed values.

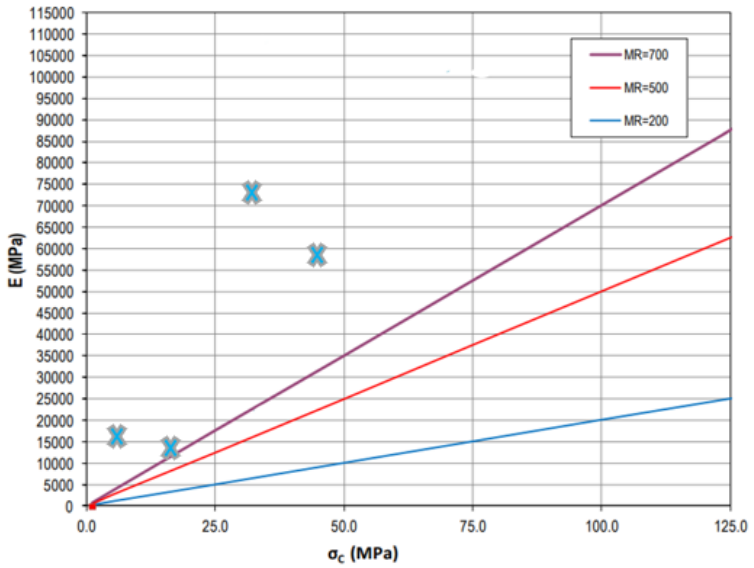


Chart 16-8 – Modulus ratio for the dtV/aV

Tensile strength has an average value of 3.10 MPa, which is also within the typical range of values for this type of lithology.

The relation between tensile strength and uniaxial compressive strength was also represented to estimate the toughness value. As shown in Chart 16-9, the obtained toughness value is 3.06.

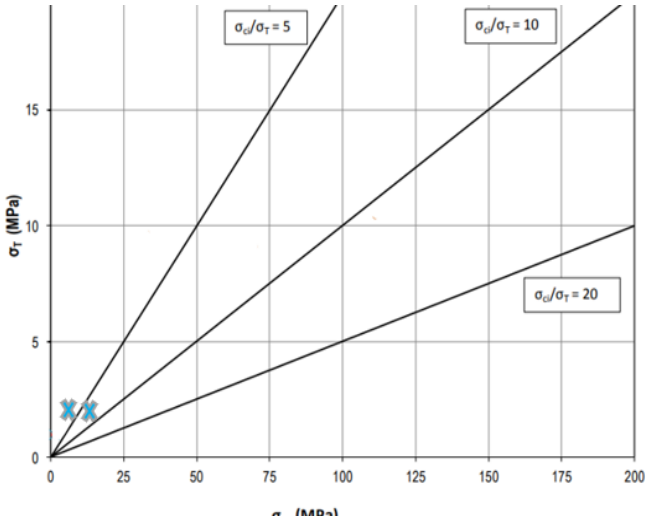


Chart 16-9 – Ratio between tensile strength and UCS for the dtV/aV

The Mohr circles for the dtV/aV are shown in Figure 16.14.

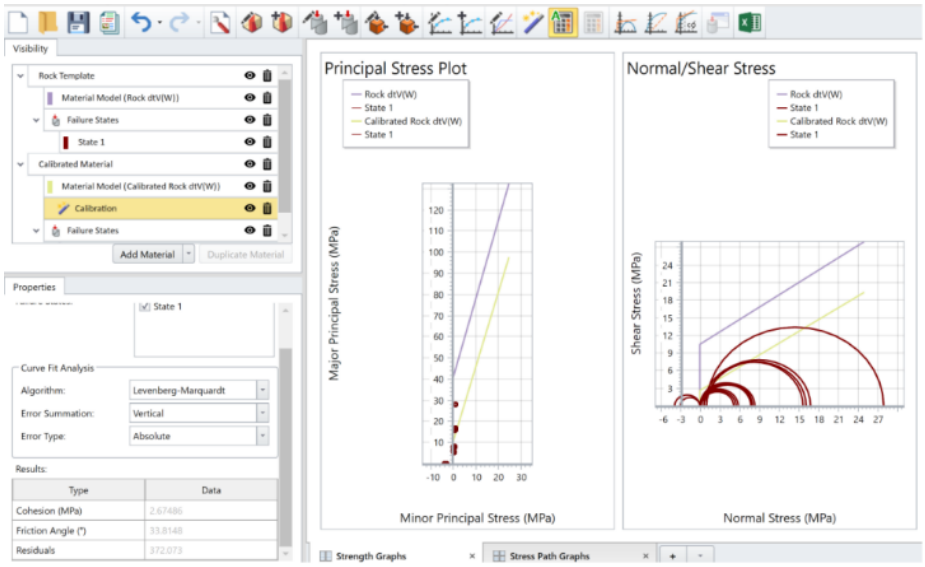


Figure 16.14 – Mohr circles for the dtV/aV (Quadrante)

Table 16.31 presents the intact rock parameters obtained from the characterisation of the dtV/aV.

Table 16.31 – dtV/aV intact rock properties

Parameter	Unit	Value
γ_b	t/m^3	2.81
σ_c	MPa	20.85
E	MPa	43 360
Poisson's ratio (ν)	-	0.20
σ_T	MPa	3.10

16.6.4.4 TERTIARY

Young's modulus for intact rock has an average value of 16 900 MPa which is an adequate value for this type of material.

The modulus ratio was calculated for all the assayed samples and an average value of 550 is obtained. Chart 16-10 plots the results obtained from the modulus ratio.

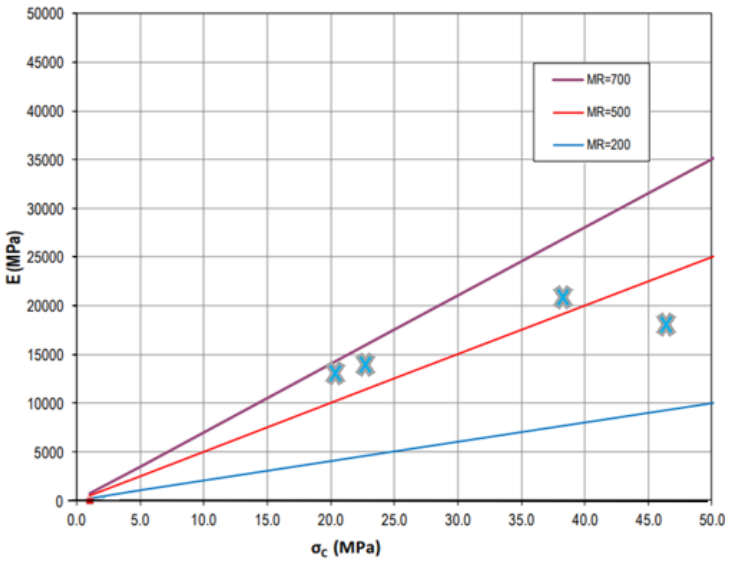


Chart 16-10 – Modulus ratio for the tertiary

Tensile strength has an average value of 7.14 MPa which is also an adequate value for this type of lithology.

The relation between tensile strength and uniaxial compressive strength was also represented to get the toughness value. As it is shown in Chart 16-11.

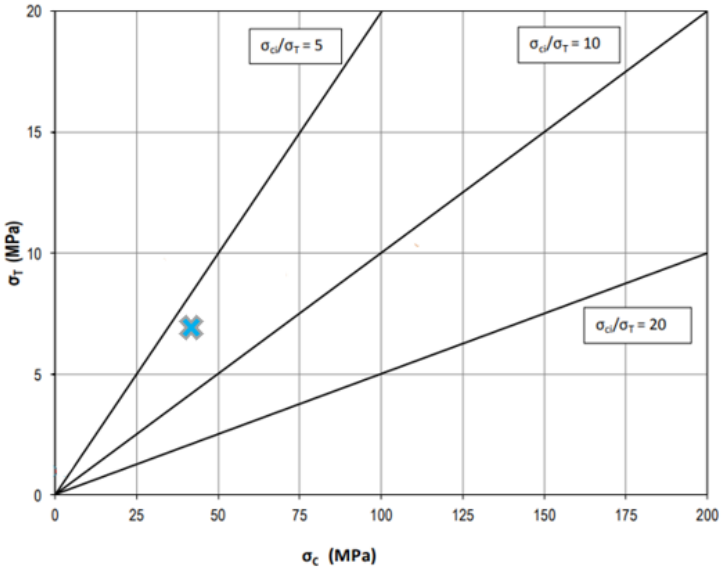


Chart 16-11 – Ratio between tensile strength and UCS for the tertiary

Figure 16.15 details the Mohr circles for the tertiary provided by Quadrante.

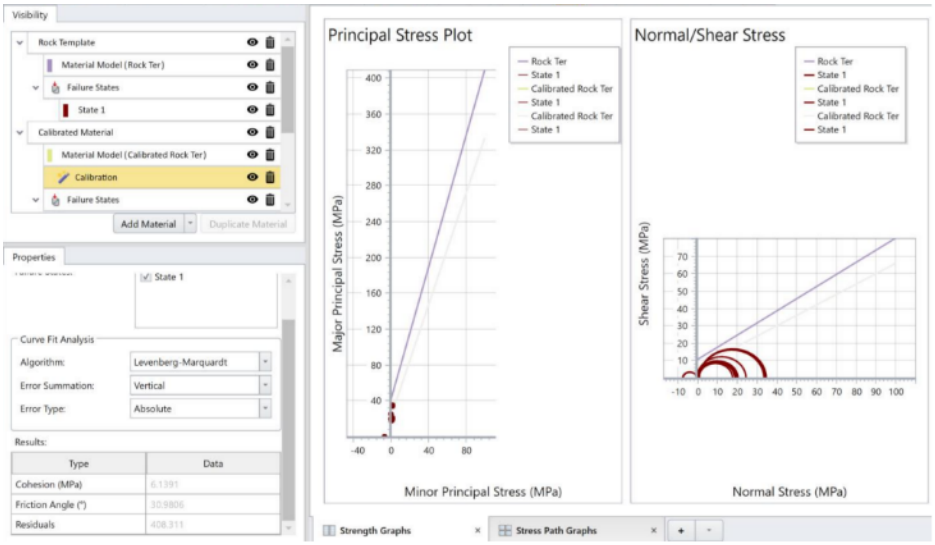


Figure 16.15 – Mohr circles for the tertiary (Quadrante)

The intact rock parameters obtained for the gossan are shown in Table 16.32.

Table 16.32 – Tertiary intact rock properties

Parameter	Unit	Value
γ_b	t/m^3	2.24
σ_c	MPa	33.06
E	MPa	16 900.00
Poisson's ratio (ν)	-	0.26
σ_T	MPa	7.14

Rock mass geotechnical parameters were derived from intact rock assay results, which were then extrapolated to the rock massif with rock mass reduction factors.

16.6.5 ROCK MASS CHARACTERISATION

Rock mass characterization was made considering the intact rock characterization and data available from the geotechnical drilling campaign. Long and transverse sections were made to identify the distribution of the Q values within the orebodies. Figure 16.16 to Figure 16.18 shows plan views of each level in the North and Figure 16.19 to Figure 16.21 in the South with the geotechnical boreholes.

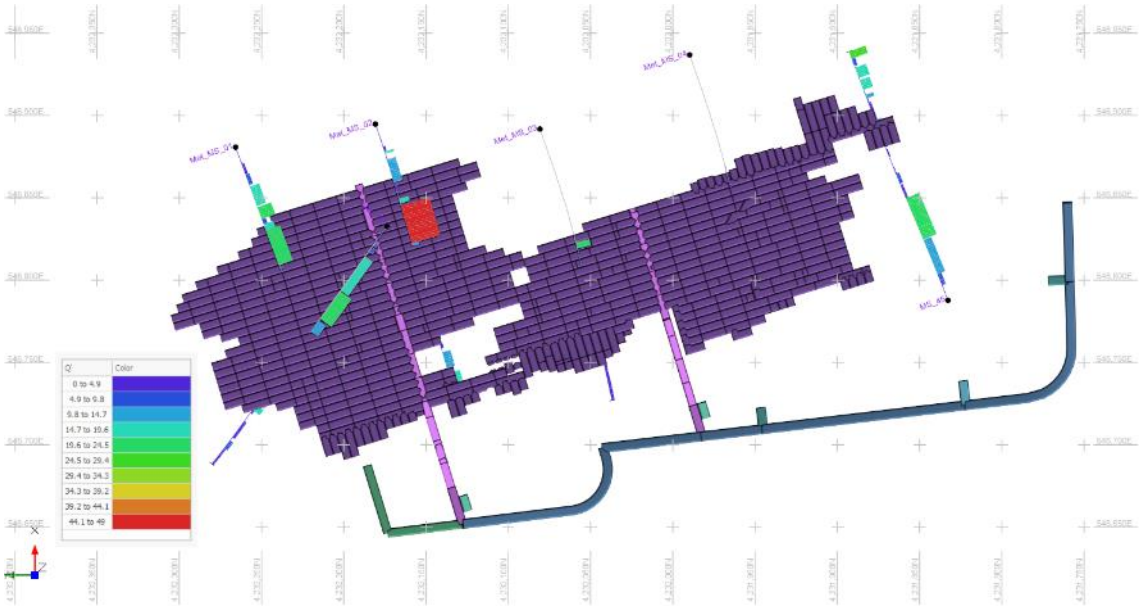


Figure 16.16 – North level 50 (plan view)

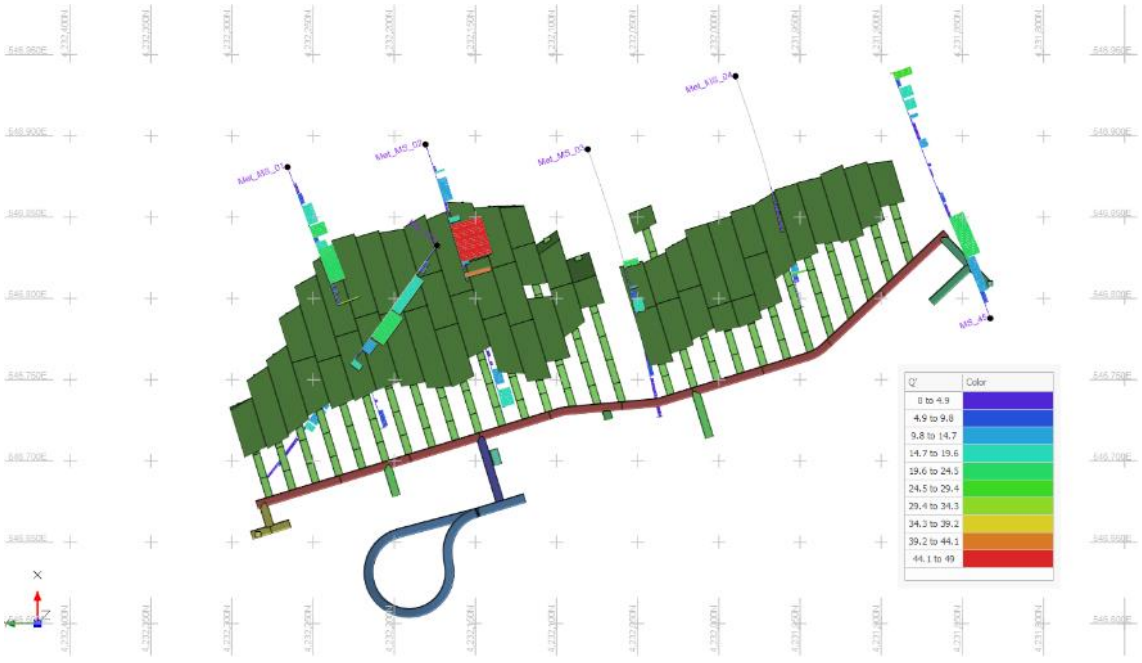


Figure 17.17 – North level 100 (plan view)

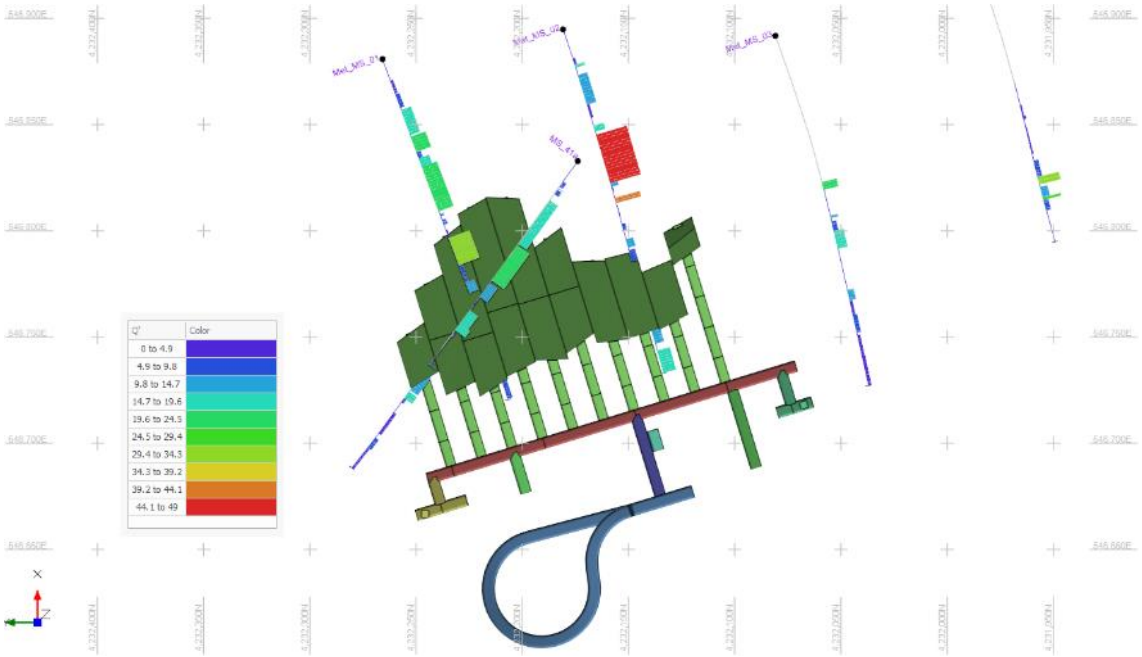


Figure 16.18 – North level 150 (plan view)

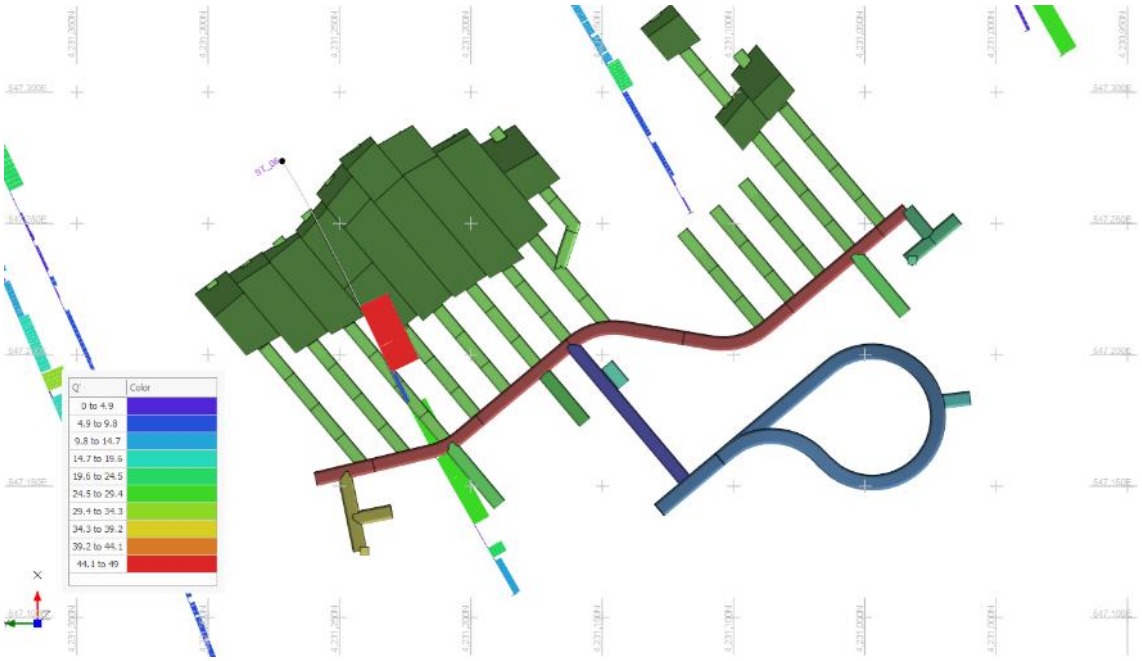


Figure 16.19 – South level 75 (plan view)

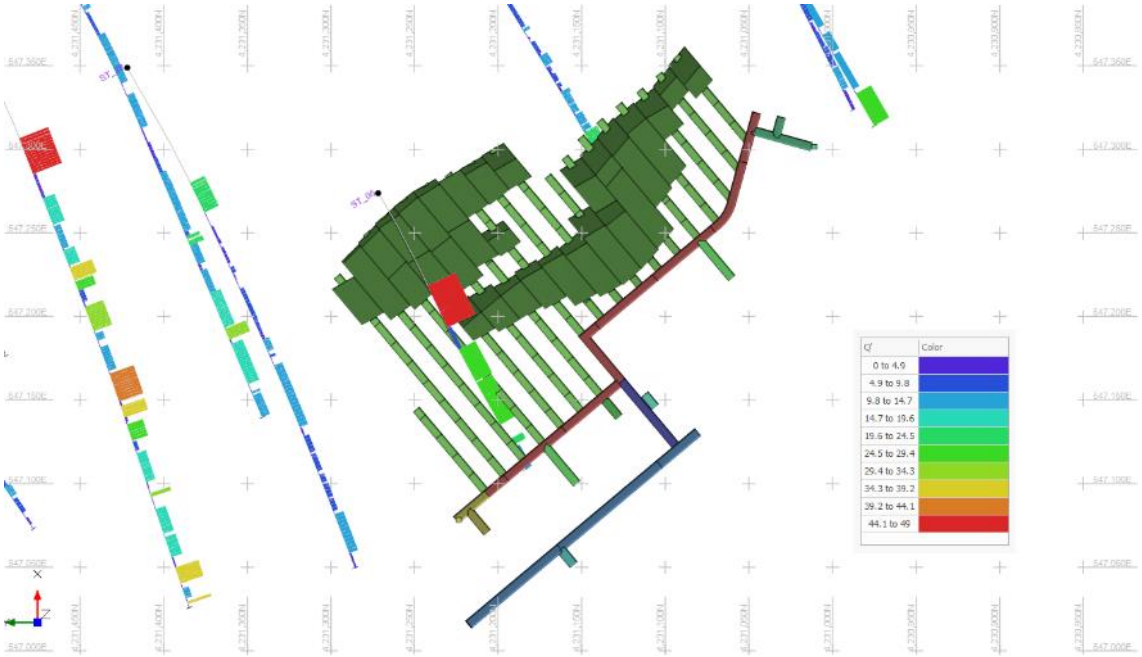


Figure 16.20 – South level 125 (plan view)

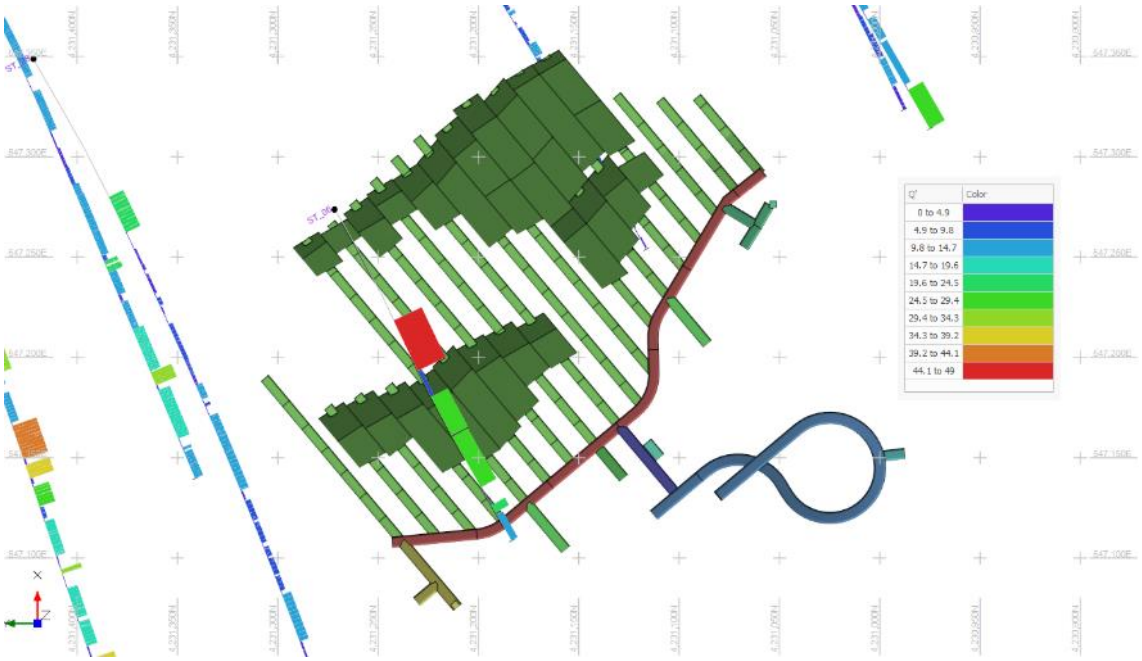


Figure 16.21 – South level 200 (plan view)

16.6.6 GEOTECHNICAL DOMAINS

Considering both the assayed Q values and the spatial distribution of the drillhole data, a total of five geotechnical domains were defined as shown in Table 16.33, dividing the mine into the different domains as shown in Figure 16.22.

Table 16.33 – Geotechnical domain definition

Domain	Type	Q
DG I	Very poor	0.01 – 0.1
DG II	Poor quality	0.1 - 4
DGIII	Fair quality	4 - 10
DG IV	Good quality	10 - 40
DG V	Very Good quality	>40

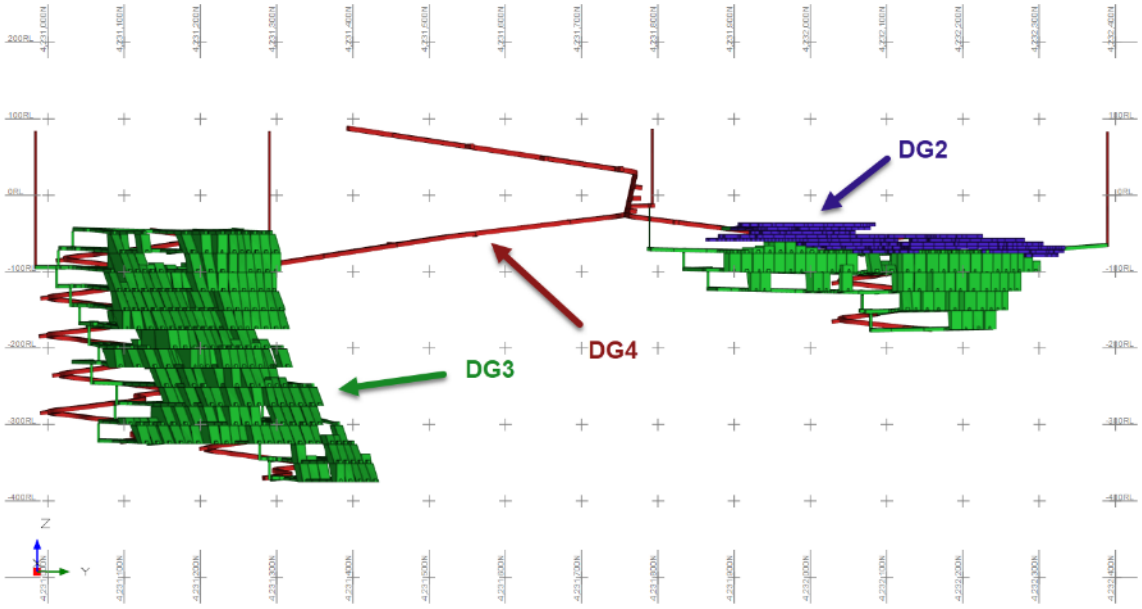


Figure 16.22 – Geotechnical domains in the mine

16.7 UNDERGROUND ROCK MECHANICS

16.7.1 MINE DRIFT DEVELOPMENT SUPPORT DESIGN

This section describes the ground support required for the underground development of the mine. It defines the minimum safety conditions that must be met by the support of the underground excavations.

Starting from the available geotechnical information, the Grimstad & Barton support chart method was chosen by IGAN to design a preliminary support. Barton's Q index, which is related to Bieniawski's RMR, was used to estimate the support in the excavations. To design the support to be installed, Barton uses the dimensions of the excavation and the use of the drift (ESR). Based on these, the index defines the equivalent dimension ("D_e") of the excavation:

$$D_e = \frac{B}{ESR}$$

Where:

- B: width of the excavation (m).
- ESR: stress level parameter obtained from Table 16.34. The value of this parameter depends on the development type.

Table 16.34 – ESR value types (Grimstad and Barton, 1993)

Domain	Type	Q
DG I	Very poor	0.01 – 0.1
DG II	Poor quality	0.1 - 4
DGIII	Fair quality	4 - 10
DG IV	Good quality	10 - 40
DG V	Very Good quality	>40

Employing the equivalent dimension and the Q value, the support can be estimated in the chart shown in Figure 16.23.

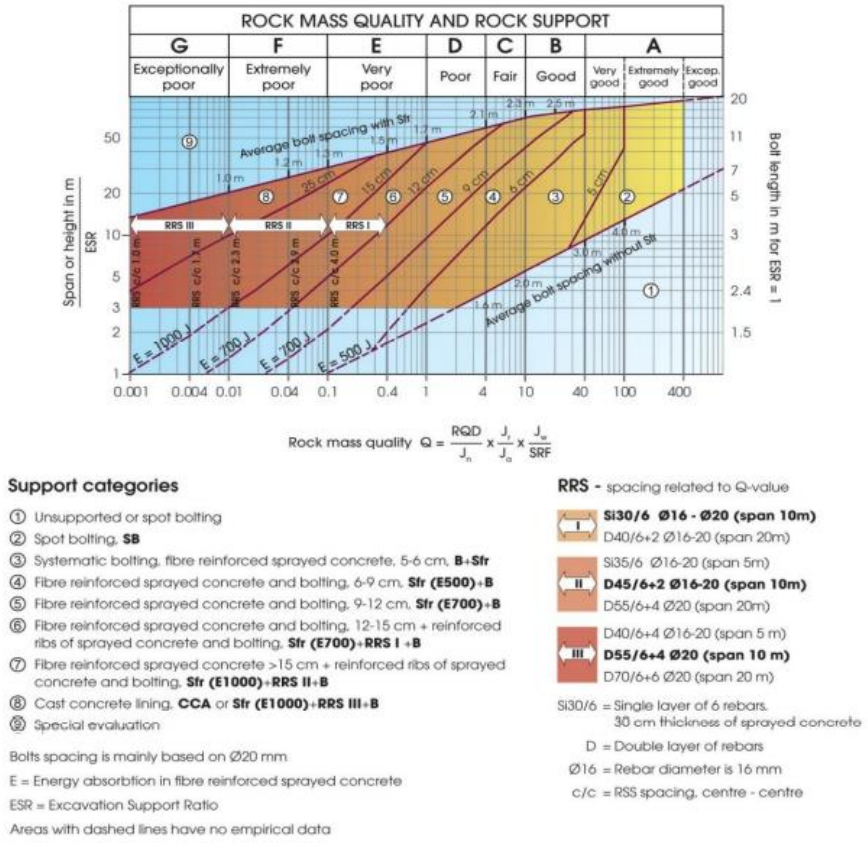


Figure 16.23 – Approximate determination of the support of a tunnel

16.7.1.1 MAIN DECLINE

The equivalent dimension of the excavation is show in **Table 16.35**.

Table 16.35 – Main decline: equivalent dimension evaluation

Parameter	Unit	Value	Comments
Section height	m	6.5	
Section width	m	5.5	
ESR		1.6	Permanently open mines, pilot tunnels
Equivalent dimension		3.4	

From the data provided above, it can be extrapolated that the value of Q is likely to indicate a rock quality between poor, fair and good, which thus establishes the types of support needed, as shown in Figure 16.24.

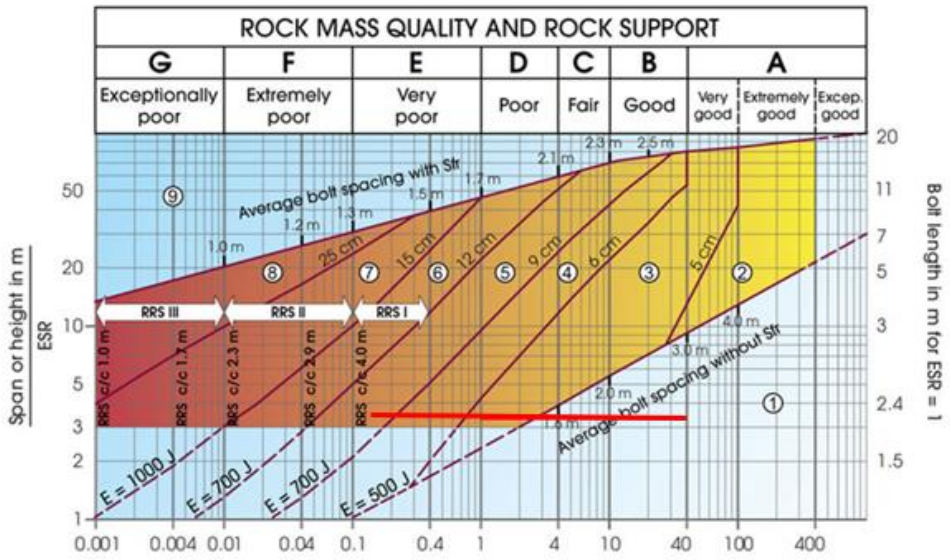


Figure 16.24 – Main decline recommended support

With the empirical results obtained from the Barton Chart and IGAN's own experience in similar operations, the recommended support for good quality material is formed by 9 bolts with a length of 2.4 m, these bolts can be galvanized split set or cemented/resin steel rebar bolts. The spacing between the sections will be between 1.5 m and 1.8 m. No mesh is required to be installed.

In ground of fair rock quality, the type of support will consist in 10 2.4 m-long bolts. These can be either galvanized split set or cemented/resin steel rebar bolts. The spacing between the sections will be 1.5 m and, additionally, 5 cm of shotcrete will be sprayed onto the walls.

In poor quality terrain, it is recommended to install 10 bolts of 2.4 m length per ring, with a reinforcement of 8 cm of shotcrete.

In cases where the decline intersects very poor quality or faulted zones, an ad-hoc support design study will be required to be conducted, so a tailored solution is developed for each case. Exceptional support items such as trusses, micropiles and self-drilling bolts could be used as needed, with drainage systems when water is present. Table 16.36 summarises the recommended support patterns for the main decline on a domain-by-domain basis.

Table 16.36 – Main decline: support pattern per domain

Domain	Type	Q	Bolt spacing (m)	Section spacing (m)	Bolts per ring	Mesh	Shotcrete thickness (cm)
DG I	Very poor	0.01 – 0.1	1.0	1.0	12	Yes	10
DG II	Poor quality	0.1 - 4	1.5	1.5	10	Yes	8

Domain	Type	Q	Bolt spacing (m)	Section spacing (m)	Bolts per ring	Mesh	Shotcrete thickness (cm)
DG III	Fair quality	4 - 10	1.5	1.5	10	No	5
DG IV	Good quality	10 - 40	1.5	1.5/1.8	9	No	0
DG V	Very Good quality	>40	2.0	2.0	8	No	0

It is anticipated that the support will be applied as follows:

- Portal support type: first 30 m.
- DG II: next 50 m.
- DG IV: general support pattern in the ramp.
- An allowance of 3% has been considered for fault areas in the ramp, that will require support pattern DG I.

16.7.1.2 CAPEX LATERAL DEVELOPMENT

The input parameters considered in the calculation of the equivalent dimension of CAPEX lateral development are shown in Table 16.37.

Table 16.37 – CAPEX development: equivalent dimension evaluation

Parameter	Unit	Value	Comments
Section height	m	5.5	
Section width	m	5.5	
ESR		1.6	Permanently open mines, pilot tunnels
Equivalent dimension		3.4	

With this, it can be extrapolated that the value of Q is likely to indicate a rock quality between poor, fair and good, which thus establishes the types of support needed, as shown in Figure 16.25.

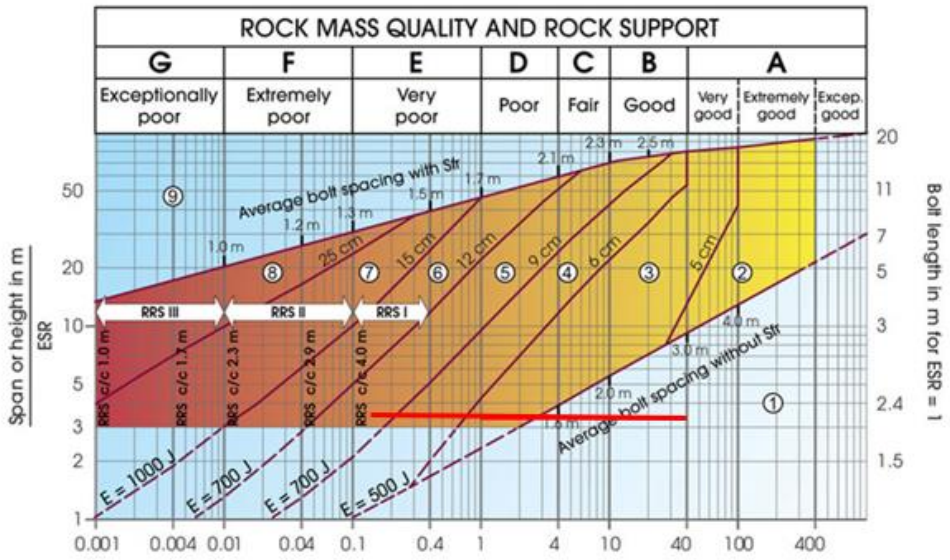


Figure 16.25 – CAPEX lateral development recommended support chart

Based on both the results obtained from the chart as well as IGAN's own experience in similar operations, the recommended support for good quality material is comprised by 8 2.4-m long bolts; these bolts can be either galvanized split set or cemented/resin steel rebar bolts. The spacing between the rings will be between 1.5 m and 1.8 m. No mesh is required to be installed.

In fair quality rock, the recommended standardised support layout will consist of 9 2.4m long bolts, installed in rings spaced 1.5 m apart. The bolts can be either of galvanized split set or cemented/resin steel rebar type. Additionally, 5 cm of shotcrete will be sprayed onto the drift walls.

In poor quality terrain, the recommended support layout is comprised by 9, 2.4 m long bolts, installed as rings that are spaced 1.5 m from each other. 8 cm of shotcrete is sprayed onto the walls as well.

It the cases where the CAPEX development infrastructure intersects very poor quality or fault zones, an ad-hoc study will be required to develop a tailored solution for each case. This may include the use of trusses, micropiles and self-drilling bolts where needed, or drainage systems if water is present. Support patters for the CAPEX development are shown in Table 16.38.

Table 16.38 – CAPEX development: support pattern per domain

Domain	Type	Q	Bolt spacing (m)	Section spacing (m)	Bolts per ring	Mesh	Shotcrete thickness (cm)
DG I	Very poor	0.01 – 0.1	1.0	1.0	12	Yes	10

Domain	Type	Q	Bolt spacing (m)	Section spacing (m)	Bolts per ring	Mesh	Shotcrete thickness (cm)
DG II	Poor quality	0.1 - 4	1.5	1.5	10	Yes	8
DG III	Fair quality	4 - 10	1.5	1.5	10	No	5
DG IV	Good quality	10 - 40	1.5	1.5/1.8	9	No	0
DG V	Very Good quality	>40	2.0	2.0	8	No	0

It is anticipated that the CAPEX development support patterns will be installed as follows:

- DG III: general support pattern.
- An allowance of 5% has been considered for fault areas in the CAPEX development, that will require support pattern DG I.

16.7.1.3 STOPE ACCESS DRIFTS

The input parameters considered in the calculation of the equivalent dimension of stope access drifts are shown in Table 16.39.

Table 16.39 – Stope access drifts: equivalent dimension evaluation

Parameter	Unit	Value	Comments
Section height	m	4.5	
Section width	m	5.0	
ESR		3	Temporary mining
Equivalent dimension		1.6	

Together with the Q values, the equivalent dimension indicates that the rock quality to be found ranges between poor, fair and good, as shown in Figure 16.26.

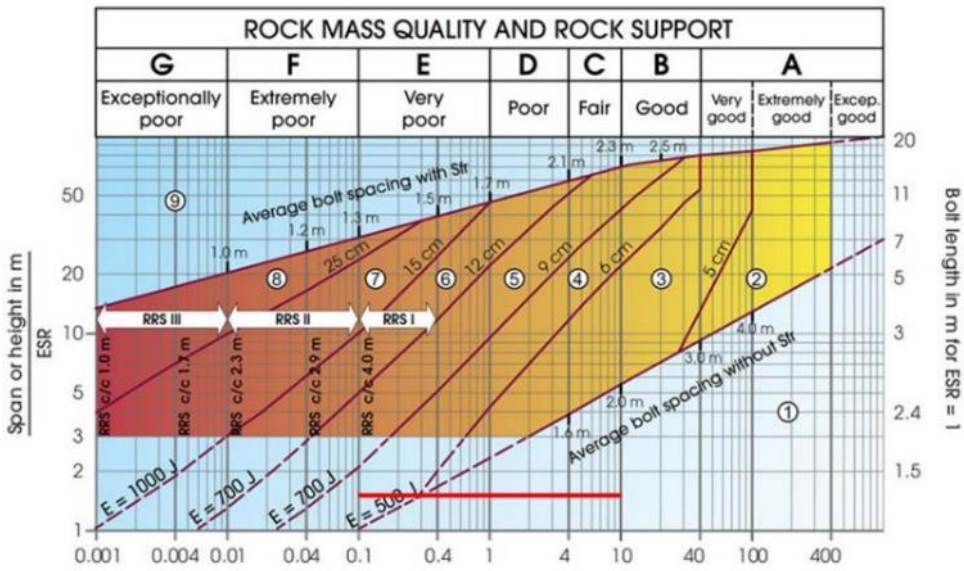


Figure 16.26 – Stope access drift recommended support

Based on both the results obtained from the chart as well as IGAN's own experience in similar operations, the recommended support for good and quality material is comprised by 8 2.4-m long galvanized split set bolts. The spacing between the rings will be 1.5 m.

In poor quality terrain, it is recommended to install 9 2.4m bolts per ring section with a 1.5 m ring spacing. Additionally, 5 cm of shotcrete will be installed as well.

It the cases where the access drift intersects very poor quality or fault zones, an ad-hoc study will be required to develop a tailored solution for each case. This may include the use of trusses, micropiles and self-drilling bolts where needed, or drainage systems if water is present. Support patters for the CAPEX development are shown in Table 16.40.

Table 16.40 – Stope access drift: support pattern per domain

Domain	Type	Q	Bolt spacing (m)	Section spacing (m)	Bolts per ring	Mesh	Shotcrete thickness (cm)
DG I	Very poor	0.01 – 0.1	1.0	1.0	12	Yes	10
DG II	Poor quality	0.1 - 4	1.5	1.5	10	Yes	8
DG III	Fair quality	4 - 10	1.5	1.5	10	No	5
DG IV	Good quality	10 - 40	1.5	1.5/1.8	9	No	0
DG V	Very Good quality	>40	2.0	2.0	8	No	0

It is anticipated that the patterns will be installed as follows:

- DG III: general support pattern.
- An allowance of 5% has been considered for fault areas in the stope access drifts, that will require support pattern DG I.

16.7.1.4 CUT&FILL DRIVES

The input parameters considered in the calculation of the equivalent dimension of cut&fill drifts are shown in Table 16.41.

Table 16.41 – Stope access drifts: equivalent dimension evaluation

Parameter	Unit	Value	Comments
Section height	m	5.0	
Section width	m	5.0	
ESR		3	Temporary mining
Equivalent dimension		1.6	

It is thus estimated that rock qualities of extremely poor, very poor and poor quality may be encountered, which thus establishes the types of support needed, as shown in Figure 16.27.

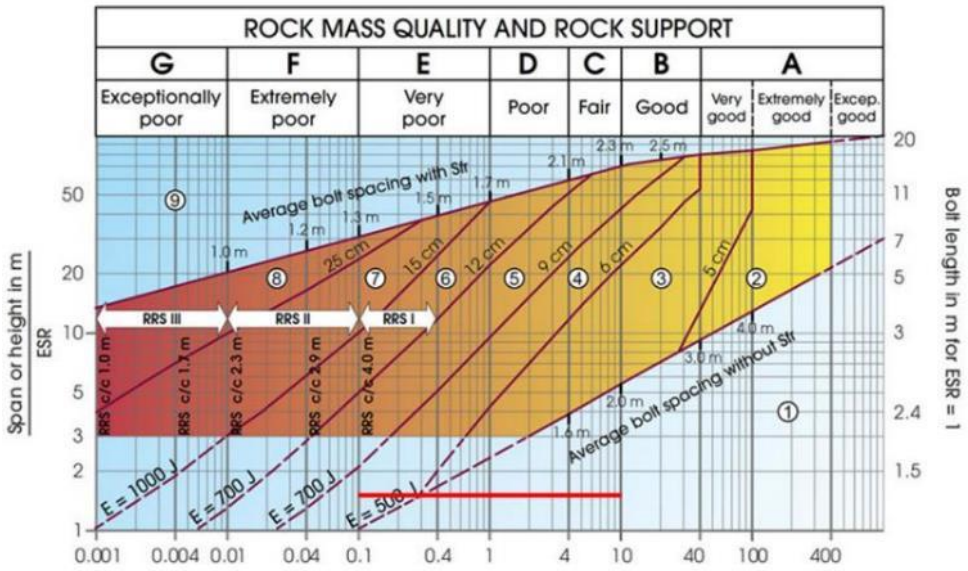


Figure 16.27 – Cut&Fill recommended support

A determining characteristic for the type of support to be placed in the cut&fill drift, is that the exposure time (the time during which these drifts need to be open) is very small. A typical cut&fill face will be excavated and filled within a few months.

It is thus recommended to install swellex type bolts, which is typically cheap and easy to install, while having an adequate load bearing capacity.

When encountering poor quality terrain, it is recommended to install 9 2.4m long swellex bolts per ring. Additionally, a 5 cm shotcrete reinforcement layer is to be sprayed onto the walls. If the material is very poor, it is recommended to install 10 2.4 m long swellex bolt rings, spaced 1 m from each other. An 8 cm shotcrete reinforcement layer and mesh are to be installed as well.

Where cut&fill development cross extremely poor material and in addition to the already recommended support, it will be required to both reduce the round length and install rib-bar shotcrete ribs. Table 16.42shows the list of standard support patterns.

Table 16.42 – Cut&fill drift: support pattern per domain

Domain	Type	Q	Bolt spacing (m)	Section spacing (m)	Bolts per ring	Mesh	Shotcrete thickness (cm)
DG I	Very poor	0.01 – 0.1	1.0	1.0	12	Yes	10
DG II	Poor quality	0.1 - 4	1.5	1.5	10	Yes	8
DG III	Fair quality	4 - 10	1.5	1.5	10	No	5
DG IV	Good quality	10 - 40	1.5	1.5/1.8	9	No	0
DG V	Very Good quality	>40	2.0	2.0	8	No	0

It is expected to have the following general support pattern distribution:

- DG II: general support pattern.
- An allowance of 5% has been considered for fault areas in the cut&fill development, that will require support pattern DG I.

16.7.2 GEOTECHNICAL STOPE DESIGN

The Matthews method was used to undertake a stope stability analysis. The method is based on the calculation of two factors: the stability number, N, and the shape factor or hydraulic radius, S. The former represents the capacity of the rock mass to resist under certain stress conditions while the latter accounts for the geometry of the mineable volume.

The stability number, N', is defined as:

$$N' = Q' \times A \times B \times C$$

Q is calculated from the data of the borehole logging of the rock mass. The same method proposed by the NGI Standard Rock Mass Classification System (Barton et al., 1974) is used, which is defined as:

$$Q = \frac{RQD}{J_n} \cdot \frac{J_r}{J_a} \cdot \frac{J_w}{SRF}$$

Barton's Q index can also be estimated through the following correlation with Bieniawski's RMR:

$$Q = e^{\frac{RMR-44}{9}}$$

The calculation of the stability number assumes that both the joint water reduction parameter ("J_w") and the stress reduction factor (SRF) are one = Q'. The Q' parameter was estimated through the reviewing of geotechnical log information as well as the characterization of intact rock characterisation.

The stress factor, A, reflects the stresses acting on the free face at depth. This factor is determined as the ratio between the intact rock strength (uniaxial compressive strength) and the induced compressive stress, measured at the centreline of the face. The induced stress can be calculated with numerical stress analysis methods or estimated from empirical stress distributions datasets and diagrams such as the one shown in Figure 16.28. The uniaxial compressive strength is usually obtained from laboratory tests.

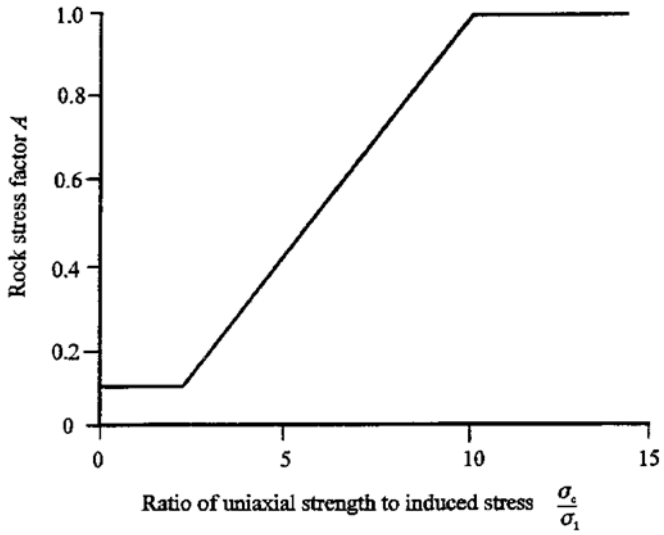


Figure 16.28 – Diagram for the estimation of the stress factor

IGAN considers that a Factor A of 1.0 is the most applicable to this project. This assumption is based on the depth of excavation (less than 500 m below surface).

The B factor, the joint orientation factor, measures the influence of the joint sets on the stability of the free faces of the stope. Generally, failure occurs along critical joints forming low angles to the free face. The influence of critical joints is highest when their alignment is parallel to the free face. On the other hand, joints perpendicular to the free faces have the least influence on stability. The B factor depends on the difference in the dip between the excavation surface and the critical joint set, as shown in Figure 16.29.

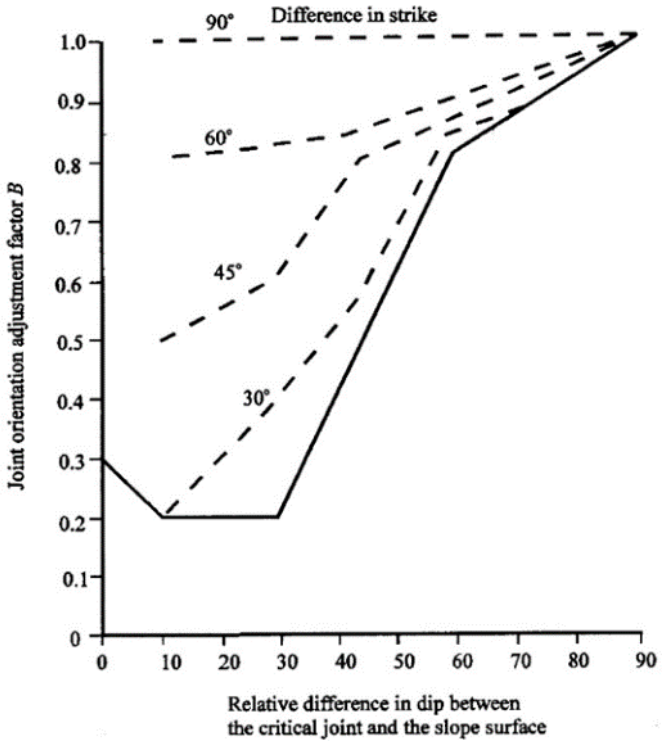


Figure 16.29 – Diagram for the estimation of the joint orientation factor

Considering both the stope surface to critical joint relative dip difference and the strike difference between discontinuity and plane trace, considered values for the B factor are 0.6 for the roof and 0.4 for the floor and sidewalls.

Due to the lack of data about joints and geological structures, these values were estimated by IGAN taking in consideration experience in similar mining operations and a conservative approach. The first value, 0.6, was estimated considering that the critical structure is vertical, and the stope roof has a maximum dip of 40°. The latter value, 0.4, considers that the critical structure is parallel to the front of the stope.

The C factor, or gravity adjustment factor, reflects the influence of the orientation of the excavation face on stability. Failures may happen from the roof by gravity-induced falls or from the sidewalls by slabbing mechanisms. According to Potvin (1988), the gravity and slabbing mechanisms depend on the inclination of the surface of the studied stope.

The C factor is calculated using the graph shown in Figure 16.30 or the following equation:

$$C = 8 - 6 \cdot \cos \alpha$$

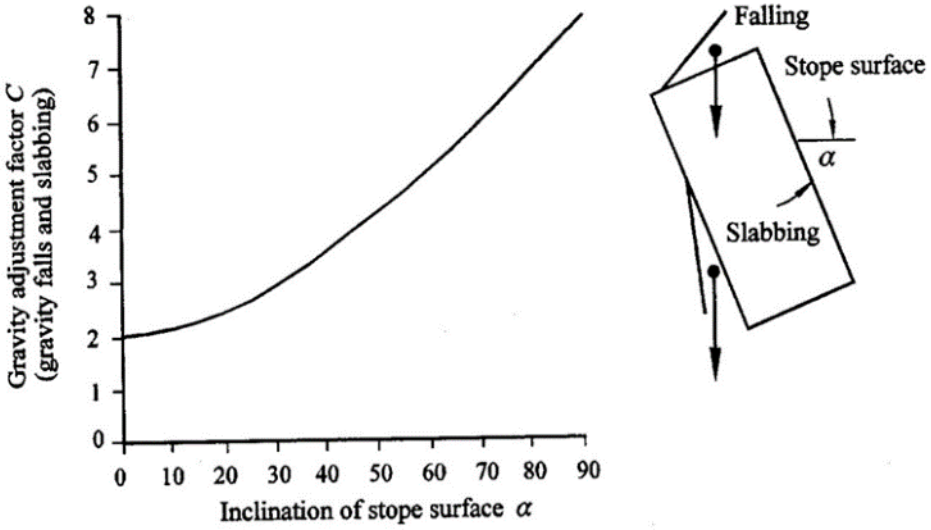


Figure 16.30 – "C" factor determination graph due to gravity or slabbing mechanisms

Sliding failures depend on the inclination (β) of the critical joints and the value of the factor C, as obtained from the Figure 16.31.

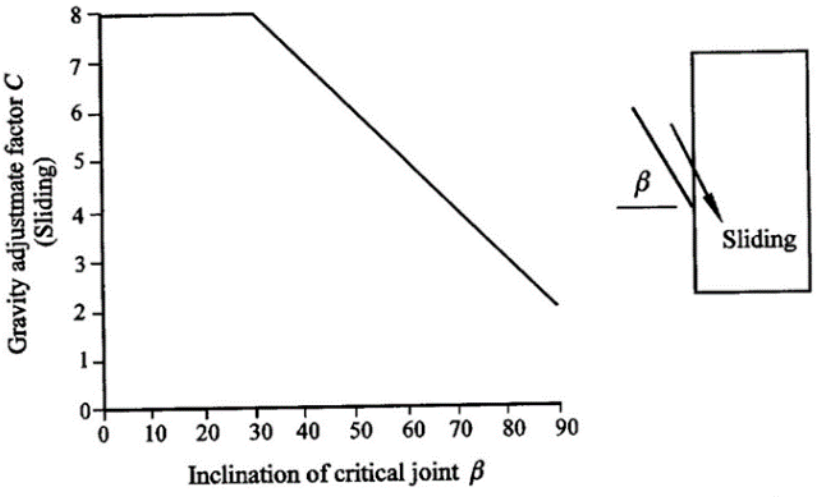


Figure 16.31 – "C" factor determination graph due to sliding mechanisms

The C factor considers the orientation of the analysed surface. The vertical wall design has a value of eight whereas the value for a horizontal roof is two. Thus, this factor reflects the inherently more stable nature of a vertical wall compared to a horizontal wall. The C factor suggests that the value of Q can be increased four times for a vertical wall compared to a horizontal roof or sidewall. In this case, values of two and six have been considered for the roof and sidewalls, respectively.

The hydraulic radius, or shape factor, for the surface under consideration is calculated as follows:

$$S = \frac{Area}{Perimeter}$$

16.7.3 STOPE DIMENSION STABILITY LIMITS

The limit hydraulic radius (HR_{limit}) is defined as the maximum hydraulic radius of an unsupported surface. This value can be used to predict the maximum surface area of the stope without support, for a given stability number, N. Using these parameters, it is possible to estimate the preliminary stope dimensions.

$$HR_{limit} = 10^{[0.573+0.338 \cdot \log N]}$$

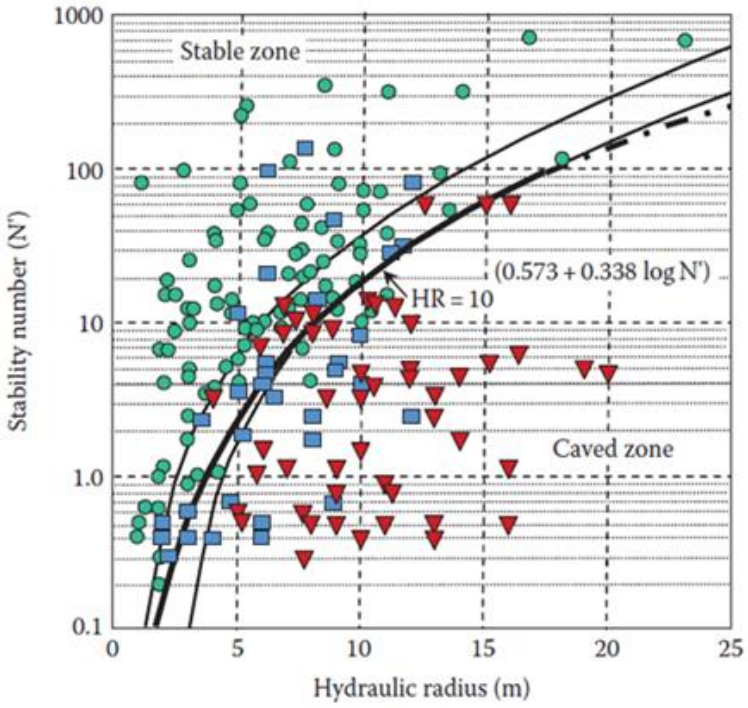


Figure 16.32 – Stability graph and determination of the stability limit (Nickson, 1992)

Based on the most restrictive value of Q' (6.0) and the determined values of A, B and C, stability numbers of 7.2 and 14.4 have been calculated for the roof, sidewalls and stope front, respectively. The corresponding values for each case are shown in **Table 16.43**.

Table 16.43 – Stope roof and sidewall stability graph parameters

Surface	N'	A	B	C
Roof	7.2	1	0.6	2
Sidewall	14.4	1	0.4	6
Front	14.4	1	0.4	6

Table 16.44 presents the values for the limit hydraulic radius for the stope roof, sidewalls, and front.

Table 16.44 – Stope roof, sidewall and front limit hydraulic radius parameters

Surface	N'	HR_{limit}
Roof	7.2	7.3
Sidewall	14.4	9.2
Front	14.4	9.2

Stope width of 15 m and height of 25 m have been selected for this project. These dimensions are consistent with those used in similar mining operations within the Iberian Pyritic Belt. This would also mean that similar equipment, work methods and operational experience could be applicable to this deposit. Table 16.45 presents the hydraulic radius for the roof considering a stope length of 50 m.

Table 16.45 – Roof: Hydraulic radius calculation

Parameter	Unit	Value
Width	m	15
Length	m	50
Hydraulic radius	m	5.8

As shown in Figure 16.33, the estimated hydraulic radius of value of 5.8 places the suggested stope size within the transition zone (i.e., the zone of semi-stability without any support).

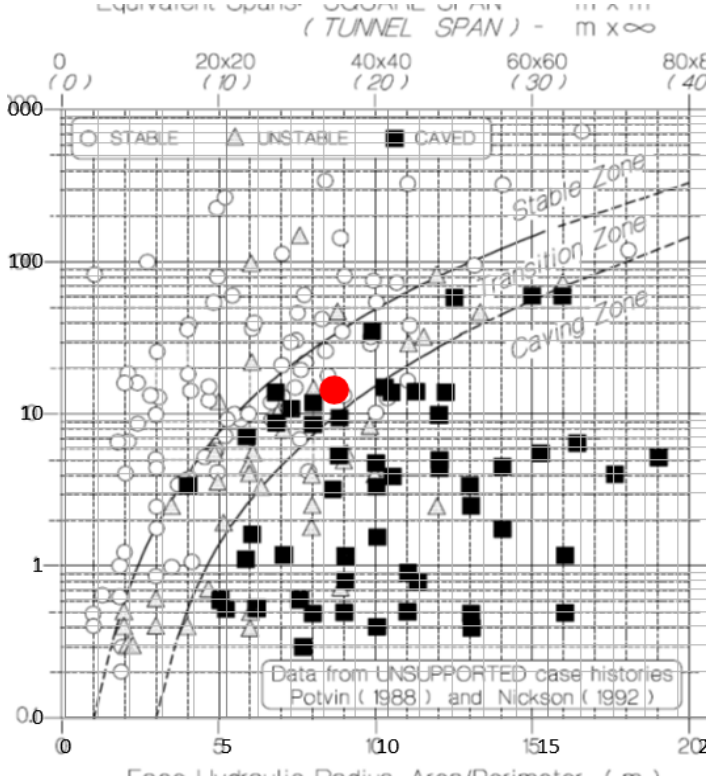


Figure 16.33 – Graph for the stope roof without support

When considering a typical support pattern for sublevel stoping, as shown in Table 16.46, the stope roof is within the stability region as it can be seen in Figure 16.34.

Table 16.46 – Recommended cablebolt support spacing and length

Parameter	Unit	Value
Support element	-	Double strand cablebolt
Spacing	m	2
Length	m	6-90

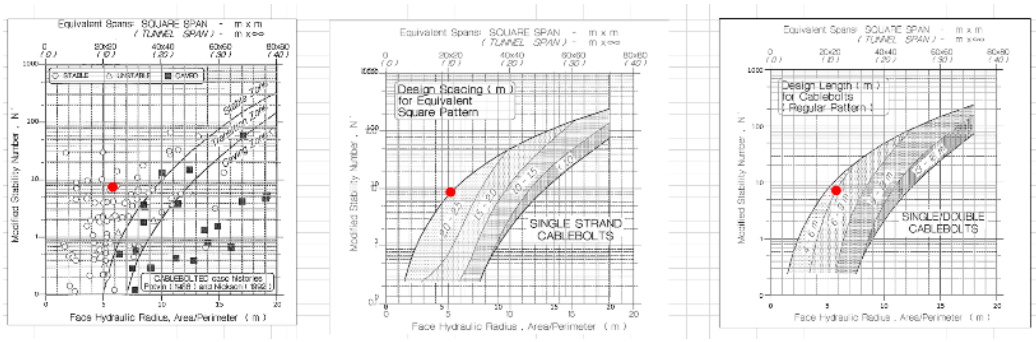


Figure 16.34 – Graph for stope roofs with support

Table 16.47 presents the hydraulic radius for the sidewalls.

Table 16.47 – Sidewalls: Hydraulic radius calculation

Parameter	Unit	Value
Width	m	25
Length	m	50
Hydraulic radius	m	8.3

As shown in Figure 16.35 the calculated value of 8.3 places the stope dimensions within the stable or transition zone. There is no need of additional support to be installed in the sidewalls.

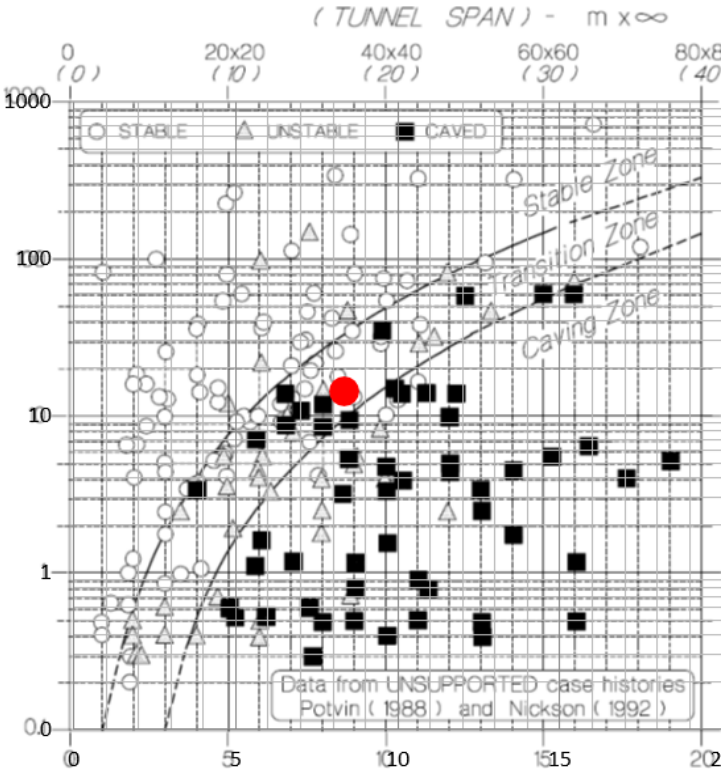


Figure 16.35 – Graph for the stope sidewalls without support

Table 16.48 presents the hydraulic radius for the stope front.

Table 16.48 – Front: Hydraulic radius calculation

Parameter	Unit	Value
Width	m	25
Length	m	15
Hydraulic radius	m	4.7

As shown in Figure 16.36 the value of 4.7 is below the theorical calculated limit value and thus, within the stable zone. There is no need for additional support to be installed on the stope front.

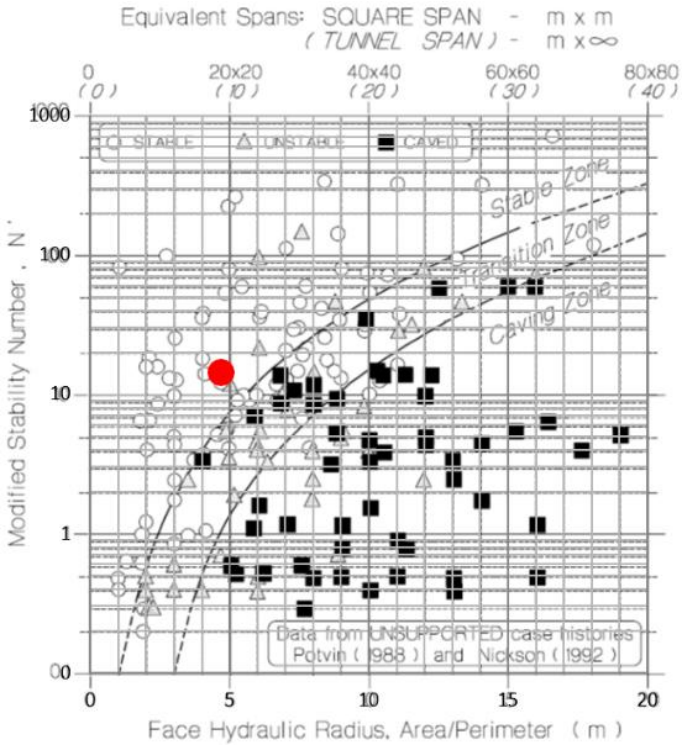


Figure 16.36 – Stope front stability without support

16.7.4 STOPE SUPPORT PATTERN

A basic design of a typical support pattern for a stope is presented in Figure 16.37 and Table 16.49, with a cable length that meets the criteria obtained from the stability analysis.

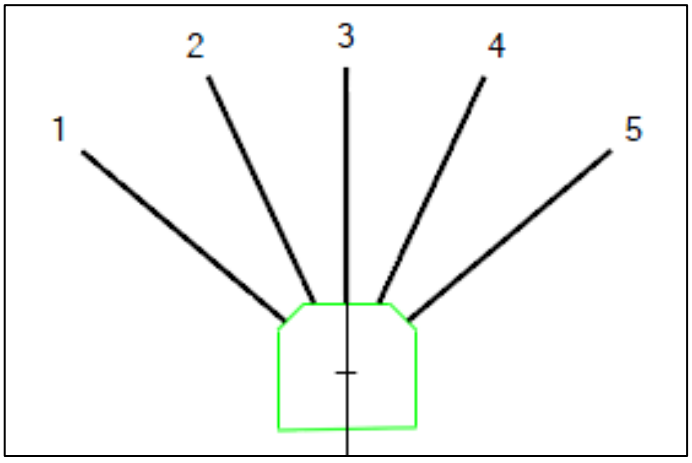


Figure 16.37 – Typical cablebolt pattern

Table 16.49 – Cablebolt support desing parameters

Cablebolt number	Length (m)	Type
1	9	Double cablebolt
2	7	Double cablebolt
3	6	Double cablebolt
4	7	Double cablebolt
5	9	Double cablebolt

16.7.5 2D MODELLING

Midas GTS was used to create a two-dimensional ("2D") stope stability model that validates the results obtained by the Mathews method.

A geometric model, as shown in Figure 16.38, was developed in the software. The model considers two primary stopes and a secondary, within the same level.

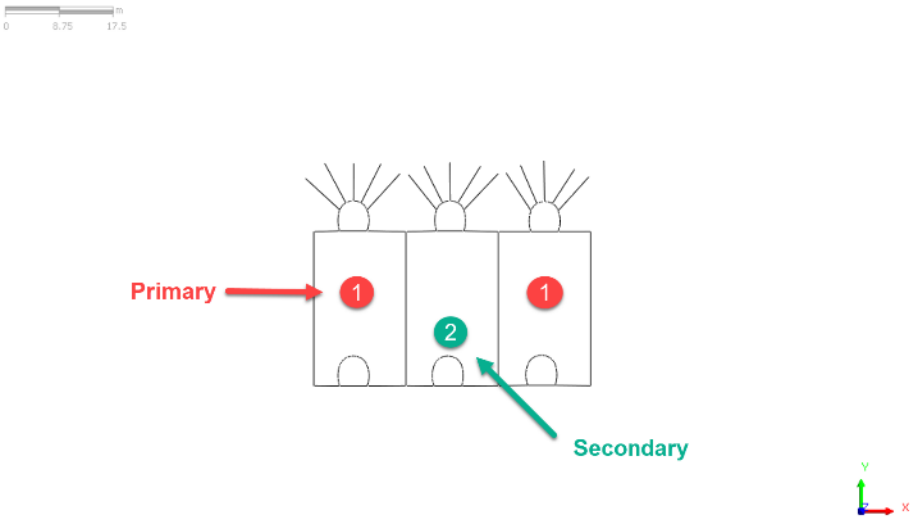


Figure 16.38 – Primary and secondary stopes (transverse section)

The extraction sequence, described as follows, was evaluated in the software as a series of steps:

- Primary stope mining: ore access drifts development; cablebolt support installation at the top sill drift, stope excavation.
- Primary stope backfill: primary stopes are backfilled with cemented paste backfill.
- Secondary stope mining: ore access drift development (top and bottom), cablebolt support and stope mining.

Figure 16.39 shows the model mesh and boundary conditions.

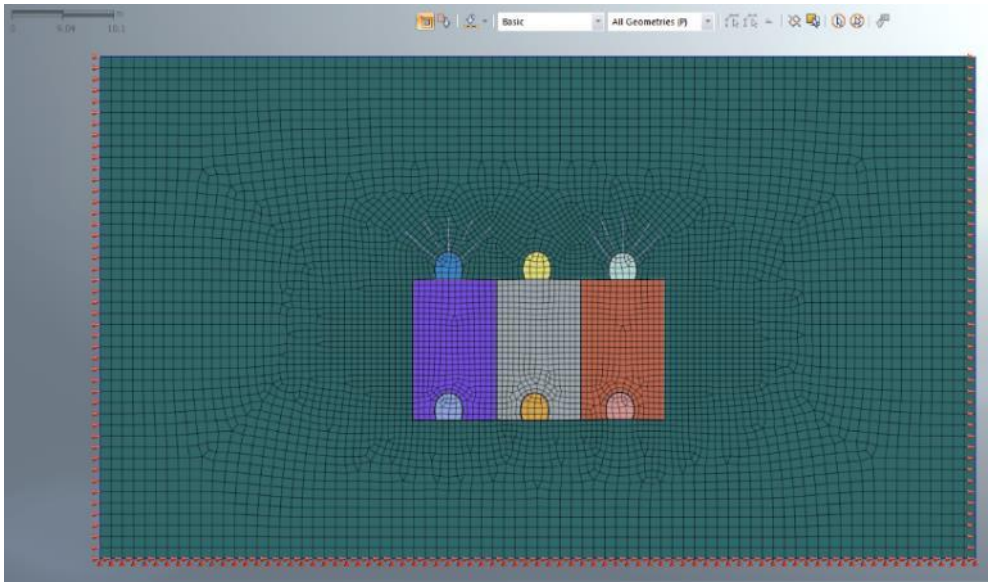


Figure 16.39 – 2D geotechnical model with boundary conditions

Table 16.50 and Table 16.51 present the properties of the rock mass and backfill material that were employed in the different analysis steps.

Table 16.50 – Intact rock properties

Parameter	Unit	Value
γ_b	t/m^3	4.64
E	MPa	46
Poisson's ratio (v)	-	0.22

Table 16.51 – Paste fill properties

Parameter	Unit	Value
γ_b	t/m^3	2.5
E	MPa	15
Poisson's ratio (v)	-	0.22

16.7.5.1 PRIMARY STOPE MINING RESULTS

Once primary stopes are mined, the maximum value of key parameters are shown in Table 16.52.

Table 16.52 – Primary stope key results

Parameter	Value	Unit
Minor principal stress	26 991	kN/m^2
Major principal stress	-25 657	kN/m^2
Effective plastic stress	0.84	MPa

The spatial distribution of the parameters are shown in Figure 16.40, Figure 16.41, Figure 16.42.

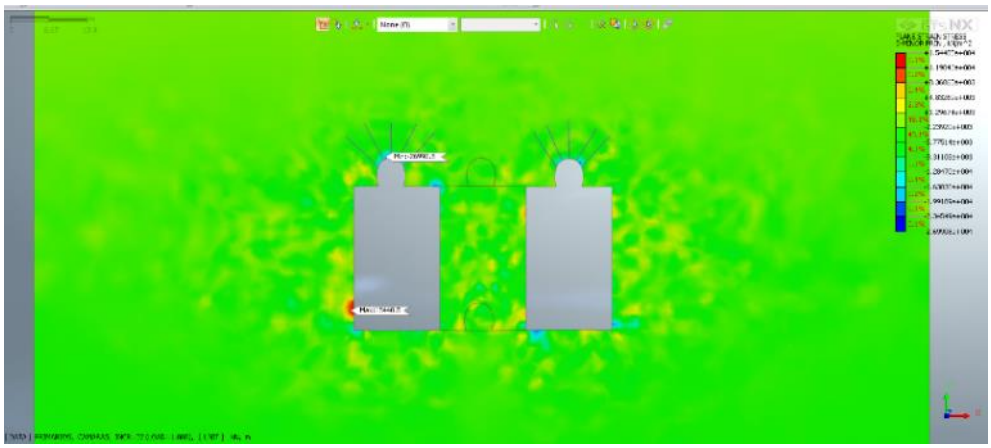


Figure 16.40 – Minor principal stress after primary stopes excavation

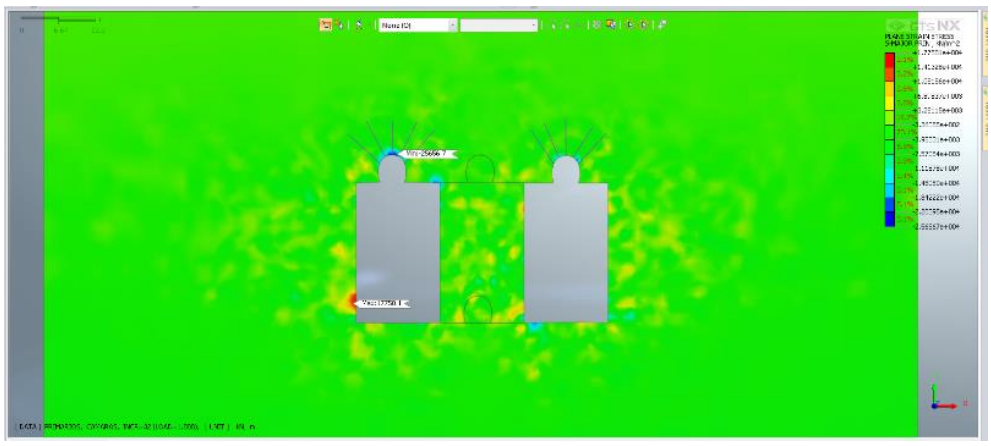


Figure 16.41 – Major principal stress after primary stopes excavation

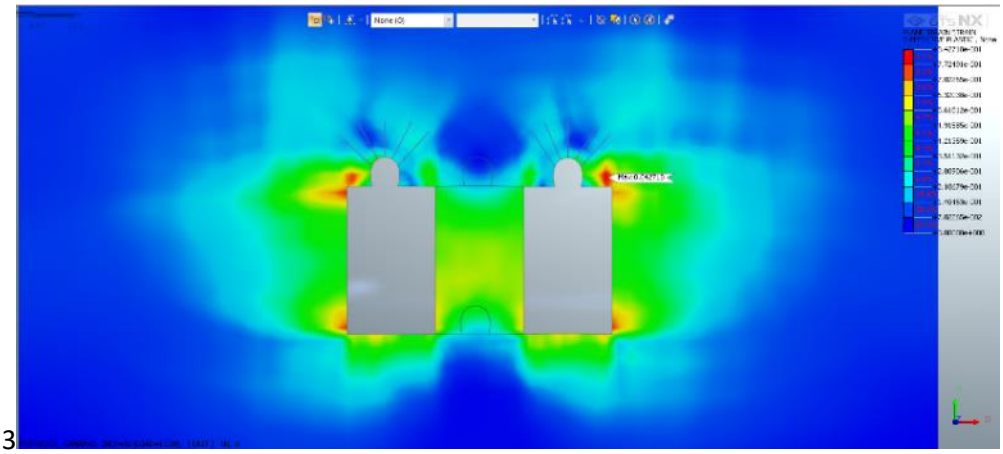
13 

Figure 16.42 – Effective plastic strain after primary stopes excavation

16.7.5.2 SECONDARY STOPE BACKFILL RESULTS

Once primary stopes are backfilled, the range of values for key parameters are shown in Table 16.53.

Table 16.53 – Secondary stope key results

Parameter	Value	Unit
Minor principal stress	-440 598	kN/m2
Major principal stress	-423 915	kN/m2
Effective plastic stress	0.84	kN/m2

The spatial distribution of the parameters are shown in Figure 16.43, Figure 16.44 and Figure 16.45.

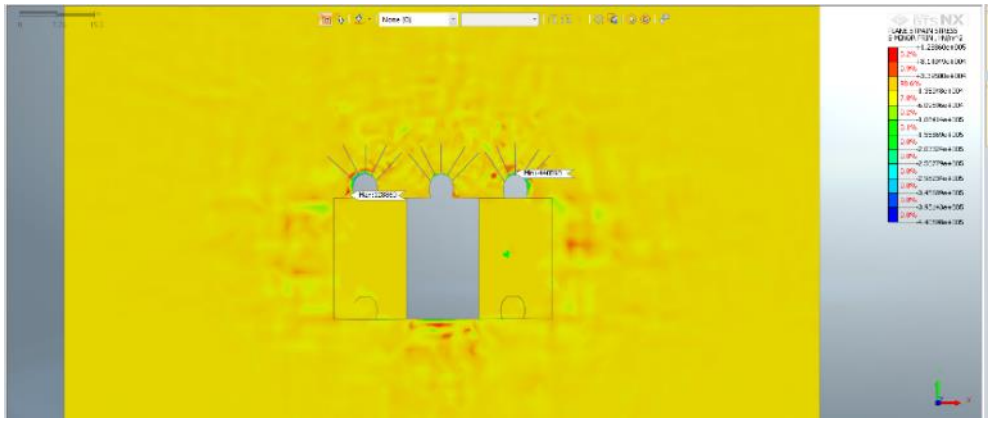


Figure 16.43 – Minor principal stress after secondary stope excavation

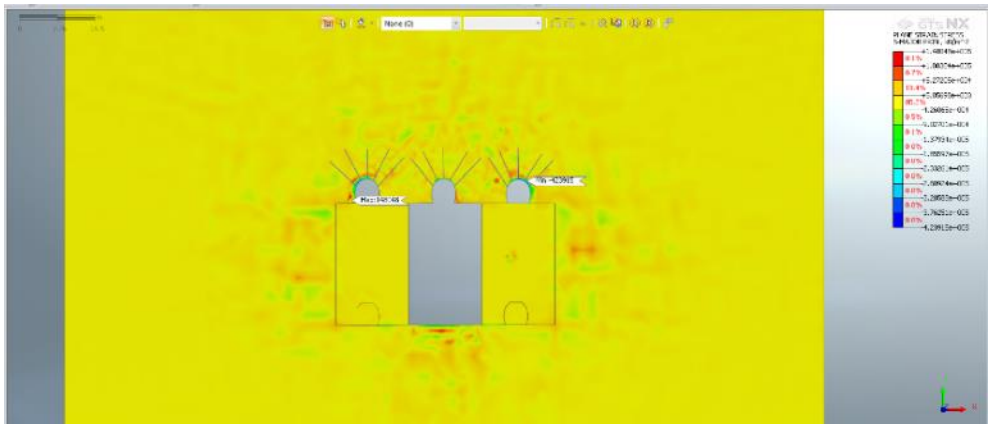


Figure 16.44 – Major principal stress after secondary stope excavation

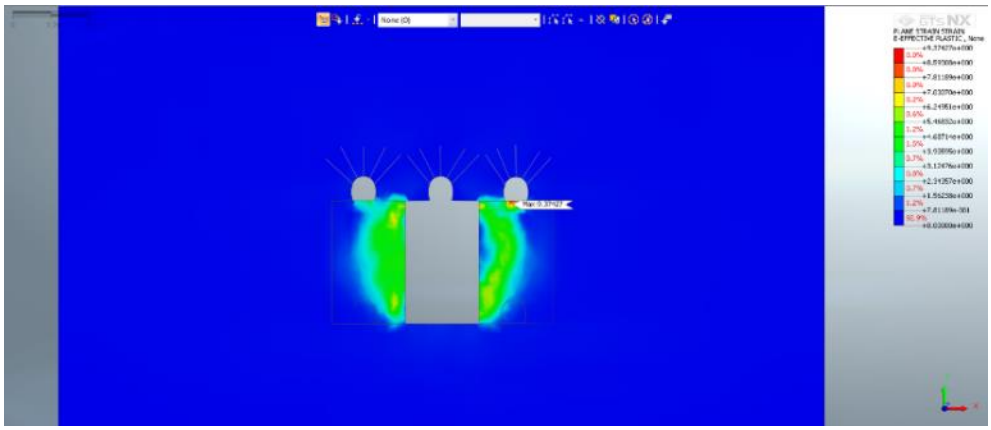


Figure 16.45 – Effective plastic after secondary stopes excavation

16.8 VENTILATION

Ventilation is required to maintain a safe and sustainable underground mining ensuring good environmental conditions. For this it is necessary to dilute gaseous, particulate pollutants and dust to concentrations which are not injurious to the health and safety of the workforce and maintain thermal comfort through the provision of adequate air velocity.

The ventilation system can be observed on Figure 16.46 , it consists in a parallel system, fresh airflow is provided by the ramp and two raises (FAW raise), one to north area and one to south area. Return airflow is exhausted by the raises located at the extreme of each area, main fans are located on surface at the top of those raises.

During ramp development FAW North Raise will be used as air extraction until RAW North raise is developed.

Fresh air is introduced to each level through the ramp and internal raises connected with the FAW raise, after driving across the footwall drive airflow is exhausted through internal raises connected with RAW Raise. Airflow quantity circulating in each level is regulated by regulators and fans located on the airway connecting the footwall drive with the internal return raise. This arrangement avoids the recirculation of air through the level and prevents contaminated air to be spread through several levels in case of fire or smoke production.

Fresh airflow is delivered to production areas with secondary fans located on the footwall drive.

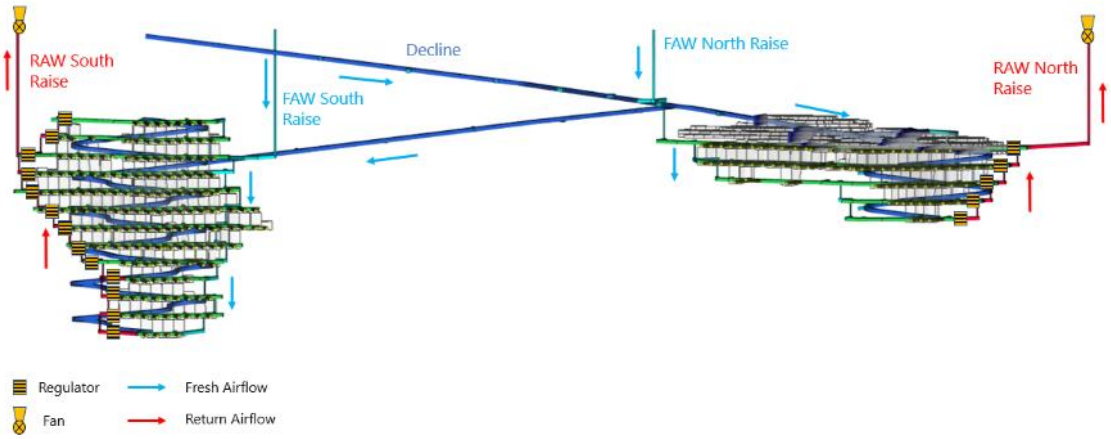


Figure 16.46 – Ventilation circuit (long section)

16.8.1 TOTAL AIRFLOW REQUIREMENT

Required airflow is calculated using a ratio of and 50 l/s per underground worker and 0.067 m^3/s per Kw of diesel equipment installed, which is more conservative than the requirement of the "Decreto-Lei nº 162/90 de 22 de Maio" From the Industry and Energy Ministry. Which requires 50 l/s per underground worker and 85l/s per installed diesel equipment CV.

Diesel equipment and personnel requirements are obtained from the mine plan plus a 20% extra flow to account for underground infrastructure, leakage, and balancing inefficiencies.

Table 16.54, shows total, North zone and South zone airflow requirement. This data has been extracted from the mine fleet estimation in IGAN´s production and financial model. The split between north and south is based on development and stoping production ratio between north and south areas.

Table 16.54 – Airflow requirements

Year	Total airflow demand (m^3/s)	North zone demand (m^3/s)	South zone demand (m^3/s)
Y1	23	23	0

Year	Total airflow demand (m^3/s)	North zone demand (m^3/s)	South zone demand (m^3/s)
Y2	77	74	3
Y3	135	126	9
Y4	147	130	18
Y5	158	140	18
Y6	159	141	18
Y7	172	116	56
Y8	170	80	90
Y9	178	40	138
Y10	169	13	156
Y11	139	12	127
Y12	136	17	119
Y13	75	0	75
Y14	69	0	69
Y15	49	0	49
Y16	10	0	10

16.8.2 RAISE SIZE AND AIRFLOW SPEED LIMITS

Airflow speed limits were obtained from “Decreto-Lei nº 162/90 de 22 de Maio”, which establishes that air speed where workers circulate should not be higher than 8m/s and not be lower than 0.2m/s.

According to IGAN’s own experience, the airspeed limits shown in Table 16.55 were considered.

Table 16.55 – Maximum airflow speed per development type

Development	Speed (m/s)
Working drifts	4
Main haulage drifts	5
Main ventilation drifts	8
Exhaust shafts	10
Ventilation shafts	20

As decline section is 38.4 square meters and maximum air velocity recommended for a transport decline is 6 m/s, the maximum airflow that should circulate through the decline is 230 m^3/s. FAW raises should provide 60 m^3/s, maximum airflow for raises on no competent rock should be 20 m/s, then FAW raises diameter designed will be equal to 2 m. RAW Raises should exhaust the maximum airflow equal to 290 m^3/s, so both are designed with 3.5 m diameter.

Raises specifications are as follow:

- Fresh air intake shafts: the South orebody has a ventilation shaft of 180 m length and 2 m diameter. The position in plan view is this: X = 547 287.193 m and Y = 4 230 983.286 m in UTM coordinates. The ventilation shaft in the North area is 103 m length and 2 m diameter. The position in plan view is this: X = 546 848.791 m and Y = 4 231 792.008 m in UTM coordinates.

- Return air exhaust shafts: the South orebody has an exhaust shaft of 180 m length and 2 m diameter. The position in plan view is this: X = 547 053.569 m and Y = 4 231 289.994 m in UTM coordinates. The exhaust shaft in the North area is 150 m length and 3.5 m diameter. The position in plan view is this: X = 546 626.264 m and Y = 4 232 389.0718 m in UTM coordinates.

- Internal ventilation shafts: they are located between production levels at the ends of the footwall drives. The average length of the shafts is 30 m. They will be executed by drill&blast with a 2.4 m diameter section. Airflow regulators will be installed at the access to the return shafts to adjust the flow circulating at each level.

16.8.3 VENTILATION MODEL

A mine ventilation model was developed using Ventsim software, for three primary purposes:

- To validate the operability of the ventilation circuit ensuring airflow can be provided to all the required areas.

- To ensure compliance with design criteria.

- To determine fans duties and energy requirements.

Peak primary fan duties will occur at full production in conjunction with maximum development activities in the lowest levels of each ventilation district.
The file “21005-21002_VEN_ventilationstudy_V1.0.0.vsm” was used as the basis for the design and simulation of the ventilation network.

Table 16.56 presents the section and friction factors used in the ventilation model.

Table 16.56 – Sections and friction factors per development type

Development	Speed (m/s)	Section (m)	Friction factor (Kg/m^3)
Main decline	Arched	6.5 x 6.0	0.016
Internal decline	Arched	6.5 x 6.0	0.016
Haulage drifts	Arched	5.5 x 5.5	0.016
Other drifts	Arched	5.0 x 5.0	0.016

Development	Speed (m/s)	Section (m)	Friction factor (Kg/m³)
Internal ventilation shaft	Circular	Ø 2.4	0.02
Surface ventilation shaft	Circular	Ø 3.5	0.0028

16.8.4 MAIN FANS

Over the LOM, there will be a multitude of settings for the ventilation circuit, depending on the type of activities and their location throughout the mine. The specifications for the North and South zone main fans are summarised in Table 16.57 and Table 16.58, respectively.

Table 16.57 – North zone fan specifications

Description	Specification
Duty	200 m^3/s @ 1865 Pa
Fan Diameter	2.5 m
Type	ZVN 1-25-500/6. Axial/Horizontal/Unidirectional
Configuration	One Forcing Fan situated on North RAW shaft at surface
Voltage	690 V
Fan Motor	500kW to 1000 rpm, variable frequency drive capability

Table 16.58 – South zone fan specifications

Description	Specification
Duty	170 m^3/s @ 4534 Pa
Fan Diameter	2.5 m
Type	ZVN 1-25-1100/6. Axial/Horizontal/Unidirectional
Configuration	One Forcing Fan situated on South RAW shaft at surface
Voltage	690 V
Fan Motor	1100 kW to 1000 rpm, variable frequency drive capability

16.8.5 AUXILIARY VENTILATION

All work areas in the mine not supplied with a split of fresh air must be ventilated using auxiliary systems. The most effective means for providing airflow to areas without primary airflow is typically with small diameter (up to 1 400 mm) axial fans combined with low leakage and flexible ducting.

16.8.5.1 INFRASTRUCTURE DEVELOPMENT

During access and level development, distances up to 800 m will be ventilated using auxiliary systems. The peak auxiliary airflow for development activity will be required to

dilute the emissions of one 65 t truck and one 21 t loader, amounting to 39 m^3/s of auxiliary airflow.

Modelling indicates that two ducts will be required. Each duct will have two 55 kW fans bolted together in series. The duct size is 1 200 mm in diameter. This will supply 40 m^3/s up to 800 m distance. This arrangement will allow for adequate overhead clearance for a fully loaded 65 t truck. Figure 16.47 shows secondary ventilation simulation for infrastructure development.

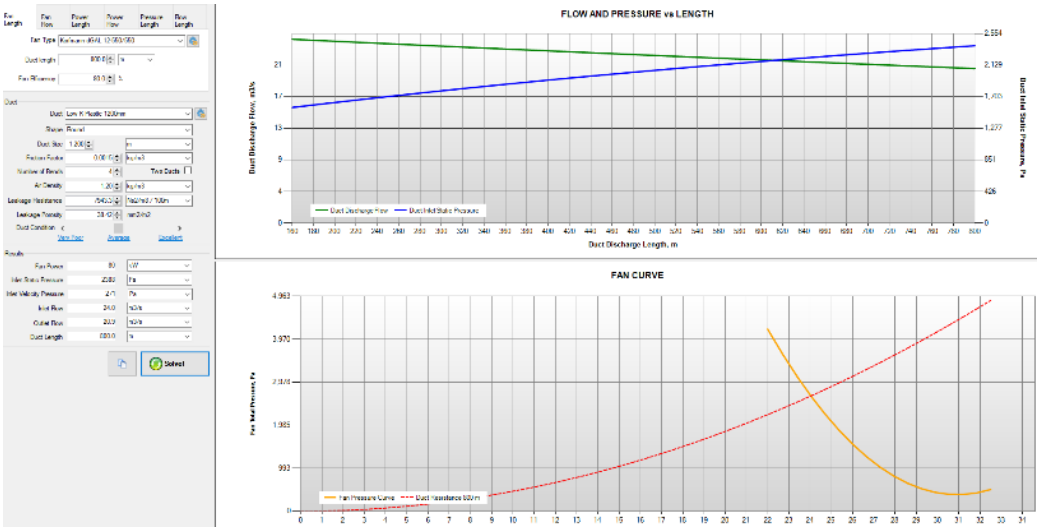


Figure 16.47 – Infrastructure ventilation

16.8.5.2 STOPE PRODUCTION

An allowance of 13 m^3/s was made for each active stope, for dust, blast fume, and diesel exhaust clearance. Modelling indicates that a single-stage 55 kW fan, with 1 000 mm diameter low-resistance, low-leakage ducting will supply the required airflow to a distance of at least 400 m. Figure 16.48 shows secondary ventilation simulation for stope production.

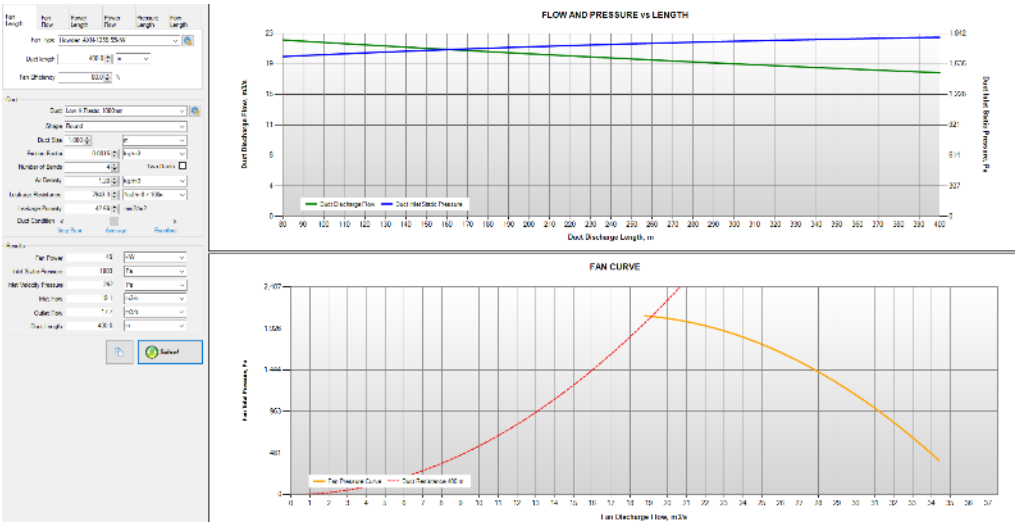


Figure 16.48 – Stope Production ventilation

16.8.5.3 EMERGENCY PREPAREDNESS

In development of the ventilation strategy for the Project, consideration has been given to the potential for mine emergencies. As such, the following criteria have been established:

- In general, ramps will be in fresh air once developed.
- On almost all levels, escape can be either to a ramp or to the escape ladderway.
- In each ramp, escape may either be up the ramp or down the ramp to a safe area.
- Fire doors will be located in accordance with legislated requirements and to isolate areas of high fire potential to ensure noxious gases are not distributed through the mine workings.
- Refuge chambers will be portable for flexibility of location at the most appropriate points in the mine.

For the production stoping blocks, a ladderway will be installed in each of the raises located next to main ramps. The raises are sized to afford easy passageway.

The exhaust raises to surface will be for ventilation only and not used as a second means of egress. Therefore, the exhaust raises will not have ladderways installed.

16.9 MOBILE EQUIPMENT REQUIREMENTS

16.9.1 PRE-PRODUCTION PHASE

During the pre-production phase, approximately 7 200 m of lateral development must be completed to meet the ramp-up production schedule. Development during this period will peak at approximately 3 700 m/quarter, but with the average over the pre-production timeframe at about 1 750 m/quarter.

The peak equipment unit requirements in the period are shown in Table 16.59.

Table 16.59 – UG equipment count during pre-production phase

Equipment	Units
Two-boom jumbo	3
LHD 21 t	3
Bolter	2
Cable Bolter	1
Shotcrete sprayer	2
Transmixer	3
Production drill 100 mm	0
Mobile raiseborer	0
Haulage Truck, 65t	3

16.9.1.1 JUMBOS

A total of three jumbos being required to meet the peak scheduled advance during the pre-production stage. All units will be two boom jumbos (Epiroc Boomer M2C22 or similar), enough for the average drift size.

16.9.1.2 LHDS

Up to three LHD units will be required to handle the blasted material from both the various development headings and production. The 21 t (Sandvik LH621 or similar) units will be used in the mucking of development drifts and for stope production mucking. The selected size classes are found within regional mining fleets and thus, presumed to be easily sourced and maintained by the operation.

16.9.1.3 TRUCKS

Up to three 65t haulage truck units (Epiroc MT65 or similar) will be required to sustain the pre-production development program. The units will be used both in development and

production efforts. As with the LHDs, the selected size class is commonly found across the mining operations in the region and thus, anticipated to be used as well in the operation.

16.9.1.4 BOLTERS

A total of two bolting units (Epiroc Boltec MC or equivalent) are required in the period. This demand will be greater than in the production phase as most of the CAPEX development will be developed early in the LOM.

16.9.1.5 CABLE BOLTING

Cable bolting will be required to support intersections in CAPEX drives and as stope ground support. Due to the number of intersections and infrastructure excavations to be developed during the pre-production period, it is anticipated that a single mobile cable bolter unit (Epiroc Cabletec LC or equivalent) will be required to fulfil this need.

16.9.1.6 SHOTCRETE SPRAYERS

It is anticipated that up to two shotcreting units (Normet Spraymec MF 050 or similar) will be required during the pre-production period. In addition to the lateral development, most CAPEX infrastructure, all of which requiring shotcreting, will be developed during this period.

16.9.1.7 TRANSMIXERS

A peak total of three transmixers (Normet Ultimec LF 600 or similar) will be required to ensure the timely supply of required shotcrete volumes.

16.9.1.8 LONG HOLE DRILLERS

Only one LH drill rig (Epiroc Simba M6C or equivalent) will be required during the pre-production period. Their number will be increased in the production period, once stoping production is at the nominal level.

16.9.1.9 MOBILE RAISE BORING

A single mobile raise bore machine (Epiroc Easer L or equivalent) will be incorporated towards the end of the pre-production period to commence stope slot raise drilling activities as well as vertical CAPEX development tasks.

16.9.2 PRODUCTION PHASE

Table 16.60 summarises the total peak equipment requirements during the production phase of the UG operation.

Table 16.60 – UG equipment peak count during production phase

Equipment	Units
Two-boom jumbo	3
LHD 21 t	6
Bolter	3
Cable Bolter	2
Shotcrete sprayer	3
Transmixer	4
Production driller 100 mm	3
Mobile raiseborer	1
Haulage Truck, 65t	6

16.9.2.1 JUMBOS

Development advance will be maintained at an average rate of 9 900 m/year during the three first years of the production period. Thereafter, the annual development rate fluctuates between 3 000 and 10 800 m, and progressively drops down in the late years of the LOM. A total of two units will be required to sustain the production rate.

16.9.2.2 LHDS

Up to six LHD units will be required to handle the blasted material from all locations, inclusive of stoping production. Stoping will demand a peak value of five 21-tonne (Sandvik LH621 or similar) units, with an additional one 21t units demanded for the drift development and other ancillary tasks.

16.9.2.3 TRUCKS

Up to six 65t haulage trucking units will be required to sustain the production program. The number of units will be increased at the start of the production period as production reaches its nominal rate and is maintained. Then, the number will decrease as development requirements ramp down.

16.9.2.4 BOLTERS

Four bolting units will be required to sustain the production activities. The demand peak coincides with an increment of cut & fill production. The remainder of the production phase requires a single bolting unit.

16.9.2.5 CABLE BOLTING

Stope drives will be stabilised with twin-cable bolts and two cable bolting unit will be required to fulfil the support installation needs during the production period.

16.9.2.6 SHOTCRETE SPRAYERS

Shotcrete will also be required for paste fill exposures in stope development, barricade construction for backfilling bulkheads and general development support. It is anticipated that up to three units will be required during the production period.

16.9.2.7 TRANSMIXERS

Up to six units will be required to supply the shotcreting units underground.

16.9.2.8 LONG HOLE DRILLERS

A total of three LH drillers will be required during the production period as well. The number will be maintained during the early years of the period, to be then reduced with the stoping production ramp-down.

16.9.2.9 MOBILE RAISE BORE

A mobile raise bore machine (Easer L) will be employed to drill production slot raises in the stopes. This machine will also be used for service hole drilling and vertical drop raise development.

16.9.3 SUPPORT EQUIPMENT

The ancillary equipment listed in Table 16.61 will be required to support all activities in the mine.

Table 16.61 – Total UG support equipment count

Equipment	Units
Emulsion Loader	3
Scissor Lift	2
Boom Truck	2
Personnel Carrier LV	12
UG Forklift	1
Grader	1
Service Truck	2
Fuel/Lubricant Truck	2

16.9.3.1 BOOM TRUCK

Boom trucks (Normet Multimec MF100 or equivalent) will be required to load materials (rock bolts, screen, resin, ventilation duct, etc.) at the surface, transport them underground and unload them at designated locations across the mine.

It is anticipated that two trucks will be able to support the activities across the life of mine.

16.9.3.2 EXPLOSIVES VEHICLES

At full production, explosives consumption is estimated to be 2.43 t/d of bulk emulsion. The explosive will be delivered to the mine in 1,100 kg capacity tanks that are to be transported to the emulsion loaders throughout the mine by boom trucks.

One scissor truck (Normet Utilift MF 540) will be tasked with the explosives loading. Most of the pre-production period needs will be met by a single unit.

16.9.3.3 SCISSOR LIFT

Two scissor lift trucks (Normet Utilift MF 540) will be required to perform daily tasks such as installation of ventilation and mine service components such as water / compressed air piping, ventilation ducts, electrical cables, and fans.

16.9.3.4 PERSONNEL CARRIER

Up to twelve personnel carrier light vehicle (Toyota Hilux or similar), each with a capacity of 5 people, will be employed to carry personnel from the surface down to the mine.

16.9.3.5 FORKLIFT

A maximum of one underground forklift (Manitou MHT 10230 or similar) will be employed to handle parts during the installation of mine services.

16.9.3.6 GRADER

An underground motor grader (Veekmas FG 5 C or similar) will be used to maintain the floor of underground trafficways.

16.9.3.7 SERVICE TRUCK

A total of two service trucks (Normet MF 100) will be deployed to provide short-term interval servicing to the production and development trackless units so they are not required to be driven to the underground maintenance workshops unless required.

16.9.3.8 FUEL/LUBE TRUCK

Two trucks (Normet MF 100 or similar) will be required to fuel and lubricate all equipment, such as LHDs, jumbos LH drills, bolters, etc that not likely to return to the surface workshop area at frequent intervals. Potential down time, as well as traffic congestion can be reduced by keeping this truck near the working locations; the service truck will travel between the equipment pieces to perform the daily servicing.

16.10 UNDERGROUND INFRASTRUCTURE

All the mining services systems, related infrastructure and equipment have been designed to sustain a nominal production capacity of 1.2Mtpa.

16.10.1 MINE DEWATERING

IGAN completed a basic study of the water flow requirements for the operation and the pumping network needed to dewater both the inflows from drill and other operating equipment and groundwater inflows. This study includes:

- Design criteria.
- Initial data and assumptions.
- Flow rates calculations.
- Pressure calculations.
- Equipment selection.

This study was completed in three phases:

1) Inflow estimation.
2) Network simulation.
3) Equipment selection.

16.10.1.1 SYSTEM DEFINITION

The first phase is to define the system by means of several parameters. These parameters are estimated according to the mine location, the flow rate required and pressure. The standard operation conditions are set based on the climate or environmental conditions of the mine location.

For this basic study, the fluid temperature is assumed to be 25 ºC. Also, the absolute atmospheric pressure is set to 1 bar, as the performance of this system will not be affected by heavy ambient temperature variations.

16.10.1.2 FLOW RATE

Mine dewatering must accommodate groundwater inflows from the Lagoa Salgada workings and inflows from drill, dust suppression and other operating equipment.

Using the Monte-Carlo method, the freshwater flow rate required to supply 95% of the possible water consumption scenarios was estimated in Section 16.10.5. The required flow, including leakages, is 9.2 l/s (or 33 m^3/h) with a peak demand of 16 l/s.

The average groundwater inflow was estimated, considering the available studies at the date of this report ("*Lagoa Salgada_ relatoriofinal_versao_draft.pdf*"), to be 30 l/s in the North and 15 l/s in the South. Thus, the total inflows are estimated to be approximately 50 l/s in the North and 40 l/s in the South. These pumping flow rates ensure that the dewatering system can pump the peak demand and has an allowance to accommodate for uncertainty in the groundwater inflow.

16.10.1.3 PUMPING NETWORK

IGAN has considered several scenarios to evaluate and configurate the final system.

1) Pumping type:
    a) Multiple, low pressure stage system.
    b) Few, high pressure stage system.
2) Evacuation point:
    a) Mine portal.
    b) Fresh air intake shaft.
3) Underground network:
    a) Service boreholes.
    b) Ramps.
    c) Ventilation raises.

IGAN has evaluated the advantages and disadvantages of each scenario and selected the following system: few, high pressure stage system with dewatering point located at the shaft, and intermediary levels draining via ventilation raises to main pumping levels. This

system maximizes the pumps efficiency and reduces the number of pumps required, allows to have enough space to decant the water before pumping and minimizes the length of the pipeline, ensuring a properly maintenance.

Figure 16.49 shows the pumping network in fuchsia. The system has two main routes: one to the pumping station located at the North in the level 150 footwall drive and other to the pumping station located at the South in the footwall drive of the level 325. Both networks converge at the main decline station (level 50). At this point, both networks are combined into one pipe and reach the surface through the North fresh air intake shaft.

Once in the surface, the pipeline continues to the decanting pound located at the south.

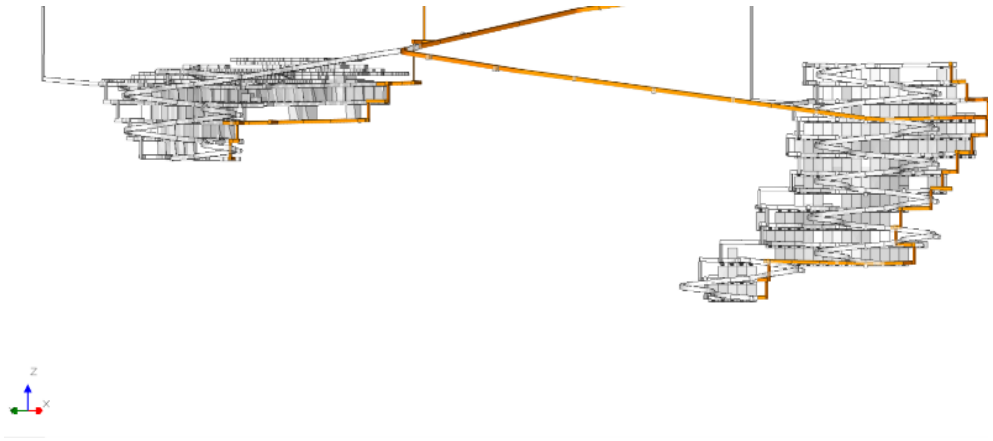


Figure 16.49 – Dewatering system (long section)

16.10.1.4 PIPE DIAMETER AND TYPE. EROSION PHENOMENA

Pipe diameter is the most important variable to calculate the resistance to the water flow of the pipe. Thus, the diameter is the most important parameter to design the pipeline.

In addition to resistance production, the fluid velocity in a pipe is the main cause of the erosion phenomenon. Erosion can be defined as the mechanical loss of material due to solid particles impacts (sand, metals, debris, etc.) present in the fluid, as well as the impact of liquid droplets or gas bubbles. To minimize the effect of the erosion, a 4 000 mg/l to 5 000 mg/l solid content was considered in the pipe selection.

Flow velocity should be limited to avoid cavitation or erosion. Velocities above 3 m/s should be avoided in IGAN's own experience. However, the velocity limit should be determined in the basic and detailed engineering phases of the project.

On the other hand, low velocities can provoke settling of solids in the pipes. Solids settling could result in increased pipe resistance or even clogging of the pipes.

Considering that the pumping stages exceed 15 bar pressure, IGAN recommends the use of steel pipes at least in the high-pressure sectors. Considering that this is a basic study, to

simplify the calculations, IGAN assumed that the entire network has Schedule 40S carbon steel pipe.

The following constrains were used to select the pipe diameter:

- Flow velocity above 1.5 m/s to avoid solids settling.
- Flow velocity below 3 m/s to prevent erosion/corrosion phenomena and cavitation.

16.10.1.5 FITTINGS AND VALVES

IGAN considers the following fittings in the simulation:

- Valve installation in every level.
- Direction changes will be made using an elbow of the appropriate dimension.
- In the case that ventilation raises are not vertical aligned, elbows will be used to redirect the flow.
- Guillotine valves are used.

Fittings are listed in Appendix 1.

16.10.1.6 PUMPS

Although more expensive than centrifugal pumps, a positive displacement diaphragm pump type was considered (Figure 16.50). They require less maintenance, are able to pump sludge directly to the surface and handle more aggressive water chemistries. Further information about this pump type can be found in Appendix 3.

For the network calculations, it was considered a 50 l/s inflow demand and 100% redundancy of pumping equipment in each station. A total of two pumps was used per station.

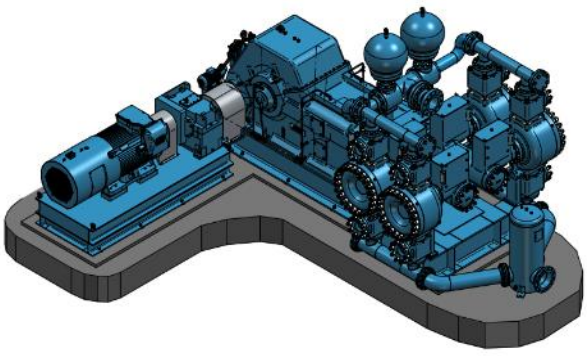


Figure 16.50 – Abbel hydraulic membrane pump HMQ-250-2000

16.10.1.7 SIMULATION

IGAN completed a simulation of the single diagram dewatering system with the Pipe Flow Expert software. This system included the basic elements of the network such as valves and elbows. Further details on the system could be retrieved from Appendix 1.

The following assumptions were made to perform the simulation:

- Temperature 25º C.
- Water viscosity is not affected for the content of solids (4 to 5 g/l).

Figure 16.51 shows the dewatering system simulated.

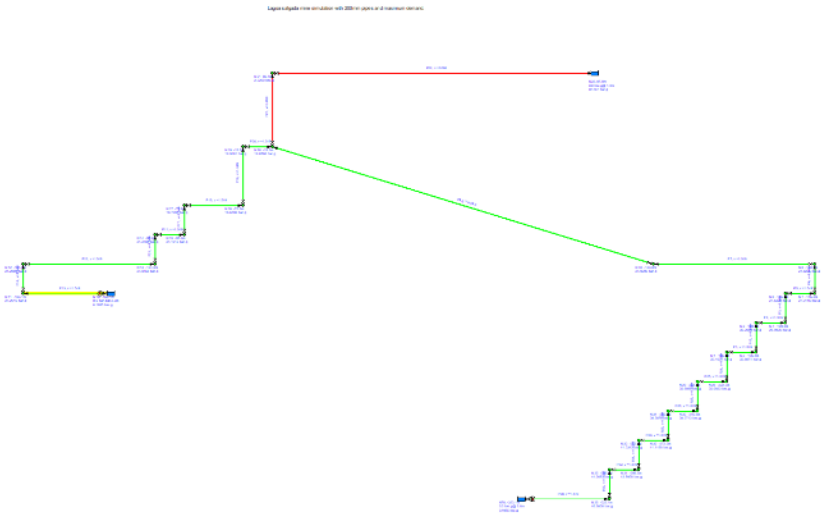


Figure 16.51 – PipeFlow simulation diagram

Table 16.62 summarises the results obtained for the selected scenario.

Table 16.62 – Scenario 1-1 parameters

Parameter	Unit	Value
Pipe size	mm	2 x 200
Level 175 pumping station flow	l/s	50
Level 175 pumping station pressure	bar	27
Level 350 pumping station flow	l/s	50
Level 350 pumping station pressure	bar	47

The main results of the simulation are as follows:

- The piping must be at least 200 mm (8”) for all pumping stations.
- Steel pipes are required in all pumping stations as both have pressures above 15 bar.

Appendix 1 includes more information about the simulation.

16.10.2 MATERIAL HANDLING

ROM material will be trammed by the LHD from the production areas into the level ore storage bays or directly loaded into mine trucks and hauled to surface. Material stockpiled in the storage bays will be scooped and loaded into mine trucks, then hauled to the surface ROM Pad.

16.10.3 POWER REQUIREMENTS AND ELECTRICAL DISTRIBUTION

The electrical distribution and installations of the mine were designed considering the following Portuguese technical regulations:

- “Decreto-Lei n.º 162/90” of the “Ministério da Indústria e Energía”.
- “Decreto-Lei n.º 18/85” of the “Ministério da Indústria e Energía”.

16.10.3.1 MAIN DESIGN

The power distribution design consists of an electrical ring comprising both exhaust shafts in the North and South orebodies.

LOW VOLTAGE DISTRIBUTION

Table 16.63 presents the low voltage power distribution characteristics.

Table 16.63 – Main parameters of low voltage network

Parameter	Comments
Voltage classification	Three-phase power network with three wires 690 V between phases
Frequency	50 Hz
Insulation resistance	≥ 0.5 MΩ
Dielectric strength	Minimum 1 500 V at 50 Hz for 1 minute

To ensure power supply continuity and protection against electrical shocks, the neutral phase distribution pattern will be of IT pattern type. This pattern is based in isolated and not distributed neutral phase; grounding the demanding installation earth using a protective earth conductor and equipotential bonding distributed inside the underground mine and grounded outside.

FIRE PROTECTION

The following protection will be installed:

- Overcurrent protections.
- Overload protections.
- Short-circuit protections.

To define the protection levels against short-circuits, the maximum and minimum short-circuits currents will be calculated in each circuit requiring a protection. This circuit is defined as the distribution segment between two protections. The short-circuit protections guarantee the following:

- All the elements in the network will be designed to withstand the electrodynamic and thermal stresses caused by the short-circuit currents until the breaking devices work.
- Automatic protective devices in charge of breaking the short-circuit current within a short-time period will be installed in the network. These devices will protect the system against fire or material degradation.
- The minimum short-circuit current will be able to active the breaking devices in a short-time period to prevent fire risks.
- 10% lower nominal short-circuit breaking capacity of the circuit breaker switches (defined following the NP standards), will be higher than the required short-circuit breaking capacity meeting the above conditions.

Short-circuit protections will be installed in the following points of the network:

- At the inlet end of each consumer or at the beginning of the exclusive power wire.
- At the beginning of the electrical shunts.
- At the outlet end of each transformer.

Fire extinguishers will be available in electrical cuddies and transformation and distribution substations. These extinguishers will have visible instructions for use and be in easy and

accessible locations even after a fire has started. The materials for extinction will not be toxic, suffocating or electrically conductive.

CABLES

The following cables will be used:

- Rigid armoured cables in fixed facilities.
- Flexible armoured cables for supplying machines and fixed, semi-fixed, semi-mobile, and mobile devices.
- Unarmoured flexible cables for supplying all types of equipment and devices, using adequate electrical protections.

Whenever is possible, cables will be installed through services holes, avoiding direct contact with live or charged conductor. If it is not possible, the cables will be installed in the opposite side of the dewatering and freshwater services in the drift.

PROTECTION AGAINST ELECTRIC SHOCK

People's protection against electrical shocks will comprise the following measures:

- Using bare wire is prohibited, except for protective and earth conductors.
- Electrically connect by protective conductors all the metal masses of the facility or the ones outside the facility but accessible from it.
- Connect the protective conductors to the earth of the power outlets. One ground will be located outdoors.
- Use isolation controller devices to warn and/or to automatic power cut off. An optic and acoustic alarm will be trigged when the isolation resistance drops below 50Ω/V. If the isolation resistance drops below 10Ω/V, all live conductors of the power supply line will be automatically disconnected.

16.10.3.2 HIGH VOLTAGE

Electrical power will be supplied to the ventilation shafts, one in the North and the other in the South, at a nominal voltage of 15 kV. Both cables will join inside the mine forming an electrical ring. The maximum average demand from the underground mine is estimated to be 5 MW.

Table 16.64 and Table 16.65 show the estimated electrical consumption for the mine operation both in the surface and underground, respectively.

Table 16.64 – Surface power consumption

Equipment	Power (kW)	Derating factor (%)	kW drawn/unit	Units	kW drawn
Main surface exhaust fans	900	100	900	1	900
Maintenance workshop	37	75	28	1	37
Freshwater pumps	8	50	4	1	8
Magazine	3	100	3	1	3
Total	**948**	**99**	**935**		**948**

Table 16.65 – Underground power consumption

Equipment	Power (kW)	Derating factor (%)	kW drawn/unit	Units	kW drawn
Ramp ventilation fans	220	100	220	1	220
Booster fans	55	100	55	1	55
Main dewatering pumps (and decline)	800	100	800	1	800
Auxiliary production ventilation fans	675	88	591	1	675
Auxiliary development ventilation fans	495	88	433	1	495
Production drilling	101	100	101	3	302
2-Boom Jumbo 160HP	135	100	135	3	405
Mobile raisebore	183	80	146	1	183
Mechanical Bolter	68	100	68	2	135
Cableboter	68	100	68	1	68
Diamond drills	75	52	39	2	150
Refuge Stations	22	20	4	1	22
Total	**2 985**	**91**	**2 721**		**3 598**

The main surface exhaust fans will be feed through an independent transformer from 15 kV to 690 V. The recommended transformer is of 1 500 kVA capacity to have an allowance to satisfy additional power demands. Once inside the mine, the cable will feed the transformation centres (15kV to 690 V).

Figure 16.52 shows the 15 kV lines in red through the mine.

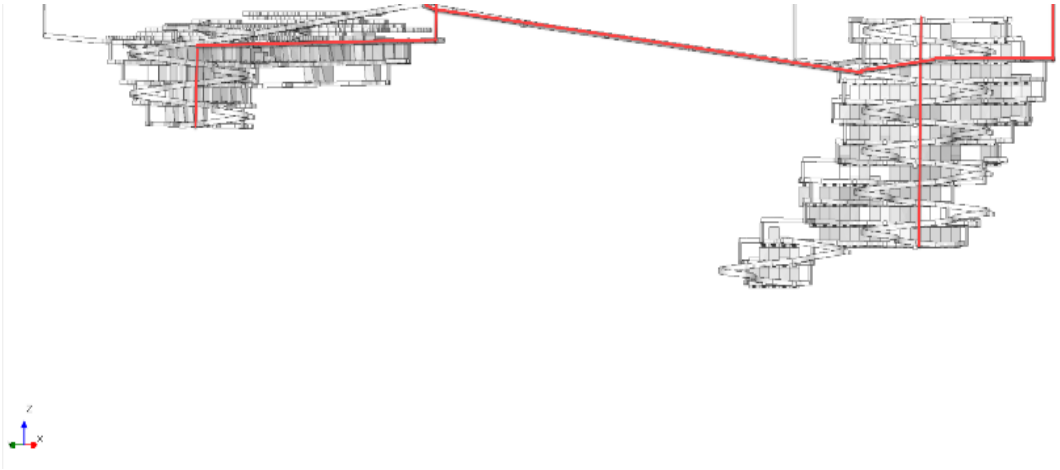


Figure 16.52 – 15kV electric lines

16.10.3.3 LOW VOLTAGE

Mobile stepdown transformers will be installed in different level accesses to provide 600 V tension to the machines working in the area while development of the mine is underway. Distribution cells will be installed on each level to allow the movement of transformers as required. On the production stage it is advisable to install a 2 000 kVA transformer per level to satisfy the power supply and even be able to supply other levels. A total of five transformer would be required to feed both areas of the mine (i.e., North and South).

Table 16.66 presents the transformer characteristics for the underground mine.

Table 16.66 – UG transformer characteristics

Parameter	Comments
Primary voltage	15 000 ± 2.5% ± 5% ± 7.5% V
Secondary voltage	690 V
Nominal power	2 000 kVA
Insulation	Class H-180 ºC copper wire
Frequency	50 Hz
Maximum room temperature	50 ºC
Average room temperature	+30 ºC
Average annual room temperature	+25 ºC
Average relative humidity	90%

16.10.4 COMPRESSED AIR

All mobile drilling equipment, including jumbos, long hole drills, bolters and cable bolters will have portable adjacent compressors to meet their elevated pressure requirements.

16.10.5 FRESHWATER SUPPLY

IGAN has conducted a basic study of the freshwater needs and the installation necessary to be able to distribute the water flow across the working levels of Lagoa Salgada mine.

This study includes:

- Design criteria.
- Starting data and assumptions.
- Calculation of flow rates.
- Calculation of pressures.
- Equipment selection.

IGAN's approach to this problem is in several phases:

1) Calculation of needs (Monte-Carlo method).
2) Simulation of the network.
3) Equipment selection.

16.10.5.1 SYSTEM DEFINITION

All calculations have been done considering the nominal production level of 2Mtpa.

The first step is to define the operating parameters of the system. These values will be determined by both the location of the installations and the equipment minimum supply pressure requirements. For simplicity in the calculations, an average water supply temperature of around 20°C (293.15 K) and absolute pressure of 1 bar has been considered by IGAN.

16.10.5.2 WATER VOLUME DEMAND ESTIMATE

IGAN has completed a freshwater demand calculation required by the production fleet and ancillary equipment consuming units shown in Table 16.67.

Table 16.67 – Freshwater consuming equipment units

Equipment	Units
Boomer M2C22 Jumbo	3
Simba M6C production drill rig	3
Easer L ITH drill	1

Equipment	Units
Cabletec LC BL17066 cable bolter	1
Boltec MC rock bolter	3
Dust suppression nozzle sprayers	8
Stope dust suppression nozzle sprayers	10
Exploration & grade control drill rig	2

The total demand estimate has been calculated considering a 15% of water flow losses.

A Monte-Carlo simulation was used to estimate the individual consumption unit demand to a 95% probability. A total of 1000 scenarios were simulated and their probability distribution is shown in Figure 16.53. This approach overcomes the issues introduced by less sophisticated methods (e.g., calculating the average usage of each piece of equipment, then calculating an average usage for the entire fleet) that tend to underestimate the demand. As a simple rule of thumb, a design capacity to the average need will result in having a capacity deficit approximately half the time.

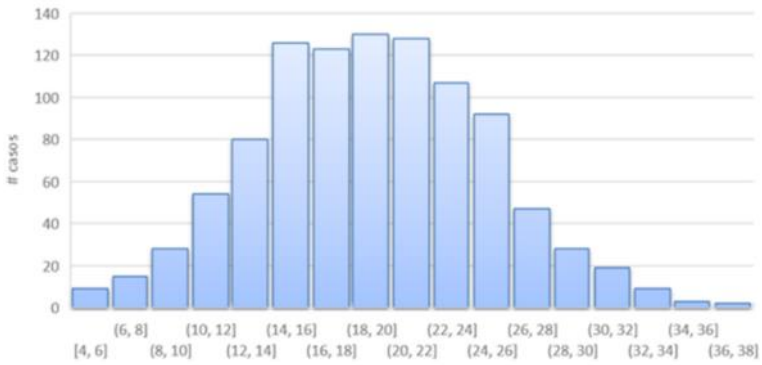


Figure 16.53 – Freshwater Monte-Carlo simulation results

As can be seen in the histogram, due to the negative histogram skewness, a large volume of water will not be necessary to cover the demand in 95% of the cases. As a matter of fact, 49% of the maximum consumption will be enough for 95% of the demand scenarios. A flow of 33 l/s of supply capacity is needed to cover 95% of the scenarios including 15% of losses in the system. The simulation main results are shown as well in Table 16.68.

Table 16.68 – Water consumption estimate

Parameter	Subtotal	Losses (15%)	Total	Units
Maximum consumption	58	8.7	66.7	m^3/h
Average consumption	19	2.8	21.8	m^3/h
95% probability consumption	28.7	4.3	33	m^3/h

Individual results per equipment item are shown in Table 16.69.

Table 16.69 – Freshwater consuming equipment units

Equipment	Parameter	Value	Units
Jumbo	Consumption per unit	3.2	m^3/h
	Maximum consumption	9.6	m^3/h
	Total utilisation	34	%
	Average consumption	3.3	m^3/h
	95% probability consumption	4.7	m^3/h
	95% probability consumption per unit	1.6	m^3/h
	Pressure	5	bar
Production drill rig	Consumption per unit	3.6	m^3/h
	Maximum consumption	14.4	m^3/h
	Total utilisation	15	%
	Average consumption	2.2	m^3/h
	95% probability consumption	7.1	m^3/h
	95% probability consumption per unit	1.8	m^3/h
	Pressure	2	bar
ITH drill rig	Consumption per unit	5.1	m^3/h
	Maximum consumption	5.08	m^3/h
	Total utilisation	30	%
	Average consumption	1.5	m^3/h
	95% probability consumption	2.5	m^3/h
	95% probability consumption per unit	2.5	m^3/h
	Pressure	2	bar
Cable bolter	Consumption per unit	2	m^3/h
	Maximum consumption	2	m^3/h
	Total utilisation	15	%
	Average consumption	0.3	m^3/h
	95% probability consumption	1	m^3/h
	95% probability consumption per unit	1	m^3/h
	Pressure	2	bar
Rock bolter	Consumption per unit	1.7	m^3/h
	Maximum consumption	4.98	m^3/h
	Total utilisation	34	%
	Average consumption	1.7	m^3/h
	95% probability consumption	2.5	m^3/h
	95% probability consumption per unit	0.8	m^3/h
	Pressure	2	bar
Dust suppression nozzle sprayers	Consumption per unit	0.7	m^3/h
	Maximum consumption	5.6	m^3/h

Equipment	Parameter	Value	Units
	Total utilisation	50	%
	Average consumption	2.8	m^3/h
	95% probability consumption	2.8	m^3/h
	95% probability consumption per unit	0.3	m^3/h
	Pressure	5	bar
Stope dust suppression nozzle sprayers	Consumption per unit	0.7	m^3/h
	Maximum consumption	7	m^3/h
	Total utilisation	40	%
	Average consumption	2.8	m^3/h
	95% probability consumption	3.5	m^3/h
	95% probability consumption per unit	0.3	m^3/h
	Pressure	5	bar
Dust suppression nozzle sprayers	Consumption per unit	3.5	m^3/h
	Maximum consumption	7	m^3/h
	Total utilisation	60	%
	Average consumption	4.2	m^3/h
	95% probability consumption	3.5	m^3/h
	95% probability consumption per unit	1.7	m^3/h
	Pressure	2	bar

16.10.5.3 WATER PRESSURE

As mine equipment requires water pressure ranging from 2 to 5 bar; the required pressure at the supply point will be the greatest value, that is, 5 bar.

The supply system is anticipated to be gravity driven; thus, the supply tank will be located at the highest part of the system and pressure controlled through pressure reducing valves. These will ensure that the hydrostatic pressure at the supply point outlet does not exceed the predefined limit value, that is, 5 bar. The location of the valves will be done in a way that that the 5-bar pressure is achieved at all mine levels and that the pressure in the main pipe does not exceed 8 bar. The location and number of required valves shall be part of a detailed engineering distribution network design exercise, but IGAN has included these in the simulation, considering that one valve is needed per level.

16.10.5.4 PIPELINE LAYOUT

Considering the designed mine layout, a high-level design of the piping network is shown in Figure 16.54. A freshwater reservoir on surface is connected to the different mine areas through gravity flow.

Unlike with compressed air or electricity, an internal distribution ring cannot (and should not) be considered. The nature of the fluid to be transported, that is water, and the

presence of pressure regulating valves can cause the appearance of very low-pressure points in the branches connecting the levels. This can cause cavitation phenomena or even destruction of the pipes due to negative pressures.

It should be noted that the design of the water distribution network has been done to the level entry points. This is since pipes in production levels may be damaged by blasting, or simply recycled for use in other levels: Effectively, they can be considered more alike to a production cost than a capital cost. However, and although they are not raised in the capital cost, they are included in the network simulations.

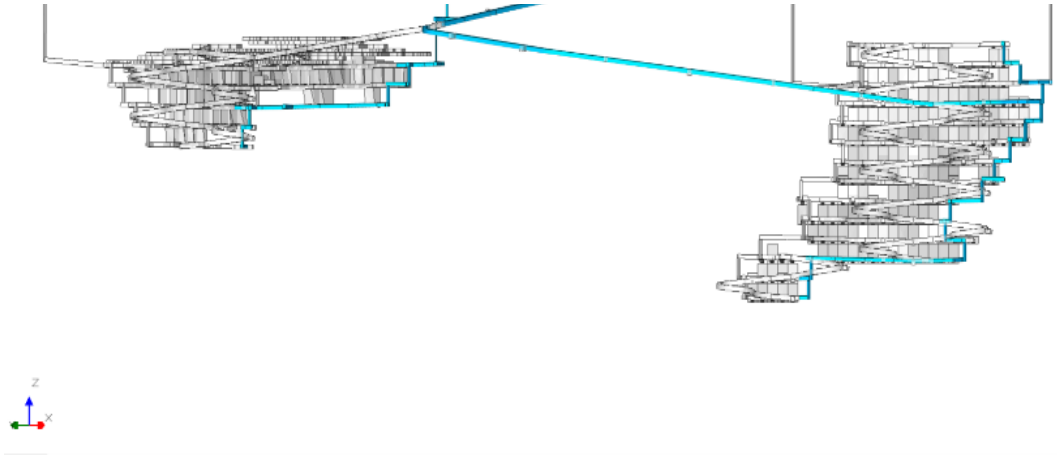


Figure 16.54 – Freshwater main piping system

16.10.5.5 PIPE DIAMETER AND PIPE CLASS

The diameter is the variable with the greatest influence on the development of flow resistance within a pipe. It therefore must be selected with care.

HDPE has been chosen as the piping material due to its ease of handling and installation (something that must be considered when planning an in-mine system) as well as its resistance to corrosion due to possible water condensation in the pipes. The HDPE class should be at least PN10 (SDR13.5) considering that HDPE has a decrease in resistance with temperatures over 20ºC.

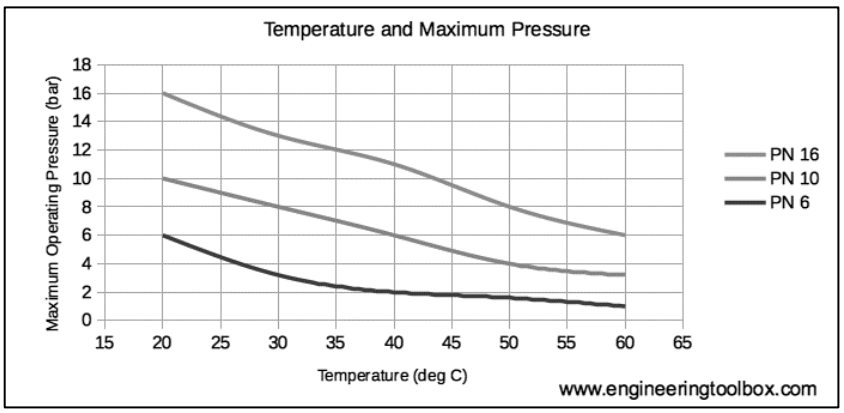


Figure 16.55 – HDPE operating resistance with temperature

IGAN has carried out several simulations with different diameters to evaluate the most suitable one, with a 100 mm diameter considered for most of the network. There are two exceptions to this rule, namely:

- The branch feeding the south through the main ramp must achieve a higher pressure so the upper levels therein can be fed by gravity. This portion of the network ought to be built using steel pipe.

- The branch feeding the raise from the freshwater reservoir must be of at least 150 mm diameter. If 100mm piping were to be used, friction losses in the branch could lead to negative pressures with a risk of cavitation or even pipe collapse.

Detail on the location of these branches can be seen in Figure 16.56.

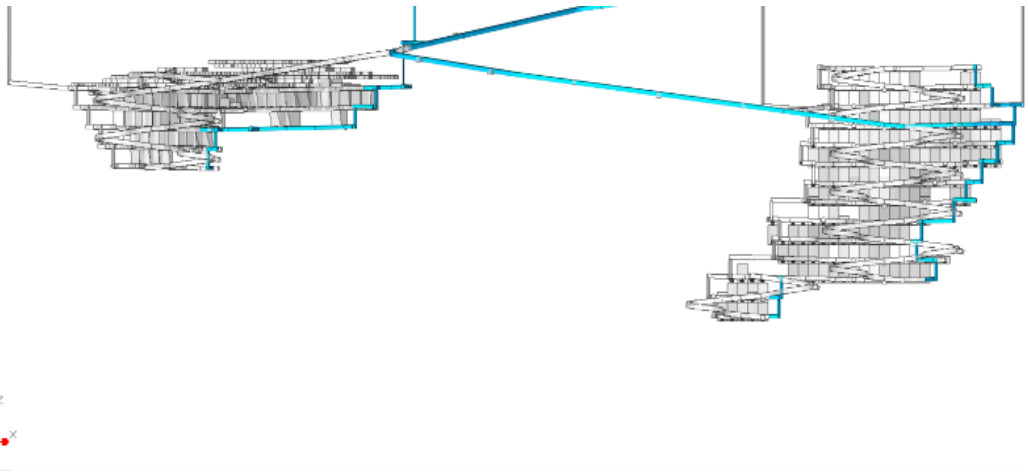


Figure 16.56 – Freshwater system. In blue 100mm HDPE, in green 150mm HDPE, in red 100mm steel. View from the west.

16.10.5.6 FITTINGS AND VALVES

The selected fittings ought to be standardized so the installation across levels is similar. IGAN has followed the following criteria for the selection and simulation of fittings and valves.

- There will be at least one valve at each level change.
- Each branch shall have one valve.
- Curves with a less than 10 m radius will have a 90º elbow.
- If raises are not aligned between levels, 90º or 45º elbows will be used to connect them.
- Gate valves will be used in all cases.
- The bottom of each raise is identified as the ideal point for the installation of pressure regulation valves.

16.10.5.7 SIMULATION

A simulation was conducted with the Pipe Flow Expert software.

A 3D wire diagram, shown in Figure 16.57, representing the network was designed and relevant network properties (pipe material, roughness, fluid properties...) assigned to all its components.

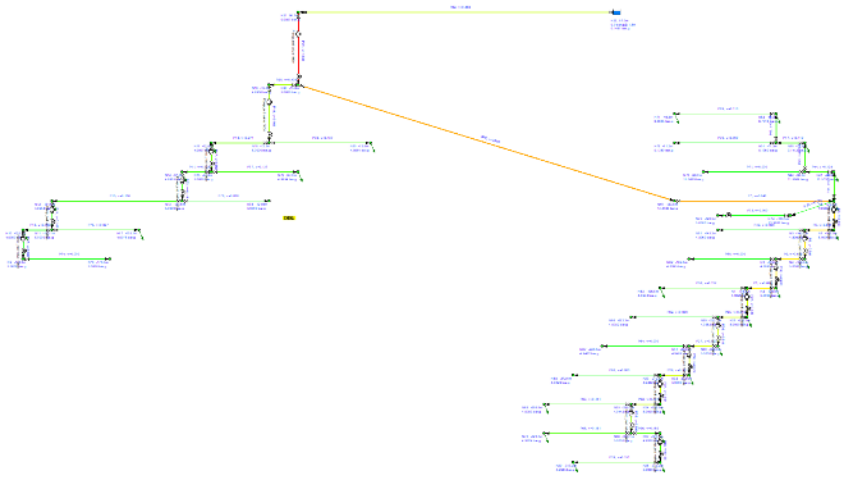


Figure 16.57 – Wire diagram of freshwater network

The main results of the simulation are as follows:

- The piping must be at least 100mm underground and 150mm on surface.
- There can be no closed nodes (rings) in the freshwater network. This leads to conflicts between pressure regulating valves, which in turn can cause cavitation in the pipes.
- Other combinations of diameters for the main and secondary network should be studied to allow cost optimization.

The need for aeration valves at low pressure points should be also considered.

16.10.6 FUELLING AND LUBRICATION

Refuelling of equipment will be done on surface at a designated fuel station. There is as well an allowance of two fuelling/lube trucks in the fleer that will refuel the fleet UG as needed. No allowance has been made for UG fuel storage.

Lubing will be done either at the surface workshop or by the fuelling trucks underground.

16.10.7 WORKSHOP AND STORAGE

Due to the size of the mine, an UG workshop deemed as not required.

16.10.8 EXPLOSIVE MAGAZINES

The explosive magazine will be installed on the surface. A set of 4 mini magazines (minipols) will be installed UG for storing detonators and small quantities of explosive that might not be consumed during a blast.

16.10.9 REFUGE STATIONS

Portable stations are to be used for the development and operation of the mine (Figure 16.58). The budgeted chambers have capacity for 15 people and have complete autonomy for 24 hours. They are built in steel with an airlock door and equipped to provide air with 100 Pa positive pressure. They will be climatized and even in the case that the compressed air does not reach the chamber, they will be equipped with built-in CO_2 scrubbers and oxygen tanks. Three units will be necessary.

Figure 16.58 – Refuge station

Additionally, permanent stations will be built along the mine to serve areas not covered by the portable stations once they are moved away (Figure 16.59). These permanent stations can be built in decommissioned drives (such as stockpiles used for mine development) or in dedicated development.

They will accommodate 40 people and will be equipped with: an airlock entrance, a battery back-up electrical system, an air conditioning unit, a carbon dioxide/carbon monoxide scrubbing unit, cache of oxygen-type cylinders, emergency supply of first aid, food, water, oxygen candles as well as connection to the compressed air system. The refuge station will be in a bay off a drift and will be separated from the drift by a concrete wall. Access to the station will be through an airlock system.

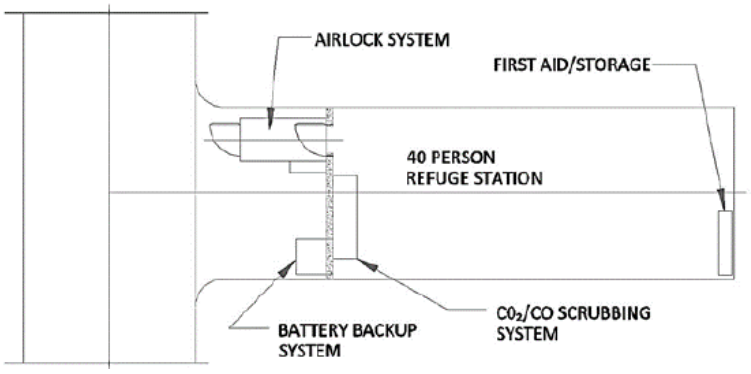


Figure 16.59 – Permanent refuge station

16.10.10 COMMUNICATIONS

Underground communications will employ a leaky feeder system installed in all the main infrastructure of the mine. This cable acts as an antenna, delivering VHF/UHF band radio as well as data transmission to all locations where the cable is installed.

Although this is a well-proven and robust system that is capable of handling communications as well as data transmission for equipment control (localization, ventilation on demand, pump control…) or other services, additional data transmission networks (e.g., WiFi) may be considered as well to enable the use of advanced technologies requiring a larger network bandwidth.

16.11 MANPOWER REQUIREMENTS

16.11.1 SCHEDULE

A 100% owner operated fleet and workforce is considered in the study, which has considered a 365-day working year with two daily shifts, each totalling 10.5 working hours. This allows one and a half for evacuation of blast fumes after end-of-shift blasting. However, the effective working time per shift per day will be less than 10.5 hours once travel time and daily pre-shift briefings are accounted for. IGAN estimates the effective working time per shift during production operations to be 9.35 hours.

A 100% contractor case, with owner retaining key roles (i.e., supervision and technical services) has been considered as well for comparative purposes.

16.11.2 ORGANISATION AND MANPOWER

The underground mining team will be organized into a Mining and Maintenance operational groups respectively. The Mining group is further broken down into mining supervision, safety & training, production, and technical services. The maintenance group is broken down into supervision and maintenance teams.

The average hired personnel head count by operational group in the nominal production period will be an average of 203 with a maximum of 266 on Q1Y5. A breakdown by group and subgroup is shown in Table 16.70.

Table 16.70 – Manpower per department and role

Group	Subgroup	Role	Head count
Mining	Supervision	Superintendent	1
		Mine captain / Shift leader	4
		Clerk	0
	Safety & Training	Safety Engineer	0
	Production	Jumbo Bolter Cable Bolter	44
		Emulsion Loader	24
		LHD	24

Group	Subgroup	Role	Head count
		Scissor Lift Forklifts Mobile Breaker Boom Truck	48
		Shotcrete	12
		Haulage Truck / Transmixer	40
		Production Drilling	12
		Slot Raise DTH Drilling	3
	Technical Services	Chief Engineer	1
		Mine Engineer	6
		Mine technician	2
		Surveyor	2
		Survey Technician	2
		Chief Geologist	1
		Senior Geologist	2
		Geologist	2
		Geology Technician	4
Maintenance	Supervision	Maintenance General Foreman	1
		Electrical General Foreman	1
		Maintenance Supervisor	4
		Electrical Supervisor	4
		Maintenance Planner	1
	Maintenance	Lampman	1
		Static Equipment Mechanic	4
		Mobile Equipment Mechanic	18
		Light Vehicle Mechanic	4
		Electrician	4
		Instrumentation	1

17 RECOVERY METHODS

17.1 INTRODUCTION

This chapter describes the Mineral Treatment Plant of Lagoa Salgada designed for the treatment of the ore coming from the underground mine and corresponds to the following five differentiated domains: fresh massive sulphide, transition sulphide, stringer, stockwork and gossan, for obtaining metal concentrates of copper, lead, zinc, tin, and an additional product in form of dore bars (silver and gold). The process plant will consist of a number of areas such as crushing, grinding, flotation, leaching, dewatering, reagents, and auxiliary services (air, fresh and process water, etc.) that will be described in detail and independently for each domain below.

Minepro Solutions Sl., a Metallurgical and Process Engineering company located in Asturias – Spain, has designed the crushing, grinding, flotation, leaching, dewatering, reagents and auxiliary services areas as well as the general infrastructure.

The design of the process plant has been based upon the design criteria document LS-PROCESS DESIGN CRITERIA, and always uses the most conservative case allowing the greatest flexibility and adaptability to different scenarios.

The following figure shows the General Process Flowsheet Diagram:

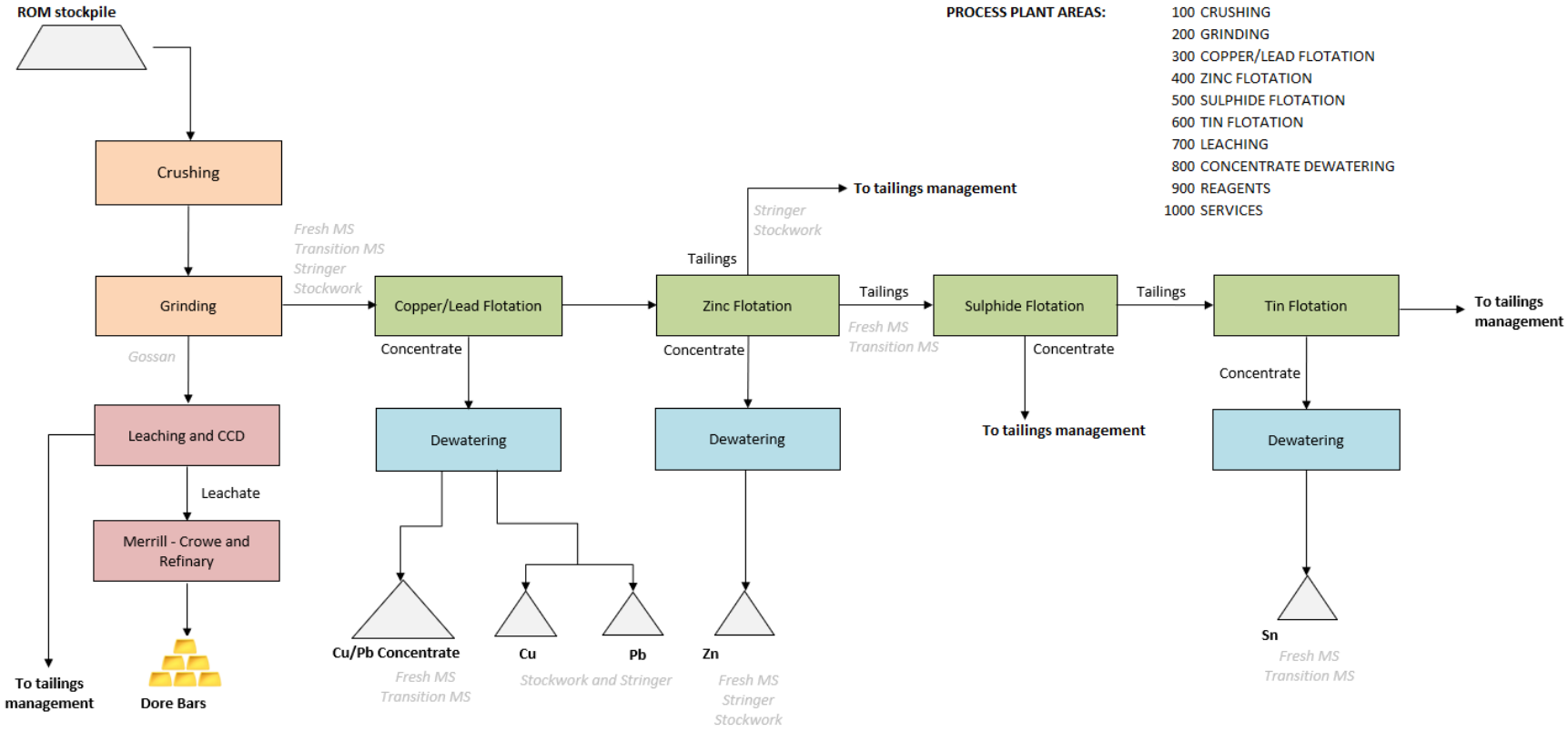


Figure 17.1 – General process flowsheet diagram

17.2 PROCESS DESCRIPTION

The mineral treatment plant will mainly consist of 10 different process areas where the mineral will be crushed, ground and then fed to several flotation processes where Cu, Pb, Zn and Sn concentrates and Au/Ag dore bars will be produced. Other than crushing, grinding, dewatering and auxiliary services, common to all mineral domains, the remaining process areas will be configured depending on the mineral domain being processed, briefly summarised below:

- Common process areas for all mineral domains:
  - Area 100, Crushing.
  - Area 200, Grinding.
  - Area 800, Dewatering.
  - Area 900, Reagents.
  - Area 1000, Auxiliary Services.
- Specific process areas for Fresh Massive Sulphide:
  - Area 300, Copper, and Lead flotation (bulk Cu/Pb concentrate will be produced).
  - Area 400, Zinc flotation.
  - Area 500, Sulphide flotation.
  - Area 600, Tin flotation.
- Specific process areas for Transition Sulphide:
  - Area 300, Copper, and Lead flotation (bulk Cu/Pb concentrate will be produced).
  - Area 400, Zinc flotation.
  - Area 500, Sulphide flotation.
  - Area 600, Tin flotation.
  - Area 700, Leaching.

- Specific process areas for Stockwork and Stringer:
    - Area 300, Copper, and Lead flotation (separate Cu and Pb concentrates will be produced, using the Cu/Pb rougher circuit for Cu and the Cu/Pb scavenger circuit for Pb).
    - Area 400, Zinc flotation.
- Specific process areas for Gossan:
    - Area 600, Tin flotation.
    - Area 700, Leaching.

The following chapters will describe the details of each process area, indicating the mechanical equipment involved and related flowsheet configuration.

17.2.1 AREA 100: CRUSHING

Throughout the life of mine, ore will be trucked from underground to the crushing area where it will be dumped through a static protection grizzly into a RoM Bin (100-HOPP-01), or alternatively fed via the ROM pad and front-end loader. A grizzly feeder will discharge the ore (100-GRIZ-01). The primary jaw crusher (100-CRUS-01) will crush ore coarser than 200 mm. Passing from the grizzly and primary crusher the ore will be discharged on a sacrificial conveyor belt (100-CONV-01) that will transfer onto the silo feed conveyor belt (100-CONV-02) and the transfer the material to a silo (100-SILO-01) which has an 8 hours live capacity. The silo conveyor belt will be fitted with a weightometer (100-WEIG-01), a metal detector (100-METD-01), and a tramp magnet (100-MAGT-01) to remove any tramp material or metallic particles present in the feed. The silo will discharge through a feeder (100-FEED-02) to bring the material on the mill feed conveyor belt (100-CONV-03) that will feed the mineral to the grinding circuit. This conveyor belt will be fitted with a weightometer (100-WEIG-02) for the plant production reconciliation.

As a contingency measure, an emergency bypass flopper gate will direct the material either directly to the silo or bypass the silo to a bypass conveyor (100-CONV-04) feeding a stockpile next to the silo (100-SILO-01). The emergency stockpile will have a 2-day production volume capacity. A front-end loader on a hopper (100-HOPP-02) will convey the emergency stockpile material with a vibrating feeder (100-FEED-01) and return the material to the mill feed conveyor belt.

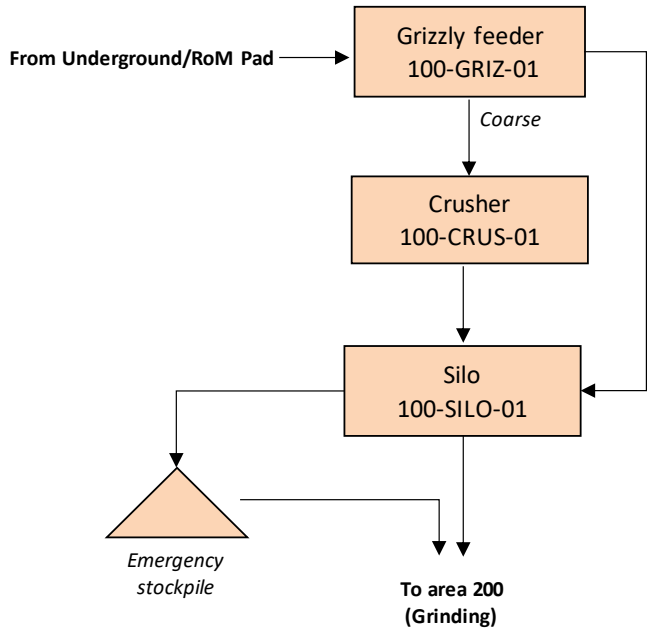


Figure 17.2 – Area 100 (crushing) diagram

17.2.2 AREA 200: GRINDING

The grinding circuit will receive the material from the crushing area and produce a final ground product with a D_{80} of 29 µm using the three following stages: SAG mill, ball mill and vertical mill in closed circuit with two batteries of hydrocyclones.

The ore from the crushing area will be discharged in a SAG mill (200-SAGM-01), where water will be added to create a slurry, at this point representing the start of the wet process flowsheet. The SAG mill will discharge the ground product through gravity to a vibrating screen (200-VSCR-01) that will reject the pebbles and recirculate the material back, via conveyor belt (200-CONV-01), to a pebble crusher. This conveyor belt will be fitted with a metal detector (200-METD-01) and a tramp magnet (200-MAGT-01) to remove any tramp steel or metallic particles in the ore. This ore will be discharged on a hopper (200-HOPP-01) installed with a feeder (200-FEED-01) to convey the material into a cone pebble crusher (200-PEBC-01), that will discharge onto the mill feed conveyor. The vibrating screen bypass will be collected through gravity in a mill discharge pump box (200-PBOX-01) with two horizontals pumps (200-HPMP-01A/B).

The horizontals pumps will drive the slurry to the primary hydrocyclones battery (200-CYCL-01) that will generate two streams, underflow (coarse ore) and overflow (fine ore). The underflow stream will discharge to the ball mill (200-BMIL-01), where water will be added to adjust the slurry density and will discharge the ground product through gravity to the same mill discharge pump box collecting what passes through the vibrating screen. The hydrocyclones overflow will be collected through gravity on a pump box (200-PBOX-02) with horizontals pumps (200-HPMP-02A/B).

Previous horizontals pumps will drive the slurry to a secondary hydrocyclone's battery (200-CYCL-02). The underflow will feed the tertiary grinding stage, with a vertical mill (200-VMIL-01) discharging the product to the same pumping units collecting the primary cyclones overflow. The overflow will represent the final grinding stream flowing to the next flotation area.

The whole grinding area will be implemented on a concrete basin with negative slope to the sump pumps (200-SPMP-01A/B) collecting any slurry spillage and returning it to the circuit. A tower crane (200-TCRA-01) will cover the entire area for maintenance purposes and grinding media charging.

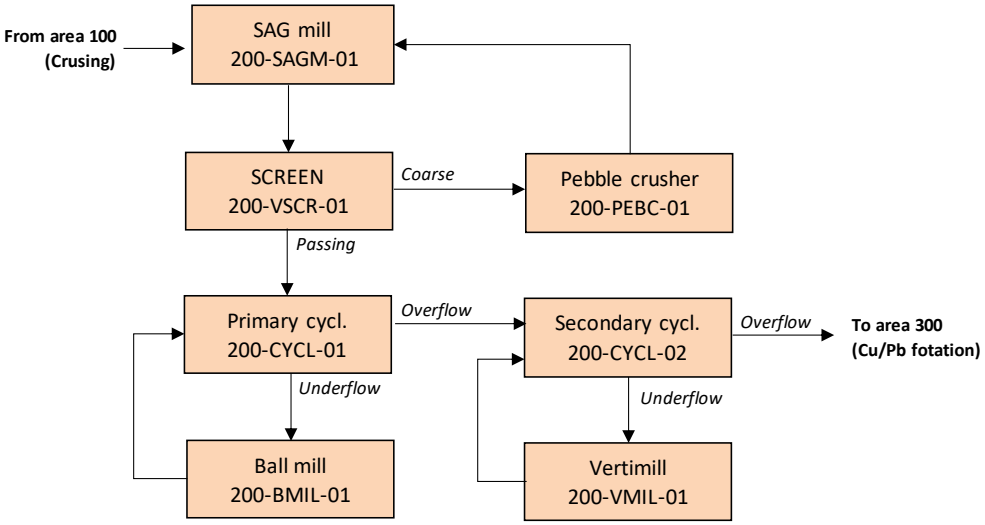


Figure 17.3 – Area 200 (grinding) diagram

17.2.3 AREA 300: COPPER AND LEAD FLOTATION

This area will be used for the flotation of Cu and Pb metals according to the mineral domain being processed. The copper and lead flotation circuit will comprise an initial stage with aeration and conditioning tanks, followed by a rougher line with three rougher cells, a regrinding mill in closed circuit with hydrocyclones and four stages of cleaning in six cells. Rougher tailings will flow to an additional conditioning tank feeding the scavenger cell, whose concentrate will be pumped to an additional regrinding stage, equivalent to the one previously described for the rougher, followed by two stages of cleaning in four cells.

The design of this circuit will enable the production of bulk or separate Cu and Pb concentrates, depending on the mineral domain being processed. For the production of bulk Cu/Pb concentrate, when processing the Fresh MS or Transition MS, the two rougher and scavenger circuits will work in parallel, producing one single Cu/Pb concentrate. In case of Stockwork and Stringer processing, to produce separate Cu and Pb concentrates, the rougher circuit will function as Cu flotation circuit and the scavenger one would do as Pb flotation circuit.

The design was considered for the most conservative case in terms of flotation volume, splitting the distinct stages as much as possible to get the flexibility required for the treatment of different domains. When processing a mineral domain that will not require the entire flotation volume, slurry will keep flowing through all the flotation cells, closing the air inlet to the ones that are inoperative, or alternatively diverted using bypass lines.

The whole area will be implemented on a concrete basin, split between the different areas to collect either high- or low-grade materials, with negative slope towards the sump pumps (300-SPMP-01A/B) returning spillages to the circuit.

17.2.3.1 CONFIGURATION TO PRODUCE BULK CU/PB CONCENTRATE WHEN PROCESSING FRESH MS AND TRANSITION MS MINERAL DOMAINS:

Material exiting the grinding circuit with a nominal D_{80} of 29 µm and approximately 32.5% of solids (w/w) will be pumped to the aeration (300-AERA-01) and conditioning (300-COND-01) tanks. The conditioned slurry will then flow through gravity to the rougher flotation cells (300-FLOT-01 to 03). In these cells, a rougher concentrate will be collected and pumped to a regrinding stage consisting of a battery of hydrocyclones (300-CYCL-01) and a regrinding mill (300-TMIL-01). Material exiting the hydrocyclone overflow (with a nominal D_{80} of 12 µm) will feed an additional conditioning tank (300-COND-03) and, through gravity, will flow to the cleaning cells (300-FLOT-05 to 10) consisting of four cleaner stages. Concentrate produced in the last cleaning stage will represent the first Cu/Pb concentrate stream. Cleaner tailings will be collected in a pump box (300-PBOX-05) with horizontal pumps (300-HPMP-05A/B), where the slurry will be pumped to the cleaner conditioning tank of the scavenger circuit (300-COND-04).

Rougher tailings will feed an additional conditioning tank (300-COND-02) feeding the scavenger cell (300-FLOT-04). The concentrate will be diverted to a regrinding circuit where a regrinding mill (300-TMIL-02) in closed circuit with a battery of hydrocyclones (300-CYCL-02) will reduce the particle size distribution to a D_{80} of 12 µm. Regrinding scavenger concentrate, together with previous cleaner tailings, will get to a conditioning tank (300-COND-04) feeding two additional stages of cleaner cells (300-FLOT-11 to 14). Concentrate from the last cleaner stage will represent the second Cu/Pb concentrate that, together with the first one, will be pumped to the dewatering area. Scavenger and cleaner tailings will be collected in a pump box (300-PBOX-03) and pumped (300-HPMP-03A/B) to the Zn flotation area.

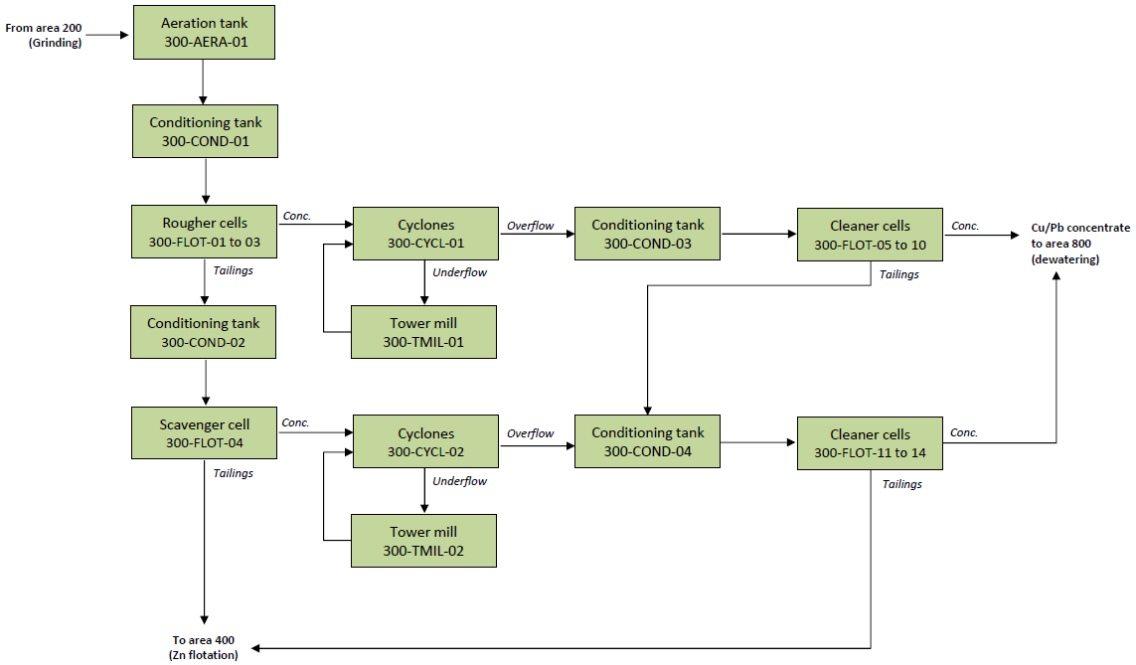


Figure 17.4 – Area 300 (Cu and Pb flotation) diagram to produce bulk Cu/Pb concentrate

17.2.3.2 CONFIGURATION TO PRODUCE SEPARATE CU AND PB CONCENTRATES WHEN PROCESSING STOCKWORK AND STRINGER MINERAL DOMAINS:

Material exiting the grinding circuit with a nominal D_{80} of 29 µm and approximately 32.5% of solids (w/w) will be pumped to the aeration (300-AERA-01) and conditioning tanks (300-COND-01). The conditioned slurry will then flow through gravity to the rougher flotation cells (300-FLOT-01 to 03). In these cells, a rougher concentrate will be collected and pumped to a regrinding stage consisting of a battery of hydrocyclones (300-CYCL-01) and a regrinding mill (300-TMIL-01). Material exiting the hydrocyclone overflow with a nominal D_{80} of 12 µm will feed the conditioning tank (300-COND-03) and through gravity will flow to the four stages of cleaning cells (300-FLOT-05 to 10). Concentrate produced in the last cleaning stage will represent the final Cu concentrate to be pumped to the dewatering area. Cleaner tailings will be pumped to the cleaner conditioning tank of the Pb circuit.

Copper rougher tailings will feed an additional conditioning tank (300-COND-02) feeding the Pb rougher stage (300-FLOT-04). The rougher concentrate will be diverted to a regrinding circuit where a regrinding mill (300-TMIL-02) in closed circuit with a battery of hydrocyclones (300-CYCL-02) will reduce the size distribution to D80 of 12 µm. Re-ground rougher concentrate, together with Cu cleaner tailings, will feed a conditioning tank (300-COND-04) feeding the two stages of cleaner cells (300-FLOT-11 to 14). Concentrate from the last cleaner stage will represent the final Pb concentrate to be pumped to the dewatering area. Rougher plus cleaner tailings will be collected in a pump box (300-PBOX-03) and pumped (300-HPMP-03A/B) to the Zn flotation area.

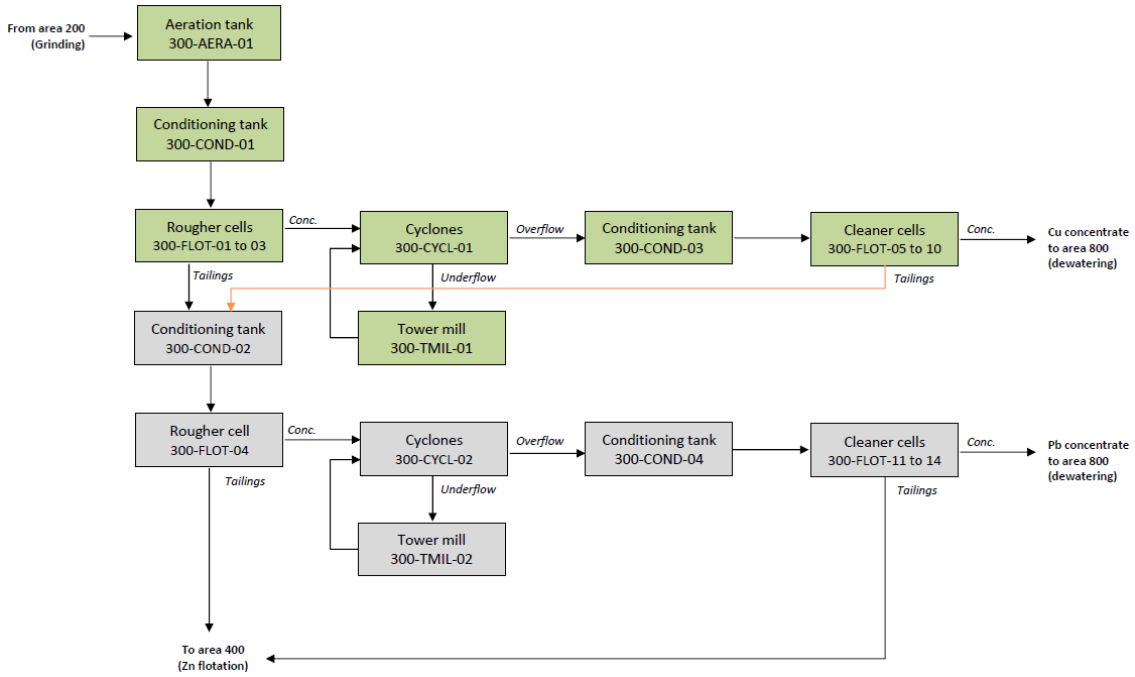


Figure 17.5 – Area 300 (Cu and Pb flotation) diagram to produce separate Cu and Pb concentrates

17.2.4 AREA 400: ZINC FLOTATION

This area will be used for the flotation of Zn metals according to the mineral domain being processed. The Zn flotation circuit will comprise an initial stage with conditioning tanks, followed by a rougher line with two rougher cells, a regrinding mill in closed circuit with hydrocyclones and four stages of cleaning in eight cells. Rougher tailings will flow to two scavenger cells, whose concentrate will be pumped to an additional regrinding stage, equivalent to the one previously described for the rougher, followed by two stages of cleaning in two cells.

Except for the Fresh MS that will flow through all flotation stages and cells in this process area, the rest of domains will only use the rougher and cleaner circuit. The design was considered for the most conservative case in terms of flotation volume, splitting the distinct stages as much as possible to get the flexibility required for the treatment of different domains. When processing a mineral domain that will not require the entire flotation volume, slurry will keep flowing through all the flotation cells, closing the air inlet to the ones inoperative, or alternatively diverted using bypass lines.

The whole area will be implemented on a concrete basin, split between the different areas to collect either high- or low-grade materials, with negative slope towards the sump pumps (400-SPMP-01A/B) returning spillages to the circuit.

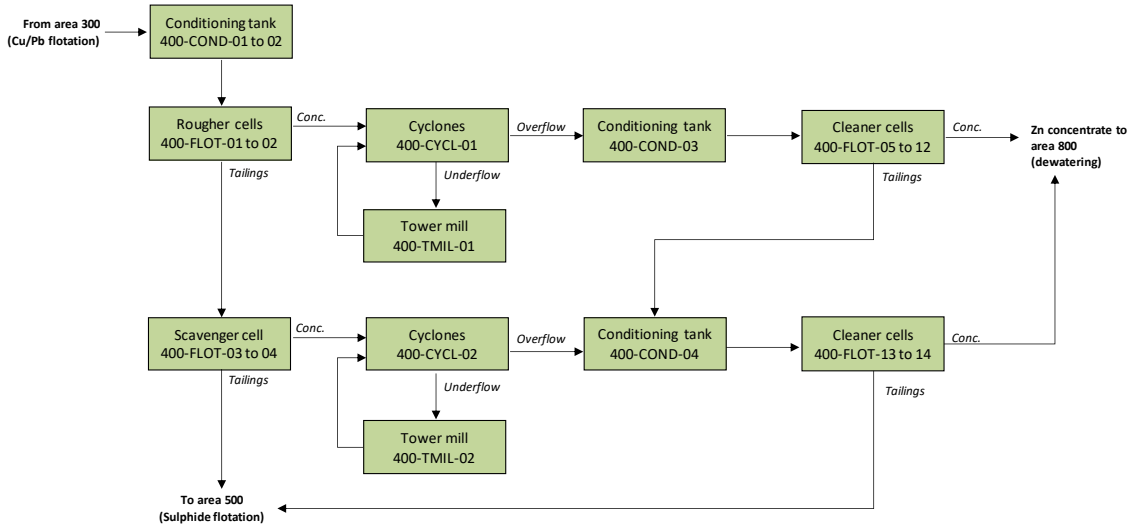


Figure 17.6 – Area 400 (Zn flotation) diagram

Material exiting the Cu/Pb flotation circuit with a nominal D_{80} of 20 µm and approximately 29.5% of solids (w/w) will be pumped to the conditioning tanks (400-COND-01 and 02). The conditioned slurry will then flow through gravity to the rougher flotation cells (400-FLOT-01 and 02). In these cells, a rougher concentrate will be collected and pumped to a regrinding stage consisting of a battery of hydrocyclones (400-CYCL-01) and a regrinding mill (400-TMIL-01). Material exiting the hydrocyclone overflow (with a nominal D_{80} of 12 µm) will feed a conditioning tank (400-COND-03) and, through gravity, will flow to the cleaning cells (400-FLOT-05 to 12) conforming to the four cleaner stages. Concentrate produced in the last cleaner stage will represent the first Zn concentrate stream. Cleaner tailings will be pumped (400-HPMP-04A/B) to the cleaner conditioning tank of the scavenger circuit.

Rougher tailings will feed the scavenger cells (400-FLOT-03 and 04). In these cells, a scavenger concentrate will be collected in a pump box (400-PBOX-03) and pumped (400-HPMP-03A/B) to another regrinding stage consisting of a battery of hydrocyclones (400-CYCL-02) and a regrinding mill (400-TMIL-02). Material exiting the hydrocyclone overflow (with a nominal D_{80} of 12 µm) will feed a conditioning tank (400-COND-04) and, through gravity, will flow to the cleaning cells (400-FLOT-13 and 14) conforming to the two cleaner stages. Concentrate from the last cleaner stage will represent the second Zn concentrate that, together with the first one, will be pumped to the dewatering area. Scavenger plus cleaner tailings will be collected in a pump box (400-PBOX-02) and pumped (400-HPMP-02A/B) to the next area.

When processing Stockwork and Stringer domains, the Zn flotation tailings will be pumped to the tailings management area. In the case of Fresh MS and Transition MS, the slurry would be transferred to the next sulphide flotation area.

17.2.5 AREA 500: SULPHIDE FLOTATION

This area will be used for the removal of sulphides when processing the Fresh MS and Transition MS domains, prior to the concentration of tin minerals in the following process area. Sulphides will be concentrated using a flotation circuit consisting of a conditioning tank, a rougher stage with four cells and three cleaner stages with one cell each.

The area will be implemented on a concrete basin, split between the different areas to collect either high- or low-grade materials, with negative slope towards the sump pumps (500-SPMP-01A/B) returning spillages to the circuit.

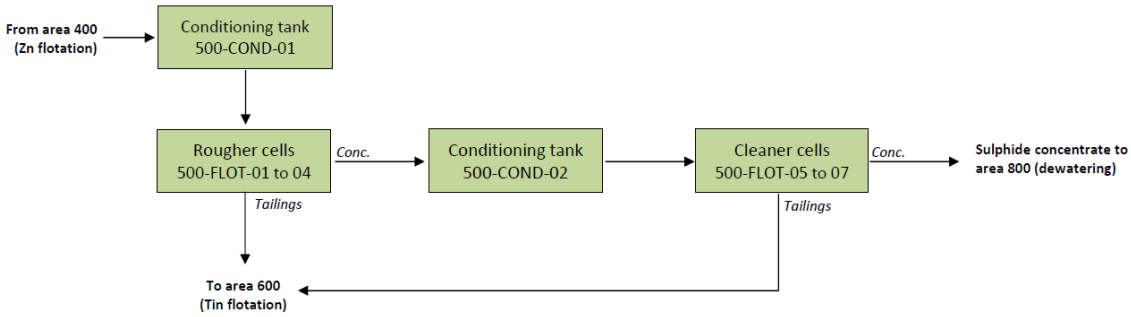


Figure 17.7 – Area 500 (sulphide flotation) diagram

Material exiting the Zn flotation circuit, with a nominal D_{80} of 20 µm and approximately 27.9% of solids (w/w), will be pumped to the conditioning tank (500-COND-01). The conditioned slurry will then flow through gravity to the rougher flotation cells (500-FLOT-01 to 04). In these cells, a rougher concentrate will be collected and pumped to the conditioning tank (500-COND-02), and through gravity will flow to the cleaning cells (500-FLOT-05 to 07) conforming to three cleaner stages. Concentrate produced in the last cleaning stage will represent the sulphide stream that will be pumped to the tailings management area. Rougher tailings will be collected in a pump box (500-PBOX-01) and pumped (500-HPMP-01A/B) to the Sn flotation area.

17.2.6 AREA 600: TIN FLOTATION

The tin flotation will concentrate the tin minerals when processing the Fresh MS and Transition MS domains, combining flotation and gravimetric concentration technologies. The flotation circuit, consisting of: conditioner and aeration tanks, two cells of rougher flotation followed by five cleaner stages, will produce an intermediate tin concentrate that will be fed to the final gravimetric concentration section. Multi-gravity separators will increase the tin content to a saleable grade, recirculating the tailings back to the head of the cleaner flotation circuit.

The area will be implemented on a concrete basin, split between the different areas to collect either high- or low-grade materials, with negative slope towards the sump pumps (600-SPMP-01A/B) returning spillages to the circuit.

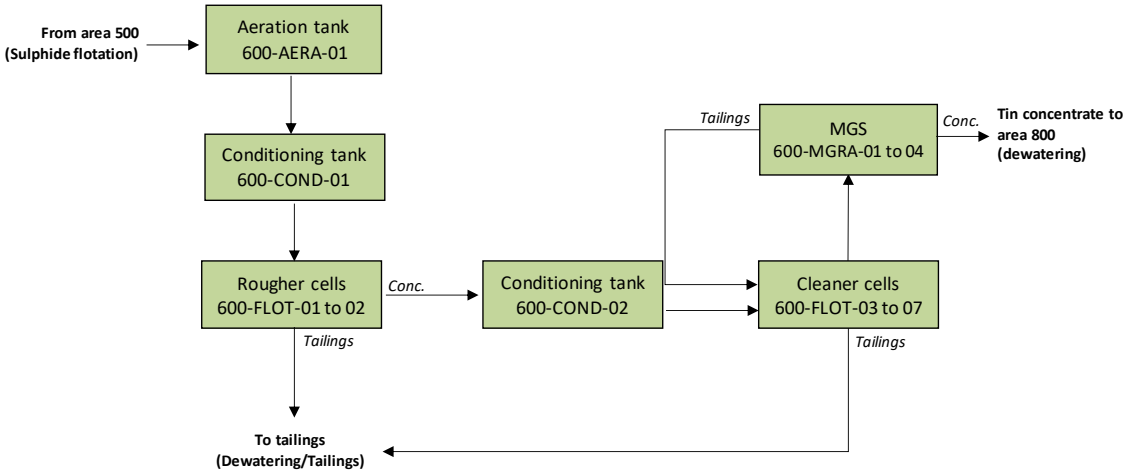


Figure 17.8 – Area 600 (tin flotation) diagram

Material exiting the sulphide flotation circuit, with a nominal D_{80} of 20 µm approximately 26.9% of solids (w/w), will be pumped to the aeration (600-AERA-01) and conditioning tank (600-COND-01). The conditioned slurry will then flow through gravity to the rougher flotation cells (600-FLOT-01 and 02). In these cells, a rougher concentrate will be collected and pumped to a conditioning tank (600-COND-02), and through gravity will flow to the cleaning cells (600-FLOT-03 to 07) conforming to the five cleaner stages. Concentrate produced in the last cleaning stage will be the intermediate Sn concentrate stream that will be pumped to the gravity section. Multi-gravity separators (600-MGS-01 to 04) will increase the tin content to a saleable grade, recirculating the tailings back to the cleaner flotation circuit. Rougher and cleaner tailings will be collected in a pump box (600-PBOX-01) and pumped (600-HPMP-01A/B) to the tailings management area.

17.2.7 AREA 700: LEACHING

This process area will recover Au and Ag present in the Gossan and Transition MS domains through a conventional cyanide leaching process followed by a CCD (Counter-Current Decantation) and Merrill-Crowe precipitation process. All the equipment used in this section will be implemented separately from the main process building.

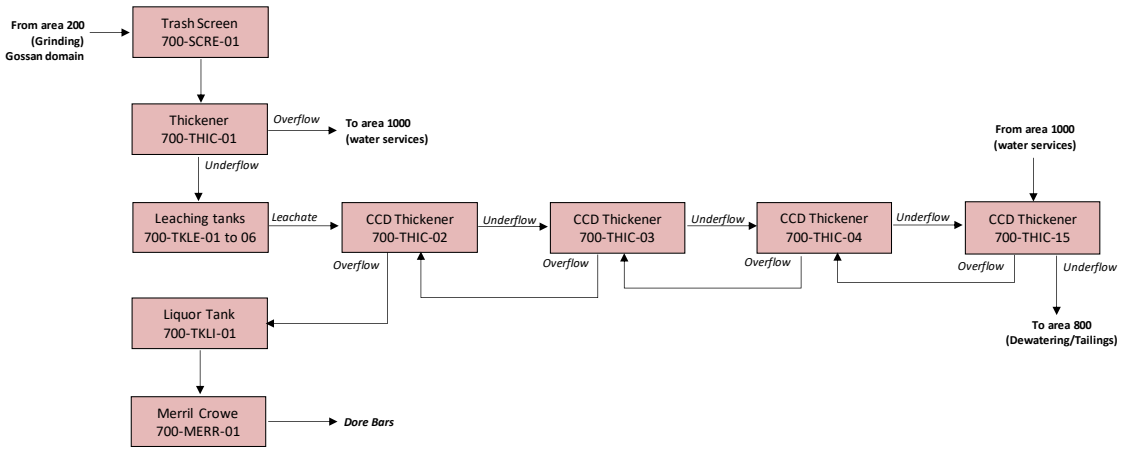


Figure 17.9 – Area 700 (leaching) diagram

Material exiting the grinding circuit, with a nominal D_{80} of 29 µm and approximately 35% of solids (w/w), will be pumped to the leaching area. The first stage will be a screening by a vibrating screen (700-SCRE-01) to remove plastics, wood or other elements that would damage the leaching process. Screen passing will be discharged through gravity to the thickener (700-THIC-01), increasing the solids content to 45% by weight. The thickener underflow will be pumped (700-HPMP-01A/B) to the six leaching tanks (700-TKLE-01 to 06) connected through gravity, where a cyanide solution will be added to leach the metals of interest.

Slurry from the last tank will discharge through gravity to the first of the four CCD thickeners (700-THIC-02 to 05). The underflow of each thickener will be pumped, advancing to the next one while overflow will discharge through gravity to the previous. The counter-current flow pattern helps to maximise the efficiency of the separation process and ensures that the highest possible concentration of the mineral is obtained from the ore.

The first thickener overflow will represent the rich liquor that will be accumulated in a storage tank (700-TKLI-01) feeding the final Merrill-Crowe and refinery area (700-MERR-01), where the addition of zinc to the solution will allow for the efficient precipitation of the metals, and a smelting process happens for obtaining dore bars.

The whole area will be implemented on a concrete basin with a negative slope towards the sump pumps (700-SPMP-01A/B) collecting any slurry spillage and returning it to the circuit.

17.2.8 AREA 800: DEWATERING AND TAILINGS

All concentrate streams coming from the Cu, Pb, Zn, and Sn flotation areas will feed the concentrate dewatering area to recover part of the contained water reducing the moisture content to a suitable level for truck expedition. All the equipment used in this section will be implemented separately from the main process building, in an independent structure with filters in the first level ground floor level for truck loading with front-end loaders.

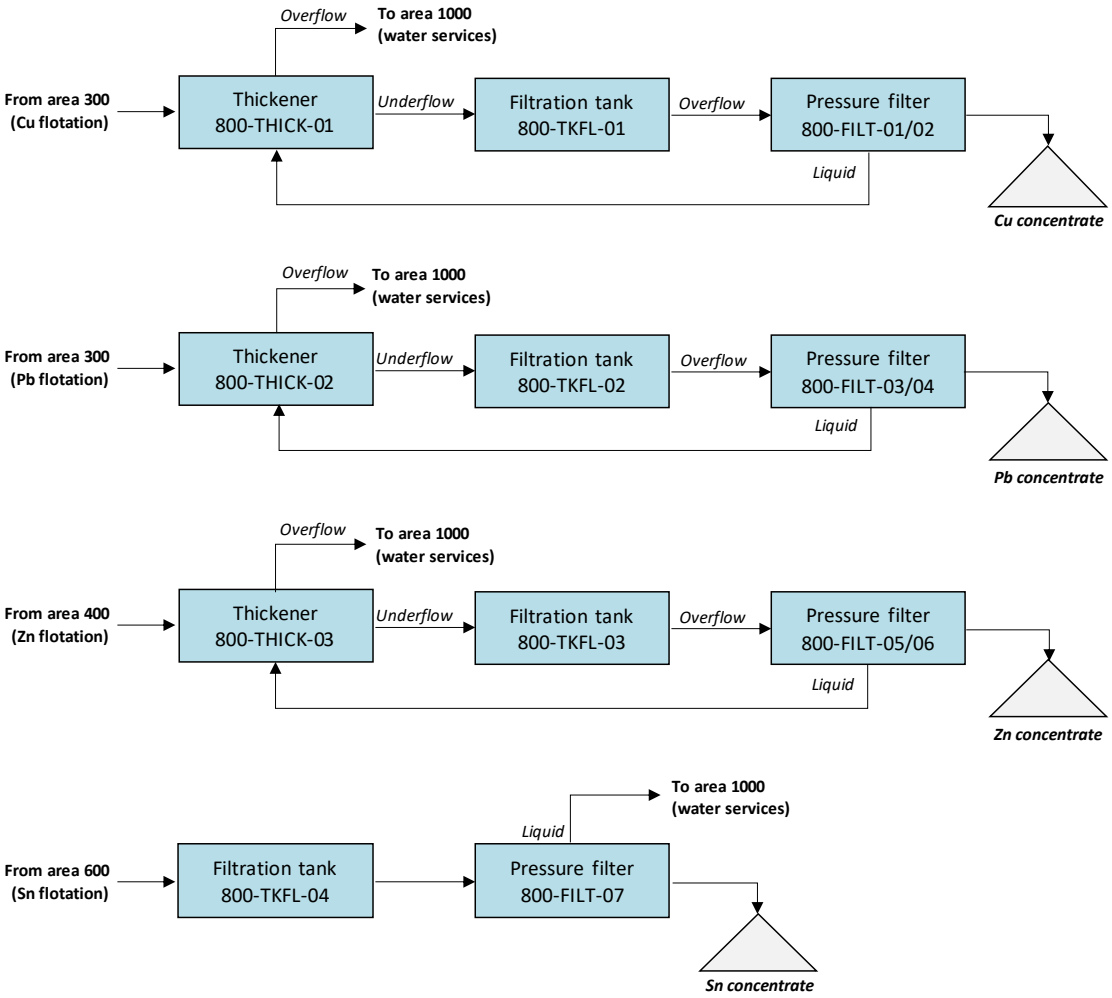


Figure 17.10 – Area 800 (dewatering) diagram

Except for the tin concentrate which will be filtered without an intermediate thickening stage, copper, lead and zinc concentrates will be dewatered using thickeners (800-THIC-01 to 03) and pressure filters (800-FILT-01 to 06). Thickener underflows, with approximately 60% of solids by weight, will be accumulated in intermediate tanks where high- and low-pressure pumps (800-HPMP-01 to 06) will be installed. Filter cakes, with approximately 87% of solids (w/w), will be accumulated in separate stockpiles for truck loading using a front-end loader. The water recovered in the pressure filters will return to the corresponding thickener and thickeners overflow will flow to the process water tank.

As previously described, tin concentrate will be dewatered in a pressure filter (800-FILT-07) without an intermediate thickening stage. The concentrate coming from the tin flotation area will be accumulated in a tank installed with high- and low-pressure pumps (800-HPMP-07 and 08) feeding the filter. Filter cake discharge configuration will be the same as established for the rest of concentrates.

A truck weighbridge will be installed in the truck loading area for concentrate production reconciliation. All thickeners and tanks areas will be implemented on a concrete basin with negative slope towards the sump pumps (800-SPMP-01 to 04) collecting any slurry spillage and returning to the corresponding circuit.

17.2.9 AREA 900: REAGENTS

The following reagents will be used in the process plant. The configuration of the supply, preparation, and distribution system is described below:

- Zinc sulphate (ZnSO4): ZnSO4 (zinc sulphate) is commonly used as an activator in the flotation of sulphide minerals such as sphalerite (ZnS) and galena (PbS) from ores. ZnSO4 is often used as an activator in the flotation of sphalerite and galena because it can increase the hydrophobicity of the sulphide minerals by forming a hydrophobic zinc sulphide (ZnS) species on the mineral surface. This makes the sulphide minerals more responsive to the collector reagents used in the flotation process, resulting in improved recovery of the valuable mineral. This will be supplied as a solid in 1,000 kg big-bag, prepared with an agitated tank (900-TKPT-01) and transferred to a dosage tank (900-TKDT-01) from where the reagent pumps (900-RPMP-02A/B) will supply the product to the consumption points.

- Copper sulphate (CuSO4): Copper sulphate (CuSO4) is commonly used in the flotation of sulphide minerals, particularly copper-bearing minerals such as chalcopyrite (CuFeS2). Copper sulphate is often used as an activator in the flotation of copper sulphide minerals because it can activate the surface of the mineral particles and improve their response to the collector reagents used in the flotation process. Copper sulphate can also form a hydrophobic copper sulphide (CuS) species on the surface of the mineral, further improving its flotation response. This will be supplied as a solid in 1,000 kg big-bag, prepared with an agitated tank (900-TKPT-02) and transferred to a dosage tank (900-TKDT-02) from where the reagent pumps (900-RPMP-04A/B) will supply the product to the consumption points.

- NASFROTH 250: is a frother reagent commonly used in mineral flotation processes, particularly in the flotation of sulphide minerals such as copper, lead, and zinc sulphides. NASFROTH 250 is a blend of aliphatic alcohols and glycols, and is known for its good frothing properties, stability, and selectivity. It is often used in combination with other frother reagents to optimise the froth properties for a given mineral and flotation conditions. The product will be supplied as a liquid in 1,000 litres IBC and transferred to a dosage tank (900-TKDT-03) from where the reagent pumps (900-RPMP-06A/B) will supply the product to the consumption points.

- DOWFROTH 200: is a frother reagent commonly used in mineral flotation processes, particularly in the flotation of sulphide minerals such as copper, lead, and zinc sulphides. DOWFROTH 200 is a blend of polypropylene glycols and alcohol, and is known for its good frothing properties, stability, and selectivity. It is often used in combination with other frother reagents to optimise the froth properties for a given mineral and flotation conditions. The product will be supplied as a liquid

in 1,000 litres IBC and transferred to a dosage tank (900-TKDT-04) from where the reagent pumps (900-RPMP-07A/B) will supply the product to the consumption points.

- AERO 7249: is a specialised collector reagent used in mineral flotation processes, particularly in the flotation of sulphide minerals such as copper, lead, and zinc sulphides. AERO 7249 is a mercaptan-based collector reagent and is known for its selectivity and effectiveness in the flotation of copper and other sulphide minerals. It is often used in combination with other collector reagents to optimise the selectivity and recovery of the target mineral for a given ore and flotation conditions. The product will be supplied as a liquid in 1,000 litres IBC and transferred to a dosage tank (900-TKDT-05) from where the reagent pumps (900-RPMP-10A/B) will supply the product to the consumption points.

- AERO 845: is a promoter or collector chemical that is commonly used in the flotation process of tin minerals. Is a thionocarbamate collector, which means that it contains sulfur and nitrogen atoms in its molecular structure that gives it the ability to adsorb onto the surface of minerals. Aero 845 is often used in combination with other collector chemicals, such as xanthates or dithiophosphates, to improve its efficiency and selectivity. The product will be supplied as a liquid in 1,000 litres IBC and transferred to a dosage tank (900-TKDT-XX) from where the reagent pumps (900-RPMP-XXA/B) will supply the product to the consumption points.

- SIBX: is a collector reagent commonly used in mineral flotation processes, particularly in the flotation of sulphide minerals such as copper, lead, and zinc sulphides. SIBX, also known as sodium isobutyl xanthate, is a xanthate-based collector reagent and is known for its selectivity and effectiveness in the flotation of copper and other sulphide minerals. It is often used in combination with other collector reagents to optimise the selectivity and recovery of the target mineral for a given ore and flotation conditions. The product will be supplied as a liquid in 1,000 litres IBC and transferred to a dosage tank (900-TKDT-06) from where the reagent pumps (900-RPMP-12A/B) will supply the product to the consumption points.

- LIME / Calcium hydroxide (Ca (OH)2): is often used in sulphide mineral flotation to adjust the pH of the slurry to the optimal range for flotation. The addition of lime in sulphide mineral flotation can play a key role in optimising the pH of the slurry and improving the selectivity and recovery of the target mineral. The product will be received solid in 25 t trucks. A silo (900-SILO-01) will storage the product and feed a preparation tank (900-TKDT-07) where the suspension will be pumped (900-RPMP-13) to the consumption points.

- 3814A: is a collector reagent used in mineral flotation processes, particularly in the flotation of copper, lead, and zinc sulphide minerals. 3418A is an alkyl dithiophosphate collector reagent and is known for its effectiveness in the flotation of copper, lead, and zinc sulphide minerals. It is often used in combination with other collector reagents to optimise the selectivity and recovery of the target mineral for a given ore and flotation conditions. The product will be supplied as a

liquid in 1,000 litres IBC and transferred to a dosage tank (900-TKDT-08) from where the reagent pumps (900-RPMP-15A/B) will supply the product to the consumption points.

- SMBS: is a commonly used depressant in mineral flotation processes. It is used to selectively depress sulphide minerals, such as pyrite, arsenopyrite, and pyrrhotite, and prevents them from floating with the valuable minerals. SMBS reacts with the surface of sulphide minerals, forming a surface layer that reduces their hydrophobicity and makes them less likely to attach to air bubbles and float to the surface of the flotation cell. This selective depression of sulphide minerals can improve the grade and recovery of the target mineral. The product will be supplied as a liquid in 1,000 litres IBC and transferred to a dosage tank (900-TKDT-09) from where the reagent pumps (900-RPMP-17A/B) will supply the product to the consumption points.

- Sodium silicate (NaSiO3): is a commonly used depressant in mineral flotation processes. It is used to selectively depress certain minerals, such as quartz and silicate gangue minerals, and prevents them from floating with the valuable minerals. Sodium silicate reacts with the surface of the silicate minerals, forming a surface layer that reduces their hydrophobicity and makes them less likely to attach to air bubbles and float to the surface of the flotation cell. This selective depression of silicate minerals can improve the grade and recovery of the target mineral. This will be supplied as a solid in 1,000 litres IBC and transferred to a dosage tank (900-TKDT-10) from where the reagent pumps (900-RPMP-19A/B) will supply the product to the consumption points.

- NaCN: is a commonly used reagent in the leaching of gold and silver ores. It is used to dissolve and extract the precious metals from the ore in a process called cyanide leaching. This will be supplied as a solid in 1,000 kg big-bag prepared with an agitated tank (900-TKPT-03) and transferred to a dosage tank (900-TKDT-11) from where the reagent pumps (900-RPMP-21A/B) will supply the product to the consumption points.

- Flocculant: are commonly used in mineral treatment plants to aid in the settling of solids and to improve the efficiency of solid-liquid separation processes. Flocculants are polymers that are added to the slurry to aid in the formation of flocs. They work by attracting and bridging the individual particles, causing them to aggregate and form flocs. The flocs settle more rapidly than the individual particles, improving the efficiency of the sedimentation process. The product will be delivered in 25 kg bags and prepared and dosed from a flocculant unit (900-FLOC-01).

17.2.10 AREA 1000: SERVICES

17.2.10.1 WATER

Two independent water systems will be implemented in the process plant: fresh and process water circuits. Fresh water will be obtained from external sources. It will be used for non-industrial purposes, such as general cleaning, irrigation, and sanitation purposes, as well as for the preparation of certain chemical reagents. It will be accumulated in an open tank (1000-TKFW-01) with 500 m^3 capacity installed with centrifugal pumps for principal supplies (1000-HPMP-01A/B). Fire protection system (1000-FIRE-01) will be also implemented in the freshwater tank, with a specific volume dedicated to that sole purpose.

Process water will be used for industrial processes in a mineral treatment plant. Process water will mainly come from aging pond and tailings management area. It will be accumulated in an open tank (1000-TKPW-01) with 1,000 m^3 capacity installed with centrifugal pumps for principal supplies (1000-HPMP-02A/B).

Apart from the two main water tanks, an auxiliary one will be implemented for gland water required in horizontal centrifugal pumps sealings. This tank (1000-TKPW-02) with 250 m^3 capacity will be installed with centrifugal pumps (1000-HPMP-03A/B) and a filtering system supplying the gland water ring.

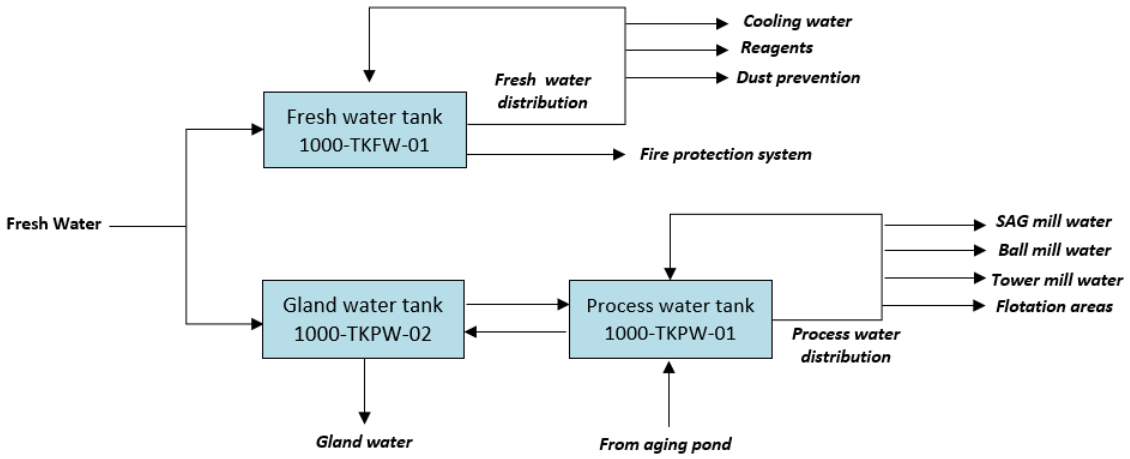


Figure 17.11 – Area 1000 (water services) diagram

17.2.10.2 AIR

Low pressure air required in the flotation cells will be supplied by three blowers (1000-BLOW-01A/C). Compressed air will be supplied from two separate circuits. The first one for cleaning, maintenance and operation purposes with two compressors (1000-COMP-01A/B) and the second one for instruments and control systems such as pneumatic actuators and valves, also with two units (1000-COMP-02A/B).

17.2.10.3 ON STREAM ANALYSIS SYSTEM (OSA)

For real-time process control, OSA system (1000-OSAS-01) will be implemented. The system will use online sensors to measure key process variables, such as the concentrations of valuable minerals and process chemicals, in the slurry as it flows through the flotation circuit. The sensor data will be transmitted to a computer system that will use advanced algorithms to analyse the data and make real-time process control decisions. The computer system can adjust key process parameters, such as the flow rate of air and process chemicals, to optimise the performance of the flotation process in real-time. OSA systems can also provide operators with detailed information about the performance of the flotation circuit, such as the recovery rates and grade of the final concentrate and can alert operators to any deviations from the optimal operating conditions.

17.2.11 THIOSALTS TREATMENT PLANT

Depending on the evolution of the quality of process water, the installation of a water treatment plant will be required to reduce the thiosalt content and allow the water to be used in the flotation processes. This facility will not be implemented from the beginning of the operation, but a reserved area for its future implementation must be considered.

Thiosalts are chemical compounds commonly present in process liquors in a plant for mineral sulphide concentration where grinding and flotation processes are applied. The alkalinity conditions used and the reagents added favour to the partial conversation of the sulphides into thiosalts. The accumulation of these thiosalts, above certain levels, leads to technical, operative, and environmental management issues. Thiosalts are metastable species that finally oxidised to form sulfuric acid through complex series reactions, and in a simplified form, can be represented for the main species, as a thiosulfate and tetrathionates.

There are different alternatives for the treatment of effluents containing thiosalts, being natural oxidation in ponds one of the best options, as a low-cost and easy implement process, neutralizing the acidity by adding alkali. However, the treatments are slow with long residence times (in the order of months) and unreliable at low temperatures due to decreased sulfoxidants activity of the microorganisms.

Other alternative for the thiosalts treatment consists of the addition of an oxidising agent, such as hydrogen peroxide (H2O2). The main objective of this process is to accelerate the oxidation of the intermediate sulphur species to sulphate. Hydrogen peroxide is one of the strongest oxidants available and it has certain advantages such us simple operation and generation of non-toxic products. This reaction has a slow and very unselective oxidation rate, for which the use of catalysts such as the Fenton Fe (II) reagent to assist in the reaction is essential. The main disadvantage of this method is the high cost associated to the reagents used.

The fundamental objective of this oxidation is to eliminate the presence of anionic sulphur species with oxidation number between elemental sulphur (0) and sulphite (+4). For this purpose, once the sulphate oxidation completed, neutralisation must be conducted by

adding an alkali and reaching a neutral pH. The final product, due to the reaction between alkali and sulphate, generates a precipitate that must be separated from the liquid.

The process would consist of the following steps:

- Oxidation from thiosalts to sulphate, using hydrogen peroxide as an oxidant, with a catalysis Fenton Fe(II) reagent to improve the efficiency of the reaction, reducing residence times.

- Alkali addition (typically calcium hydroxide Ca(OH)) to neutralise the solution and precipitate the sulphate in the form of CaSO4 as well as another metal present in the solution.

- Solid and liquid separation with a conventional thickener, to obtain a clarified overflow as the resultant treated water is recirculated to the process plant.

18 PROJECT INFRASTRUCTURE

18.1 MASTERPLAN

The general site masterplan was defined considering several considerations. These included reducing the layout footprint, facilitating flows for people and equipment, separating the industrial area from the social area, and considering the prevailing wind direction to prevent dust contamination of the social area from ore processing.

The figure below depicts the general masterplan, which considers the following considerations:

- A parking area near the main gate is designated for heavy trucks to make temporary stops, such as for documentation control purposes.
- The parking area for employees' light vehicles is located outside the main gate, with no access permitted to the industrial area.
- A weighbridge is situated near the main gate to monitor all incoming and outgoing goods on the property.
- Social buildings are positioned close to the main gate to minimize travel distance for people. Social buildings primarily occupy the left-hand side of the main access, while industrial buildings are situated on the right-hand side.
- Considering that the prevailing wind comes from the northwest, the social buildings are designed to face the southwest. High-speed winds, albeit less frequent, originate from the south, so the social buildings are positioned to the south of the ROM pad.
- The ROM pad and surface crusher are situated as close as possible to the mine portal to minimize travel distances for mining trucks.
- The processing plant is near the ROM pad, and the loading area for concentrates is positioned near the main internal access.
- Buildings serving the mine operation, such as the batch plant, fuel farm, wash-bay, workshop, and warehouse, are situated close to the mine portal to reduce travel distance and centralize services in those areas.
- The brake test ramp is positioned near the mine portal to allow access for light vehicles on their way to the underground area. Heavy vehicles will be equipped with a negative brake system, rendering access to the test ramp non-mandatory. However, the ramp is prepared to accommodate heavy vehicles if necessary.
- The main electrical substation is located near the plant to minimize cable usage and is positioned away from the prevailing wind direction to prevent dust contamination from ore handling.

- The paste fill plant is situated in the optimal location to meet the backfill requirements for the underground area.
- Water utilities are placed near all other buildings to minimize the footprint and consider water flow and treatment criteria.
- The tailings storage facility is situated in the only available area on site that can accommodate a facility of such dimensions while adhering to environmental constraints.
- A heliport is projected near the medical centre to facilitate emergency transportation of individuals if needed.

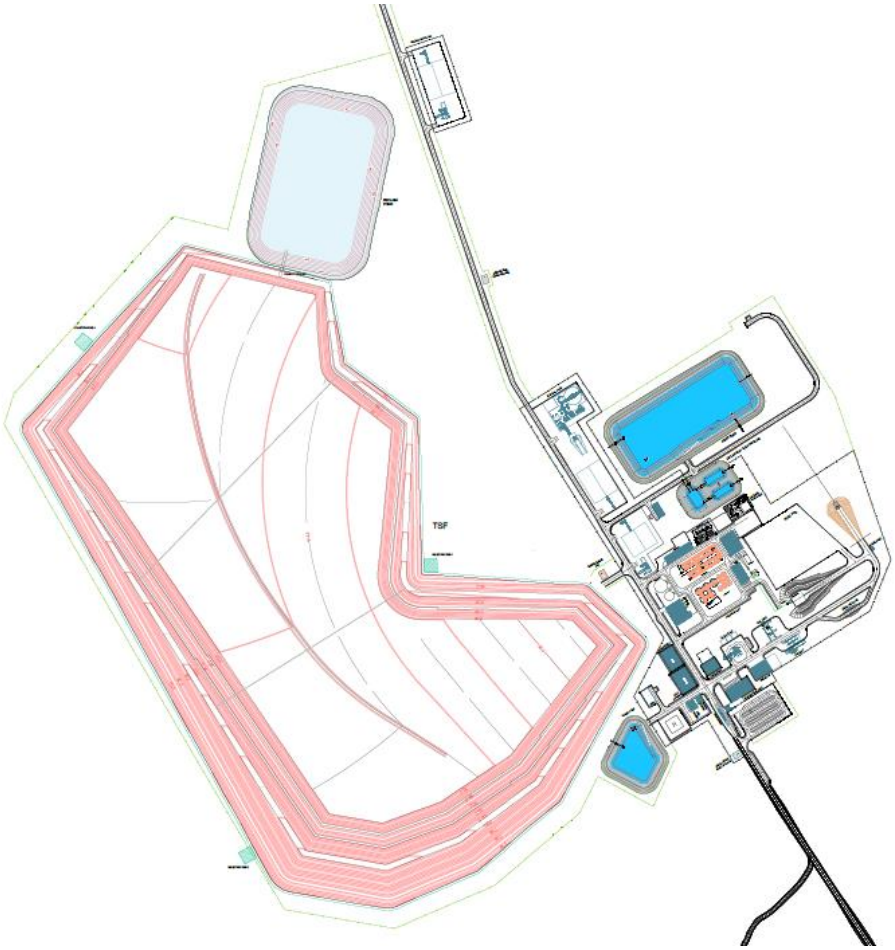


Figure 18.1 – Project Masterplan

As previously mentioned, another important consideration in the master plan was optimizing the flow of people, equipment, and emergencies. The figure below illustrates the flows that were considered for this study.

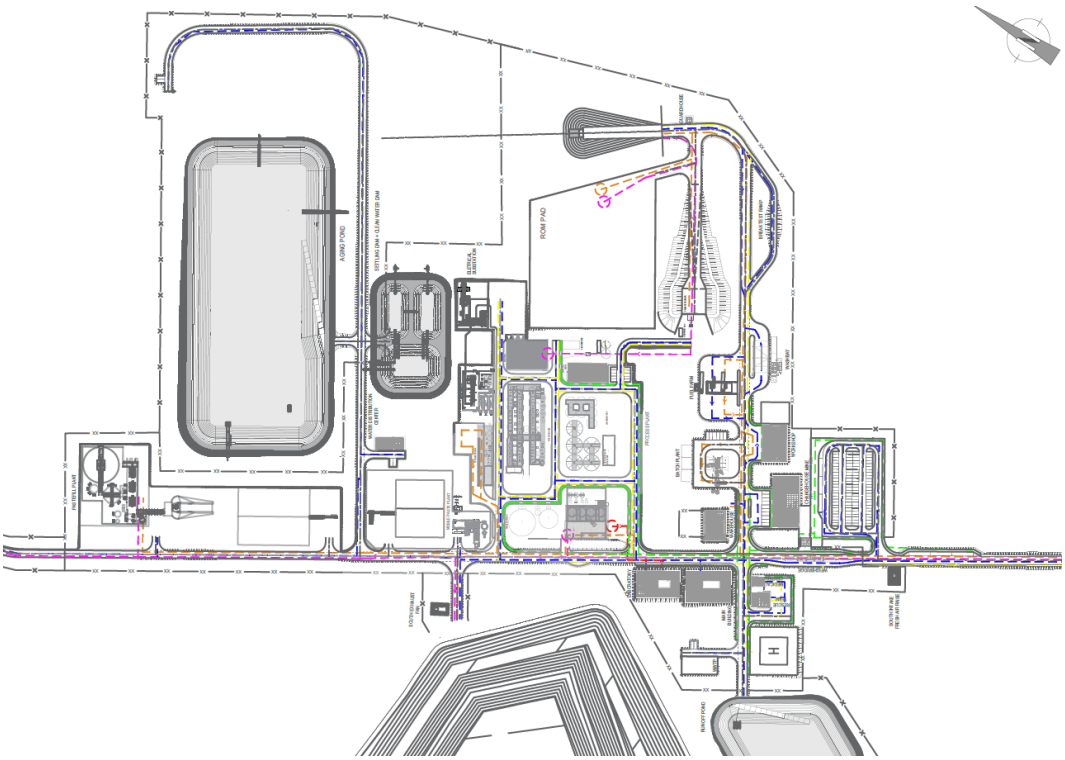


Figure 18.2 - Flow Diagrams

18.2 EXISTING INFRASTRUCTURE

The Project area is a zone where there is no infrastructure whatsoever. The current access is through gravel roads from the nearest paved road. There is no water or power connection on-site. For mining exploration works, water from wells and electrical power from diesel generators were used. In terms of buildings, there are none on-site, with the current core shed and support offices located in the industrial area of Grândola (the nearest village to the project site).

The project envisages connecting the project to the national road network, electricity, and water supply, as well as the construction of all supporting infrastructure, which will be described in the following chapters.

18.3 PROJECT SITE PREPARATION

To carry out a plan for preparing the project implementation site, it was necessary to start by defining the project implementation area within the mining lease area. The starting point was the Preliminary Economic Assessment (PEA), where a high-level master plan was developed. Following this point, environmental constraints in the area were identified, to accommodate the infrastructure in a way that minimally affects the sensible areas.

The comprehensive site preparation works required for construction are detailed in various sub-chapters of this section. These works encompass a range of activities necessary to establish a suitable foundation and infrastructure for the upcoming construction phase at the mine site. The sub-chapters provide specific guidelines and instructions for tasks such as land clearing, earthmoving, grading, excavation, drainage system installation, and other essential preparations. These meticulous preparations are essential to ensure a well-prepared site that meets the requirements for safe and efficient construction operations.

18.4 PROJECT ACCESS

The Lagoa Salgada Project access contemplates the Project access and internal roads. The Road Infrastructure is divided into the following main components:

- Geometric road definition.
- Earthworks.
- Definition of typical cross sections.
- Drainage.

The conditions, options taken, solutions adopted, and the methodology followed in preparing this Project are described below.

18.4.1 ROADS

The study that is currently being presented intends to systematize and justify the design adopted for the road network for access to the Project, as well as the circulation network within the perimeter of the Lagoa Salgada Mine.

In this phase, the feasibility of the execution of the platforms, the accesses to the surrounding roads, and the network of internal roads within the mine complex were studied, to ensure good articulation between the network of internal roads and the external roads.

This document describes the conditions, options taken, solutions adopted, and methodology followed in the preparation of this project, for the description of adopted structures, traffic signs, and road markings, please refer to document T2021-620-01-DFS-VIA-RPT-00-EN_01, on the support documentation.

The road network was designed according to the topographical characteristics of the terrain and the existing road network, having determined three accesses to the mine, one access preferably for light vehicles (Secondary Road access) and two alternative accesses for heavy vehicles (Mine Access Road Sol. A & Sol. B).

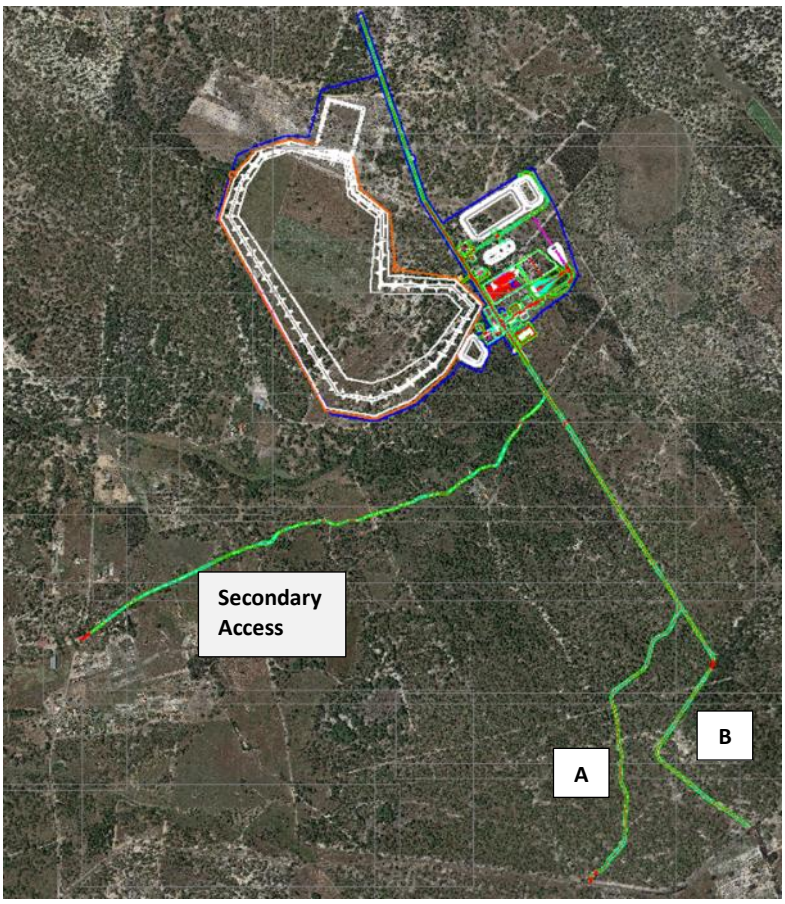


Figure 18.3 – General Layout

18.4.2 LAYOUT OF THE ROADS

For the geometric definition of the accesses, the primary objective was to achieve an efficient overall functionality, taking into account both the layout and rainwater runoff. The considerations included the limitations inherent to circulation functions, such as reduced speed, and the orography of the terrain was also taken into consideration.The roads to be built are characterised by low speeds and are intended for the circulation of heavy vehicles. The layout of the external access roads, whenever possible, is conditioned, both in plan and in longitudinal profile, to the existing roads.

18.4.2.1 GEOMETRIC DESIGN

EXTERNAL ROADS

External Roads (Secondary Access - Light Vehicles, Mine Access Road Sol. A & Sol. B)

The external roads projected are made up of straight alignment coincided with circular curves with radii varying between 18 m and 5000 m.

The longitudinal profiles are constituted by grades between 0.3% and 3.5%.

INTERNAL ROADS (VI 01 TO VI 21)

The projected internal roads consist of straight alignments combined with circular curves, featuring radii that vary between 8 m and 60 m. The longitudinal profiles include grades ranging from 0.3% to 19.5%.The plan and longitudinal profile are shown on the annexed drawings, making it possible to observe their geometrical characteristics.

18.4.2.2 TYPICAL CROSS-SECTIONS

The study encompassed 9 typical cross-sections to define the characteristics of the proposed network. This section presents the general cross-sections.External roads

The standard cross-section of the Secondary Access for light vehicles is characterised by:

- 5.0 m wide carriageway (two lanes of 2.50 m each).
- On the left, shoulder with 0.50 m wide.
- On the right, shoulder with 0.50 m wide.

The typical cross-section profile of the Mine Access Sol. A & Sol. B is characterised by:

- 7.0 m wide carriageway (two lanes with 3.50 m each).
- On the left, shoulder with 1.00 m wide.
- On the right, shoulder with 1.00 m wide.

The shoulders have the same transverse slope and pavement structure as the carriageway. Generally, the carriageways have a transverse slope of 2.5%, while car parks and sidewalks have a transverse slope of 2.0% along the axis.Internal roads

There are several types of internal roads, with different cross-sectional profiles, that have the following characteristics:

- Transverse slopes of the carriageways differ from 2.0% to 2.5%.

- 5.0, 6.0, 8.0 and 10.0 m wide carriageway
- Sidewalks with a width of 2.0 m.
- Car park with a width of 7.0 m.

18.4.3 EARTHWORKS

Prior to the execution of the earthworks there will be topsoil and clay removal, to a minimum thickness of 0.20m.

A slope of 1:1.5 (V:H) is proposed in this study for the excavation slopes to be executed. The slopes adopted in this study should be confirmed when the prospecting works are carried out and analysed.

In relation to the embankment slopes, slopes of 1:1.5 (V:H) were adopted, considering the application of materials that enable such slopes.

The following earth movements will be necessary for the implementation of the planned roads:

Table 18.1 – Earthworks Balance for Project Roads

Infrastructure	Top Soil (m^3)	Cut (m^3)	Fill (m^3)	Earthwork Balance (m^3)
External Roads Acess	16 701	9 282	24 472	15 190
Internal Roads Acess	24 040	2 634	83 529	80 895
Total	40 741	11 916	108 001	96 085

18.4.3.1 CAPPING LAYER

The 0.30m thick compacted subgrade layer will be made of S4 type materials to ensure a design CBR of 10% and a pavement foundation modulus of deformability with 100 MPa.

18.4.4 PAVEMENT

18.4.4.1 METHODOLOGY

The method that was used to verify the structure of the new pavements is the analytical method, which is based on the structural analysis of the pavement layers, determining the states of stress and strain induced in the pavement structure and its foundation when it is subjected to the axles of heavy vehicles.

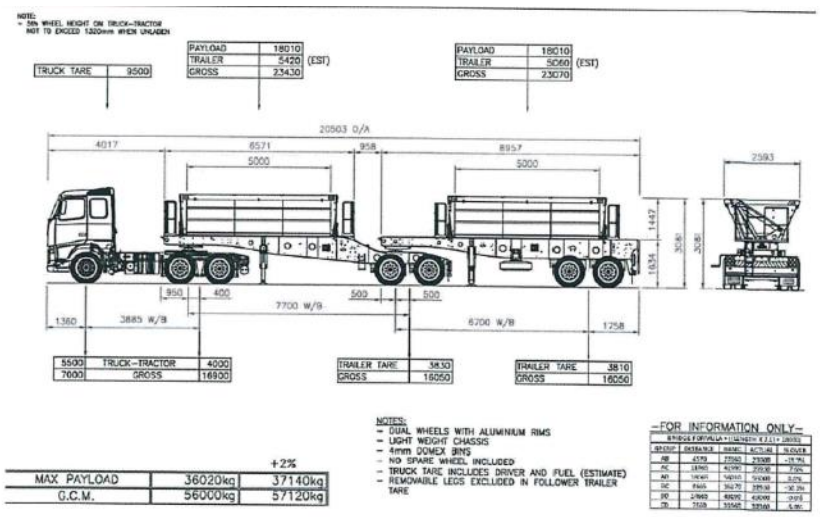


Figure 18.4 - Heavy truck considered in design with number of passes (8/per day)

The methodology followed consists of arbitrating a model structure and optimizing it until the values of stresses or strains induced by the application of a unit load are lower than the permissible limit values obtained from the failure criteria of bituminous mixtures and the pavement bed.

To be able to appreciate a given pavement solution, it is necessary to establish design criteria that reflect the limit states of the failure of the pavement. The limit states of failure of the structure of a floor are considered the same as when the previous checks were made:

- The traction rupture of the bituminous layers resulting from heavy vehicles passing over it repeatedly (fatigue effect), which results in excessive cracking of the pavement surface.

- Excessive deformation (ditches of wheels) of the wear layer surface associated with excessive deformations transmitted to the respective bed.

For design purposes, these failure limit states are considered using the following criteria:

- Limit the maximum traction extent of the lower zone of the bituminous layers (ε_t), preventing the fatigue rupture, in traction, of those layers during the dimensioning period.

- Limit the vertical extent of compression at the top of the pavement foundation (ε_z), which aims to reduce the contribution of permanent deformations of the foundation to the formation of tread grooves.

To calculate the stresses, strains, and displacements induced in the pavement by the vehicle wheels, it is necessary to have a model that simulates the structural behavior of the pavements.

The adopted behavior model consists of assimilating the floor to a set of overlapping layers based on the respective foundation, in which each layer has a given constant thickness and is considered infinite on the horizontal plane.

18.4.5 STORMWATER DRAINAGE

The objective of this study is to define and design an effective drainage system that will protect the site from the damaging effects of water and provide safe circulation on the access roads to the mine as well as on the internal roads.

An important objective of the internal roads drainage is to direct runoff water, contaminated with polluted materials from mining activities, collected by both platform and footwall ditches, to the Runoff Pond for subsequent treatment and pumping to the Aging Pond.

Runoff water not subject to contamination with the mining materials will drain naturally to water lines and land adjacent to the mine area.

The design of the stormwater network will ensure that there is no mixing of contaminated and uncontaminated water due to environmental constraints.

All the hydrological calculations that were executed in this study, cross drainage dimensioning, and longitudinal drainage dimensioning can be found in the support information.

18.4.5.1 LONGITUDINAL DRAINAGE

The drainage of the roads is ensured by a set of longitudinal, surface, and internal works, which will collect the water that falls in this area.

So, considering the characteristics and the constitution of the cross sections, the following works were defined:

- Platform and border ditches, placed in the roads to collect and conduct the runoff water from the platform.
- Trapezoidal slope foot ditches, intended to collect water from the slopes and the land adjacent to it.
- Slope descents.
- Manholes with cast iron cover.
- Manholes with cast iron grilled cover.
- Concrete collectors, with the diameters shown in the plan.

18.4.5.2 CROSS DRAINAGE

The cross drainage of the road will be ensured by the implementation of hydraulic crossings in low points of the ground.

The cross-drainage structures are fundamentally made up of the following elements:

- Concrete pipes or reinforced, with inner diameters of 0.60 m to 1.50 m.
- Inlet and outlet manholes for circular concrete culverts.
- Trapezoidal ditches for diverting water lines, lined with riprap.
- Energy sinks at de base of embankment slopes

18.5 MINE ACCESS

The access to the underground mine (mine portal) has been defined with a solution of cut & cover. A solution of standard excavation was accessed but due to costs, hydrogeological, social and environmental reasons the cut & cover solution was selected. The present section refers to the cut & cover and slope stabilization solution for the Lagoa Salgada Mine box cut.

A Geotechnical Prospecting Plan, specifically designed for the Box-Cut deployment zone, was utilized to guide the development and definition of excavation geometry, as well as the implementation of containment and stabilization measures associated with the excavation.In addition, hydrogeological analysis and evaluation of water levels were carried out in their own study. Thus, the solutions now presented were based on the following documents:

4) Bulletins of The Prospecting and Laboratory Tests. Prepared by Triáguas and Geoalgar.

5) Basic Hydrogeological Study - Preliminary Report.

The geomechanical zoning and parametrization calculations, as well as the general pre-dimensioning criteria of the access gallery can be found in the support information, document T2021-620-01-DFS-GEO-RPT-00-EN_00 and T2021-620-01-DFS-EST-RPT-00-EN_00.

18.5.1 RECOMMENDED SOLUTIONS

Given the hydrogeological conditionality and the geological and geotechnical scenario, it is recommended to execute a waterproofing curtain along the perimeter of the Box Cut excavation, to minimize the impacts mentioned above.

The excavation for the mining tunnel, should start below the cutaneous aquifer, that is, at 17.00 m deep and in areas of the Miocene, where the geomechanical behavior is more favorable compared to the layers above.

Thus, it is recommended that the start of the underground excavation occurs, approximately 24 m deep, and for this it will be necessary to carry out an open excavation under a containment.

In order to optimize the cost and execution time of this containment, the decision was made to implement a provisional approach using the "cut and cover" method.The sections below describe the provisional containment solution as well as its waterproofing barrier.

18.5.1.1 PROVISIONAL CONTAINMENT

For the stabilization of two slopes, in the initial phase, a soil nailing wall type solution is recommended, characterized by the execution of nails and a sprayed concrete coating.

The highest excavation height occurs on the front slope, where the mining work reaches the portals, with a maximum height of approximately 24 m, materialized by 3 slope panels 8 m height, interspersed by a 3 m long bench.

The defined excavation geometry uses a slope geometry of 1:2 (V:H) in the superior excavation slope, without using containment. The lower level of the excavation geometry is vertical, adopting a slope of 2:1 and 3:1 (V:H) combined with the execution of four levels of soil nails, in A500NR ∅25mm steel rebars, arranged in a quincuncial pattern, spaced 2.0 m apart and 12 m long.

About the slope surface, it will be coated by applying three layers of shotcrete concrete (5+5+5 cm thick) combined with AQ50 wire mesh.

The soil nails should be drilled with a NX diameter drill hole (∅76 mm) with a hole length larger than approx. 20 cm in theoretical value.

The soil nails should be drilled, as a rule, with a slope of 15° to horizontal.

Geodrains, 6.0 m long and 80 mm in diameter, should also be made of rigid, ribbed P.V.C., coated with a geotextile of 150 g/m². These drains will ensure the internal drainage of the soil rock mass, preventing the eventual generation of hydrostatic impulses caused by the infiltration of rainwater. To ensure the gravitational flow of the collected water, these elements should have an upward slope of 5 to horizontal.

18.5.1.2 WATERPROOFING CURTAIN

To ensure that the excavation area is waterproof, it was necessary to define a solution that would guarantee that it would not interfere with the aquifers, both in terms of their piezometry and the possibility of them being contaminated.

The project therefore recommends a solution using injection technology (jet-grouting) of clay-based material which, due to its low permeability, allows the formation of a waterproofing curtain.

This way, it will be possible to carry out dry excavations, as well as to dispense with the waterproofing solution along the “cut and cover” section of the access gallery.

The materialization of the waterproofing curtain will be carried out through two alignments of injection holes, with a plan distribution # 1.20 x 1.20 m and a depth of approx. 17 m.

18.5.1.3 ACCESS GALLERY

The purpose of adopting a closed gallery in the initial section of the mine access ramp is to ensure that the mine operation is supported by a definitive lower volume access trench, with the associated advantages (Figure 18.5, Figure 18.6, Figure 18.7).

The gallery that will precede the mining tunnel will be held in situ, inside a temporary excavation.

The geometry of the gallery will be in a closed frame, to be built in reinforced concrete, with a free jig of 6.5 m, and an internal width of 6.0 m.

The thicknesses of reinforced concrete will vary by stretches along the 137 m length, in order to optimise the use of materials without excessively complicating the construction. The thickness of the walls and slabs will vary between 0.50 m and 1.0 m.

The height of soil over the gallery structure varies between 3 m and 17 m.

On the outside of the walls, a draining screen is installed to conduct any water which have not been contained by the surrounding cofferdam. The existence of this draining screen allows to disregard the presence of a groundwater level when detailing the frame. The flow rates captured by these screens should not be significant, and will be conducted into the first pumping well inside the mine.

The circulation of the vehicles will take place on a ribbed/groove concrete floor positioned over the lower slab of the gallery.

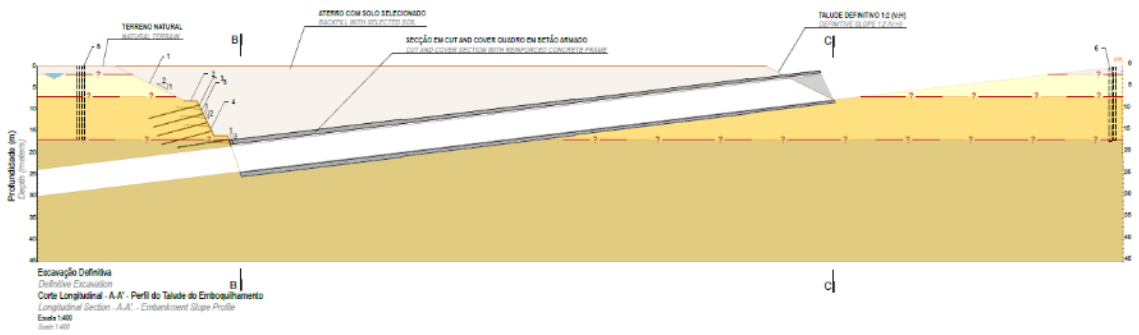


Figure 18.5 – Access Gallery – Longitudinal cut

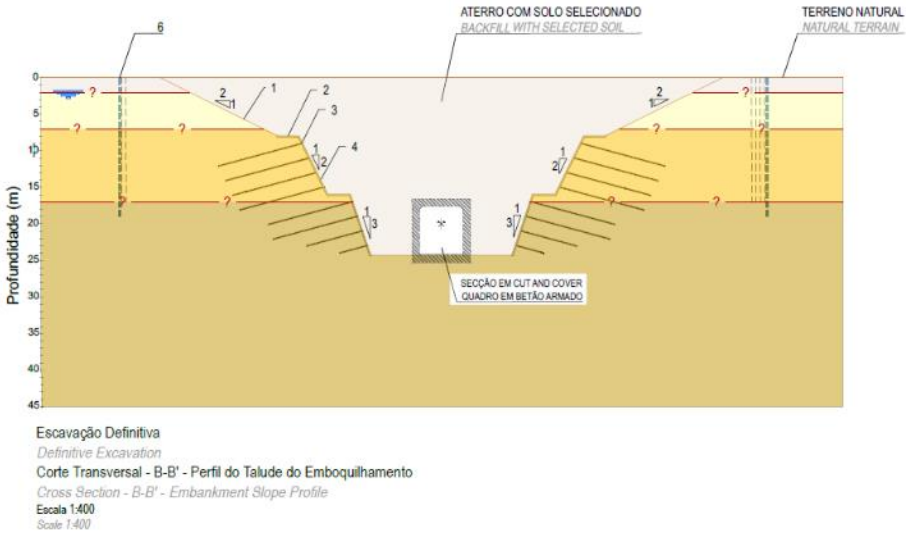


Figure 18.6 – Access Gallery – Cross section B

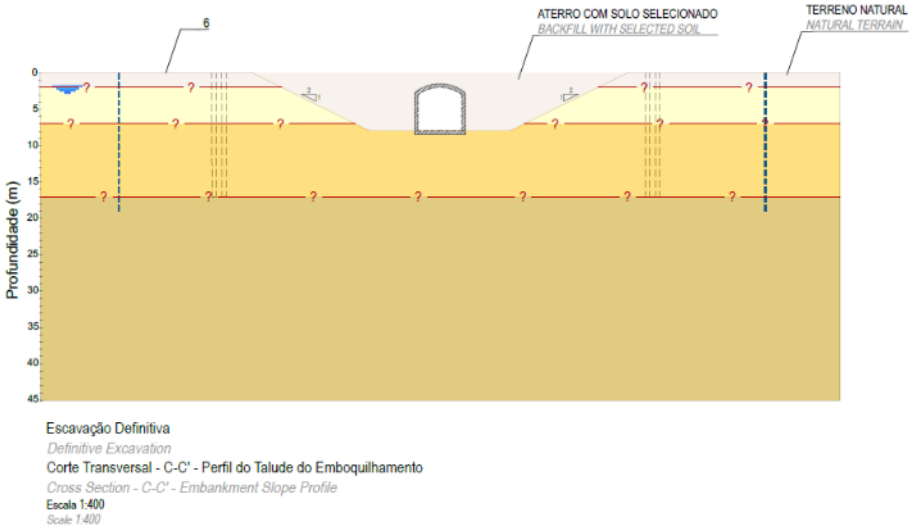


Figure 18.7 – Access Gallery – Cross section C

18.5.2 SLOPE STABILITY ANALYSIS

In terms of overall stability, it was used limit equilibrium methods, such as Bishop's method. It is a method that determines the safety factor of breaking a slice along a sliding surface, by dividing the sliding mass into a variable number of slices, depending on its geometry and the different types of materials that constitute it, and admitting some simplifications regarding the interaction force between the slices.

The overall stability was evaluated through the possibility of forming a rupture mechanism simulated by a potential circular slip surface.

The program also allows a pseudo-static analysis to check the safety conditions in case of an earthquake. Pseudo-static methods constitute simplified analyses that simulate the effect of seismic phenomena by increasing inertial forces, determined by increasing the weight of each slice by the seismic coefficients considered.

The analyses were carried out according to Eurocode 7 (NP EN 1997-1:2010) using the partial safety coefficients method.

All the support calculations can be found in the support information.

18.5.2.1 CROSS SECTIONS

The overall stability analysis covered the frontal scale, both for the provisional excavation, as well as for the definitive excavation (Figure 18.8).

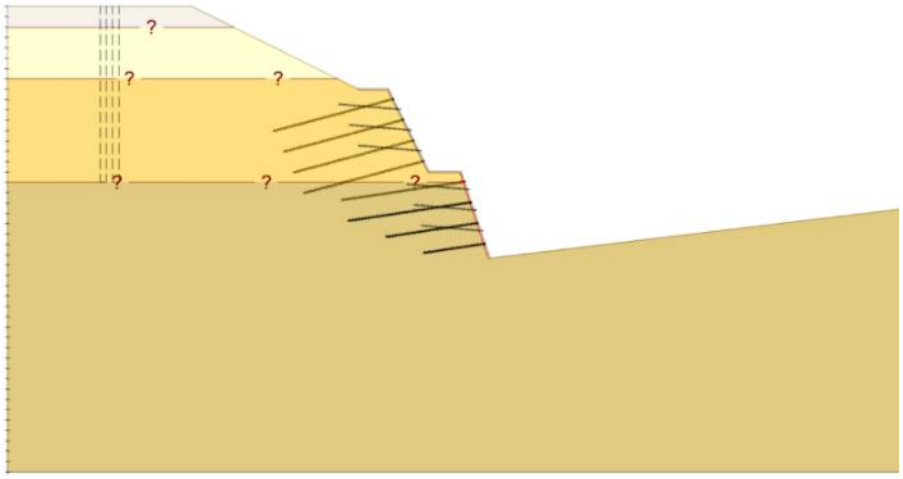


Figure 18.8 - Global stability analysis. Calculation sections considered - Temporary excavation

18.6 BUILDINGS AND ANCILLARY ITEMS

The buildings and ancillary items considered in the Project were: the gate house, main building, change house, medical centre, laboratory, rescue team, workshop, warehouse, guardhouse and control room.

All the regulations and rules (applicable standards) applied when elaborating the project for the following section can be found in the support information.

A total of 10 buildings will be constructed, all of them one-storey high, with floor areas ranging from 14m^2 to 942m^2. The buildings will be used by various mine support services and have the following designations and functions:

Table 18.2 – Building numeration

• Building	Type of Construction
01 – Gate House	Prefabricated
02 – Main Building (Administration & Offices)	Prefabricated
03 – Change House	Prefabricated
04 – Medical Centre	Prefabricated
05 – Laboratory	Prefabricated
06 – Rescue Team	Prefabricated
07 – Workshop	*In-situ* construction
08 – Warehouse	*In-situ* construction
09 – Guardhouse	Prefabricated
10 – Control Room	Prefabricated

In addition to the presented information, the structural study of the buildings (T2021-620-01-DFS-EST-RPT-00-EN_00) and hydraulic networks (T2021-620-01-DFS-HIP-RPT-00-EN_00), can also be found on the support information. Also, the HVAC, electrical, telecommunications and active security studies were also conducted, for further information please refer to reports T2021-620-01-DFS-ACV-RPT-00-EN_00 and T2021-620-01-DFS-ELE-RPT-00-EN_00.

18.6.1 ARCHITECTURE

The designed architecture intends to create a set of buildings of industrial architectural language where technical spaces and functional paths are designed to be conceptually and functionally adopted.

18.6.1.1 CONCEPT DESIGN

The proposal aims to meet a practical and functional architectural language that responds to the program in a pragmatic and efficient way, both in terms of spatial design and in the choice of materials to be adopted, in accordance with the language of the industrial environment in which it operates.

The buildings as a whole share the same architectural image to create unity, distinguished by the colours and numbering of each building, making it easier to identify/distinguish them.

The Architectural design aims to comply simultaneously with the following basic principles:

- Cost and energy-efficient design.
- Adoption of best architectural practices and functional design.
- Design low-risk buildings with simple construction methods and proven techniques.
- Generally, life cycle cost, durability, and economics are the main factors in material selection.
- All chosen materials, where possible, are available locally and sustainability issues have been taken into consideration.
- *In-situ* buildings to be designed for a minimum life of mine of 20 years.
- Design principles endeavour to minimize the risk of pollution.
- Design principles are in line with the regulations.
- Minimize the construction periods.

In the next chapters, only the in situ buildings are illustrated/depicted with images; for images of the prefabricated buildings, please refer to the support information.

18.6.1.2 FUNCTIONAL SPECIFICATION

GATEHOUSE (01)

The building is designed to be accessed by security personnel and to regulate the entrance and exit flows of mine workers and visitors. This building will have 3 security staff permanently and will have one reception for visitors, a weighbridge control room, an office, a toilet, and kitchenette.

MAIN BUILDING (02)

The Main Building has two separate programmes, Administration & Offices and Control & Automation.

The Administration & Offices program will function as a main office centre for the mine managers and the estimated number of building users is 72 permanent workers. It will have three distinct work areas, the manager's area, the mine workers' area and the G&A workers' area, in an open space. It will have a reception with a waiting room, several office spaces, meeting rooms, briefing rooms, a kitchenette, and toilets.

The Control & Automation zone is designed to be accessed by restricted personnel and to regulate the entrance and exit flows. This will have staff permanently, an emergency room, a control room, a kitchenette, and a toilet.

CHANGE HOUSE (03)

The building is a change house of the mine, with two change houses for miners with a capacity for 40 miners/shift, 172 lockers, two change houses for technicians and visitors with a capacity for 31 users, and two change houses for supervisors with a capacity for 13 users. To fulfill the functional purpose, it complies with reception, laundry, men's and women's change rooms, storage, a briefing area, and a lamp room.

MEDICAL CENTRE (04)

The main function of this building is to serve as the mine's primary medical center, providing occasional medical services. For severe traumas or special medical needs, patients will be redirected to nearby hospitals. The facility is estimated to accommodate four permanent workers and will include a reception area, waiting room, medical room, infirmary, storage, toilets, and an emergency entrance.Laboratory (05)

The building will house a laboratory equipped to analyze products derived from mining, a processing plant, and water facilities, with a capacity for accommodating six people. To fulfill its functional purpose, the building includes dedicated spaces for men's and women's toilets and change rooms, laboratories, sample storage areas, handling rooms, offices, a kitchenette, and additional toilets.

RESCUE TEAM (06)

The building will accommodate a rescue operation center for firefighting equipment, related vehicles, and personal protective equipment to help in any crisis/emergency. The estimated number of building users is 3 permanent workers. It will have office spaces, toilets, and a storage room.

WORKSHOP (07)

The building (Figure 18.9) will work as a workshop for fixed equipment and/or mobile vehicles and the estimated number of building users is 4 permanent workers. It will have four large spaces for maintenance as service bays, equipped with an overhead crane, office spaces, storage rooms, a kitchenette, and a toilet.

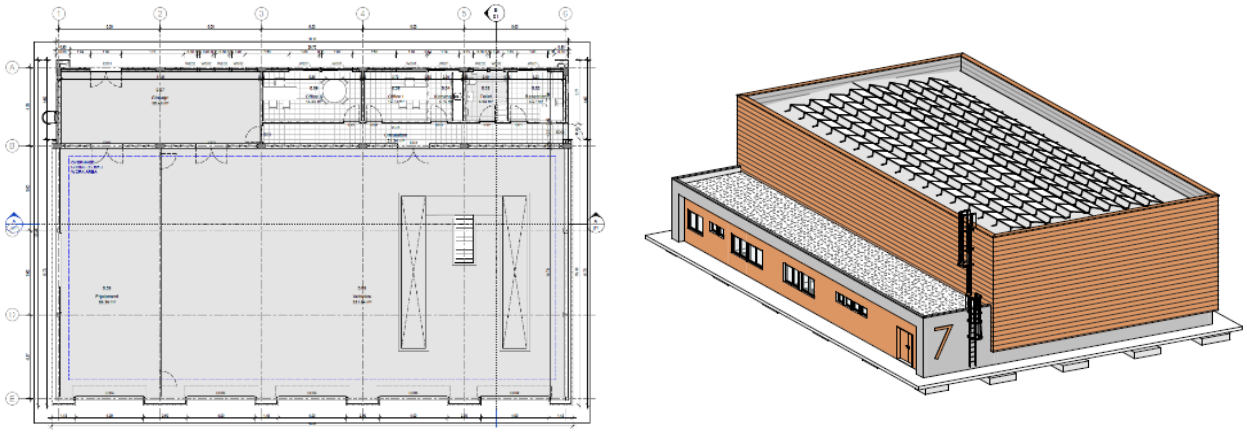


Figure 18.9 – Warehouse – Ground Floor Plan (left); 3D View (right)

WAREHOUSE (08)

The building (Figure 18.10) will serve as a warehouse for equipment and consumables. The estimated number of building users is 2 permanent workers. It will have a large interior and exterior space for storage, office space, other storage areas, an entrance and a reception for receiving products and materials, a kitchenette, and a toilet.

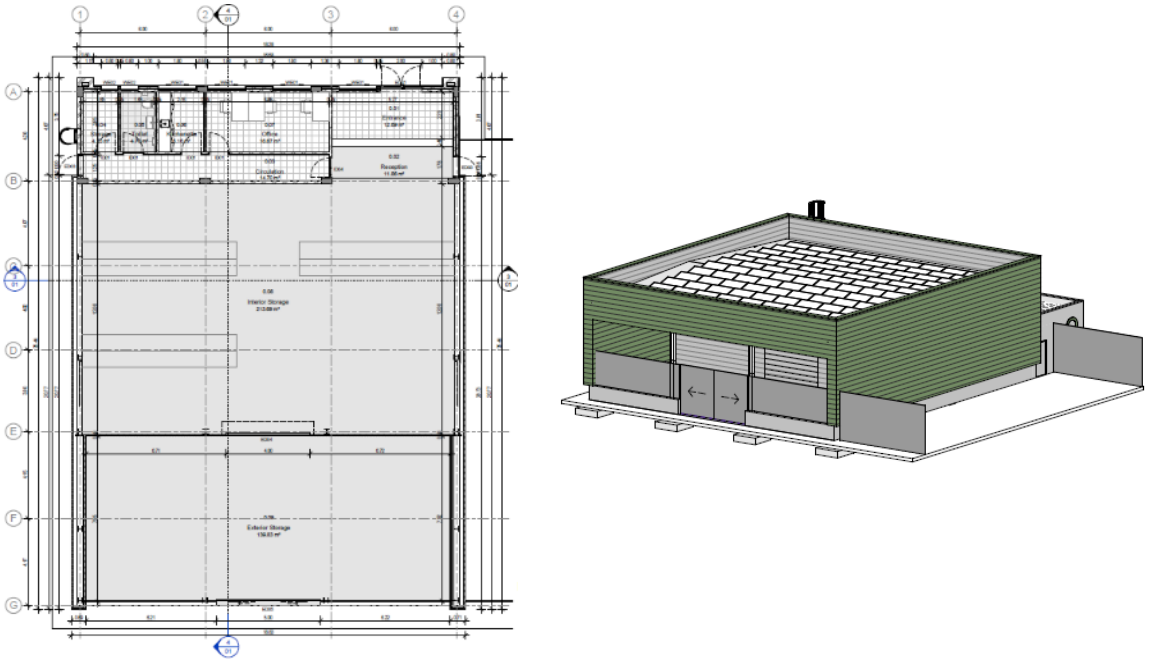


Figure 18.10 – Workshop – Ground Floor Plan (left); 3D View (right)

GUARDHOUSE (09)

The building is designed to be accessed by security personnel and to regulate the entrance and exit of the mine. This building will have 1 permanent security staff member and it will have a control room and a toilet.

CONTROL ROOM (10)

This building will serve as the control center for the Process Plant, responsible for managing the operation of the ore treatment process. Additionally, it will house the changing rooms dedicated to the workers in this area.

On one side, the building will feature a designated cleaning pathway for the workers, equipped with a shoe-washing area and separate changing rooms for both men and women. On the other side, there will be an administrative area intended for the permanent staff. This area will include a reception area, meeting rooms, offices, restroom facilities, and a kitchenette. The building design also incorporates the necessary technical areas.

18.7 PROCESS-BUILDINGS

The Project encompasses the mineralurgical processing of five distinct domains (MS Fresh, MS Transition, Gossan, Stockwork, and Stringer) individually. Each ore type requires a unique sequence for processing, and the design of the Process Plant complex ensures that the necessary modifications can be made efficiently (Figure 7.1).

Figure 7.1 - Process Plant 3D

The Process area comprises multiple sections, including crushing, milling, flotation, leaching, and filtration. In addition to these primary sections, the facility will feature an administrative area, changing rooms, a reagent storage area, thickeners, electrical rooms, a compressor room, and a water treatment area utilizing Fenton reaction for theosalt treatment.

Consequently, the following areas have been defined (Figure 7.2):

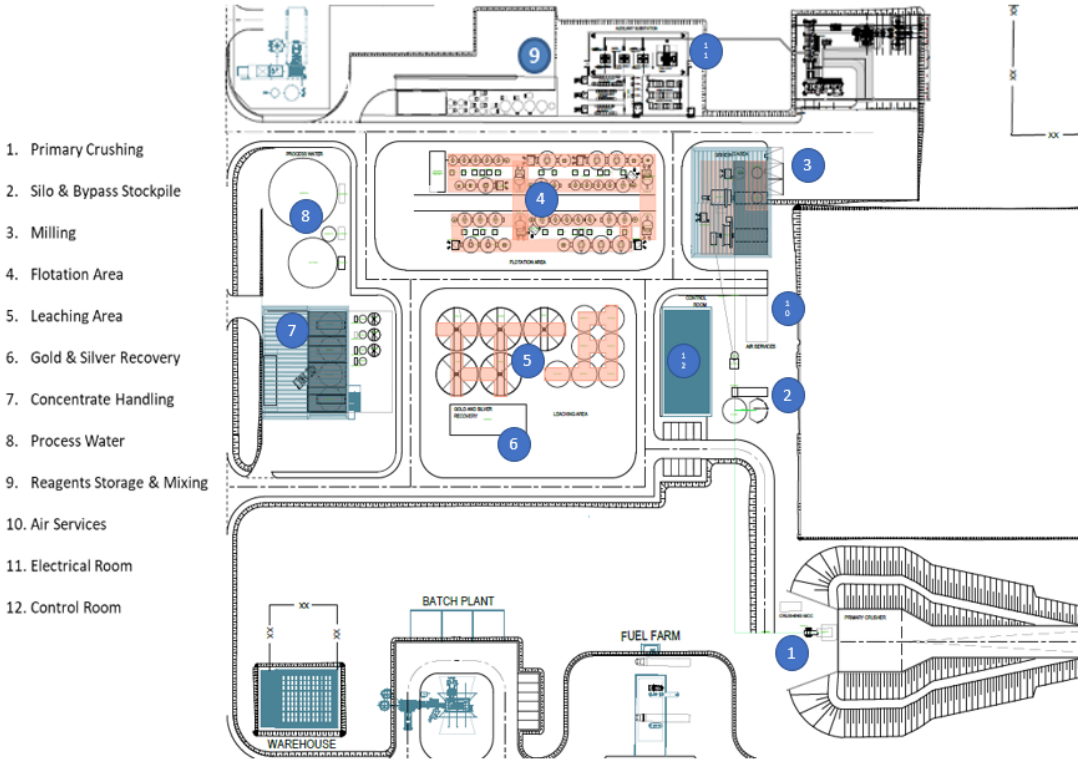


Figure 18.2 - Process Plant Layout

In addition to the presented information, the study for structures, hydraulic networks, HVAC, electrical, telecommunications and active security was also conducted, and can be found in the support information. The conveyor belts study and piping can also be found in the following reports, DFS-GEN-CON-01 - DESIGN REPORT CONVEYOR BELTS and T2021-620-01-DFS-PIP-RPT-00-EN_00, respectively.

18.8 ELECTROMECHANICAL PROJECT

The main report referring to the electromechanical project and electrical & electromechanical design, with the design criteria and all the technical information, can be found in the support information, report DFS-HVE-301-01-EN_01, DFS-HVE-LCC-00-EN_01 and DFS-HVE-MDJ-00-EN_01.

18.8.1 GRID CONNECTION

The object of this chapter aims to characterize the 60 kV overhead line, interconnecting the Public Service Electrical Network (PSEN) to the Lagoa Salgada substation. This interconnection will be made at the Arregota Nova substation of E-REDES, which is under construction.

There are plans to install 52 supports along the alignment of the route in Grândola, Setúbal, Portugal, as shown in Figure 18.11. The High Voltage Overhead Line (OHL) of Lagoa Salgada has a length of approximately 15.1 km.

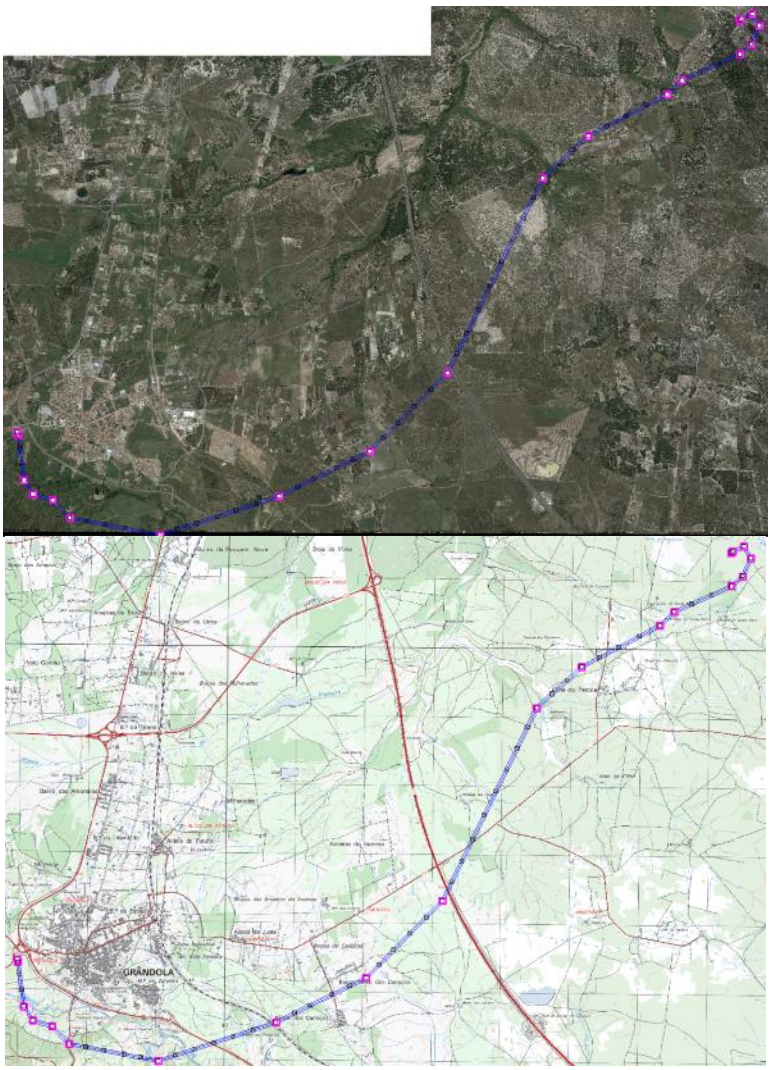


Figure 18.11 – OHL layout on orthophoto and military map

18.8.1.1 LINE CHARACTERISTICS

The line to be constructed will be a single line, over its entire length, consisting of an Aluminum-Steel 325 mm² conductor and an OPGW AS/AA/St 34/170 guard cable, with a three-phase alternating current, a frequency of 50 Hz and a voltage of 60 kV.

It consists of 52 F-type metallic supports, characterizing 51 topographic spans, with the objective of interconnecting the above-mentioned installations.

For further information regarding the mechanical design & calculations, general technical criteria, and equipment, please refer to the T2021-0620-004-LN60KV MD report, in the support information.

18.8.1.2 IDENTIFICATION OF AREAS INTEGRATED INTO THE NATIONAL AGRICULTURAL RESERVE (NAR)

This layout presents 5 supports in an area that is considered the National Agricultural Reserve.

Figure 18.12 – Identification of NAR areas

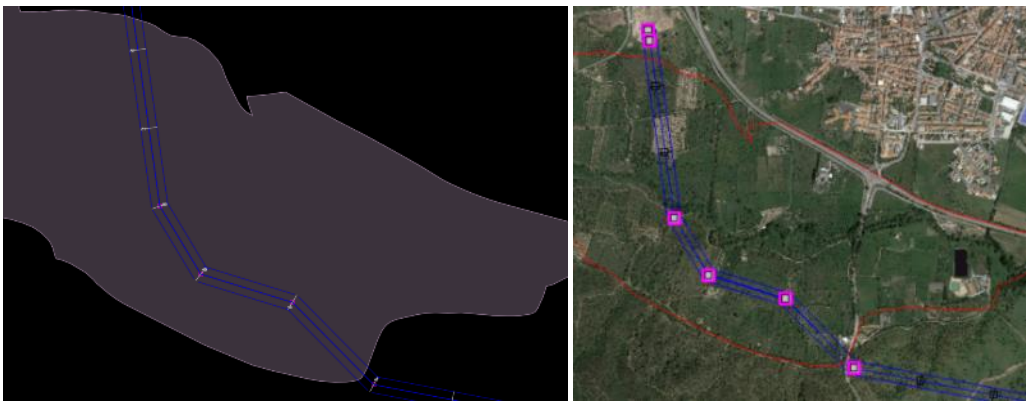


Figure 18.13 – Support implemented in RAN areas

Table 18.3 – Location of the supports in the NAR area

Support Nr.	Coordinates (Datum 73)	
	X[m]	Y[m]
47	-38452.57	-166548.82
48	-38732.78	-166462.62
49	-38858.17	-166255.28
50	-38891.67	-166017.87

Support Nr.	Coordinates (Datum 73)	
	X[m]	Y[m]
51	-38925.43	-165778.73

18.8.1.3 INTERSECTIONS AND CROSSINGS

WATER LINE

There are approximately 33 crossings of water lines/courses, with more than one crossing per span. These are illustrated in Figure 18.14. In all these crossings, the supports are more than 10m away from the axis of the water lines.

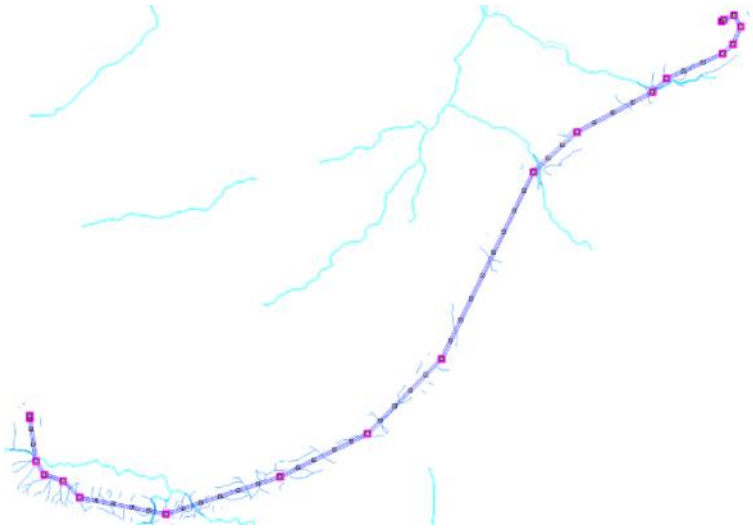


Figure 18.14 – Crossings of water lines/courses

ELECTRICAL LINES

Regarding electric line crossings, there are 9, 8 being MV crossings and 1 HV.

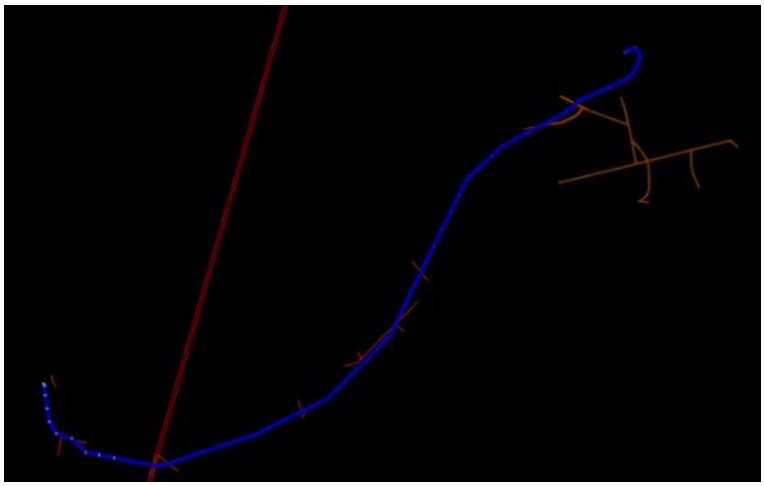


Figure 18.15 – Power line crossings

ROAD NETWORK

There is an overhead crossing of the IP1 (A2) highway. This crossing will be carried out according to the regulations and standards in force (safety distances, signalling, sizing of supports, etc.).

Figure 18.16 – IP1 Crossing

RAIL NETWORK

There will be an overhead railroad crossing in the village of Bairro dos Cadoços, in Grândola. This crossing will be carried out according to the regulations and standards in force (safety distances, signalling, sizing of supports, etc.).

Figure 18.17 – Railroad Crossing

CORK OAK TREES

There are cork oak crossings, this crossing will be aerial, and, in the worst case, only crown thinning will be required to ensure safety distances.

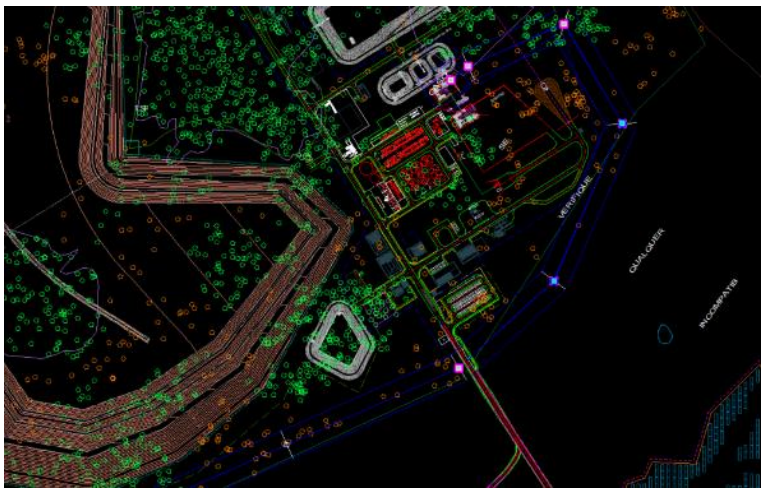


Figure 18.18 – Crossing of cork oak trees

It should be noted that this study is only valid for the cork oaks already surveyed in the field. Only after the survey of the total corridor can the situation be verified (elevations of the cork oaks + protected trees not surveyed).

18.8.2 BASIC PRINCIPLES

The general design of this project was ruled by the simultaneous satisfaction of the following basic principles:

- General security for people and goods.
- Simplification and standardization of construction.
- Ease of maintenance.
- Application of solutions to minimize the environmental impact.
- Optimization of installation conditions to rationalize the operating costs.

18.8.3 SUBSTATION GENERAL CHARACTERISTICS

18.8.3.1 FACILITIES DESCRIPTION

The electrical system of the Lagoa Salgada will have the following general characteristics presented in Table 18.4. All equipment, to be installed in this project, shall be compatible with these characteristics, not compromising the perfect operation of the whole electrical system.

Table 18.4 - General characteristics of the electrical system

Characteristics	60 Kv Level	15 Kv Level	690 V Level	400 V Level
Number of poles/phases	3	3	3	3
Nominal voltage (U_n)	60 kV	15 kV	690 V	400 V
Highest voltage (U_m)	72 kV	17,5 kV	1000 V	1000 V
Initial symmetrical short-circuit current	TBD	TBD	TBD	TBD
Peak short-circuit current	TBD	TBD	TBD	TBD
Nominal frequency	50 Hz	50 Hz	50 Hz	50 Hz
System neutral circuit	Resistance Earthed	Resistance Earthed	Solidly Earthed	Solidly Earthed

From the power supply point of view, it is considered a new connection to the Distribution System Operator (DSO), namely E-Redes, at the distribution level of 60kV.

Inside the mine, a new step-down substation will be built to reduce the grid connection voltage level (60 kV) to the internal distribution voltage levels (15 kV) and to distribute the power from the substation to the loads.

The configuration of the proposed Electrical Power Supply system can be seen on the project Single Line Diagram: DFS-HVE-301-00.

In the substation switchyard, the following has been installed:

- All the HV and MV outdoor equipment.
- One 60/15 kV Power Transformers with maximum rated power of 45 MVA (ONAN/ONAF).
- One Auxiliary Transformer.
- One Neutral Reactance.
- Emergency Generator Group.

The buildings have the following rooms:

- MV Room where the MVS are installed.
- Cabinet Room with a WC.

An electrical interlocking system between the two voltage levels is provided to allow the substation to operate under safe conditions and to prevent false maneuvers with the HV and MV equipment.

The neutral system considered at the HV level is neutral, connected directly to earth.

The neutral regime considered at the MV level is connected by creating an artificial neutral, which is ensured by a three-phase reactance limiting the phase-to-earth fault current that is connected to the MVS.

The supply to the substation auxiliary services is ensured by one 15/0,4-0,23 kV auxiliary service transformers, with a rated power of 50 kVA.

The substation location is considered, in order to define insulators, cable terminations and external overvoltage surge arresters, a place with a heavy level of pollution, according to the levels defined in the IEC 60815 standard.

All electrical equipment to be installed in the substation has dielectric characteristics that are compatible with the altitude at which they will be installed.

18.8.3.2 SUBSTATION DESIGN

The 60/15 kV Lagoa Salgada Substation consists of the installation shown in

Figure 18.19.

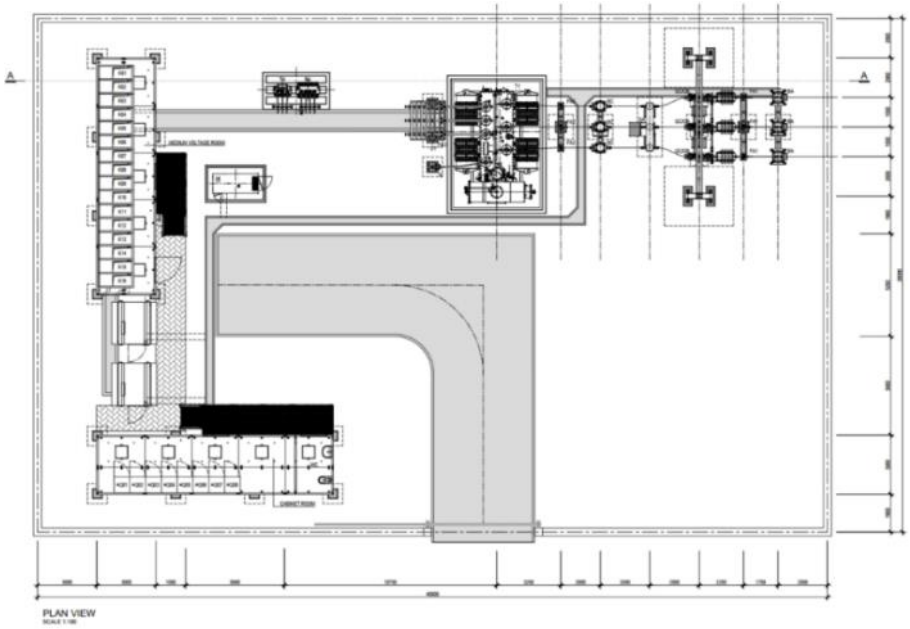


Figure 18.19 – Substation layout

18.8.4 SCADA AND AUTOMATION SYSTEM

The SCADA and Automation system of the Lagoa Salgada Project is based on digital micro processing equipment, including:

- SCADA Servers.
- HMI Servers.
- Engineering Station Servers.
- Programmable Logic Controllers.
- Remote Input / Output equipment.
- Multifunction Protection and Control IED.
- Digital Meters.
- Specific Equipment Control IED.
- Specific Field Devices.

The Field, PLC and Control Room communication equipment will be connected by means of an FO/Ethernet LAN infrastructure.

The communication protocols that will be used to exchange information between the various plant sub-systems - SCADA, Control, Protection and Field Equipment - should be based on industrial standard protocols supported by Ethernet.

18.8.4.1 LAN

To increase reliability, the LAN will be implemented using a logical ring architecture, ensuring a N+1 redundancy in communication paths.

To ensure the same level of reliability at the communication level, the LAN switches that interface directly with the backbone shall be able to implement the Rapid Spanning Tree Protocol (RSTP) for communication path redundancy, according to the IEEE 802.1D-2004 standard.

Each contractor shall design their sub-system's LAN architecture to ensure proper interface with the LAN backbone, regardless of the internal LAN architecture or the equipment used internally in their sub-system.

18.9 WATER UTILITIES

The main report referring to the water infrastructures, with other relevant information, Fire Water System, design criteria and all the technical information, can be found in the support information, report T2022-620-DFS-HID-RPT-00-EN_01.

18.9.1 WATER BALANCE AND MANAGEMENT

A deterministic water balance model has been carried out for the operating phase to:

- Verify that the process make-up water demands can be met.
- Determine the design criteria for the water management infrastructure and equipment.
- Simulate the transfer of water between the water management dams for storage, to use the water collected on site to support the mining activities (process, dust suppression, etc.) and to discharge excess water to the receiving environment.

Site-wide water balance simulations have been developed for the average annual precipitation conditions (the precipitation data for the Lagoa Salgada Project site is sourced from the Grândola climate station) as well as dry and wet annual precipitation conditions based on three different return periods: 10 years, 25 years, and 100 years.

The water balance for the whole operation has been calculated on a monthly timeline to reflect the extreme wet and dry seasons experienced on site. Figure 9.1 and Figure 9.2 presents the total water balance of the site, demands and discharges, as well as discharge volumes during the life of the mine, for average conditions, respectively.

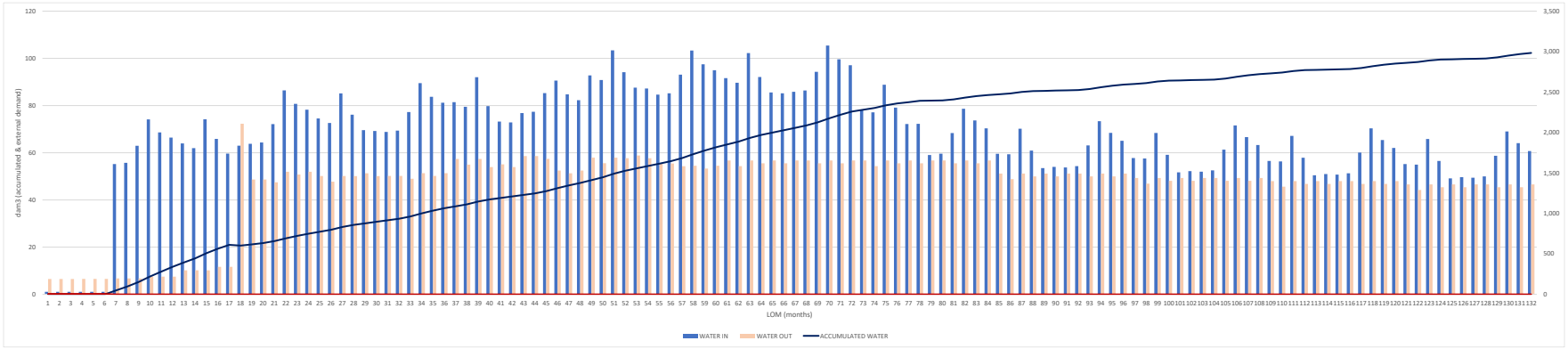


Figure 18.20 – Water Balance - Average conditions

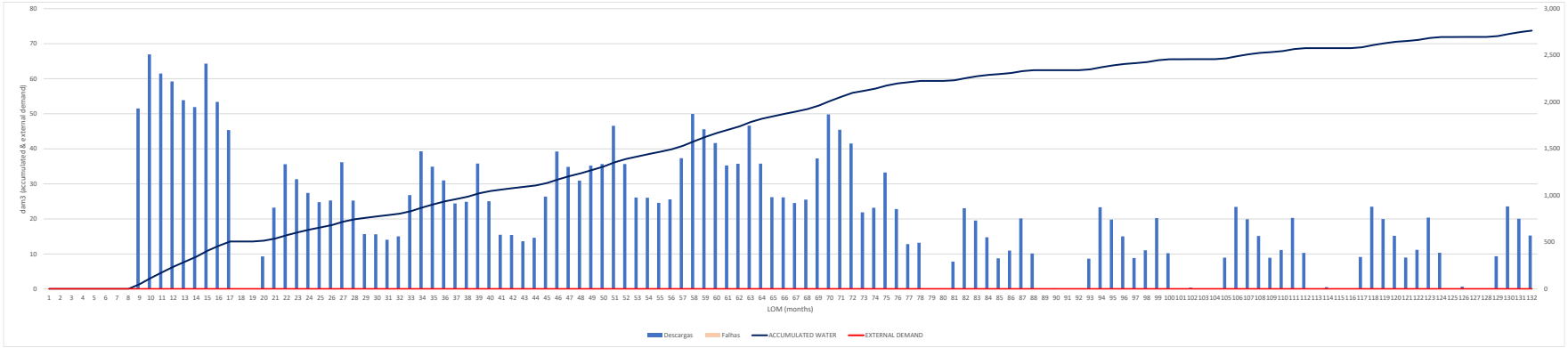


Figure 18.21 – Water Balance – Demands and discharges values

The model results demonstrate that recirculated water can (on average) meet mine and process water needs without drawing additional water from aquifers or other water sources. However, there is a need to discharge water into the environment.

18.9.1.1 Water Management

The site water management strategy is to divert or deflect non-contact surface runoff water away from the Project site to the extent possible and to collect and treat site-influenced contact water. Where applicable, best management practices for sediment and erosion control will be employed.

The majority of the water that will need to be managed at the site comes from the dewatering of underground operations, followed by surface run-off around the site, especially during major precipitation events (such as 1-in-20-year storms). Additional aspects of water management around this site will include the collection of water from ditches and the treatment of water for potable and process use.

NON-CONTACT WATER MANAGEMENT

As the Project site is relatively flat, perimeter deflection berms and diversion ditches will be constructed around it to minimise the catchment area and to facilitate drainage of non-contact water away from the site.

CONTACT WATER MANAGEMENT

Contact water refers to water that has been in contact with mining activities, mined material, and/or underground mine infiltration. The contact water will be collected, treated, tested and reused for processing and mining operations.

Collected surface contact water will be transferred to a containment dam (Aging Pond). The collection of underground contact water and mine dewatering will be managed through a series of pumps and sumps that transfer water to the containment settling and balancing dams. Four dams will be constructed on the surface to manage the contact and process water. The details of the ponds are summarised below:

- Aging Pond – A lined containment pond with a capacity of approximately 80 000 m^3 will be constructed to store excess mine dewatering water, and rainfall from the Runoff Pond. An emergency overflow spillway allows for controlled overflow if the design storm were to be surpassed.

- Settling water dam – A lined containment dam with a capacity of approximately 2 x 1000 m^3 will be constructed for the containment, settling and polishing of surface contact and underground mine dewatering. The settling pond has two sides separated by a berm with an overflow spillway. An emergency overflow spillway allows for controlled overflow if the design storm were to be surpassed.

- Clean water dam – This dam is located close to the mina entrance (ramp) and will have a capacity of approximately 1000 m^3. This dam stores water for use in the mining operations;

- Runoff Pond – A lined containment dam with capacity of 10 000 m^3 will be constructed to retain and divert rain that falls in contaminated areas. It will have a pumping station to drain water into the Aging Pond. An emergency overflow spillway allows for controlled overflow if the design storm were to be surpassed.

The following diagram (Figure 9.3) presents the above considerations and summarises the process Flow Diagram of Lagoa Salgada Project.

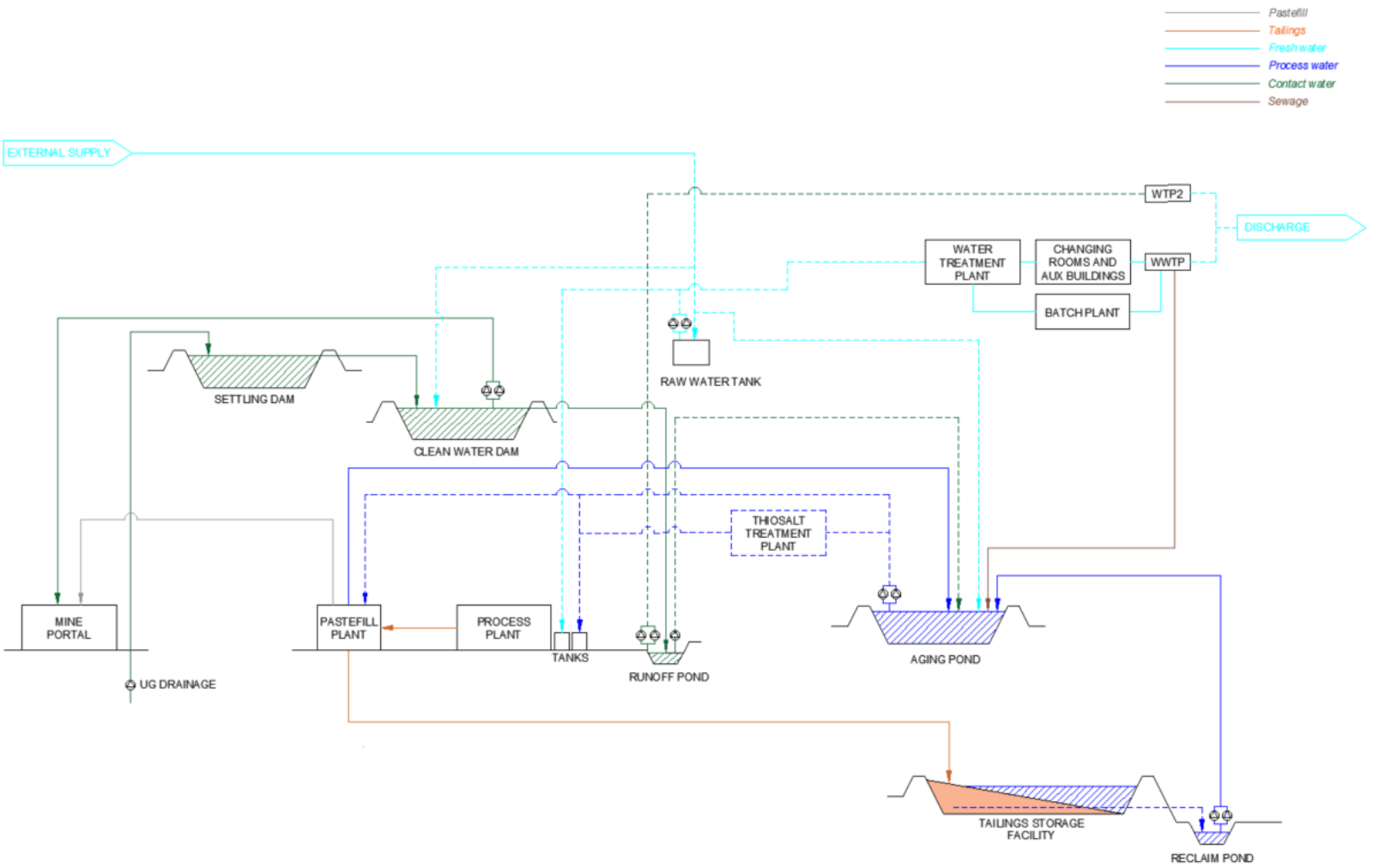


Figure 18.22 – Process Flow Diagram

18.9.2 WATER INFRASTRUCTURES

18.9.2.1 RAW AND PROCESS WATER SYSTEMS

Information related to the design of pipes, fittings and valves, and pumps, refer to the document T2021-620-DFS-HID-RPT-00-EN-02.

RAW WATER

To ensure the availability of potable water in the mine facilities, the process involves pumping raw water from the Vale do Gaio Dam through a network of pipes. The water is first directed to a raw water tank and then sent to a water treatment plant before reaching the process plant and other infrastructure and equipment within the mine.

An alternative method involves pumping raw water from the Vale do Gaio Dam using a floating pump station. This approach breaks down the total bulk water transfer into two stages. First, the water is pumped through a shorter rising main to an intermediate storage tank. From there, a second pump station (referred to as a low lift pump) transfers the water from the storage tank to the raw water tank.

The intermediate storage tank has enough capacity to store one day's worth of water, ensuring that the system has some flexibility in selecting pumping tariff bands. This storage facility is conveniently located near the junction of the road EM1119 with EN 543 at an elevation of 100 meters above sea level (m.a.s.l.).

Table 18.5 - Main Characteristics

Component	Main Characteristics
Pump station 1	18 m^3/h@132 m
Pipeline 1	DI DN 100 mm, L= 10 250 m
Storage tank	500 m^3
Pump station 2	18 m^3/h@64 m
Pipeline 2	DI DN 100 mm, L= 9 750 m
Raw Water tank	1000 m^3

PROCESS AND CONTACT WATER

Water needed for mine activities and its recirculation around the mine field was presented in the previous chapter of this document as well as the Process Flow Diagram (PFD). Its transportation and storage sizing are presented in the next chapters and in the support documentation.

SETTLING DAM

The settling dam is sized to retain water for the time enough for large particles mixed in water coming from mine activities to settle in the bottom of the reservoir. It will be equally divided in 2 for maintenance and cleaning purposes.

Scour valves will be provided for cleaning and draining purposes and designed considering acceptable times to complete drain the reservoir. The dam will also have a minimum freeboard of 1.0 m related to the dam crest. Spillway will also be provided at maximum water level storage and will be designed considering the maximum discharge of the 10-min peak rainfall that falls in the reservoir added to any inflows from other sources, such as activities from inside the mine.

According to the design criteria the Settling Dam has the following characteristics (Figure 18.23):

- Net volume – 2 x 1 000 m³;
- Retention time – 11.6 h;
- Minimum water level – 89.0 m;
- Maximum water level – 90.5 m;
- Scour valve:
    - Discharge into Clean Water Dam;
    - 2 x PEAD DN200;
    - Complete Discharge Time - 6 h;
- Spillway:
    - Discharge into Clean Water Dam;
    - Concrete U-shape open channel;
        - Base – 1.0 m;
        - Height – 0.8 m;
    - Expected maximum discharge flow – 0.17 m³/s
    - Expected maximum water elevation related to the spillway base – 0.22 m.

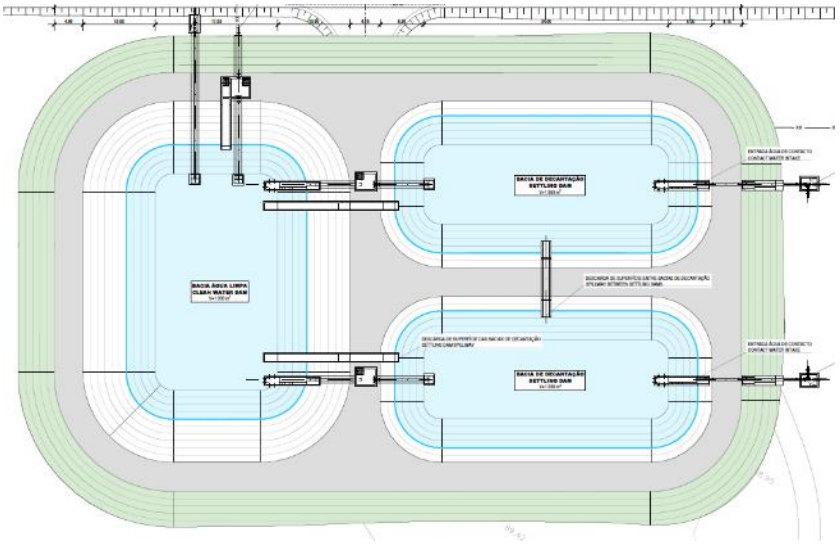


Figure 18.23 – Settling and Clean Water Dam – Plan View

CLEAN WATER DAM

The clean water dam will receive water from the settling dam free from large particles that can cause mine equipment to malfunction.

Scour valves will be provided for cleaning and draining purposes and designed to allow acceptable times to complete the drainage of the reservoir. The dam will also have a minimum freeboard of 1.0 m from the dam crest. A spillway will also be provided at the maximum water level storage and will be designed considering the maximum discharge of the 10-min peak rainfall that falls in the reservoir added to any inflows from other sources, such as from the settling dam.

According to the design criteria the Clean Water Dam has the following characteristics (Figure 18.24):

- Net volume – 1 000 m³;
- Minimum water level – 87.0 m;
- Maximum water level – 88.5 m;
- Scour valve:
  - Discharge into Aging Pond;
  - PEAD DN200;
  - Complete Discharge Time - 6 h;
- Spillway:
  - Discharge into Aging Pond;
  - Concrete U-shape open channel;
    - Base – 1.0 m;
    - Height – 0.8 m;
  - Expected maximum discharge flow – 0.25 m³/s

- Expected maximum water elevation related to spillway base – 0.28 m;

AGING POND

The Aging Pond will receive contaminated water from mine processing activities as well as discharges from all other reservoirs in the mine.

Scour valves will be provided for cleaning and draining purposes and designed considering acceptable times to complete drain the reservoir. The dam will also have a minimum freeboard of 1.0 m related to the dam crest. Spillway will also be provided at maximum water level storage and will be designed considering the maximum discharge of the 10-min peak rainfall that falls in the reservoir added to any inflows from other sources, such as from the settling dam.

According to the design criteria the Aging Pond has the following characteristics:

- Net volume – 80 000 m^3;
- Minimum water level – 84.5 m;
- Maximum water level – 87.5 m;
- Scour valve:
  - Discharge into TSF;
  - PEAD DN900;
  - Complete Discharge Time - 48 h;
- Spillway:
  - Discharge into TSF;
  - Concrete U-shape open channel;
    - Base – 2.0 m;
    - Height – 0.8 m;
  - Expected maximum discharge flow – 1.72 m^3/s
  - Expected maximum water elevation related to spillway base – 0.65 m;

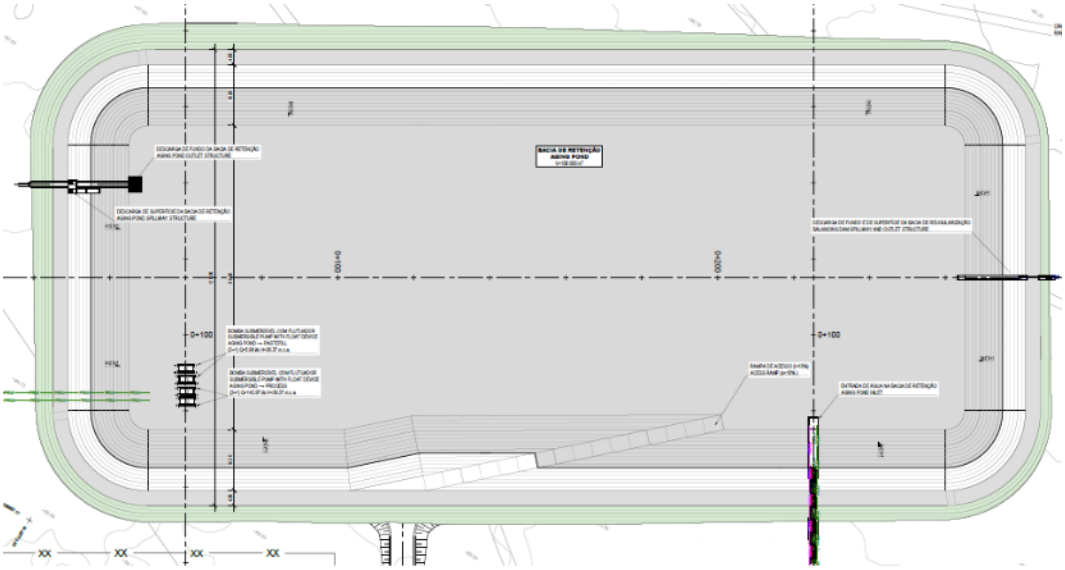


Figure 18.24 – Aging Pond – Plan View

RUNOFF POND

The Runoff Pond will receive contaminated water from rainfall that falls into possibly contaminated areas such as the Process Plant. It is sized to provide sufficient storage for a 20-year return period for an extreme rainfall event of 1 full day.

Scour valves will be provided for cleaning and draining purposes and designed to allow acceptable times to complete the drainage of the reservoir. The dam will also have a minimum freeboard of 1.0 m from the dam crest. A spillway will also be provided at the maximum water level storage and will be designed considering the maximum discharge of the 10-min peak rainfall that falls into the reservoir plus any inflows from other sources, such as from the settling dam.

According to the design criteria the Runoff Pond has the following characteristics (Figure 18.25):

- Net volume – 10 000 m³;
- Minimum water level – 82.5 m;
- Maximum water level – 85.0 m;
- Scour valve:
    - Discharge into TSF;
    - PEAD DN400;
    - Complete Discharge Time - 12 h;
- Spillway:
    - Discharge into TSF;
    - Concrete U-shape open channel;
        - Base – 1.0 m;

- Height – 0.8 m;

- Expected maximum discharge flow – 0.29 m³/s
- Expected maximum water elevation related to spillway base – 0.31 m;

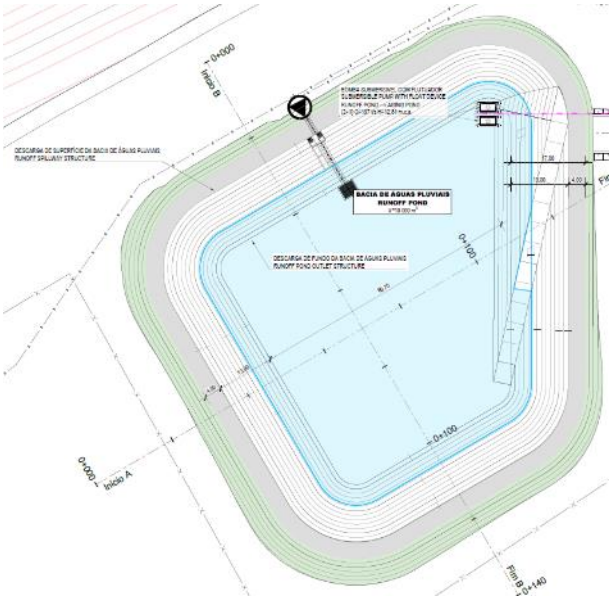


Figure 18.25 – Runoff Pond – Plan View

18.9.2.2 POTABLE WATER

Potable water will be supplied to all buildings such as offices, toilets and change houses. Additionally, some equipment will be supplied with potable water to avoid having parallel service water lines that may unnecessarily increase costs and maintenance.

The potable water demands and peak flows are present in the support information, report T2022-620-DFS-HID-RPT-00-EN_01.

WATER TREATMENT PLANT

The water treatment plant will be modular including any mechanical and electrical devices for easier assembly and maintenance.

The WTP is prepared to treat 116.2 m³/day of raw water in 16h/day (7.5 m³/h flowrate).

The main functions are described below:

- Removal of solids in suspension, organic matter, turbidity and pathogenic microorganisms.
- Reduction in content of calcium, magnesium, and bicarbonates, therefore, avoiding encrustations in pipes and equipment.
- Average quality water treatment.

18.9.2.3 SEWAGE SYSTEMS

Peak discharge is determined by the number of fittings at each location multiplied by the flow each fitting can deliver, summing all flows and applying a simultaneousness coefficient. Wastewater is calculated at 90 % of the daily potable consumption.

Raw sewage will be treated in a Wastewater Treatment Facility. Treated sewage effluent should be recycled or needs to comply with the applicable discharge water requirements.

SEWAGE TREATMENT PLANT

The wastewater treatment plant will be modular including any mechanical and electrical devices for easier assembly and maintenance.

The WWTP is prepared to treat 112.6 m^3/day of domestic sewage.

It includes a Pumping Well Unit and a Fat Separator Unit, to separate oils and fats from organic nature (animals and vegetables) from water, depending on the difference in density. As well as an Activated Sludge Debugger Unit and a Chlorination System Unit to remove pathogenic elements from the effluent.

18.10 STOCKPILE DIMENSIONING

The Lagoa Salgada project will stockpile material at the surface, in the ROM Pad. The ROM Pad is located west of the Mine Portal, near the Process Plant and Surface Crusher. The stockpile analysis defined 3 options for one, two, and three stockpiles in the ROM Pad. For all these options, a maximum angle of 35 degrees, 1/5 inclination for the stocked material was taken into consideration.

The objective of this infrastructure will be to temporarily store the ore coming from the mine before being processed in the processing plant. The ore stockyard has been designed to accommodate the various domains to be processed, as the plant will operate in campaigns for each domain (MS Fresh, MS Transition, Stockwork and Gossan). However, this storage will never exceed 3 to 4 weeks due to ore oxidation issues that have a negative impact on mineral processing. Ideally, the ore extracted from the mine will be crushed at the surface facility and not stored in the ore stockyard.

As a good mining operation practice, the permanent stock in the ore stockyard should constantly be equivalent to 4 to 5 days of the plant operation to serve as a buffer that prevents the shutdown of the ore processing facility in case of any stoppage or production issue at the mine.

18.10.1 STOCKPILE OPTIONS

For option 1 – single stockpile – the maximum height of the stockpile will be 30 meters, with a volume of 130,000 m^3 (Figure 18.26).

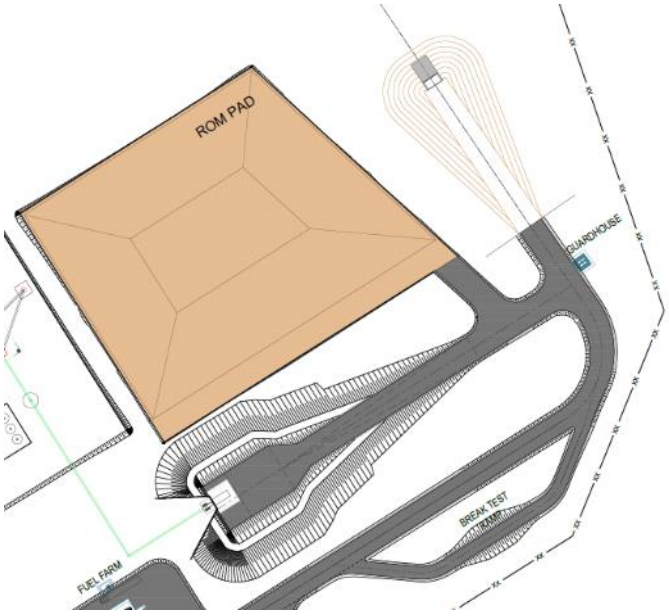


Figure 18.26 - Stockpile Option 1

The second option (Figure 18.27) consists of two stockpiles with a maximum height of 30 meters, the left and right stockpiles have the same capacity, 55,300 m^3.

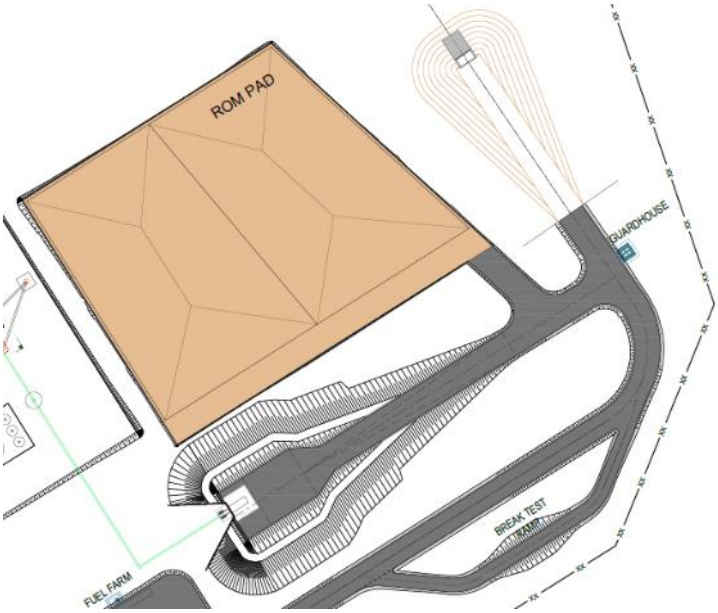


Figure 18.27 - Stockpile Option 2

Regarding the third option (Figure 18.28), it consists of three stockpiles with a maximum height of 13.5m meters, the three stockpiles have a volume of 24,900 m^3 each.

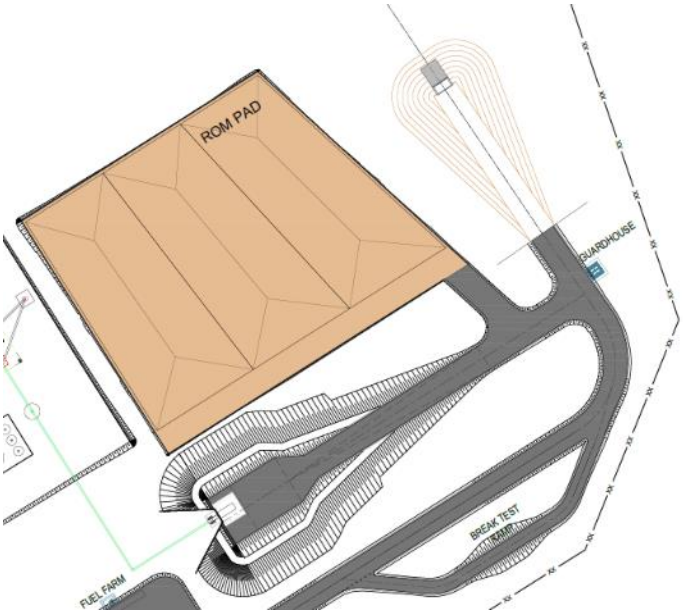


Figure 18.28 - Stockpile Option 3

The volume per stockpile information is summarized in the table below (Table 18.6), these values represent only one of the stockpiles for options 2 and 3.

Table 18.6 - Volume for each Stockpile

Stockpile	Volume (k m^3)
Option 1	130
Option 2 (x2)	55
Option 3 (x3)	25

To calculate the maximum tonnage for each stockpile, it was assumed the density values presented in the table below - Table 18.7, where this information is divided into the different domains.

Table 18.7 - Density by domain

Domain	Density (t/m^3)
Gossan	3.0
Massive Sulphide	4.6
Stockwork	3.0
Waste	2.8
Tertiary	2.2

Converting the volume of the stockpiles into the different domains, the stockpiles capacity is the following:

Table 18.8 - Tonnes by Domain/Stockpile

Stockpile	Tonnes (k t)				
	Gossan	Massive Sulphide	Stockwork	Waste	Tertiary
Option 1	390	603	390	364	291
Option 2 (x2)	166	257	166	155	124
Option 3 (x3)	75	116	75	70	56

Using a swelling factor of 1.6, due to the material voids, the stockpiles have the following capacity (Table 18.9):

Table 18.9 - Tonnes by Domain/Stockpile with Swelling Factor

Stockpile	Tonnes (k t)				
	Gossan	Massive Sulphide	Stockwork	Waste	Tertiary
Option 1	244	377	244	228	182
Option 2 (x2)	104	160	104	97	77
Option 3 (x3)	47	72	47	44	35

Applying a 0.8 error coefficient to the Table 18.10 values, once the stockpile does not include ramps for stocking and all the stocks are a perfect pyramidal stockpile, the final values are (Table 18.10).

Table 18.10 - Tonnes by Domain/Stockpile with Swelling Factor and Error Coefficient

Stockpile	Tonnes (k t)				
	Gossan	Massive Sulphide	Stockwork	Waste	Tertiary
Option 1	217	335	217	202	162
Option 2	92	143	92	86	69
Option 3	42	64	42	39	31

18.11 TAILINGS AND WASTE MANAGEMENT

18.11.1 GEOCHEMISTRY

A geochemistry assessment for the tailings and development rock was conducted for both Venda Nova North ('Northern Zone') and Venda Nova South ('Southern Zone'), where the mining will take place. Static and long-term leaching was assessed using kinetic testing to establish the onset time for acid leaching and to predict the leachate quality and to check for the presence of dissolved metal and non-metal ions.

The conclusion from the Northern Zone tests is that there is pyrite in the tailings and as a result of this and the lack of buffering minerals, the tailings will produce acid quickly, with the onset of acid conditions within 2 weeks, based on the kinetic testing.

The Southern Zone tailings consist of aluminosilicate and carbonate minerals which buffer the pH and prevent the immediate onset of acid conditions, despite the presence of sulfides. Kinetic testing indicates that these tailings will not produce acid.

The results of the geochemical testing for the development rock indicate that the rock could potentially be acid forming (PAF), except for the rock from the Tertiary and Gossan deposits that are non-acid forming (NAF). Based on the contact leach testing and kinetic testing, the heavy metals will not leach in high concentrations, provided that the waste rock management is conducted in such a way that acid-generating rocks are combined or placed with non-acid generating rocks, using a 2:1 NAF:PAF ratio.

The various geochemistry tests conducted are detailed in the geochemistry report (SLR, 2023a).

18.11.2 TAILINGS MANAGEMENT

The mine will process multiple ore streams, which will produce different streams of tailings with varying characteristics, ranging from massive sulphide (almost zero clay) to high-clay-types (sericite, illite, etc.). Tailings from the Northern Zone are expected to be potentially acid-generating (PAG), while those from the Southern Zone are expected to be non-potentially acid-generating (non-PAG).

The tailings will be dewatered to a non-segregating (high-density) thickened slurry consistency in the process plant. The thickened slurry will be discharged to the Tailings Storge Facility (TSF) at full throughput when not backfilling, or at a reduced rate when backfilling. When backfilling, a partial stream between 15% to 30% of the total tailings stream (depending on the ore type) will be blended with sand and binding agents before it is distributed underground as backfill. The remainder will be pumped from the thickener agitated storage tank to the surface TSF. Overall, an estimated 20% of tailings will be used for backfill and 80% will be deposited in the TSF.

The average in-situ dry density of the facility governs the total tailings capacity which the TSF will be able to store. The weighted average for the dry density of the deposited tailings is 1.9 t/m^3, but will vary depending on the ore type, specifically sulphide versus clay-like tailings, ranging between 1.5 and 2.6 t/m^3.

18.11.3 TAILINGS STORAGE FACILITY

The Tailings Storage Facility (TSF) will contain all the tailings that are not used for backfilling as well as all the development rock from the mine. Development rock that is non-PAG may be used as dam fill while all PAG rock must be stored within the TSF. The TSF is a lined ring embankment facility constructed using downstream methods. The planned TSF and the section is presented on Figure 18.29 and Figure 18.30. For full details about the TSF design, please refer to the TSF Design Report (SLR, 2023b).

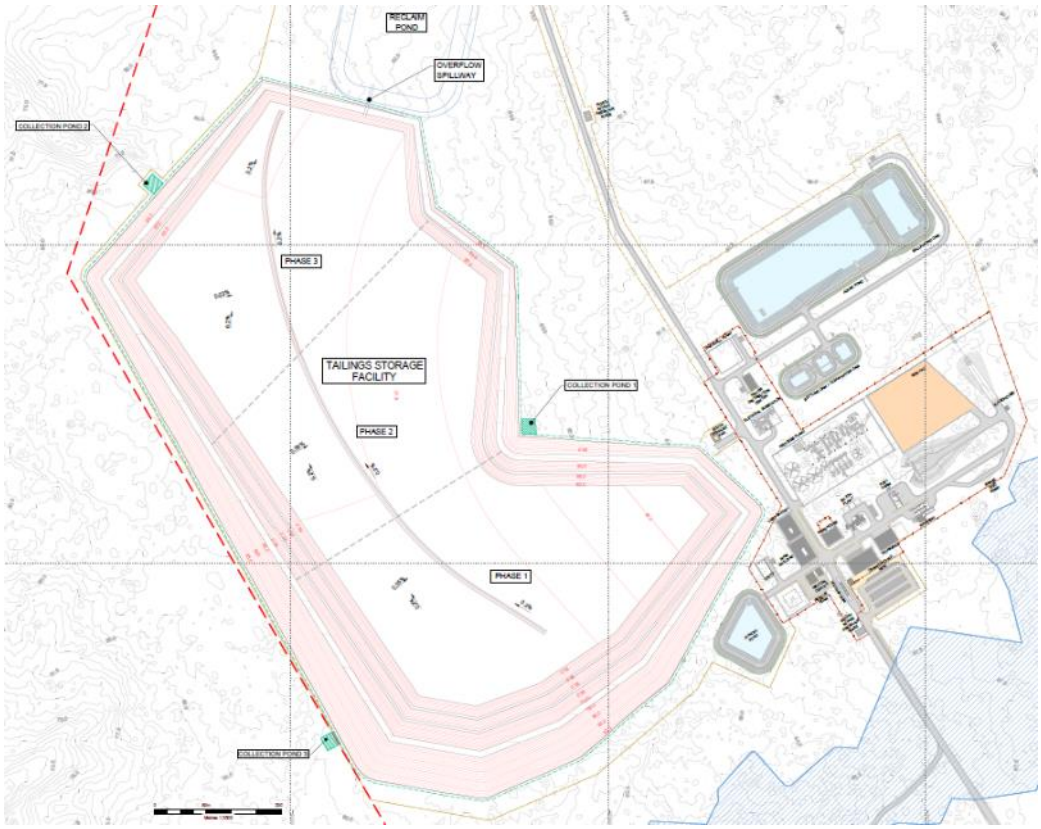


Figure 18.29 – TSF Plan

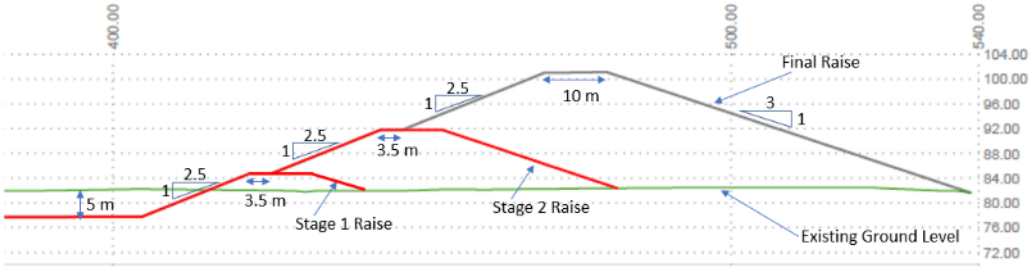


Figure 18.30 – TSF Cross-section

18.11.3.1 TSF Site Selection

A site selection study was conducted that assessed eight potential locations within a 10 km radius of the mine. The selection study considered locations that could feasibly contain the design's tailings tonnage, followed by a systematic and rational approach of assessing a balance in the factors over the life cycle to identify a preferred site.

Of the eight sites studied, two emerged as viable options (termed Site E and Site H) worthy of consideration for the candidate site. Both sites afforded relatively good storage efficiencies and were generally close to the mill. Ultimately, Site H was selected for this project because it is entirely on the mine property and within the area of the current environmental study. However, Site E remains an appealing alternative because of its higher storage efficiency (i.e. requires less dam construction) and it should be further assessed as the project advances.

18.11.3.2 Basis of the Design and the Design Criteria

The overall design objective of the TSF is to safely contain mine waste (tailings and development rock) in a single facility while protecting the groundwater and surface water resources during operations and long term. A design basis memorandum (DBM) has been prepared to summarize the drivers and the basis of the TSF design (SLR, 2023c)

The principal drivers of the design basis include:

- Geochemical characteristics of the tailings and development rock that require storage in a saturated state to inhibit acid generation and to provide a geosynthetic liner to protect groundwater resources;
- Environmentally sensitive and protected areas on site that restrict the location and the extent of the TSF;
- Management of contact water and runoff within an external reclaim pond; and
- Design for closure approach that facilitates the possibility of early suspension or shut-down of operations.

A non-segregating, thickened tailings consistency has been selected to support the underground backfilling and for tailings disposed on the surface to be maintained in a saturated state to prevent acid-generation by inhibiting oxidation. Tailings will be discharged via pipelines and spigots around the TSF forming a consistent beach slope from the point of discharge, predicted to be approximately 2%.

The TSF will be sloped from high ground in the south to lower ground in the north to allow bleed water from the deposited tailings and runoff from precipitation to drain passively to a lined reclaim pond from where water can be recirculated to be used as make-up water for the mill.

The mine plan indicates that 14.8 Mt of ore will be processed, and 1.9 Mt of development rock will be generated. After accounting for the concentrate and underground backfill, a total of 11.2 Mt of tailings will be deposited in the TSF, in addition to the 1.9 Mt of development rock. The design of the TSF provides storage for these with an estimated storage capacity of 6.3 Mm^3 for tailings and development rock.

18.11.3.3 PRELIMINARY CONSEQUENCE CLASSIFICATION

Because of the lack of a surface pond, the consequences of a potential dam breach would not be extensive, and the runout of tailings would be limited. Nonetheless, a preliminary consequence classification for the TSF has been assigned as follows to provide design conservatism and allow for optimization opportunities as the project advances:

- Global Industry Standard on Tailings Management: Very High
- Canadian Dam Association dam safety guidelines: Very High

It should be noted that environmental losses have been conservatively assessed, and further work may demonstrate a downgrade in classification to High under both GISTM and CDA if environmental losses can be demonstrated to be less severe.

18.11.3.4 TSF DESIGN ELEMENTS

The TSF embankments are constructed in a series of raises with a 3H:1V downstream slope and a 2.5H:1V upstream slope. At the crest of each raise, a minimum crest width of 10 m is provided for operation and maintenance of the tailings delivery pipeline.

The embankments are constructed of sand and clay soils excavated from within the TSF basin.

The embankment slope stability was assessed against minimum factors of safety as recommended in the CDA guidelines (2013) and are summarized below.

- Static end of construction: 1.3
- Static long-term: 1.5

- Seismic pseudo-static: 1.0
- Seismic post-earthquake: 1.2

BASAL LINING SYSTEM

A lining system will be required to reduce the impact of the consolidation water from the TSF and percolation, which could result in contamination of the groundwater or impact the quality of the water in rivers downstream of the TSF. The lining system will be installed over the basal area and upstream slopes of the TSF.

The lining system is presented on Figure 18.31 and consists of (from top to bottom):

- 500mm thick sand protection/ballast layer
- High-Density Polyurethane (HDPE) geomembrane
- Geosynthetic Clay Liner (GCL)

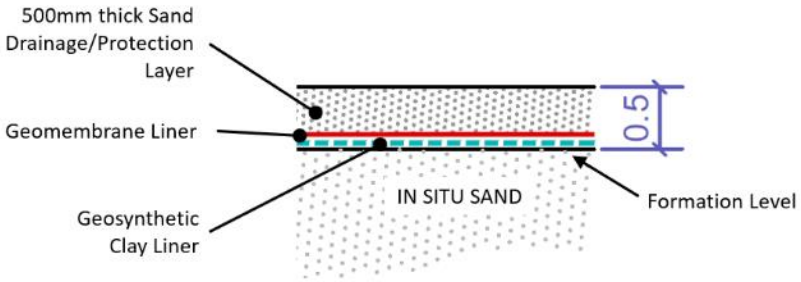


Figure 18.31 – Basal lining system arrangement

Preliminary test pitting at the TSF has encountered a natural low permeability clay layer; however, there is currently insufficient data to demonstrate that it will form a continuous low permeability layer beneath the TSF. Further investigations to demonstrate that the clay layer is continuous beneath the entire TSF and has a consistent low permeability may provide an opportunity to change this design and replace the geosynthetic elements of the lining system.

Tailings generated from stockwork ore may also be suitable for basal lining or capping given the lack of sulphides and high clay content (i.e. low permeability). This has not been incorporated at this time and represents an opportunity for further consideration as the project advances.

GROUNDWATER DRAINAGE SYSTEM

There is evidence from the test pits that a clay layer underlies the TSF and that this layer acts as an aquitard with groundwater perched above it. A groundwater drainage system is

intended to control any local groundwater perched above the clay, preventing uplift forces acting on the lining system that could result in instability.

The design of the drainage system comprises a drain located around the upstream slope of the perimeter embankment at the interface between the clay and the overlying sand, which drains to a sump in the northwest corner of the TSF. Water collected in the sump would be pumped back to the TSF or Reclaim Pond. A typical section of the groundwater drain is presented on Figure 18.32.

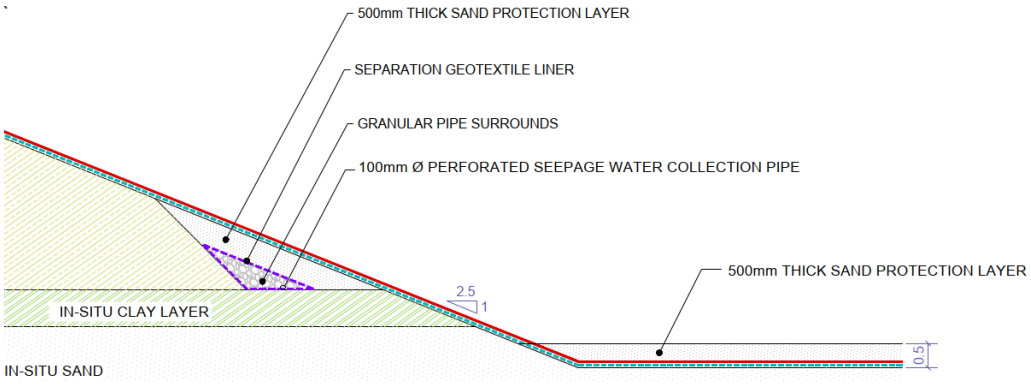


Figure 18.32 – Typical section through groundwater drain

OVER LINER WATER DRAINAGE SYSTEM

The base of the TSF is graded from the upstream toe of the TSF embankment to the centre of the basin so surface water will be drained in a central channel flowing south to north and discharging into the Reclaim Pond. A typical section of the drain is presented on Figure 18.33.

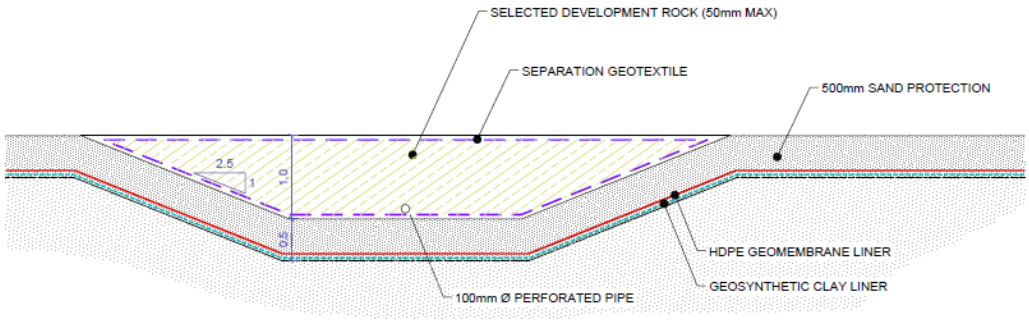


Figure 18.33 – Typical section of over liner drainage channel

The drainage channel has a minimum depth of 1 m, a 3 m wide base, and 2.5H:1V side slopes. The lining system in the channel is overlain by a protective layer of sand, above which a perforated pipe will be installed, and the channel backfilled with selected

development rock. A geotextile separator shall be wrapped around the development rock to prevent plugging the drain.

A 500mm layer of protective sand will be placed over the geomembrane liner which will also function as a drainage path to channel bleed water along the TSF base to the drainage channel. The interphase berms formed from either sand or development rock also form part of the bleed water drainage system, with the free draining granular fill allowing bleed water to drain through the berm and collect downstream of the berm.

COLLECTION DITCHES AND PUMP SUMPS

Runoff from the outer slopes of the TSF will be collected in ditches at the toe of the perimeter embankments and passively drain to sumps located at low points around the facility. Water collected in the sumps will be pumped into the Reclaim Pond. A typical section of the collection ditch is shown on Figure 18.34.

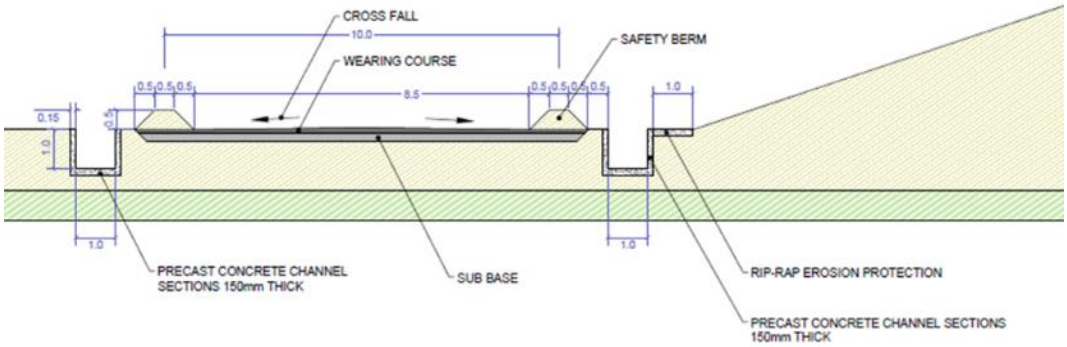


Figure 18.34 – Typical section of collection ditch

The collection ditches will be concrete lined with a rectangular cross section having 1.0m base width and 1.0 m depth. Ditch bed slopes will vary between 0.2% and 1.2% depending on the natural topography. The collection ditches have been designed to convey the peak flow from the critical duration of a Probable Maximum Precipitation (PMP) event without overtopping.

The sumps constructed in the low points around the TSF will consist of lined excavations and will be sized to contain two or more submersible or vertical turbine pumps, to manage the critical duration of a 1:100 year flood. Each sump will contain a standby pump in addition to the operating pumps.

18.11.3.5 TSF Development Sequence

The development of the TSF will take place in 3 phases of basal lining, which will be followed by vertical raises of the perimeter embankment, with the number of raises dependent on the height of the embankment. The TSF embankment is tallest along the south end (up to

20 m above existing ground) and shortest along the north end to maintain passive drainage to the external Reclaim Pond.

In general, the TSF basin will be cut to an average depth of 5 m. Topsoil will be stripped and stockpiled for use as restoration soil at closure. The remainder of the cut soil will be used to construct the perimeter embankments and are expected to be comprised of sand and potentially clay, should the cut extend into the deeper clay deposit. Where there is an excess of soil from the basal cut beyond what is required for the perimeter embankments, the soil(s) will be temporarily stockpiled within the ultimate TSF footprint and used later for construction or may be used for blending sand with tailings for backfill, provided the soil gradation meet the backfill specifications.

To provide greater operational control and flexibility, the inside of the TSF will be subdivided into small paddocks, separated by permeable internal berms constructed from either development rock or sand. The number of paddocks and orientation of the berms will be determined by site operations with the intention of managing PAG tailings saturation and maintaining areas to develop rock deposition.

The following describes the development of the phases and deposition plan of the TSF.

PHASE 1

Phase 1 forms the southern portion of the TSF covering an area of 22 ha and will be constructed first. The basal area of Phase 1 will be excavated, and the excavated soil will be used as fill to construct the starter embankment. The lining system will be installed over the whole of the basal area and upstream slope of the starter embankment. Phase 1 in plan view is presented on Figure 18.35.

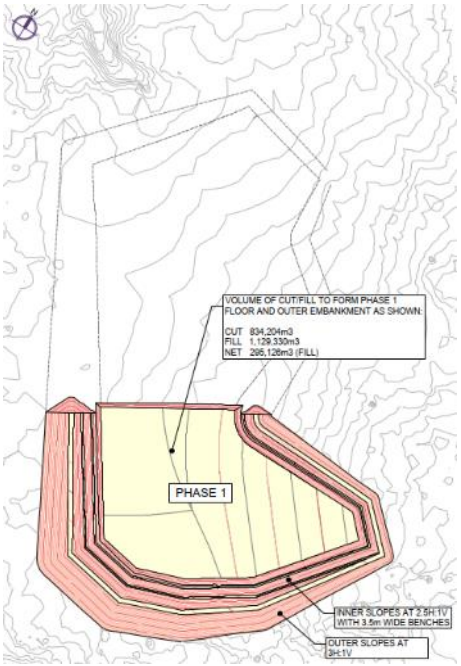


Figure 18.35 – TSF Phase 1 plan

A temporary lined reclaim pond will be constructed within the Phase 2 area for water management (not shown on Figure 18.36). A 3 m high permeable interphase berm, constructed from development rock along the interface between Phases 1 and 2, forms the northern limit of the Phase 1 area and is intended to allow water to flow through while containing deposited tailings.

Tailings will be deposited from the southern and western embankments of Phase 1 falling in a northerly direction. This will ensure that the seepage water and contact water always drains to the reclaim pond and is not retained on the surface of the tailings.

PHASE 2

Phase 2 forms the central portion of the facility covering an area of 15 ha. Whilst Phase 1 is operational, the remaining area of Phase 2 north of the temporary reclaim pond and the temporary reclaim pond in Phase 3, will be excavated to the design levels. Excavated sand and clay will be used to construct the Phase 2 embankments and raise the embankments in Phase 1. Phase 2 in plan view is presented on Figure 18.36.

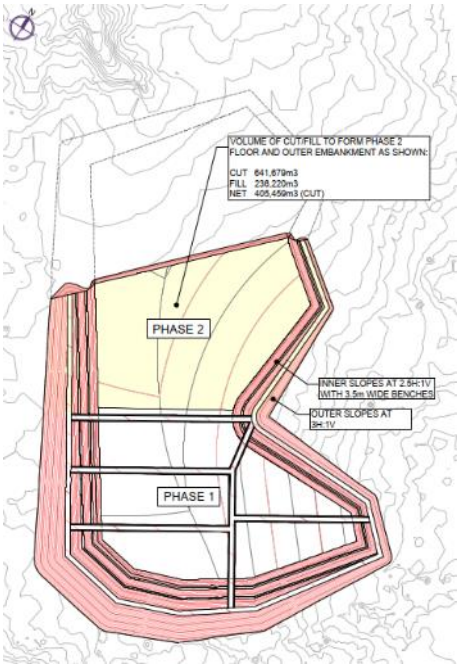


Figure 18.36 – TSF Phase 2 plan

The phase will be lined in the same manner as Phase 1, with an interphase berm and a temporary reclaim pond constructed within the Phase 3 footprint. Water held in the temporary reclaim pond will need to be released and allowed to flow into the new temporary pond.

Once the tailings on the northern edge of Phase 1 have reached the top of the interphase berm, the tailings will be deposited in Phase 2. The tailings in Phase 2 will be deposited from the western embankment to maintain the beach slope towards the reclaim pond. The placement of the tailings for the Stage 2 vertical raise in Phase 1 will start once the tailings are placed across the base of Phase 2.

PHASE 3

Phase 3 forms the northern-most portion of the TSF covering the area of 14.5 ha. Whilst Phase 2 is operational, the remaining area of Phase 3 and the external reclaim pond will be stripped of its topsoil and excavated to design levels. The excavated sand and clay will be used to construct the Phase 3 embankments and the excess material stockpiled. The phase will be lined in the same manner as Phases 1 and 2. Phase 3 in plan view is presented on Figure 18.37.

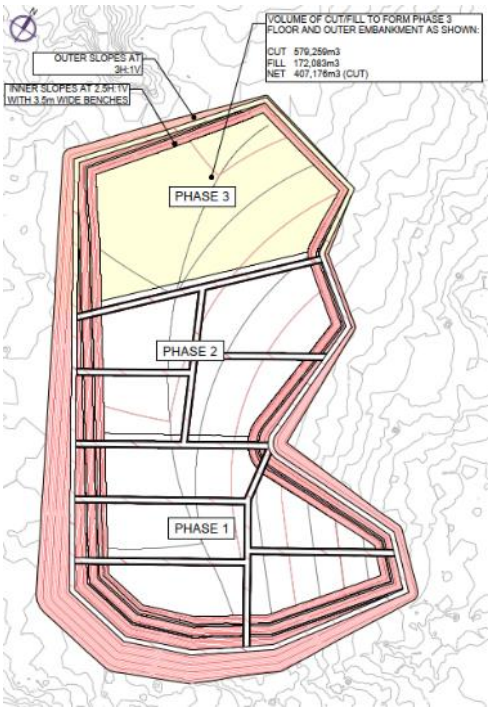


Figure 18.37 – TSF Phase 3 plan

Similar to Phase 2, water held in the temporary reclaim pond will need to be released and the liner which covers the upstream face of the temporary reclaim berm removed.

Once the tailings placement on the northern edge of Phase 2 has reached the top of the interphase berm, the tailings will be deposited in Phase 3. The tailings in Phase 3 will be deposited from the western embankment.

Once the tailings have been deposited across the entire TSF basin, the process of filling vertical stages from north to south to maintain a consistent beach slope will continue until

the TSF has reached the final design levels or the mine cease production, at which point the TSF will move into the closure phase.

18.11.3.6 TSF Water Management

A lined Reclaim Pond will be maintained throughout the operating life of the TSF to collect runoff and bleed water from the deposited tailings for return to the process plant. The location of the pond will change during the development of the TSF, with the pond initially maintained within the facility's footprint but over time moving outside the TSF to a final location in its northeast corner. An external Reclaim Pond is preferred to reduce the risk of a tailings breach and to minimize water losses due to evaporation by storing it in an excavation to reduce the surface area.

The spillway is sized to safely convey the peak flow from the critical duration Inflow Design Flood (IDF), that is the Probable Maximum Flood (PMF), from the tailings beach area to the Reclaim Pond at all stages of TSF development.

The Reclaim Pond has been sized to contain the following water volumes with no release to the environment:

- The maximum operating water volume in the pond, based on modelling of the average annual TSF water balance on a monthly time step.
- The critical duration of the volume of a 1:100 year Environmental Design Flood (EDF).

At all stages of TSF development, the Reclaim Pond will be provided with freeboard above the EDF level and an emergency spillway to safely convey the peak flow from the critical duration IDF (that is the PMF) through the facility.

The permanent reclaim pond will be constructed north of the TSF and will be lined to the same specification as the TSF. The pond is 315 m by 215 m, covers an area of 6.5 ha, and will be excavated to a depth of 7m with 3H:1V side slopes.

18.11.3.7 Development Rock Disposal

Development rock can be used for internal temporary roadways into the TSF to enable tailings piping/spigotting to generate thin-layer disposal for enhanced desiccation/stability and achieving dry density.

Development rock would be infilled and covered with tailings to mitigate air rainfall entrainment thereby limiting leachate generation. The slurry density for infilling is in alignment with the thickener underflow density; paste is too thick for this purpose.

18.11.4 MINE BACKFILL PLANT

18.11.4.1 TAILINGS DEWATERING

BACKFILL PLANT DESIGN

The project was initiated by RMS with a site visit kick-off meeting in January 2022; over the course of the last 12 -14 months, additional exploration, geological updates, process testing and process design update, along with mine engineering studies have been continued or developed. As a result, on-going tailings and backfill design studies under RMS have also evolved with mining layouts, major changes to process flowsheet to accommodate different ore processing needs, and as samples became available for testing and process design.

The selected Paste Backfill System (Option 2) was based on the lowest Net Present Cost (NPC). The solution consists of a partial stream of thickened tailings, combined with sand and binder to produce a paste backfill; large differences in tailings resulting from the varying massive sulphide to stockwork ore types will be managed with sand (between 60% to 70% of the mix) and binder component to ensure the backfill strength meets mining requirements.

The options evaluated are shown below in Table 18.11. The Base Case option, which was rejected based on the NPC, consisted of a typical paste backfill plant (PBP) using a thickener and filter dewatering flowsheet along with paste pump for the underground distribution systems. The lower Net Present Cost (capex/opex time series) solution was selected to utilize local sand sources at USD$6/tonne (identified by Ascendant) blended with thickened tailings to produce a paste for the majority of the mining; the gossan zone currently based on cut and fill (C&F) mining method can use uncemented or cemented sandfill (SF or CSF). It may also be possible to use cemented sand/tailings blend, though latest testwork will need to confirm. (This evaluation, including capital and operating along with a discounted cash flow series, included:

- Sub-aerial disposal of non-segregating thickened tailings (NSTT) to support stacked tailings (potential to crest above the dam elevation), with separate process/contact water and tailings storage facilities;
- Elimination of tailings filters and paste pumps, unless justified by increased sand cost, larger mineral resource, mineral reserve and payback;
- Thickened tailings slurry pump system from agitated storage tank to each North and South receiving tanks
- Sand storage system and front-end-loader (FEL) to feed the mobile backfill plant;
- Single trailer-mounted mobile backfill plant to be relocated as needed to North and South Zones as required, the system discharges into cased boreholes by gravity to stopes;

- Backfill recipe to include tailings, sand and binder, used with each ore type (with exception of gossan, which is to be determined); and
- Independent Underground Distribution Systems (UDS) to both North and South Zones.

RMS executed several studies that are outlined below:

- Design criteria and operating data;
- Paste backfill plant (PBP) design; and
- Backfill options

The selected Paste Backfill System uses thickened tailings, sand and binder to produce a paste backfill; since there is an enormous difference between the ore types, from massive sulphide to stockwork, the quality of the fill will change, however the sand component will vary to ensure the quality underground is in alignment with the backfill strength required, along with binder.

The Base Case design of the paste backfill plant developed relied on the paste thickener and vacuum disc filters to dewater the tailings received from the concentrator in order to prepare the paste backfill to support underground mining operation. Paste backfill would be distributed to North and South Zones using high-pressure paste pumps. However, due to high capital cost of the filter plant and possible risks associated with the filtration of stockwork tailings, RMS provided alternative solutions using blended tailings and sand, while eliminating filters, and placing mobile plant over the mine zones as needed to eliminate paste pumps. This alternative allows for approximately 26% of all tailings to be deposited underground, a priority given the constrained small footprint of the site.

The backfill options study considered a few different dewatering arrangements including a paste backfill mobile plant, paste backfill transport configurations (paste pumping versus gravity) and backfill mixtures with ranges of tailings and sand blends. Each option was compared with Base Case design. The Net Present Cost (NPC) in Figure 18.38 reflects the time series of CAPEX, OPEX and discount value to select the most cost-effective solution (Option 2), as determined in close collaboration with Ascendant.

Table 18.11 – Trade off Paste Backfill System

Base Case	Option 1	Option 2	Option 3
Paste Thickener	Paste Thickener	Paste Thickener	Cyclones
2 Vacuum Filters	Sand System	Sand System	2 Vacuum Filters
2 Paste pumps	2 paste pumps	Mobile Mixing Plant	2 paste pumps

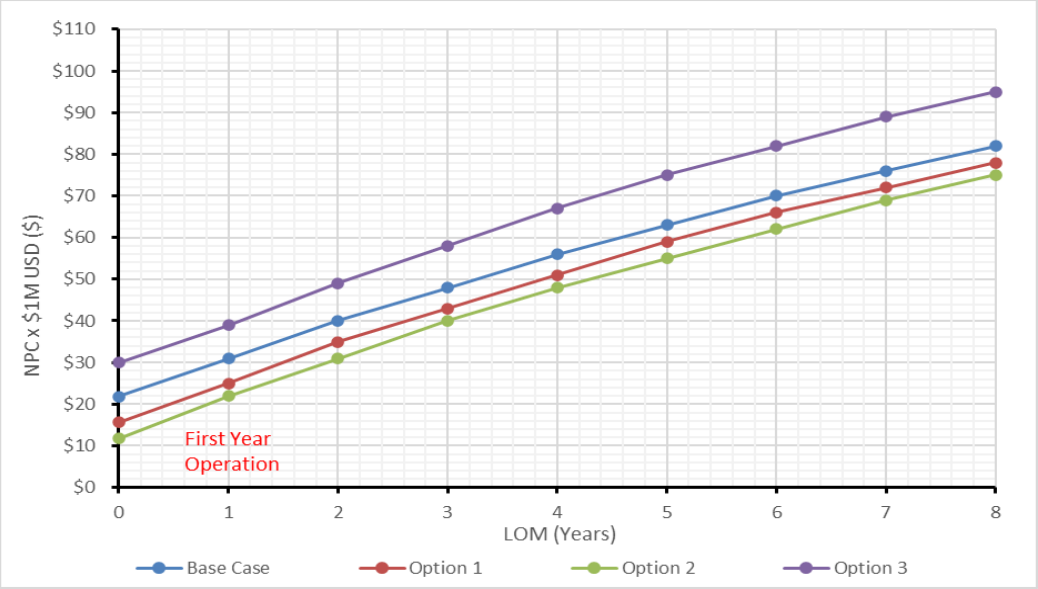


Figure 18.38 – NPC Estimate

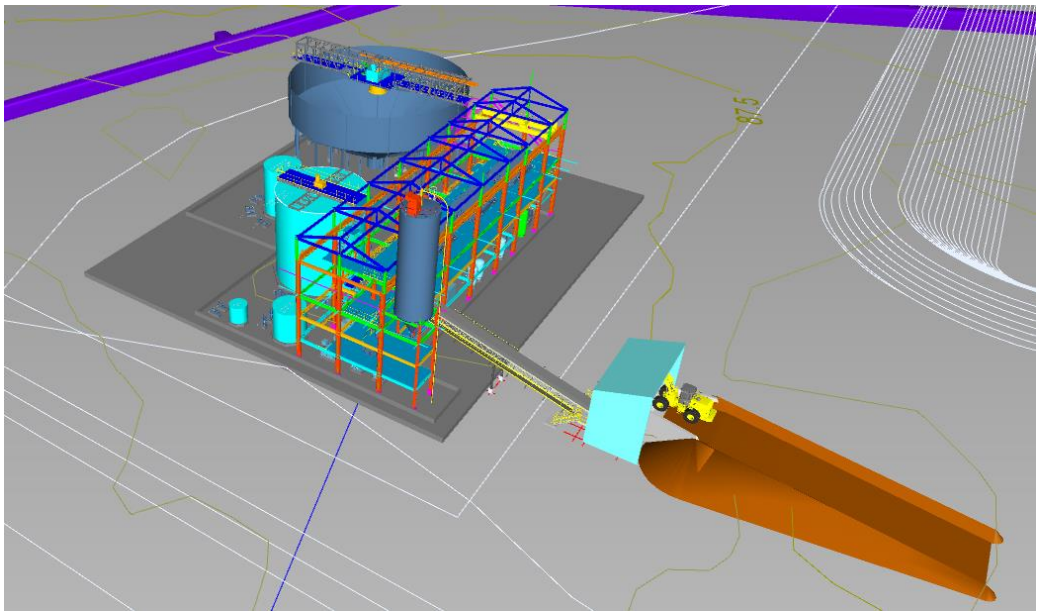


Figure 18.39 – Base Case (Thickener, Filter, Paste Pumps)

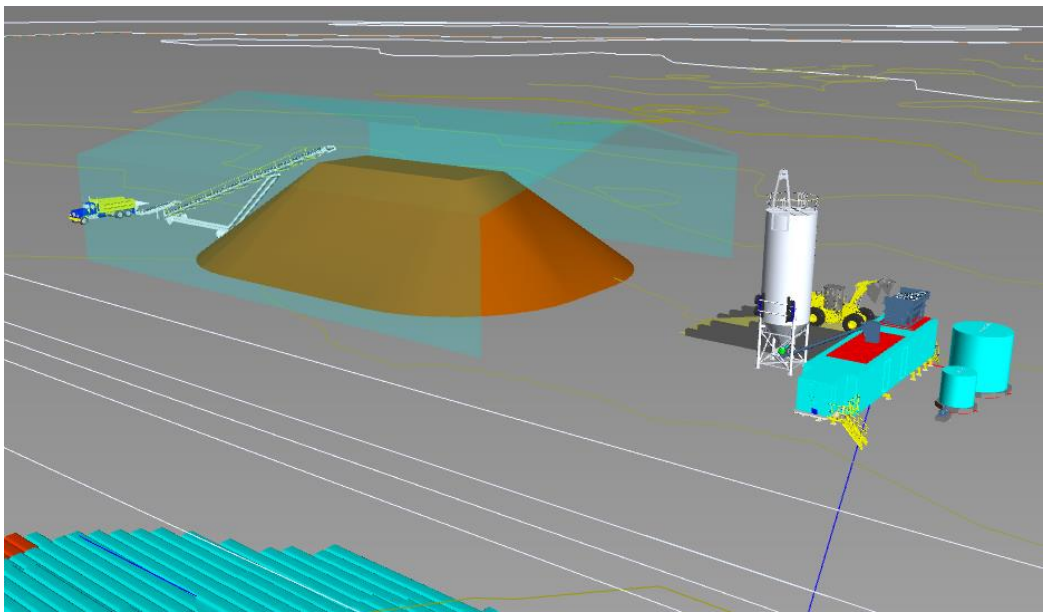


Figure 18.40 – Mobile Paste Backfill Plant (Option 2)

Based on the trade-off study outcomes, Ascendant decided to move forward with Option 2 strategy (paste thickener, mobile mixing plant, gravity distribution to stopes); should the cost of sand increase, or Ascendant be successful in finding additional resources/reserves thus warranting greater capacity requirement in the TSF, consequently necessitating a greater reliance on eliminating sand in backfill (i.e. more UG tailings disposal), then the development of single paste plant with filters and paste pumping in the future may be justified. This increase in capex has not been included, as it is assumed that Option 2 will be carried to the end of the current Life of Mine Plan (LOMP).

Should sand costs increase the NPC to be less competitive to the Base Case, it may be that the constrained footprint for tailings capacity is the likely determinant constraint in the future, when considering non-economic aspects including the permitting of new or expanded TSF. Thickened tailings sub-aerial stacking within this study will enable the TSF to contain the full tailings within the resource/reserves provided herein.

Should greater capacity be needed, additional studies may be warranted to assess the filtered tailings stacking, however rates of reactivity of the sulphide tailings including through air-entrainment during the filtration process, handling via trucks or conveyors, loaders, dozers and packers would need to be considered to limit the negative environmental impacts and potential permitting issues. Furthermore, stockwork ore types having high clay and low sulphides may be used to encapsulate reactive tailings by sequencing these zones to produce sufficient tailings to cap cells of reactive material within the mill reconfiguration changes/schedule to limit reactivity and metals leaching/acid rock drainage (ML/ARD).

This technical report outlines RMS' approach for PBP design to support underground mining operation and surface disposal of thickened tailings to meet the tailings synergy between the underground backfill and the TSF within the current LOMP.

18.11.5 PROCESS DESIGN CRITERIA

The operating data, design criteria and backfill plant design are presented in paste backfill design report (RMS 2023). It should be noted that the Gossan testing is ongoing, as the sample was recently received (February 2023).

According to discussions with the relevant parties, the operating data and LOMP, the process plant throughput is stated as 1.2 Mt/y; considering that frequently planned shutdowns of 2-3 days required to reconfigure the mill between processing of the different primary ore types, as per SCYMPI). In accordance with LOM, the tailings dewatering thickener would be designed for the full plant tailings (FPT) throughput of ~170 t/h on dry solid basis, while the equipment design factor of 1.1 has been included, as per discussions with the project team.

Based on Ascendant's decision to proceed with Option 2 (mobile PBP design without vacuum filtration arrangement), the PBP will rely on the blend that consists of sand and thickened tailings at the ratio of approximately 70%/30% to 60%/40% (this ratio varies by ore type, since the SG of sulphides vs stockwork ranges from 4.9 to 2.9). In this arrangement, the independently operating paste plant can readily alter backfill rate as needed as it can be considered largely disconnected and not tied strictly to the milling rate. Additionally, mineralogical variation across the ore bodies will affect the thickened tailings density and paste backfill properties, including final placement densities, which would result in different tailings synergies and operational strategies between the underground backfill and the TSF, since sand displaces tailings from underground disposal. In order to maintain the paste backfill transport velocity range from 1 m/s to 1.5 m/s along the underground distribution system (UDS), with 6" (150 mm) pipes, the mobile PBP would be designed to process paste backfill with throughput of 100 to 170 t/h on dry solid basis.

More specifically, based on current level of testing, the Stockwork zone paste rheology ranges from weight% concentration (C_w) of 67% to 70% for 250mm to 175mm slump, whereas the MS will be at C_w of 82% – 83% solids. However, due to the paste backfill blend design at ratio of nominally 60/40, Stockwork and MS densities would be at C_w of 75% and C_w of 83%, respectively. Due to lower SG, fine clay minerals within the stockwork retains significant water in the paste, thus have a relatively high volumetric flow rate. The ratio of sand: tailings and throughput rate will be managed to suit mining needs while maintaining balanced UDS flow velocities. Nonetheless, stockwork will retain higher water in the fill, which results in a greater percentage of stockwork tailings being displaced to surface. Tailings storage to surface based on placed backfill densities with 60% sand blend are presented in Table 18.12.

Table 18.12 – Tailings Synergies and Placed Densities for Paste Backfill Blend (Sand/Tailings)

Description		Placed Densities (t/m³)			Tailings Synergy Per Orebody TSF/Backfill	
Ore Type	Paste Slump (mm)	Wet RD	Dry RD	Water	%Surface	%UG
MS Fresh	175	2.94	2.44	0.50	70	30
MS Fresh	250	2.88	2.36	0.52		
MS Trans	175	2.81	2.33	0.48	70	30
MS Trans	250	2.75	2.25	0.50		
Gossan	175	2.16	1.63	0.53	100	0
Gossan	250	2.09	1.53	0.56		
Stringer	175	2.25	1.84	0.41	85	15
Stringer	250	2.22	1.79	0.43		
Stockwork	175	1.97	1.48	0.49	85	15
Stockwork	250	1.97	1.48	0.49		
Sandfill	N/A	1.91	1.76	0.15	-	-

18.11.6 PROCESS DESCRIPTION AND PLANT DESIGN

18.11.6.1 PLANT LOCATIONS

The Mobile Paste Backfill Plant (PBP) locations for South and North ore bodies and Thickening Plant location are presented in Figure 18.41 and Figure 18.42. North and South locations for the mobile PBP are positioned about 650 m north and 300 m south, respectively from the tailings dewatering plant. Both locations for mobile PBP are well-situated to provide gravity transport from their surface locations to underground stopes, eliminating the need for paste pumps.

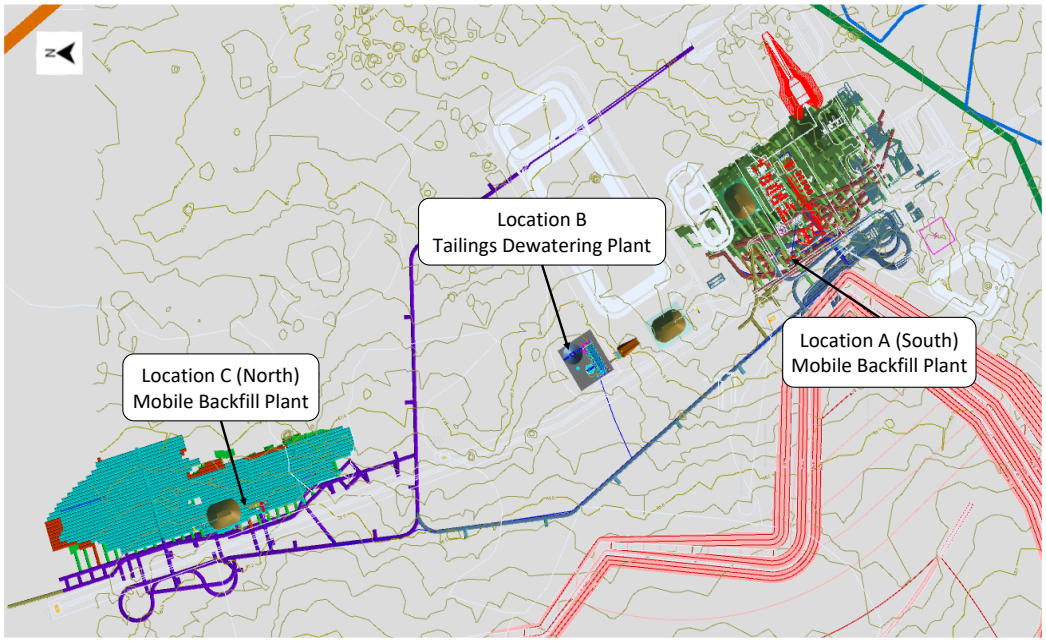


Figure 18.41 – Paste plant locations evaluated in this assessment (Top View)

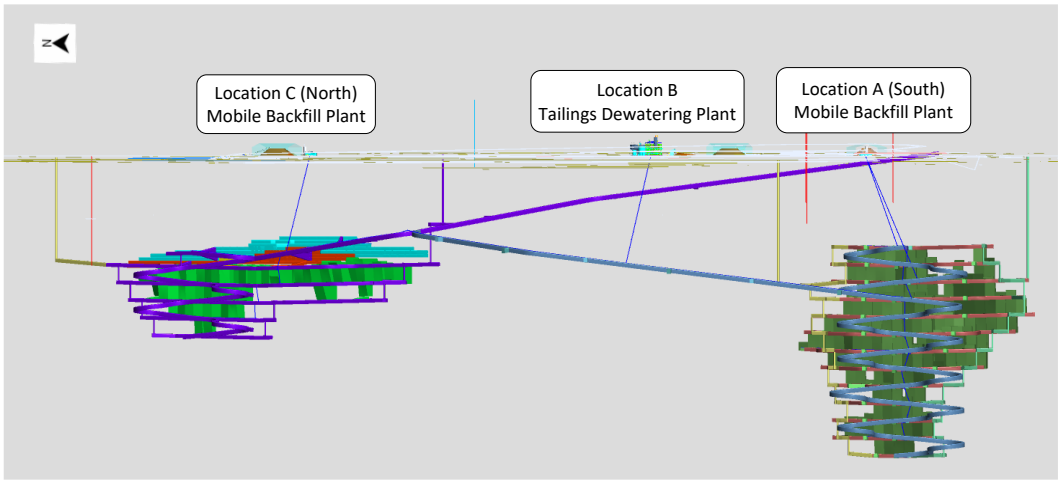


Figure 18.42 – Paste plant locations evaluated in this assessment (Isometric View)

18.11.6.2 PROCESS DESCRIPTION

The process flow diagram (PFD), mass balance, equipment list, motor list and overland piping list can be found in paste backfill design report (RMS 2023). The full plant tailings (FPT) discharged from concentrator would be dewatered by paste thickener (PT) installed between North and South orebodies as shown above (Fig. 5.4). Concurrently, the overflow process water would be discharged to process water tank, hence it will be delivered to the process water pond (on the TSF), for recycling into the mill (dependent on dissipation of reagent). Backfilling will cycle between North and South orebodies as needed, depending on the orebody that is mined and requires backfill.

When the MS Fresh and MS Transition ore is processed, the PT would densify the respective FPT at C_w to approximately 77%, that would be stored in the agitated storage tank (nominally minimum of 8hr live storage, plus dead volume). All pumping of tailings to the TSF or the backfill plant will be managed following the agitated storage tank to mitigate drawing down the thickener and negatively impacting underflow density.

The agitated storage tank would decouple the PBP from the process plant and provides a minimum 8hr steady backfilling, should the process plant experience operating difficulties or unexpected malfunction occur. From the agitated storage tank, one portion of the thickened tailings would be pumped via 650 m long double-walled HDPE pipeline, that comprise 4" (100 mm) DR 11 inner pipe and 8" (200 mm) DR 17 outer pipe, to each agitated storage tank locations at both North and South Zones. This double-wall containment of the tailings transfer pipeline would ensure the safe transport of the thickened tailings and prevent potential environmental spillages. Agitated storage tank at each location would act as a transfer point with 1hr storage capacity. The system of two agitated storage tanks provides approximately 9hr storage capacity in total. Concurrently, the process water would be delivered from the process water tank at the thickening plant via 16" (400 mm) HDPE DR 13.5 to process water tank at North/South locations as needed. The sand would be delivered by trucks and stockpiled in a covered containment system to mitigate dusting. The sand stockpile required is estimated at 12,000 tonne capacity providing a steady backfilling operation for 2-3 days continuous filling A single front-end-loader (FEL) would be used for feeding the mobile PBP with sand. The sand would be added to the receiving bin on to a weigh conveyor to feed a controlled rate of sand to the twin shaft mixer to be blended with the tailings, binder, and process water to the desired backfill recipe. All three components: receiving bin, weigh conveyor and twin shaft mixer will be mounted on a single trailer that will be moved to North and South location as needed. A stationary binder silo would be set-up at each location, along with process water tanks, flush systems, compressed air and C/A receiver to push the residual flush water through the piping system and pumps to support backfill preparation. Additional binder storage can be provided using guppy/pig would be added along with a blower to support the binder consumption should road transport by the supplier be restricted.

When backfill is required, the thickened tailings would be delivered by centrifugal pumps from the 1 hr agitated storage tank to the twin shaft mixer via the double-walled pipe. Based on an established backfill recipe, the process water, sand and binder would be added concurrently to twin shaft mixer and mixed with thickened tailings. Prepared backfill mixture at a targeted binder and slump with corresponding density would be discharged from twin shaft mixer to borehole and transported by gravity via UDS to targeted stope location, within North or South orebody.

In parallel with backfilling operation and during the TSF's Phase 1 operating stage, the remained portion of the thickened tailings would be pumped continuously from agitated storage tank at thickening plant to TSF, by centrifugal pump and 1.2 km long overland pipeline. This tailings disposal pumping arrangement would be constructed in accordance with TSF developed over time in phases and time investment management basis. The 1st

centrifugal disposal pump would be installed at the project onset (TSF's Phase 1) along with overall support i.e., electrical, concrete etc. for the entire pumping arrangement. The thickened tailings transfer pipeline would comprise 6" (150 mm) HDPE DR 9.0 (TSF's Phase 1) that would be installed from the thickening plant to TSF crest (disposal points). In accordance with TSF Phasing, the first section of the tailings transfer pipeline, that spans from the thickening plant to TSF would be replaced by 4" (100 mm) Sch 40 pipe, while the other section that runs over the TSF crest would be the same 6" (150 mm) HDPE DR 9.0 The thickened tailings transfer pipeline route for Phase 1 to 2 and Phase 2 to 6 is presented in Figure 18.43

As the TSF progresses according to the phases shown in Figure 18.44, an additional two centrifugal pumps (2nd and 3rd) arranged in series would be added to the existing tailings disposal pumping trains to make an ultimate configuration that consists of 3 pumps installed in series. The entire configuration would consist of 1 operating and 1 standby tailings disposal pumping train. This pumping configuration would be able to meet the TSF's Phase 6. In accordance with this the tailings transfer pipeline 6" (150 mm) HDPE DR 9.0, installed on the TSF's crest would be expended with an additional length of about 2.1 km.

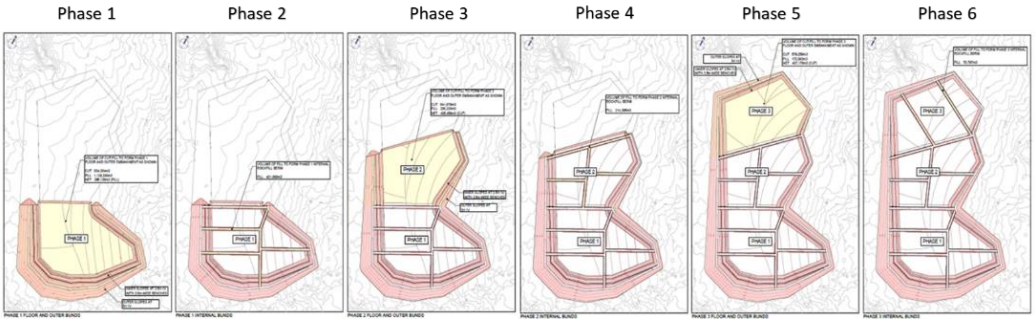
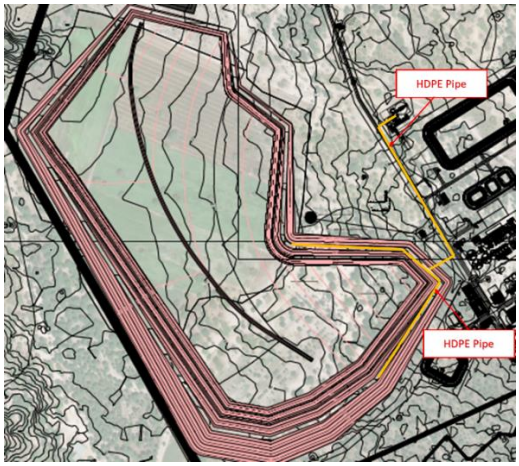
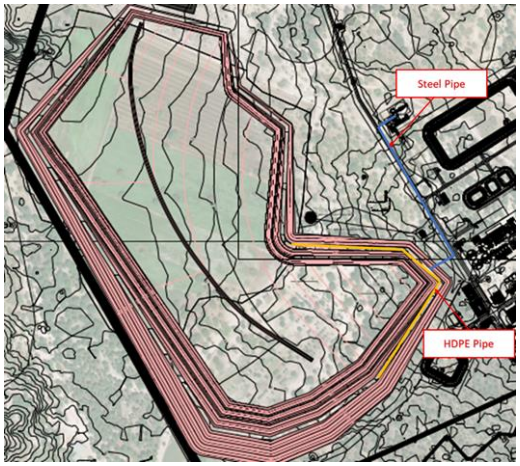


Figure 18.43 – TSF Phasing Provided by SLR Consulting





- **TSF Phase 1 - 2**
- **TSF Phase 2 - 6**

Figure 18.44 – Tailings Transfer Pipeline Route – TSF Phase 1

When the Stringer and Stockwork ore is processed, the backfilling procedure would be executed similarly to that above-described strategy. Before backfilling operation

commences, the mobile PBP would be moved from the North to the South location and prepared for paste backfill preparation.

In this case, the PT at the thickening plant would densify the thickened tailings at C_w ~63% and stored in the main 8hr agitated storage tank. From this tank, the tailings would be split with one portion of the thickened tailings pumped via 400 m long double-wall containment HDPE pipeline, that comprise 4" (100 mm) DR 11 inner pipe and 8" (200 mm) DR 17 outer pipe, to the smaller 1hr agitated storage tank at the South location. This double-walled containment tailings transfer pipeline would ensure the safe transport of the thickened tailings and prevent potential environmental spillages. At the same time, the process water would be delivered from the process water tank at the thickening plant via 16" (400 mm) HDPE DR 13.5 to process water tank at South location. The same sand storage and dosing system described for North zone (MS Fresh and MS Transition ore body), would be applied at the South zone (Stringer and Stockwork ore bodies). Also, the sand containment would be built at the South location to accommodate 12,000 tonne storing capacity. The same FEL used at North location, would be used for feeding the mobile PBP at the South Location.

When backfill is required, the thickened tailings would be delivered by centrifugal pumps from agitated storage tank to twin shaft mixer and mixed with the other ingredients: sand, water and binder as per targeted backfill recipe. Prepared backfill mixture at a targeted density would be discharged from twin shaft mixer to borehole and transported by gravity via UDS to targeted stope location, within South orebody.

In parallel with backfilling operation and during the TSF's Phase 1 operating stage, the remaining portion of the thickened tailings would be pumped continuously from the main agitated storage tank at thickening plant to TSF, by a single centrifugal pump (TSF Phase 1) and the same tailings transfer pipeline described above for the MS Fresh and MS Transition tailings (Figure 18.44).

The Gossan ore will undergo cyanide leaching (CIL) to recover gold, and not go through flotation processes as it retains a small amount of economic base metals. The testing program is completed and major mineral constituents such as quartz, hematite, goethite were identified. Goethite and hematite are oxide zinc minerals that would affect the backfill strength. In this case, Gossan ore body would be backfilled with sandfill alone, while the tailings would be disposed on the TSF impoundment.

However, should the next testing campaign proves that the Gossan tailings do not contain detrimental oxide minerals and the backfill strengths are acceptable, then the CIL tailings would undergo cyanide destruction before processed for backfill. The arrangement of the UDS readily enables gravity flow of high-density sandfill (HSF). It was estimated that the HSF at C_w ~ 73% to 75% would be sufficient for backfilling purposes. In some cases, the authors have used HSF as high as 83% at other operations, however, this is in a well-controlled system with relatively easy direct piping to stopes. Detailed investigation and careful phased implementation would be needed if this has economic and logistics value.

The thickened tailings would be processed through the same thickening plant and continuously pumped to the TSF by the same tailings disposal pumping arrangement and

tailings transfer pipeline for Stringer and Stockwork tailings. It would be also possible for Gossan tailings to be pumped via pipeline reserved for MS Fresh and MS Transition tailings, which is dictated by solid tailings throughput.

18.11.7 PASTE BACKFILL UNDERGROUND DISTRIBUTION SYSTEM

RMS has evaluated three paste plant location (Figure 18.42) listed in the Table 18.13 and modelled the UDS that will be used to service paste backfill to the stopes based on the LOMP presented in this report.

Table 18.13 – Options Considered for UDS

Plant Location	Description	Mine Area
• A	Mobile Paste Backfill Plant (Option 2)	South Zone
• B	Future Permanent Paste Backfill Plant (Base Case)	Center Zone
• C	Mobile Backfill Plant (Option 2)	North Zone/Mill

The paste plant location 'A' and 'C' are temporary installations with a mobile paste backfill plant (Option 2), and location 'B' will be the thickening plant which will provide Ascendant the flexibility to install a permanent plant (Base Case) if needed in the future. The location of Option B, over the ore zone, as well as the Concentrator positioned directly over the South Zone, is within the scope of other team members evaluating the crown pillar stability. RMS has recommended other locations; however, the footprint is limited.

The steady state hydraulic analysis only considers the farthest deposition points from borehole breakthrough to the stopes on key levels of the mining areas to assess the expected duty head and paste density (slump operating range 175 to 250 mm slump) operating envelopes when operating the UDS as pumped or gravity system.

The UDS was modelled with DN150 (6-inch) carbon steel schedule 80 and schedule 40 paste piping based on ASME B36.10 dimensions. The pressure ratings for each pipe type were calculated according to ASME B31.4, including coupling type and reasonable allowances for wear. The nominal flow rate of 75 m^3/h (±12%, 66 to 84 m^3/h) will promote suitable filling ratios and productive mining rates while maintaining a velocity envelope of 1 to 1.5 m/s.

Considering that the mobile backfill plant will have the flexibility to move either South or North orebody, RMS evaluated the locations A and C so that a gravity system be used to deliver paste to the UDS. No pump has been specified for this option since the available static head in the system is sufficient to reach all deposition points at nominal flow rate of 75 m^3/h over the typical paste friction losses of 4 to 8 kPa/m.

Flow models for the gravity system were developed for the two routes (South and North) as presented in Figure 18.45 through Figure 18.48. For each location, the reticulation system Hydraulic Grade Line (HGL) was analyzed to understand the flow characteristics throughout the pipeline. The maximum operating pressure and pressure rating of the system during steady state operation were assessed. The resulting flow model HGLs present the data of the UDS design considering the farthest stopes of each system operating with

relevant type of tailings (MS and Stockwork). Based on RMS experience with similar projects in Iberian Pyrite Belt, the tailings:sand blend recipe is expected to have rheological properties ranging between the two type of tailings, respectively, MS and Stockwork.

The friction losses used in the flow model are calculated based on the testwork results from Ascendant tailings performed by RMS laboratory and experience with similar systems. RMS strongly recommends Ascendant to perform flow loop tests in the next phase to measure the actual friction losses for each recipes with various tailings types, sand blend that will be produced by the mine.

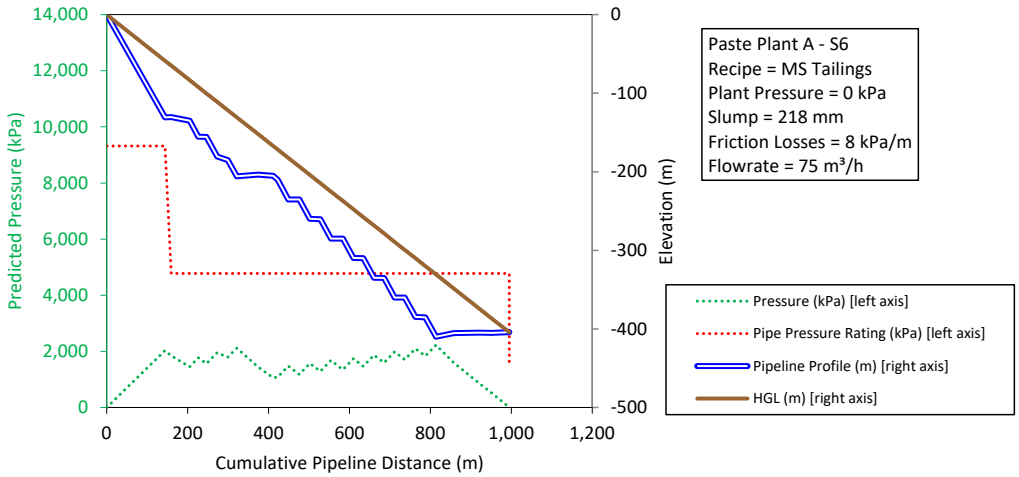


Figure 18.45 – HGL Stope S6 - MS tailings

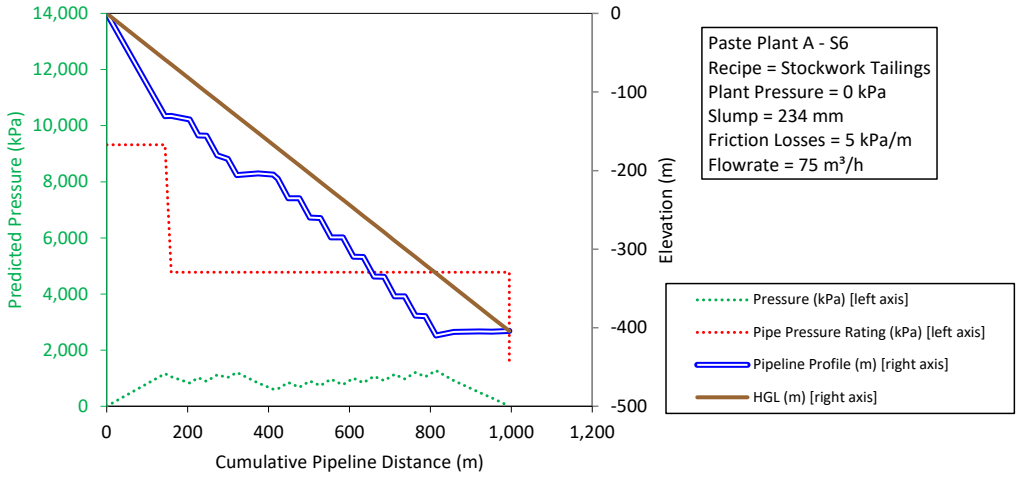


Figure 18.46 – HGL Stope S6 - Stockwork tailings

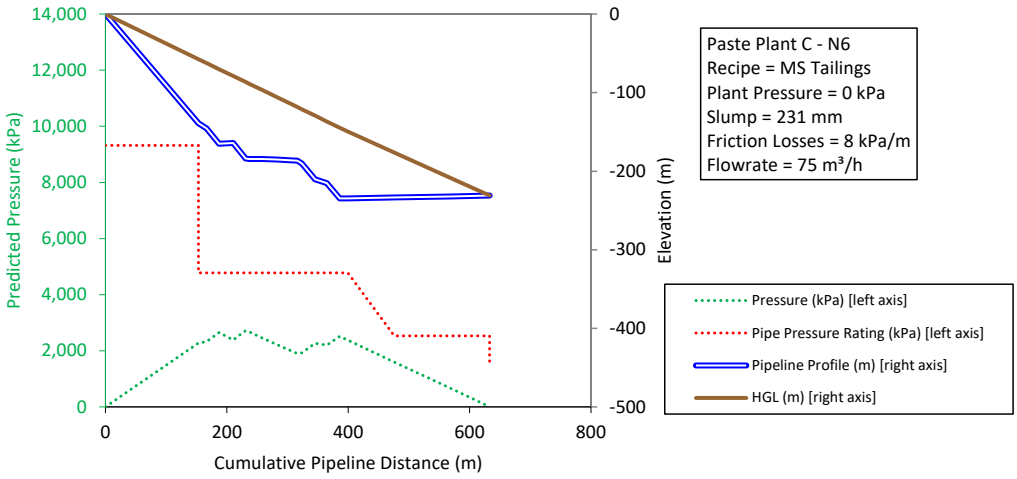


Figure 18.47 – HGL Stope N6 - MS tailings

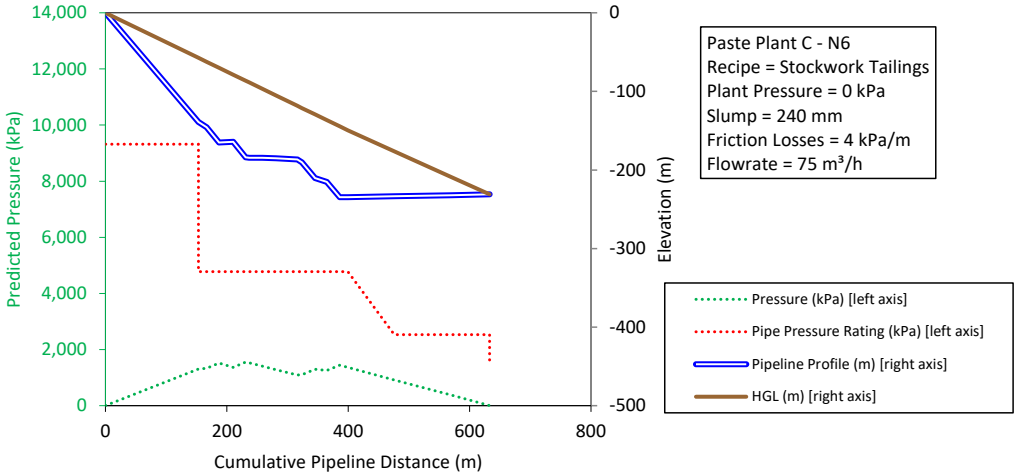


Figure 18.48 – HGL Stope N6 - Stockwork tailings

A flow meter will be installed at the discharge pipe to the surface borehole and pressure transmitters are installed at the bottom of the surface borehole which provide real-time indication of the system flowrate, pressure and friction losses.

To protect against over-pressurizing the pipeline, rupture discs or dump valves or a combination of both are recommended to be installed near the base of surface borehole and key levels of the mine depending on the pressure rating requirements of the UDS. RMSCorp. recommend performing a careful analysis of over-pressure devices and key locations of each system in the next phase.

Distribution of paste to the various underground levels will be performed by manual switchovers on each level from the main piping system with a combination of carbon steel and HDPE pipes. Manual switchovers, or swap-out spools, can be used to direct the flow into the required stope areas.

19 MARKET STUDIES AND CONTRACTS

The Lagoa Salgada Project will produce a gold/silver dore and [4] saleable metal concentrates including:

- Zinc concentrates.
- Copper concentrates.
- Lead concentrates.
- Tin concentrate.

Annual production of each concentrate produced will represent a small portion of supply in a large and liquid global market with transparent daily market pricing. As such, production from Lagoa Salgada is expected to be freely delivered into the market with minimal impact on overall supply/demand dynamics for their respective global markets.

The concentrates are expected to be delivered to smelters and refineries within Europe and transportation costs have been assumed on this basis. However, the concentrates are expected to be salable into any of the major markets or smelters globally and are not dissimilar to products being delivered by currently producing mines in the Iberian Pyrite Belt.

Currently, no contracts for offtake for any products have been executed. It is expected that these will be negotiated as part of future financing decisions as is normal for a project at this stage of development. However, the Company has confirmed the salability of the products with several large commodity traders and smelters within Europe. The results of these discussions have been incorporated into this study, specifically as it relates to market terms for payability, by-product credits, treatment and refining charges and penalty rates for any deleterious elements.

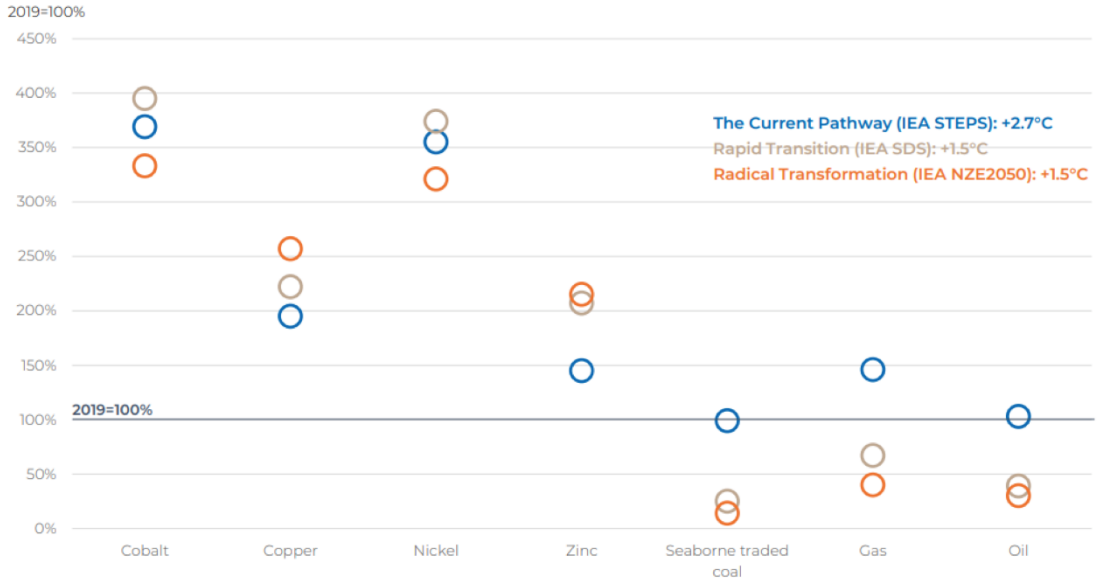
19.1 MARKET OUTLOOK

While global growth is currently predicted to slow due to increasing interest rates, higher energy prices and a strong dollar, the medium to longer term outlook for metals demand remains extremely robust. Combined with current low levels of metals inventory and the slow buildout of new supply as well as the continued and global push towards decarbonisation suggest the emergence of another period of strong demand and thus prices for commodities.

According to the International Energy Agency (“IEA”), *“a concerted effort to reach the goals of the Paris Agreement (climate stabilisation at “well below 2°C global temperature rise”, as in the IEA Sustainable Development Scenario [SDS]) would mean a quadrupling of mineral requirements for clean energy technologies by 2040. An even faster transition, to hit net-zero globally by 2050, would require six times more mineral inputs in 2040 than today.”*

The world's major economies appear to be supporting the transition and policy changes and commitments appear to be accelerating. China, the world's largest greenhouse gas emitter, recently pledged to become carbon neutral before 2060 with an accelerated peak greenhouse gas emissions target before 2030. The target for peak emissions was previously 2040. The European Union also pledged to cut its greenhouse gas emissions by 55% by 2030. Under the Biden administration, America rejoined the Paris Agreement, in April 2021, and pledged to become a net zero emissions economy by 2050. Moreover, the country's Inflation Reduction Act is anticipated to support energy transition investment.

Glencore plc. a major producer and one of the largest metals producers globally anticipates extreme demand increases over the coming decade depending on the pace of the transition. Figure 19.1 below outlines that base metals demand for nickel, copper, and zinc are expected to increase by ~325%-375%, ~200%-260%, and ~150%-225% by 2050 as compared to 2019 levels driven by various global energy and emission scenarios.



Source: Glencore plc

Figure 19.1 – Commodity Demand in 2050 vs. 2019 Under IEA Global Energy and Emissions Scenarios

The above, coupled with historic underinvestment in the minerals sector and long lead times from discovery of new deposits to first production suggest a looming supply gap across the metals sector.

A recent article by S&P Global Market Intelligence concluded that the average lead time from the discovery of a deposit to the start of production is over 15 years. Increasing regulations are expected to maintain long lead times for development.

Source: S&P Global Market Intelligence, 2023

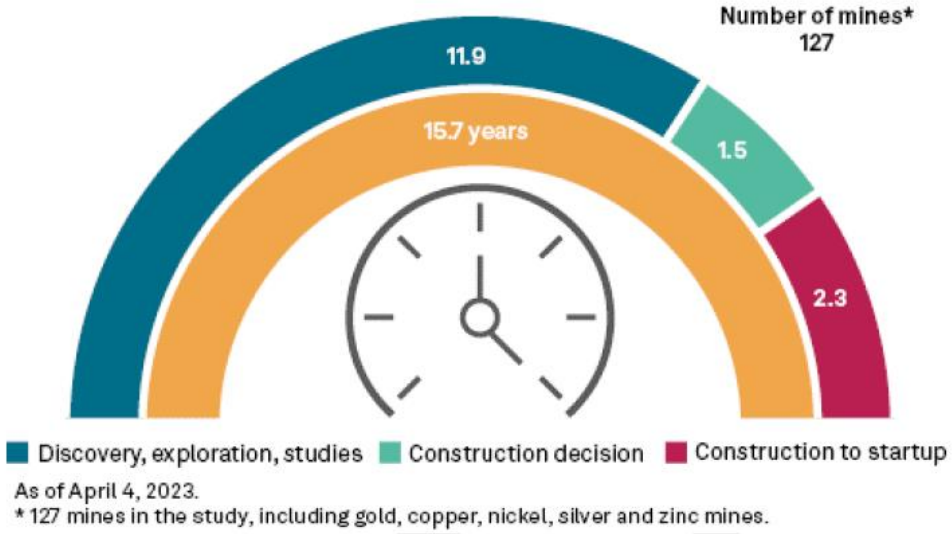
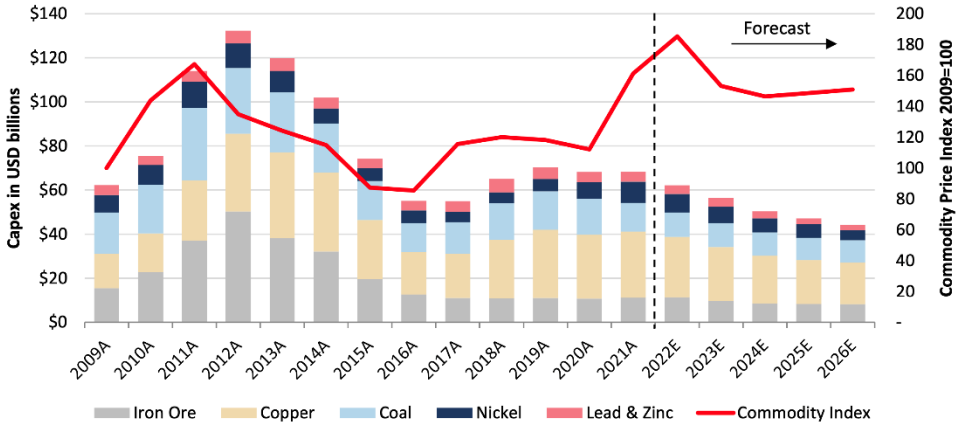


Figure 19.2 – Average lead times of mines from discovery to production, 2002-2023

Investment in new production remains subdued and is expected to continue. A recent report from Scotiabank highlights investment levels remain well below that in previous cycles despite strong commodity prices. Additionally, there is a lack of "shovel-ready" projects available to provide a near-term supply response across the commodities sector. "*We note that total planned mining capex (growth and sustaining) in the 2021A-2026E period of $54.8 billion per annum remains 35% below the 2010-2020 average of $84.7 billion per annum despite elevated commodity prices (average growth spending is down 70% on a standalone basis). Although ongoing inflationary pressures are likely to further elevate these estimates, we anticipate mining capex to remain relatively subdued in the years ahead.*"



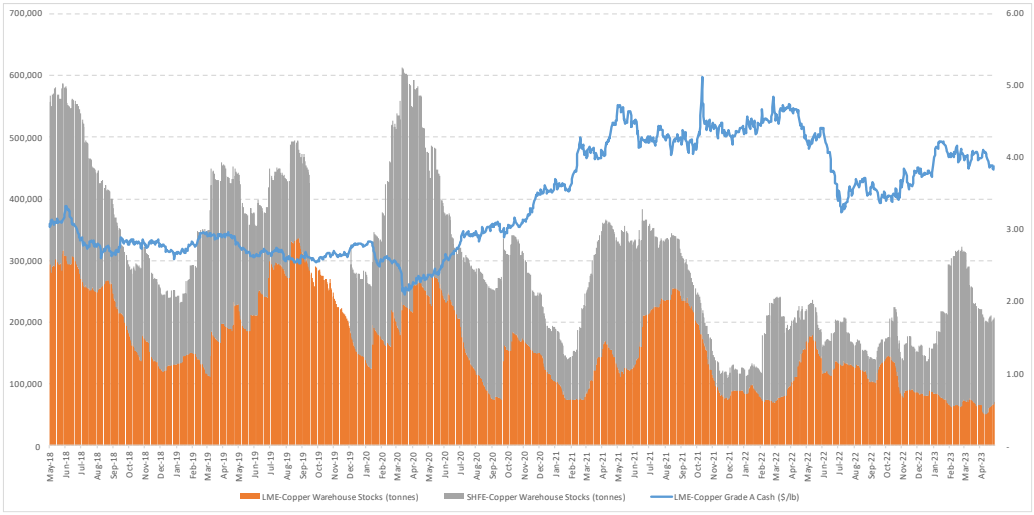
Source: Scotia Bank Global Equity Research, January 16, 2023.

Figure 19.3 - Total Planned Capex by Commodity vs. Equal-Weighted Commodity Price Index (2009-2026E)

A lack of new discoveries, declining grade profiles, underinvestment and long lead times suggest a significant supply gap across the metals sector. Under even moderate demand growth, this would normally be sufficient to drive price increases. Couple this with the drive for electrification and the expectations is for a robust and sustained price response.

19.1.1 COPPER SUPPLY AND DEMAND

Despite near-term slowing demand due to expectations for slower global growth the copper market and prices have held up primarily on the back of supply-side challenges. Visible LME inventories have hit all time lows in 2023 and are currently at critical levels. In the medium to longer term the analyst community and industry are predicting a global supply shortage in the copper market, driven by both supply-side disruptions from major production centres in South America and increasing demand from the energy transition. This is expected to support above average prices.



Source: S&P Capital IQ.

Figure 19.4 - Historic Copper Price and Warehouse Inventories (5 years)

Currently, China represents approximately 54% of global copper demand (although ~15% of production is exported) and as China emerges from lockdowns due to Covid-19 an increase in demand and consumption is expected to rebound towards the later part of the year. Continued investment in the national grid and China's "three arrows" policy announced in November 2022 is expected to drive a recovery in the local property market both of which are expected to drive moderate growth in copper consumption from the largest consumer.

In the more medium to longer term, global decarbonisation efforts are viewed as extremely bullish for copper demand. Demand from the auto sector currently comprises ~9% of all copper use, this however, could add an incremental 1.5 million tonnes of demand by 2025

and 2.7 million tonnes by 2027, according to Wood Mackenzie. In 2021 total refined copper demand was just under 25 million tonnes.

On the supply-side, the copper market is supplied principally from new production and is dominated by production from a few key countries, namely Chile, Peru, China (a net importer) and the Democratic Republic of the Congo. Supply risks from increasingly nationalist policies in South America, principally Peru and Chile coupled with low levels of capital investment in the sector generally point to a muted supply-side response. In fact, most of the expected supply-side growth is coming for expansions contemplated prior to 2019 such as the Grasberg block cave, Quebrada Blanca II and Kamoa-Kakula Complex. The dearth of new projects globally means incentive prices will likely remain well above the cost curve longer.

The copper price been propelled to all-time highs over the last year reaching a record of US$4.87 in March 2022. It is expected that prices will remain high over the medium term as highlighted by analyst forecasts below.

Table 19.1 - Analyst Consensus Copper Forecasts

Copper *(US$/lb)*					
Date	**2023**	**2024**	**2025**	**2026**	**LT**
Analyst Consensus	**$3.83**	**$3.91**	**$4.12**	**$4.15**	**$3.71**

Source: CIBC Global Mining Group, Analyst Consensus Prices, April 2023

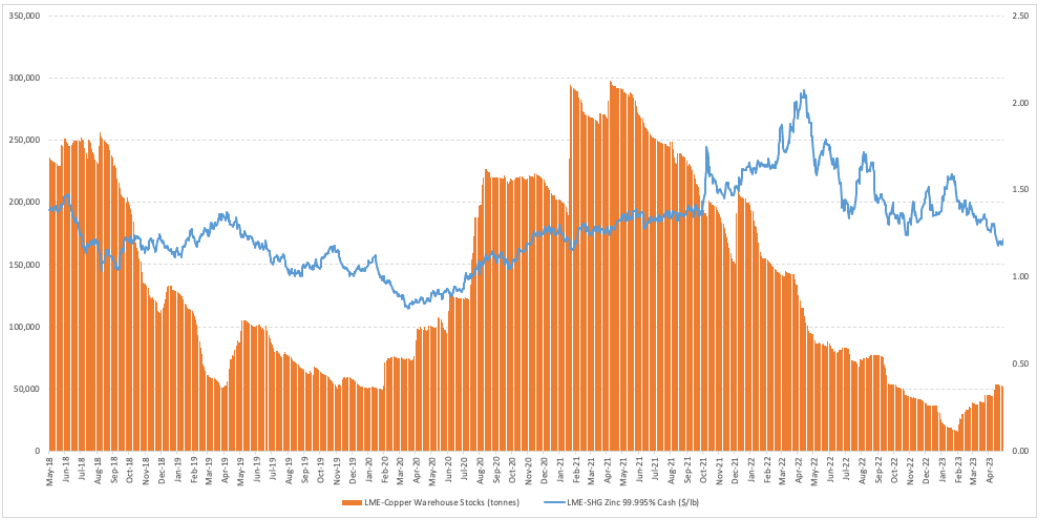
19.1.2 ZINC SUPPLY AND DEMAND

Currently the zinc market remains in deficit, ongoing smelter capacity issues in Europe due predominately with elevated energy prices as a result of the Russian invasion of Ukraine has sidelined much production capacity. In the medium term, as this production returns minor surpluses may result, but this is not likely to outweigh the structural deficiency in primary mine supply. As a result, longer elevated pricing is expected in the medium and longer term.

Primary demand for zinc is dominated by its use in the galvanisation process through the construction and automotive industries. Other end users consist of the rubber, chemical and agricultural industries.

China is currently the largest producer of mined zinc representing approximately 34% of global production. From 2015 through 2020 Chinese zinc production slowed as a result of depleting reserves, a lower commodity price and a country-wide focus on improving safety conditions. Despite recent increases in mine production from China in 2021 and 2022 the market remains relatively opaque and continued growth is uncertain. Non-Chinese production growth seems to have peaked over the 2019-2021 period as idled capacity and expansions were brought online as a result of price strength in 2017-2019 period.

In near-term growth will be supplied by several ongoing expansions and greenfield starts, including Aripuana, Century Tailings, Gamsberg, Neves Corvo and Zhairem. It should be noted however, that the risk of ongoing Russian sanctions likely delays development of a portion of production growth. Additionally, as major producers and markets have been in favour of commodities such as copper and lithium, investment in large greenfield zinc projects has lagged.



Source: S&P Capital IQ.

Figure 19.5 – Historic Zinc Price and Warehouse Inventories (5 years)

Table 19.2 - Analyst Consensus Zinc Forecasts

Zinc *(US$/lb)*					
Date	2023	2024	2025	2026	LT
Analyst Consensus	$1.39	$1.34	$1.28	$1.27	$1.19

Source: CIBC Global Mining Group, Analyst Consensus Prices, April 2023

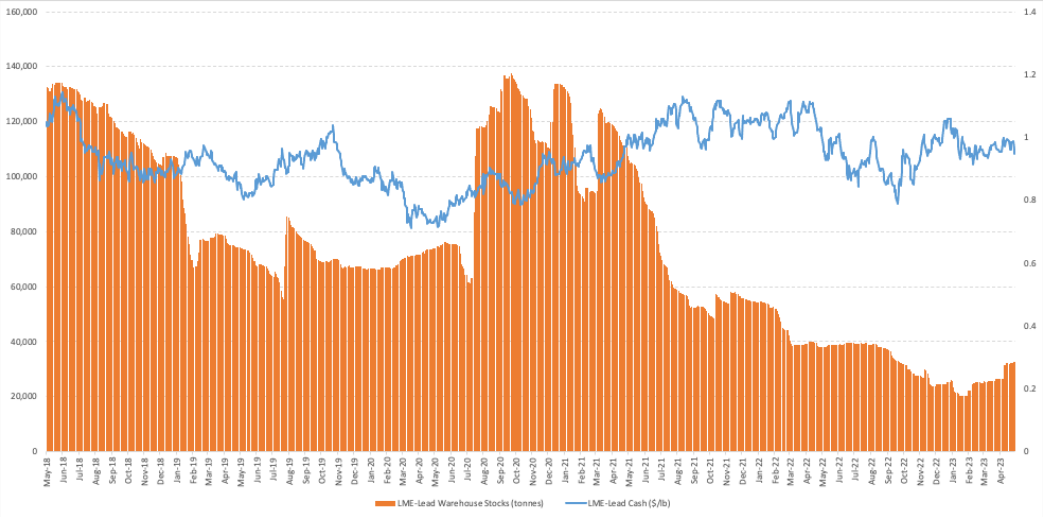
19.1.3 LEAD SUPPLY AND DEMAND

Lead demand is driven primarily by global battery demand. With automotive production recovering post-lockdown, demand has remained stable in recent years. China's increase in demand in 2021 suggests that the global recovery will continue, but there are some doubts about the true strength of Chinese lead consumption in the coming years. The demand for Chinese batteries looks to be strong, and this trend is potentially driven by export markets. That said, with the transition to new EV's globally the potential for a slow decline in demand is expected as this transition occurs.

In terms of supply, lead mine production is not expected to not fully rebound to pre-pandemic levels for a couple of years due to lack of new mine development and uncertainty over the longer-term demand profile. Global stocks are still low due to deficits accumulated

in the past few years, meaning the total global stocks will only revert to average levels. It is thus expected that the market will not be awash with lead, and that the lead concentrate market will continue to be tight. Given that the bulk of lead supply is driven by the development of associated zinc mines growth, new supply is expected to be seen as slow for zinc supply growth.

Treatment charges for lead concentrates are already low, and they are likely to remain so as the capacities of primary smelters exceed the available mine supply. Additionally, the lead recycling industry – which produces two-thirds of total global output – is heavily dependent on scrap batteries but cannot do much to influence the rate of supply.



Source: S&P Capital IQ.

Figure 19.6 - Historic Lead Price and Warehouse Inventories (5 years)

Table 19.3 - Analyst Consensus Lead Forecasts

Lead *(US$/lb)*					
Date	2023	2024	2025	2026	LT
Analyst Consensus	$0.98	$0.95	$0.94	$0.95	$0.95

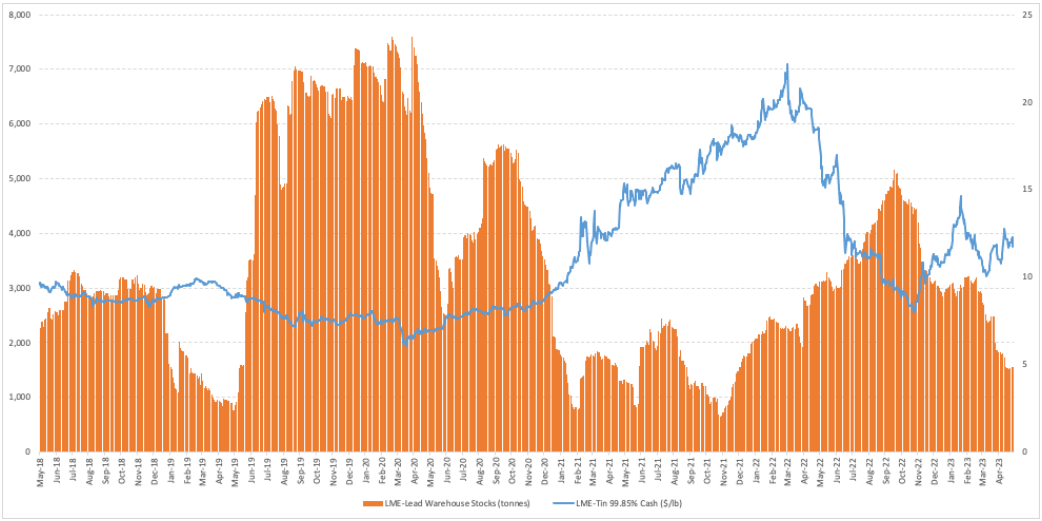
19.1.4 TIN SUPPLY AND DEMAND

In recent years, the tin market has been faced with a consistent supply deficit, which is forecast to continue as a result of increasing regulations in producing countries and the depletion of ore reserves. Ample opportunities exist for tin demand to grow, which will exacerbate the already existing demand-supply gap. Globally, most pipeline projects for tin production are not expected to commence in the near future.

The future of tin within the electric vehicle and battery storage markets is bright, which will help to drive demand. Its role in the electric vehicle market is growing and that demand

remains strong for electronic solders. Another reason for the growing demand is the use of tin in lithium-ion batteries. It is clear that the versatility of tin is on the increase, which bodes well for the price and the market outlook going forward. One other favourable aspect is that tin's "cost in end-use" is typically small, meaning demand is relatively inelastic to prices which also limits substitution risk.

Tin supply is fairly concentrated, as China remains the world's largest producer followed closely by Indonesia and Malaysia. The country has also been the main driver of demand for the metal for many years. While demand for tin remains stable, supply issues are expected to likely cause the price to rise.



Source: S&P Capital IQ.

Figure 19.7 - Historic Tin Price and Warehouse Inventories (5 years)

19.1.5 PRECIOUS METALS SUPPLY AND DEMAND

The Lagoa Salgada project is expected to produce gold and silver both as a by product in concentrate production, and from the production of Gold & Silver dore.

Historically, gold is a haven for investors during turbulent times as seen by the recent strength in prices which are now at historic highs. This recent strength is a result of global geopolitical events, rising interest rates and uncertainty of over economic growth.

Recently, Macro asset allocators have shown signs of a pivot back to both Gold with increasing inflows into ETFs. Central bank buying has also seen a recent uptick with the Chinese, Singapore and Turkey all making additions year-to-date.

Similar to gold, silver prices have remained strongly driven by rising industrial and jewellery demand as the global economy recovers but also from increased investment demand. Silver is used in a variety of industrial applications including applications in the production of solar energy, while jewellery still accounts for about one-third of consumption. Whereas demand

from industrial users usually does not fluctuate very much, the next four years are likely to see a substantial increase due to new green policies. As the single most conductive metal, Silver is a key component in many green technologies, and if industrial demand grows as expected, the silver price is likely to increase.

Figure 19.8 – Historic Gold & Silver Prices (5 years)

Table 19.4 – Analyst Consensus Precious Metals Forecasts

Date	2023	2024	2025	2026	LT
Gold (US$/Oz)	$1,831	$1,840	$1,794	$1,763	$1,672
Silver (US$/Oz)	$22.93	$23.43	$22.86	$22.53	$22.11

Source: CIBC Global Mining Group, Analyst Consensus Prices, April 2023

19.2 SMELTER LOCATION AND SALES INFRASTRUCTURE

19.3 PRICE FORECASTS

For the purposes of this study, we have assumed Base Case commodity prices in line with long term consensus from the analyst community. Table 19.5 below summarises the analyst commodity price estimates at the time of writing the report, the prices assumed for financial modelling and the spot price for each of the principal commodities of the Lagoa Salgada Project.

Table 19.5 – Price Forecasts and Assumptions

	Units	Long Term[1]	Spot Prices[2,3]	Modelled Price
Copper Price	US$/lb	$3.71	$3.87	$3.75
Zinc Price	US$/lb	$1.19	$1.18	$1.20

	Units	Long Term[1]	Spot Prices[2,3]	Modelled Price
Lead Price	US$/lb	$0.95	$0.96	$1.05
Tin Price	US$/lb	N/A	$12.20	$15.00
Silver Price	US$/oz	$22.01	$25.39	$22.50
Gold Price	US$/oz	$1,672	$2,008	$1,700

Source: 1. CIBC Global Mining Group, Analyst Consensus Prices, April 2023. 2. LME cash contract bid price dated 5 May 2023. 3. Kitco.com dated 5 May 2023.

19.4 CONCENTRATE SALES AND PRICING

Concentrates produced at Lagoa Salgada are expected to be sold directly to European smelters and refineries or through a third-party offtake arrangement. At this time, no formal contracts for concentrate sales have been entered into. Concentrates are planned to be trucked to a local port for shipment to smelters. As such a fixed US$35/t transport cost per tonne of concentrate has been assumed for Lead and Copper and US$60/t for Zinc.

Concentrates are sold to end users net of treatment and refining charges (TC/RC'S) on payable contained metals and transport costs to deliver the Net Smelter Return per tonne of concentrate shipped.

Penalties could be levied for several reasons, including deleterious elements, concentrate grades that are out of the agreed specification range and concentrate moistures that are outside of the agreed specification range. Iberian Pyrite typical elements that may be at risk of penalties include Arsenic and Mercury. A blending strategy is planned to be used at Lagoa Slagada, particularly with Zinc concentrates, to reduce the probability that concentrates specifications fall out of the limits that would attract penalties.

19.4.1 PAYABILITY AND BY-PRODUCT CREDITS AND TREATMENT AND REFINING CHARGES

The company has engaged in detailed discussions with several commodity traders and smelters to arrive at market estimates for payability rates for the metals contained in the concentrates as well as market rates for treatment and refining charges.

The assumptions used in this study for NSR calculations and financial modelling based on life of mine average concentrate grades can be found on Chapter 22.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

To comply with the established in Canadian Norm NI-43-101, an environmental section was considered in the present DFS, however it should be noted that, under the Portuguese Legal Environmental Regime applicable, the Environmental Impact Assessment is still ongoing and its main report can only be delivered to the authorities after the DFS (that, in this particular case, is the Project that will be evaluated during the Environmental Impact Assessment Procedure) is concluded.

In this context, the next sections introduce the environmentally applicable information available at this stage of the Portuguese environmental process of the Lagoa Salgada Project.

20.2 SITE SETTING

The project area lies in the Alentejo Region, one of the five regions of the Portuguese mainland.

Administratively speaking, the Project of Lagoa Salgada will be implanted on land belonging to the Municipality of Grândola as referred to in Chapter 4 of the DFS – Property Description and Location.

The study area (AE) considered for analysis in the EIA was much wider than the area where the masterplan of the Project will be implanted. Effectively, the minimum study for the EIA includes:

$$\text{EIA Study Area } (AE_{EIA}) = AEb + C.Ad.Ag. + C.Ac.P + C.Ac.Sec. + C.LE$$

Where:

AEb – includes the Masterplan implantation area for the Project plus a buffer of 500m.

C.Ad.Ag. – environmental corridor of 200m wide centred on the water supply main pipeline intervention area.

C.Ac.P - environmental corridor of 200m wide centred on each option of the main access road.

C.Ac.Sec. - environmental corridor of 100m wide centred on the secondary access road.

C.LE - environmental corridor of 400m wide centred on the electric power line layout.

Additionally, for some environmental factors under analysis, the study area was even bigger regarding, for instance:

- Water resources – where all water basins communicating with the project area were considered;
- Landscape – which considered a buffer of 3km outside the project area;
- Noise and vibrations - which consider all sensitive receptors that can potentially be affected by noise and/or vibrations caused by the project;
- Social component – which considered an integrated local, regional and national analysis of the project's potential effects.

With this approach, a correct environmental assessment of the Project can be guaranteed.

20.3 ENVIRONMENTAL AND HERITAGE BASELINE STUDIES

The Lagoa Salgada Mine Project, under development in a Preliminary Study phase, was subject of an Environmental Impact Assessment (EIA) Scoping Proposal ("Proposta de Definição de Âmbito" - PDA) conducted in 2019, based on the Preliminary Economic Assessment (PEA) of that year, which analysed two mineral deposits. In 2021, the decision was made to proceed with the exploitation of the Lagoa Salgada deposit, and a new PDA was submitted based on a new PEA, and approved by the Portuguese Environmental Agency (APA, I.P). The ongoing environmental studies are based on the study areas and methodologies proposed in 2022.

The study area (AE) has undergone desktop analyses and field visits to characterize the current situation of the project area and the surrounding region.

In addition, various field monitoring campaigns have been conducted for environmental factors, including:

- **Air Quality** - PM10/PM2.5 Monitoring - Eight monitoring campaigns are planned between February and December 2023. As of now, four campaigns have already been conducted (Feb, Mar, Apr, May).
- **Soil** - Site visits were conducted in February 2023 to determine the soil's chemical status in the infrastructure implementation area.
- **Noise** - Campaigns were conducted in June and November 2022.
- **Landscape** - A visit was conducted in February 2023.
- **Forest Component** - Four campaigns were conducted (Feb, Mar, May, Jun 2023) to identify and characterize forest plantation areas, inventory cork oak/holm oak trees, and conduct a forest inventory.
- **Heritage** - Visits were conducted in January and February 2023.

- **Ecology** - Campaigns were conducted between July and November 2022, and in February, April, and July 2023, with the main objective of identifying plant species, habitats, and vegetation, as well as fauna species.

Additionally, a **Hydrogeological** study has already been developed. Its main results are presented in Chapter 20.7.

Following, a summary of the current state of each environmental (physical and biological) factor under assessment for the project area.

20.3.1 CLIMATE

The climatic characterisation of the region was based on data from the Alcácer do Sal weather station (Climatology Normal for 1971-2000 and 1981-2010). The average temperature is 16.4ºC, with the most extreme temperatures occurring in August and January, respectively. Precipitation is mainly distributed over five months of the year (Jan-Mar and Oct-Dec). The average wind speed is approximately 7 km/h. The climate classification of the studied region according to Köppen is CsA (Humid temperate climate with dry and hot summers).

The most prominent climate risks identified are associated with drought, high temperatures, heatwaves, and concentrated rainfall in short periods of time, leading to high runoff and consequent floods and rapid inundations.

20.3.2 GEOLOGY

In terms of geomorphology, the AE is integrated into the morpho-structural unit known as the "Cenozoic Basin of Baixo Tejo," which includes the sedimentary basins of Baixo Tejo and Sado, covered by tertiary sediments with similar characteristics. Vast exceptionally flat horizons dominate this unit, corresponding to either the terminal surface of sedimentary accumulation or more recent planation levels progressively embedded within it.

The base study area is part of an extensive flattened surface that extends from the west bank of the Sado River to the vicinity of Serra de Grândola.

The AE is in the Portuguese South Zone (ZSP), one of the geotectonic units of the Iberian Massif, bordering the Ossa-Morena Zone to the north through first-order tectonic faults.

No active, certain, or probable faults or geological lineaments cross the AE.

20.3.3 HYDROLOGY

According to the Management Plan for the Sado and Mira Hydrographic Region (PGRH6), the Sado and Mira Hydrographic Region - RH6, with a total area of 12,149 km2, includes the hydrographic basins of the Sado and Mira rivers, as well as the coastal stream basins, including their corresponding groundwater and adjacent coastal waters.

The main surface water bodies covered by the AE are the Sado River (PT06SAD1278) and the Sado River Tributary (PT06SAD1272), which intersect the study corridor of the power line. Both have an overall "Good and High" quality. The AE covers the hydrogeological unit "Tejo and Sado Basin," which is also classified overall as "Good."

No groundwater catchment protection perimeters for public water supply are identified within the AEEIA or its immediate surroundings.

20.3.4 HYDROGEOLOGY

Refer to Chapter 20.7.

20.3.5 SOILS

In the study area, the pedological units are mainly composed of podzolised soils (74%), followed by incipient soils (17%), litholic soils (7%), and finally, hydromorphic soils (2%). In more detail, in the implementation area, there is a strong presence of podzolised soils (93%), compared to the remaining soils, which contribute to 7% of the pedological composition when combined. Both in the study area and in the implementation area, the dominant soil element is identified as Pz. The analysis of the potential intervention area shows the same trends, specifically, pedological units primarily characterized by podzolised soils (77%), followed by incipient soils (13%), litholic soils (6%), and finally, hydromorphic soils (3%).

Regarding land use, forests (*Pinheiro Manso* and *Pinheiro Bravo*) (63%) and agroforestry surfaces (27%), with a particular presence of cork oak agroforestry systems (23%), prevail in the AE.

20.3.6 ECOLOGY

The AE does not fall within any conservation area of interest, although some conservation areas are referenced in its surroundings, although at relevant distances greater than 10 km.

The AE is an area with a potentially significant presence of rare species, particularly endemics. According to the conducted research, a high number of species important for conservation could potentially occur in the AE, including 12 species considered threatened, three of which with the status of "Endangered (EN)" and the remaining "Vulnerable (VU)," according to the Red List of Vascular Flora of Continental Portugal. However, based on the ecology of the various species, the most relevant for conservation and with a higher

likelihood of occurrence in the AE are the following: *Caropsis verticillato-inundata* (VU), *Centaurea exarata* (VU), *Juncus emmanuelis* (VU), *Utricularia gibba* (VU), *Wolffia arrhiza* (VU), *Armeria rouyana* (NT), *Elatine brochonii* (NT), and *Juniperus navicularis* (NT).

The analysis of fauna encompasses various components, and the most ecologically relevant faunal species include:

- Amphibians: 11 potentially occurring species, with 10 classified as "Least Concern" and one as "Near Threatened."

- Reptiles: Information collected on potentially occurring reptiles in the AE allowed for listing only 12 taxa. All identified taxonomic entities are considered "Least Concern" species in Portugal, except for the Lataste's viper (Vipera latastei), which is "Vulnerable".

- Birds: After analysing various information sources, combined with a preliminary study of ecosystems and biotopes, a list of 112 potentially occurring bird species in the AE was obtained. There is no intersection of sensitive bird areas with the AE.

- Mammals: Based on the available habitats and conditions in the AE and its immediate surroundings, as well as the consulted bibliography, 28 species of terrestrial mammals have the potential to occur. According to the Red Book of Vertebrates of Portugal, most of the listed terrestrial fauna species for the AE have a status of "Least Concern" in Portugal, with some classified as "Data Deficient" (DD) or "Not Applicable" (NA). The AE is relatively close to some nationally important shelters but does not overlap with any critical shelter area. Among terrestrial mammals, notable species include Oryctolagus cuniculus, classified as "Near Threatened" (NT), and Cabrera's vole (Microtus cabrerae), which is "Endangered" (EN).

- Fish: In the AE, the existence of several fish species has been recorded, particularly in the watercourses and reservoirs in the region. However, no species of conservation interest were identified.

- Invertebrates (Lepidoptera): In the fauna list, there are 51 potentially occurring species in the AEEIA, with their conservation status. Only three species have a more unfavorable conservation status (Endangered).

From the samplings conducted for fauna and flora, it can be concluded that the most important areas are the Montado (cork oak forest), pine forest, and mat-covered cork oak ecosystems (2150*) and 2260 (Pinhal Manso), as well as the interdunal depression areas and reed beds. These areas harbor the most unique and important ecosystems in the AEEIA, as well as greater diversity in terms of fauna and flora.

20.3.7 AIR QUALITY

The air quality in the AE and its immediate surroundings is primarily influenced by emissions from road traffic and the presence of six areas of mineral exploitation (quarries) in the immediate surroundings of the AE of the EIA.

To analyse air quality in the AE region, data from the QUALAR[1] platform was used for the Rural Station de Monte Velho in the municipality of Santiago do Cacém, in the Alentejo Litoral zone. In this area, most days in the last five years (2017-2023) have had air quality indices ranging from "very good" to "good." Analyzing the levels of PM_{10}, $PM_{2.5}$, O_3, and SO_2 for the same period, although some exceedances of the legislated values were recorded (Decreto-Lei n. º 102/2010, as currently amended), it can be concluded that the limit values for the protection of human health have mostly complied with the legislation.

20.3.8 NOISE

From the assessment made, the main sources of noise responsible for the local sound environment are: natural phenomena, road traffic on EM 543, road traffic on local roads, human activities.

For the characterisation of the current situation, five acoustic assessment locations were selected near residential houses, which are considered (i) the closest and most exposed to noise emitted by the normal operation of the mining infrastructure, and (ii) representative of the local sound situation, where various recordings were made for the LAeq index. The sound environment in most locations within the influence area of the mining infrastructure - Mina da Lagoa Salgada under study - is quiet, typical of rural areas or locations far from noteworthy or relevant sound sources.

20.3.9 VIBRATION

Technical visits to the site of the mining infrastructure - Mina da Lagoa Salgada under study - did not identify significant vibrational sources, meaning that there are no sources responsible for noise transmission mechanisms that could influence the local environment or cause discomfort to the population.

20.3.10 HUMAN HEALTH

Based on the study of the age structure of the resident population in the municipalities where the project is located, it is observed that 39.79% of the population in Grândola and 42.2% of the population in Alcácer do Sal are considered vulnerable, as these groups are under 15 years old or over 64 years old, based on estimates of the resident population in 2021.

[1] HTTPS://QUALAR.APAMBIENTE.PT/

Based on Local Health Profile data of the region (ULS Litoral Alentejano), from 2019, respiratory diseases are among the leading causes of mortality for individuals under the age of 75. It should be noted that the population with respiratory diseases is vulnerable to air pollution, an environmental factor influenced by the implementation of the project.

The influence of noise on human health, as well as air quality, was also analysed. The data obtained for the reference situation of both environmental factors was compared with values recommended by the WHO, and compliance was observed.

20.3.11 LANDSCAPE

To delimit the study area for landscape characterisation, a 3 km wide strip around the project elements was considered, including access roads, power lines, and water supply structures. Main conclusions, regarding landscape characteristics of the region are:

- Elevation: It was observed that the study area encompasses elevations ranging from around 3 meters in the Sado floodplain to 107 meters at the Fresta vertex, near Água Derramada.
- Slopes: The study area predominantly features low slopes, up to 3% (sometimes 5%).
- Slope Orientation: The analyzed area generally exhibits a balanced distribution in terms of slope orientation.
- Landscape Units: The analyzed territory falls within the Q. Terras do Sado group of Landscape Units. The study area includes the Landscape Units UP 94: Charneca do Sado, UP 95: Pinhais do Alentejo Litoral, UP 96: Vale do Baixo Sado, and UP 97: Montados da Bacia do Sado, each comprising several subunits.
- Visual Quality: This is a landscape of overall medium to high scenic and aesthetic value, with some areas of low value.
- Landscape Sensitivity: It can be concluded that the study area is predominantly characterised by a high visual absorption capacity. Regarding landscape sensitivity, the area exhibits variable sensitivity, with a predominance of medium and low sensitivity classes, but still significant areas of high sensitivity.

20.3.12 HERITAGE

For the definition of the reference situation, in addition to the AE, two distinct areas were identified: the area of direct incidence (AID) and the area of indirect incidence (AII). The conducted work identified three archaeological sites within the AE and nine in its immediate surroundings (up to 2 km from the AE).

20.3.13 SOCIAL COMPONENT

According to data from the 2021 Census, the resident population in the municipalities covered by the project is 11,112 habitants in Alcácer do Sal and 13,822 in Grândola. Both municipalities have a similar age structure, with the 25 to 64 age group being the most representative, followed by the 65 years and older group, the 0 to 14 age group, and finally the least representative group, the 15 to 24 age group. In fact, the aging index of the population in Portugal is highest in the Alentejo region.

Housing in the immediate vicinity of the project area is characterised by single-story houses with one floor, often with a portion of land for self-consumption food production. The availability of internet and telecommunications services is limited to satellite. The local public transportation network has a low range of options in terms of available routes and schedules.

Regarding public infrastructure and facilities, the closest ones are located about 10 km away (healthcare, education, security, sports, and leisure facilities). As for road infrastructure, the nearest national road (N5) is about 8 km away, the main itinerary (IP8) and the highway (A2) are about 4 km away.

In the Alentejo region, in terms of the number of companies, the extractive industry represents about 0.2% (186 companies) of the total number of companies (84,838 companies). However, in terms of turnover, it represents a weight of 2.3% (€482 million) of the total turnover (€20,971 million), reflecting the significant role of the extractive industry in the Alentejo region. In the Alentejo Litoral, this importance is less reflected. Companies linked to the extractive industry represent only 0.08% (10 companies) of the total number of companies (12,532 companies), which translates into a weight of 0.09% (around €3 million) of the total turnover (€3,211 million), indicating a less significant presence of this sector of activity in this sub-region.

20.4 ENVIRONMENTAL RISKS/ISSUES

Potential environmental impacts, both positive and negative, were identified, described and assessed for the construction, operation, and closure stages of the Project.

During the construction and operation stages of the Project, there are potential moderate impacts to the environment, some of which can be reversed or mitigated in the short, medium, and long term. The following are the key potential impacts expected:

- removal of the vegetation for the sitingof Project facilities.
- alteration of wildlife habitat due to vegetation reduction in some sectors.
- significant impacts on the use of forest areas – loss of vegetation and decrease in productivity and consequent loss of income for the population.
- change in and diversification of land use.

- increased noise levels due to the use of equipment, machinery and vehicles, and plant process operations.

Despite the undeniable negative impacts associated with exploration, some positive impacts arising from the project are identified, namely at the social level, with the strategic positioning of Portugal in relation to European mining policies or the increase in dynamism of the socioeconomic framework.

Project impact mitigation measures are divided into two main groups: general and/or cross-cutting measures - applicable to most project intervention areas and associated with various environmental factors - and specific measures - established in response to identified and evaluated impacts within a particular environmental factor, with the aim of mitigating the impacts on that specific environmental factor.

Compensatory measures will also be defined, with a special focus on the forest and social components of the project. The main measures include:

- Forest component: afforestation of cork oak and stone pine, with particular emphasis on the productive aspect associated with the stone pine in the short term (5 years); partnerships with universities to study cork oak loss and implement mitigating measures; definition of a Forest Management Plan, utilising natural regeneration and implementing wildlife enhancement.

- Social component: Stakeholder Engagement Plan, with a benefit-sharing agreement based on the development of strategies at the community, environmental, and negative impact management levels of the project.

20.5 ENVIRONMENTAL ASSESSMENT PROCESS & PERMITTING

20.5.1 MAIN STEPS TO ACHIEVE ENVIRONMENTAL COMPLIANCE

Under the Portuguese environmental legal framework, some steps are required to achieve environmental compliance to operate a mining project. For the Lagoa Salgada project the steps required are:

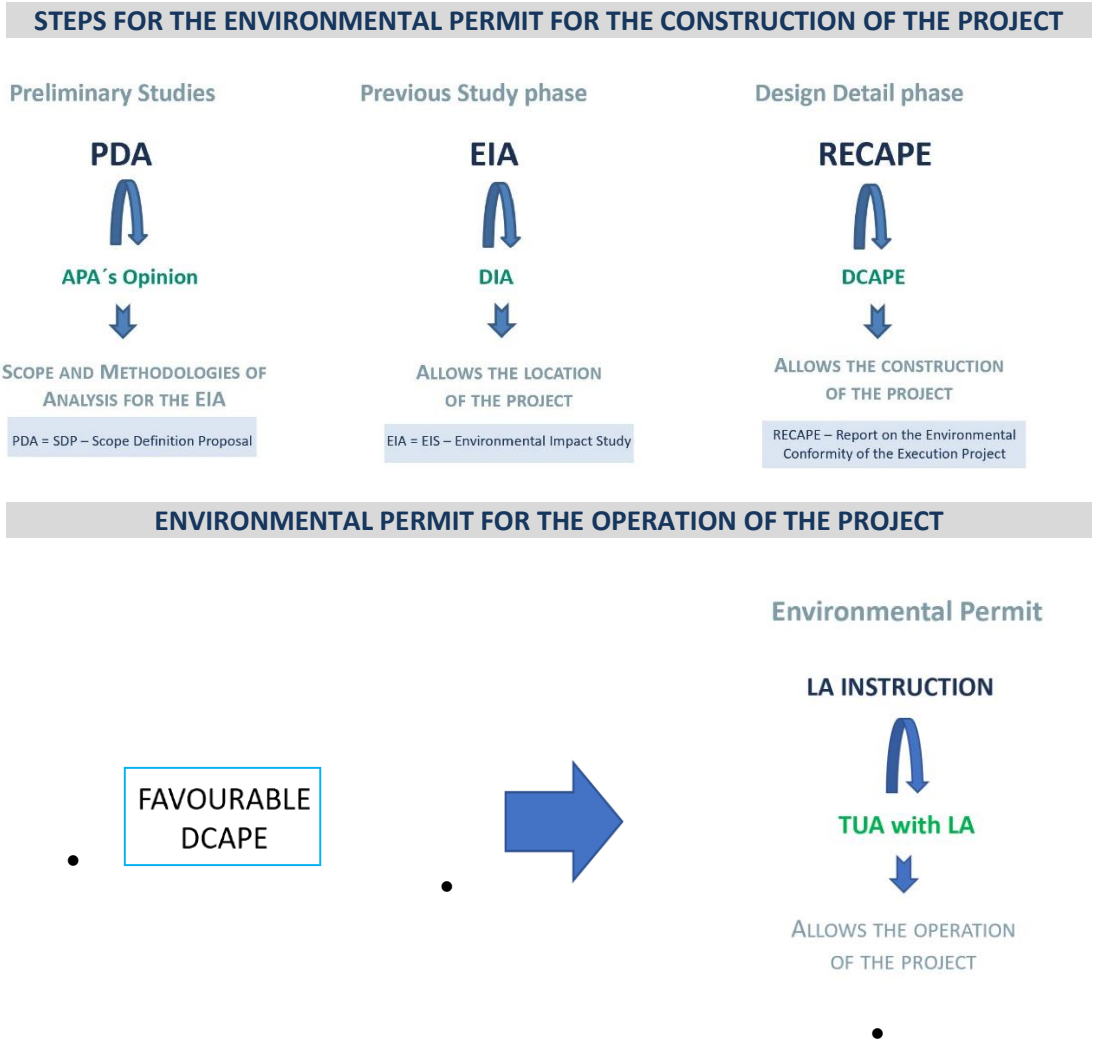


Figure 20.1 – Steps required for Lagoa Salgada environmental permitting

The following sections will legally frame each step, pointing to those that are optional (but can be recommended) and those that are compulsory, after choosing a particular path. The path chosen for Lagoa Salgada Project will be identified and the main applicable documents, considering its characteristics and location, as well as the legal options available to reach

the final environmental permitting that will allow the construction and operation of the Project, will also be pointed out.

Within the context of environmental permits, the following sections will deal, with:

- Relevant terms and definitions.
- Legal Regime of Single Environmental Permitting (LUA).
- Legal Regime of Environmental Impact Assessment (RJAIA).
- Legal Regime for Industrial Emissions (REI) applicable to the Integrated Prevention and Pollution Control (IPPC).

20.5.2 RELEVANT TERMS AND DEFINITIONS

To understand the Portuguese legal environmental permitting process, which will be explained subsequently, some definitions need to be presented:

Related to the Project

- **DGEG** – *Direção Geral de Energia e Geologia* = General Directorate for Energy and Geology, the permit coordination authority for the Lagoa Salgada Project
- **EP** – *Estudo Prévio* = **Previous Study Phase of a project** (for impact assessment purposes, this stage of a project can be a pre-feasibility study, definitive feasibility study, base project, or any other stage of the project except the design detail / execution project stage)
- **PE** – *Projeto de Execução* = **Detail Design Project** (project for construction)

Related to Environmental Impact Assessment

- **APA, I.P.** – *Agência Portuguesa do Ambiente* = Portuguese Environmental Agency, the AIA authority for the Lagoa Salgada Project
    - **SILiAmb** - the electronic platform of APA for environmental issues
    - **TAU** – *Taxa Ambiental Única* = single environmental fee payable when submitting a process for environmental assessment to the authorities
- **RJAIA** – *Regime Jurídico de Avaliação de Impacte Ambiental* = **Legal Regime of Environmental Impact Assessment**
- **TUA** – *Título Único Ambiental* = Environmental Single Title. A kind of "environmental duties' bulletin" for the project where all environmental obligations of the Proponent related to a particular Project will be registered, being the first issuing of this document obtained with the DIA (presented below)

- **AIA** - *Procedimento de **A**valiação de **I**mpacte **A**mbiental* = **Environmental Impact Assessment Procedure** (EIA). This procedure may have 3 stages, the PDA, EIA and RECAPE:
    - **PDA –** *Proposta de Definição de Âmbito* = **Scope Definition Proposal**
    - **EIA –** *Estudo de Impacte Ambiental* = **Environmental Impact Study** (EIS)
        - **DIA** - *Declaração de Impacte Ambiental* = Environmental Impact Statement
    - **RECAPE** – *Relatório de Conformidade Ambiental do Projeto de Execução* = **Environmental Conformity of the Detail Design Project.**
        - **DCAPE –** *Declaração de Conformidade Ambiental do Projeto de Execução* = Statement on the Environmental Conformity of the Detail Design Project

<u>Related to Environmental Permitting</u>

- **APA, I.P.** – *Agência Portuguesa do Ambiente* = Portuguese Environmental Agency, the AIA authority for the Lagoa Salgada Project
- **LUA** – *Licenciamento Único de Ambiente* = Legal Regime of Single Environmental Permitting
- **LA** – *Licença Ambiental* = Environmental License or Environmental Permit
- **REI** – *Regime de Emissões Industriais* = Legal Regime of Industrial Emissions

<u>Related to other Legal Regimes directly associated to Environmental Permitting</u>

- **RJURH** – *Regime Jurídico de Utilização dos Recursos Hídricos* = Legal Regime for the Use of Water Resources
    - **TURH** – *Título de Utilização de Recursos Hídricos* = Title authorising Water Resources Use
- **RJGRM** – *Regime jurídico de gestão de resíduos das explorações de depósitos minerais e de massas minerais* = Legal Regime for Mining Waste Management
- **CELE** – *Regime de comércio europeu de licenças de emissão de gases com efeito de estufa* = European Union Emissions Trading Scheme - EETS
    - **TEGEE** - *Título de emissão de gases com efeito de estufa* = Title for **Greenhouse Gas Emissions**

20.5.3 LEGAL REGIME OF SINGLE ENVIRONMENTAL PERMITTING (LUA)

The Legal Regime of Single Environmental Permitting (LUA) - Decree-Law No. 75/2015 of May 11th – establishes the proceedings for receiving the TUA (single environmental title), a document that brings together all the environmental requirements for the construction, operation, monitoring and deactivation/closure phases of a specific project, as well as all the administrative authorizations required by all environmental regimes to which the project is subject.

The LUA regime integrates, among others, the obligations under the following regimes (when applicable):

- Legal Regime of Environmental Impact Assessment (AIA)
- Legal Regime for Industrial Emissions (REI) applicable to the Integrated Prevention and Pollution Control (IPPC)
- Legal Regime for the Prevention of Major Accidents involving Hazardous Substances (RPAG)
- Legal Regime for Use of Water Resources (RJURH)
- European Union Emissions Trading Scheme (CELE)
- Legal Regime for Mining Waste Management (RJGRM)

Other important legal documents related to the LUA are:

- **Ordinance No. 332-B/2015 of October 5th** – establishes the value of the **TAU** – *Taxa Ambiental Única* (single environmental fee)
- **Ordinance No. 399/2015 of November 5th** – establishes the elements that must be presented for each environmental request to the authorities, for projects that include industrial or similar activities and also indicates which parts of the **SILiAmb** platform (the electronic platform of APA for environmental issues) must be filled in in order to receive eachenvironmental title/authorisation.

Once a project receives its first TUA, all future environmental permits and obligations will be added to that TUA for each update that is required for that document.

The TUA that will allow the operation of Lagoa Salgada Project will include the environmental obligations resulting from all environmental applicable legal regimes above mentioned.

20.5.4 LEGAL REGIME FOR ENVIRONMENTAL IMPACT ASSESSMENT (RJAIA)

The Legal Regime of Environmental Impact Assessment (**RJAIA**) is **established under Decree-Law No. 151-B/2013**, of October 31st, altered and **republished in the Annex XII of Decree-Law No. 11/2023 of February 10th**.

The following legal documentsmust also be considered during an AIA procedure development:

- **Ordinance No. 395/2015 of November 4th** – establishes the technical and formal requirements for various documents to be produced within the scope of AIA, such as PDA structure and minimum contents (Annex III) and the structure of the post-evaluation monitoring reports to be presented to APA.I.P. (Annex V).

The environmental impact assessment of a project can be performed at two different stages:

1 – Under previous study stage (PS)

• **Status**	• **PDA**	• **EIA**	• **RECAPE**
Required?	• Optional	• Compulsory	• Compulsory
Official statement	Opinion of APA, I.P.	DIA_{PS}	DCAPE
Consequences	**Methodologies** and depth of analysis **defined** for each environmental factor	**Project location approved**	**Project construction allowed**
Result and consequences	Commitment between APA, I.P. and Proponent for the EIA development	Definition of guidelines to be followed: During project design During construction During operation Establishment of some requirements to accomplish during RECAPE	Establishment of measures to be implemented: During construction During operation At closure phase Establishment of Monitoring Plan for construction and operation phases

2 – Under design detail stage (DD)

• Status	• PDA	• EIA	RECAPE
Required?	• Optional	• Compulsory	No
Official statement	Opinion of APA, I.P.	DIA_{DD}	• -
Consequences	**Methodologies** and depth of analysis **defined** for each environmental factor	**Project construction allowed**	• -
Result and consequences	Commitment between APA, I.P. and Proponent for the EIA development	Establishment of measures to be implemented: During construction During operation At closure phase Establishment of Monitoring Plan for construction and operation phases	• -

The pathway chosen for the Lagoa Salgada Project until construction was:

EIA_{PS} => **DIA_{ps} ->** RECAPE => **DCAPE**

In synthesis, the chosen path involves:

EIA_{PS} + approval by APA, I.P. => **DIA_{ps}** – **<u>approves the location</u>** of the project and its main characteristics and establishes some constraints to be considered in the Detail Design Project BUT requires a second moment of environmental impact assessment, after design detail project development -> RECAPE:

- RECAPE + approval by APA, I.P. => **DCAPE** – **<u>allows the project construction</u>** in the location evaluated; may establish some constraints to construction and/or operation phases. The DCAPE will also establish the final Monitoring Plan to be executed along the project life (although it might be adjusted along the operation phase, depending on the monitoring results)

However, to be able to operate, the Lagoa Salgada Project must still comply with the Legal Regime for Industrial Emissions (REI) applicable to the Integrated Prevention and Pollution Control (IPPC), to receive its environmental permit.

20.5.5 LEGAL REGIME FOR INDUSTRIAL EMISSIONS (REI)

The Legal Regime for Industrial Emissions (REI) applicable to the Integrated Prevention and Pollution Control (IPPC) is established under Decree-Law No. 127/2013 of August 30th in its version in force.

As a result of the REI application, the proponent fills in several fields in the SILiAmb platform (the electronic platform of APA for environmental issues) to request an environmental license for the project. The information required to fill in the platform fields is identified in Ordinance No. 399/2015, of November 5th as previously referred.

When the process is approved by APA, I.P., an environmental license (LA – Licença Ambiental) is issued and its constraints are registered into the TUA of the Project.

When a project must be submitted to an AIA procedure, which is the case with the Lagoa Salgada Project, the LA cannot be issued without a DIA_{DD} or a favorable DCAPE, the latter being applicable in this case.

After receiving its Environmental License, annexed to the TUA, the operation of the Lagoa Salgada Project can legally start as concerns environmental authorisations and permits.

The environmental process needed for the permitting of Lagoa Salgada Project operation can be summarised as:

EIA_{PS} => **DIA_{ps} ->** RECAPE => **DCAPE ->** LA instruction **=> TUA with LA**

20.6 CONSIDERATIONS OF SOCIAL AND COMMUNITY IMPACTS

Aware that the success of the project is closely linked to public opinion and acceptance by local communities, transparently clarifying the scope of the project and adopting a receptive attitude towards the concerns of the local communities is required.

To better understand the social and economic impacts of the project, studies have been carried out with the aim of identifying the main social issues and perceptions, both positive and negative, associated with the mining development of Lagoa Salgada for the affected communities.

The Key Informant Interviews (KIIs) study, conducted by the Community Insight Group (CIG) was developed in May 2023. The objective of the KIIs was to identify the main social issues related to the community's relationship with the project, the population's concerns about negative impacts, and potential positive impacts. The results are expected to inform the future impact assessment and plans for interacting with stakeholders, although they are not statistically representative.

Interviews were conducted with a total of 12 stakeholders, including service providers, interest groups, government authorities, and communities directly affected by the project. Additionally, four internal stakeholders were interviewed, although their results are not reflected in the study's conclusions.

The main results of the KIIs study are reflected in the following table:

Table 20.1 - Conclusions of KIIs study (CIG, 2023)

Community's relationship with the project	Lack of information about the project and contact with the communities. Distrust towards the proponent is associated with the lack of information and involvement. Mixed opinions about the project.
Population's concerns about negative impacts	**Reduction in water availability and impact on agricultural and forest activities.** **Impacts on ecosystems due to air, water, and soil pollution and contamination.** **Increased sources of discomfort such as blast vibrations, noise, and dust.** **Difficulties in integrating new people into the community due to the housing market.** **Impacts on infrastructure quality such as roads and healthcare services.** **Cumulative impacts of the railway logistics platform and new tourist developments.** **Unfair distribution of wealth generated by the project.**
Potential positive impacts	**Local development and improvement of economic capacity in the region.** **Opportunity for repopulation of communities with increased job opportunities.** **Improvement of transportation systems, telecommunications, and healthcare services.** **Increased profitability of businesses (e.g., restaurants, accommodations).**

As the project progresses, efforts will always be made to align the nature of the project with the needs and concerns of the neighbouring populations, to minimise the negative impacts of the project and enhance the value to the region. As mentioned in Chapter 5, measures are being developed to minimise and compensate for the social component of the project, as well as socioeconomic studies to better understand all the underlying issues related to the project in terms of its social and economic impacts.

20.7 BASELINE HYDROGEOLOGICAL STUDY

The Baseline Hydrogeological Study (BHS) was carried out by CONSMAGA - Geólogos e Engenheiros Consultores Lda and by FRASA - Ingenieros Consultores S.L. in partnership with QUADRANTE, the responsible for the elaboration of the DFS.

The information utilized in this chapter is derived from the executive summary of the BHS, which can be accessed in the supporting documentation.

20.7.1 OBJECTIVES

The BHS intended to evaluate, understand and characterise the dynamics of the hydrogeological systems, including the cover formations of the Cenozoic and the underlying formations of the Palaeozoic, where the mineral deposits (mineralisation in polymetallic sulphides of Venda Nova North and Venda Nova South) are located, considering the role of the various hydrogeological units present, their complexities and influences, and the interrelationship with surface waters, natural and industrial ones produced by the Project, within the scope of the design of the future mining operation.

The hydrogeological conditions of a mineral deposit and its surrounding setting are decisive for the correct design of the mining operation, the preventive drainage system of the mining area, and its optimisation, all essentially to the extent that it affects the general context of the mining project. In addition, it is necessary to estimate the potential impact of the mining operation on water wells and holes, mainly for public supply, and in the aquifer systems (both in quantity and quality), in the establishment of measures to protect the water resources, the objective is also to estimate the possible available water resources that can be used during the production operations such as the mineral extraction and treatment.

To carry out this hydrogeological characterization study of the Project area, it was necessary:

- to establish, based on available data, the hydroclimatic parameters required to estimate the water balances under different climatic conditions and to determine the probable likely maximum rainfall;
- to characterize the conditions of the surface water runoff, in the Project area and its surroundings, to define the infiltration rate into the soil, the water surplus and/or water deficit, and the floodable areas;
- to define the hydrogeological characteristics of the Cenozoic and Palaeozoic lithological materials and the predictable impacts on the groundwater aquifer systems.

The specific objectives are:

- the identification of the geological formations and their structural characteristics, which contribute to groundwater storage and flow (aquifers, aquifuges, and aquitards);
- the determination of the hydrodynamic parameters that characterize the different hydrogeological systems;
- the surface water and the groundwater physicochemical characterization, to aid in the establishment of flow dynamics, and the relationship between the different hydrogeological systems and surface waters;
- the determination of recharge rates in the underground systems, considering heterogeneities and the presence of sedimentary coverings (multi-layered hydrogeological system);
- the establishment of the conceptual hydrogeological model for the area of potential mining effect/influence.

20.7.2 SOCIAL AND ENVIRONMENTAL KEY ASPECTS

From a landscape environment point of view, the area of the Lagoa Salgada Project is integrated into the landscape unit defined by Ribeiro (1987) as the Sado Depression. This landscape unit covers the NW part of Alentejo, from Serra do Cercal to the area north of Águas de Moura village. Basically, this area corresponds to the Sado Basin.

The Sado River hydrographic basin has an average altitude of 130 m and is situated between 37º25' and 38º40' N latitude and 07º40' and 08º50' W longitude (Loureiro, et al., 1982, in Quevauviller, et al., 1989). This basin covers an area of 7,640 km2, which is the largest of its kind on the Portuguese territory, of which 7,258 km2 is located in the Alentejo region.

In particular, the area of the Hydrogeological Study assigned to the Lagoa Salgada Project corresponds to a dominant sandy flattened plateau and a cut-out area between rivers, that is, between the Sado River and its two main affluents on the left bank, the Ribeira do Arcão to the north and the Ribeira de Grândola to the south, forming a more or less circular elevation, which is limited by the valleys of the main rivers, and by all the third order affluents that contribute to them, and cut by tributaries with radial drainage pattern.

In the Project area, there is a preponderance of U-shaped valleys, where the lands of “lezíria", which are wetlands for rice production, are installed. In particular the Sado River Valley is dominated by the same rice fields. There is some tide influence up to an intermediate sector of the Sado River, next to Porto Rei in the study area.

The existence of lagoons (superficial and temporary) particularly fed during the seasonal wet hydrological cycles is associated with some undulations of the land surface, marked by intercalations of a more clayey nature between sandstones and recently deposited aeolian sands.

The profusion of small dams and weirs along many talweg lines of the area shows the tendency for pellicular to cutaneous and sometimes even deeper flows of the infiltrated waters into these porous media, which in depth may be recharging aquifers through the intersected valleys.

The pedological units present were identified based on the Soil Map, edited in digital format at 1:25,000 scale (SNIS, 2022), and on bibliographic references. In general, they are characterised by their low agricultural potential, due to the lack of organic and mineral nutrients. On the other hand, the floodplains adjacent to the Sado River and in the downstream segments of the Arcão and Grândola streams are characterised by their fertility and good hydric conditions.

Regarding land use, a large part of the area is occupied by forestry, with cork oak forests, stone pines, and eucalyptus. There are also large areas of a mix of cork oak and stone pine forest. In the near water stream areas (Arcão and Sado) there are extensive floodplains with rice paddies. In the northwestern part, there is some irrigation with the structuring of pivots for irrigation. There are also some dispersed vineyards. In areas where there is no forestry or irrigation, the land is occupied by arable cultures and rainfed land, where cattle and sheep graze.

20.7.3 HYDROMETEOROLOGICAL SETTING

The available climatological data was sought and collected, structuring data series from 1939 to 2020 from the data meteorological station of Grândola (24F01C) although there are gaps in the registration of precipitation observed there. The year 2000 was filled with the results of simple linear regression Azinheira Barros/Grândola, while the gaps in the precipitation record in the years 2001 and 2008 were filled with the results of simple linear regression Torrão/Grândola, and those of 2009 with the results of simple linear regression Ferreira do Alentejo/Grândola.

The temporal distribution of precipitation and temperature (monthly) was defined, and the characterisation of dry, average, and wet years, and the water balance was carried out according to the Thorntwaite and Mather formulas, establishing the useful rainfall, surface runoff, and infiltration.

Climate change was considered for several future scenarios, in the short, and intermediate terms, considering the less restrictive level RCP 4.5 scenario, and the most restrictive RCP 8.5 scenario, according to information made available by the Climate Portal (http://portaldoclima.pt).

20.7.4 DEMAND AND USE OF WATER RESOURCES

The volume of water collected for the different local activity sectors is presented according to the available information in the PGRH of RH6 - Sado and Mira, both for the 2nd planning cycle 2016/2021, and the 3rd planning cycle 2022/2027, respectively.

20.7.5 HYDROLOGY: SURFACE WATER

The streams and hydrographic basins in the Project implementation area and their vulnerability to the in-depth aquifer recharge, which could be felt throughout the Project area, have been characterized.

20.7.6 HYDROGEOLOGY: GROUNDWATER

The hydrogeology of the aquifer systems was characterised, i.e., the general and detailed hydrogeological characteristics of the study area have been established, based on 12 core drillings intersecting the Tertiary formations including the 5 initials, mentioned above, plus a new set of 7 holes since carried out by REDCORP in 2022 and, even more, data from old drillings carried out by LNEG and REDCORP.

Four aquifer systems have been defined in the Project area, with variable thickness and productivity, namely:

- **Cutaneous Aquifer System**. The upper young formations (Holocene and Plio-Pliocene); contain two free aquifers in unconsolidated fine sandstone separated by a thin impervious clayey metric layer intercalated between the top of the sandstone and the surface aeolian sands, at a depth of 0-20 m. It is an aquifer system of great environmental importance as it supports the plant habitat dominated by cork oak trees and is of social importance for its use by local inhabitants and landowners.
    - **Intermediate Aquifer System**. In the Miocene formations; porous sandstone, conglomerates and calcarenites; multi-layered semi-confined aquifer system (up to 4-5 m thick each), the average depth between 30 and 80 m. It includes the top of the Palaeogene formation (top of the Vale do Guizo Formation) when it is sandstone type within argillites (80-90m). It contains at least two important and identified aquifers each with its own characteristics, but there may be other multi-layered aquifers of lesser expression and importance throughout this formational sequence. It is the main groundwater catchment and supply aquifer system for various public and private users.
- **Deep Aquifer System**. The two lower systems merge into a single deep aquifer system because, although with their own specificities, they are contiguous and interfacing, and can only be easily identified where there is no overlap with mineralization.
    - **Basal System**. Paleogene/Palaeozoic contact corresponds to the important stratigraphic hiatus that emerged; heterogeneous aquifer, at variable depths between 120 and 150 m in the deposits.
    - **Deep System *ss***. Palaeozoic within the Volcano-Sedimentary Complex and massive, disseminated, or venular sulphides; corresponds to fractured and/or mineralisation-related rocks: polymetallic sulphides, stockwork and

gossan; mixed heterogeneous porous/fissured confined to semi-confined system; indefinite depth.

Large diameter bores, aligned along sections, have been drilled and prepared as piezometers, in which pumping tests have been carried out and their future planned for use as piezometers and hydrochemical control.

In the Venda Nova Norte deposit, three bores were drilled, a short one of 22 m deep for the cutaneous aquifer, another of 90 m to capture the intermediate multi-layer system, isolated from the cutaneous aquifer, and a third one of 157 m to capture the deep system, intersecting the Tertiary/Palaeozoic contact isolated from the above-drilled sections, corresponding to the cutaneous and intermediate aquifers.

In turn, four bores were drilled in the Venda Nova South deposit; one short 22m deep, to catch the cutaneous aquifer, one 40m deep, to catch the first aquifer of the intermediate system together with the catchment of the cutaneous system, another of 82m, to catch the multi-layered intermediate system, isolated from the cutaneous aquifer and a fourth one 150m deep to catch the deep system, intersecting the Tertiary/Palaeozoic contact with isolation of the upper sections, corresponding to the cutaneous and intermediate aquifers.

Pumping tests were carried out in each of the 7 hydrogeology holes with observations at the adjacent piezometers installed. In addition, some controlled pumping tests were performed in the private hole WH-HCC01. The physical conditions were defined and the characteristic hydrogeological parameters for each test (transmissivity (T) and storage coefficient (S) were calculated with typical methodologies applicable to each case (Neuman, Hantush with shielding, Theis, etc).

Also in the public supply hole, renamed WH-ADA01, data made available by AgdA - Águas Públicas do Alentejo was used.

Thus, the influence was defined, regarding the intercommunication between aquifer systems, of pumping on the deep and intermediate systems and vice versa, and probable effects of the drainage of the deposits, if not by natural causes, then by the proper construction of the respective piezometers.

20.7.7 FIELD SURVEYS, WATER POINT INVENTORIES AND SAMPLING

The inventory of observation points includes water points to the north, where the Project is fully integrated and to the southern half of the study area, which will be progressively integrated in future field surveys and monitoring campaigns, in the same way as also applicable to the later stages of the Project and to the continuation of the hydrogeological studies in the future.

As part of REDCORP's policy and mission and in order to intensify good relations between the public partners (entities) and social partners (population and non-governmental organizations) of the Project (Stakeholders) and sharing with the partners of the Project (Shareholders), in a perspective of partnership, mutual respect, and accountability and

transparency, the owners of houses, lands, and farms were directly contacted with explanation and sharing of knowledge, given and received, on the scope and nature of the studies in progress, which allowed the permission, without exceptions, of accessibility in the field. AdgA which has control of exploitation in a catchment borehole and a former well in the Project area, was also contacted, with whom information was shared. A technical presentation was made to the Municipality. Contacts were also established with APA - ARH Alentejo, which regulates procedures and licensing, with a view to the procedures for reconverting the Project's evaluation drill holes (*na*: into hydrogeology control holes.

The inventory of relevant surface water (108) and groundwater (104) observation points (also the basis for the conceptual hydrogeological model), included: rivers and streams, lagoons/ponds, dams/weirs, springs, fountains and wells built on residential land, farms and estates, for various uses, such as self-consumption, and agricultural or livestock consumption, etc., public and private water supply drilling holes, and research drilling carried out by the Project, from which the most representative have been selected for the scope and purpose of the BHS and for piezometric, hydrochemical or other control.

A database of points of hydrological interest was created, suitable for the current study phase of the BHS, and also to constitute a future monitoring and control grid system for the EIA, and for the necessary continuous follow-up controls throughout the evolution of the Project, in order to anticipate the monitoring and control network of surface and groundwater on a PVCASS - Surface and Groundwater Monitoring and Control Plan for Lagoa Salgada, in the longer term, until its closure.

Points have been selected that may be relevant to ensure its effective control, the establishment of the baseline reference values for hydrochemical and hydrophysical quality expeditious *in situ* recording parameters, and for laboratory analytical parameters, but also for all the records of complementary observations such as producible, circulating and/or draining flow-volumes, depending on hydrological and hydroclimatic cycles.

This is an inventoried point network, which required field validation to establish the overall point network, individual characterisation spreadsheets, ETRS89 coordinates, photographs, and measurements of water levels, as well as hydrochemical sampling, for both *in situ* and laboratory determinations.

The field validation started after the first contact with owners, the first quarterly sampling was carried out in April and the second in July 2022, which continued until the beginning of August conditioned by the restrictions and prohibition of field work due to the high temperatures recorded and fire risks.

A total of 209 monitoring campaigns (105 surface water and 104 groundwater) were carried out in April/22 and 212 (108 surface water and 104 groundwater) in July/22. Among them, the parameters measured *in situ* were recorded (31 in April/22 and 42 in July/22).

Laboratory chemical analysis (ALS) was carried out (25 in April/22 and 35 in July/22), plus one more rainwater sample for tritium content analysis. The samples were collected following the protocol defined by the laboratory.

Whenever possible, points located outside the Project's API areas of potential interest, in sectors not directly related to the Project, were also considered to establish a representative reference framework of the original background conditions ("baseline" or "background" reference values), in the sense of contributing to future assessment/comparisons.

20.7.8 WATER PHYSICOCHEMISTRY (NA: CHARACTERISATION)

Geochemical techniques facilitate interpreting the relationship between surface water and groundwater to elaborate the conceptual hydrological-hydrogeological model and, based on it, to be able to predict changes that may cause anthropic impacts in the hydrological systems *sensu lato*.

A study was conducted for each parameter recorded, both *in situ* and laboratory (ALS), and different hydrochemical conclusions based on the analytical results are presented in the report.

The chemical facies and the temporal and spatial variations of the parameters between the sampling dates were studied.

The results are presented for the main cations (calcium, magnesium, sodium, and potassium) and for the main anions (chlorides, hydrogen carbonates (bicarbonates), sulphates, nitrates, and boron), as well as other parameters of interest (free CO2, nitrites, aluminium, antimony, arsenic, barium, beryllium, lead, copper, chromium, iron, dissolved iron, bivalent iron, trivalent iron, phosphorus, phosphorus as P2O5, fluorides, lithium, manganese, mercury, molybdenum, nickel, vanadium, zinc, cadmium, cobalt, silver, selenium, and thallium.

Regarding the radiological parameters, for tritium, no conclusion was obtained due to the detection limit which needs to be much lower than that performed by the laboratory. It is recommended that a new dating with a lower limit be performed and that the isotopes oxygen 18 (δ18O) and deuterium (2H) be studied to be incorporated in the determinations in future campaigns to obtain a better understanding of the hydrodynamic relationships.

20.7.9 HYDROLOGICAL-ENVIRONMENTAL IMPACT OF MINING

Starting with the chemical characterisation of the natural environmental conditions, according to which some parameters already exceed the maximum recommended value (VMR) of Decree-Law 236/98 and even the maximum admissible value (VMA), we are in the presence of an environment with a natural chemical background that already exceeds these limits in the situation 0 (state zero) or baseline reference of the pre-project phase which has nothing to do with any mining activity, but that already sets reference values.

Although there are apparently no major natural impacts related to mineral deposit oxidation to highlight on the water chemistry, the chemical conditions will change when these deposits are exposed to free oxygen mixing conditions when water pumping starts.

Different hydrochemical conclusions are presented based on the analytical results obtained in the campaigns. Different water groups are distinguished, the quality of which may be related to their origin, the degree of influence of the sulphide oxidation process, and the degree of neutralization of the acidity generated.

20.7.10 PIEZOMETRY

A monthly piezometric monitoring dataset of up to 39 points is presented, to be progressively improved in the following phases of the project, within the scope of the piezometric monitoring plan, both at the level of the Project's focus area (AI_PDA), and at a more general and regional level involving the current BHS area in a total of 65 points, to which progressive access is expected.

Different conclusions are drawn from crossing the observations with the general piezometric behaviour of the area covered by these measurements, regarding aspects such as the differentiation and interrelationship between groundwater systems, hydrogeological divisions, its relationship with the hydrographic network (condition of providing or receiving water to or from the fluvial flows).

20.7.11 CONCEPTUAL HYDROGEOLOGICAL MODEL: MODEL FLOW

The conceptual hydrogeological model of the study area is presented, focussing on the following items:

- Interpretation of the model of flows in the cutaneous system.
- Flows in the intermediate aquifer system.
- Flows in the deep aquifer system.

Numerical modelling was not carried out. A numerical model cannot be made without historical piezometric data, nor with the pumping and testing carried out, which will contribute to it, in the future, without first having the coverage of a whole hydrological year, with both dry and wet seasons, and long-lasting tests, and as such already the subject of the recommendation.

The BHS is only a starting point, not an endpoint, and a lot of work and future commitment will have to be assumed, until the mining start-up phase, without forgetting that WATER is one of the most determining factors in the Project's decision taking.

20.7.12 WATER STORAGE AND SUPPLY STRUCTURES

Regarding the water supply sources for the Project, different alternative and/or interactive complementary possibilities are raised, with reference to the distance relative to the Project.

Besides these alternatives, the different types of water to be produced from the groundwater mining drainage systems are distinguished and included in future underground works plans.

These are some possible contributions, regardless of their technical and financial feasibility, to be determined, which may be a basis for work, to be developed in due course during the Project, as they will have an impact on the preparation of the water balance.

20.7.13 HYDROLOGICAL-MINING INTER-RELATIONSHIPS AND WATER BALANCE

The BHS is an approach as a first step towards establishing the relationships between the different aquifer systems and the Lagoa Salgada Mining Project itself, to be implemented as the Project develops and progresses over time.

20.7.13.1 HYDROGRAPHIC SYSTEM

Different conclusions are presented regarding the behaviour (gaining or losing) of the hydrographic network, climate estimates related to precipitation, its monthly evolution over the year, temperature, evaporation, surface runoff, infiltration, etc.

20.7.13.2 CONCEPTUAL MODELLING OF HYDROGEOLOGICAL SYSTEMS FLOWS, CAPACITIES AND EVOLUTIONS

For each hydrogeological system (cutaneous system, intermediate system, and deep system) the flow model, their aptitudes as supply sources, and the evolution that they will face under drainage conditions are presented.

20.7.13.3 CUTANEOUS SYSTEM

The hydrogeological characteristics of the Holocene and Plio-Pleistocene formations that constitute a free aquifer in unconsolidated sand and fine sandstone are summarised. Recommendations are made for its full protection and preservation intact from mining drainage and from the designing of protection systems against infiltration (na: water inflow and/or seepage control) in the industrial area and in the planned stockpiles and reservoirs, due to the high vulnerability of this system.

20.7.13.4 INTERMEDIATE AQUIFER SYSTEM

The hydrogeological characteristics of the Miocene formations are synthesised; porous sandstone, conglomerates, and calcarenites interbedded with argillites, forming a multi-layered semi-confined aquifer system. Recommendations are made to protect it from the impact (influence) of mining drainage, also by the study of the possibility of reducing it, with aquifer recharge near the supply water catchment boreholes. Thus, the implementation of sealing and ground consolidation systems against unwanted collapse and infiltration during the construction of the access ramp, chimneys, or shafts, when the most transmissive materials are intersected by the sinking works, is recommended.

20.7.13.5 DEEP AQUIFER SYSTEM

The hydrogeological characteristics of the materials of the Palaeogene/Palaeozoic contact (Basal System) are summarised, where they can be seen to present a high hydraulic load, which can produce a strong irruption of water when they are intersected by the mining works, reducing considerably once they are depressurised. Consideration is given to the Palaeozoic fractured deep system and the advantages of draining It in advance of the respective mining access to assist the depressurisation process as mentioned above.

20.8 CLOSURE PLAN

20.8.1 INTRODUCTION

After the completion of mining activities, interventions will be necessary to prepare the project's activity area to return the site to proper safety conditions and alignment with the surrounding environment, allowing for future use.

Mine closure is a complex and careful process that must be planned to ensure the safety of workers, environmental protection, and compliance with applicable laws and regulations. The mine closure plan should be developed before the start of operations and should be regularly updated throughout the life of the mine. Recovery operations should be carried out in conjunction with mine exploitation and completed after its closure, representing an important way to reduce liabilities during the closure phase.

Therefore, the following points will describe the measures to be implemented during and after the termination of operations regarding mobile equipment, deactivation of fixed facilities, environment, security systems, monitoring plans, and human resources, to achieve these objectives.

20.8.2 MOBILE EQUIPMENT

After the termination of operations, the mobile equipment will be sold for future use in another operation or sold as scrap, thus being fully decommissioned, and removed from the project site.

During the closure phase, there will still be activities requiring the use of some equipment (Table 20.2) that will remain on-site after the end of operations until the completion of the entire closure plan and cessation of activities.

Table 20.2 – Equipment used during decommissioning phase

Equipment	Model	Un
Dumper	Epiroc MT	2
Dust Suppression	Tractor + Joper	1
Front End Loader	CAT 980	1

Other equipment that is not part of the project's fleet will be temporarily used, being requested specifically for use during the closure phase.

20.8.3 FACILITIES DEACTIVATION

20.8.3.1 SURFACE BUILDINGS AND PROCESSING PLANT

The social and support facilities, as well as the workshop, laboratory, and warehouse, will be dismantled and relocated outside the mine area during the deactivation phase. These facilities will be dismantled, and concrete structures demolished by specialized external contractors. However, if there is interest from local entities and/or other organizations, the retention of some facilities may be considered.

The processing plant will be dismantled in sectors and phases. Prior to dismantling and removing the buildings, all mineral processing equipment, electrical installations, water, and sewage facilities will be removed. The remaining industrial support facilities will be relocated outside the mine area, and the soil will be leveled.

The destinations of the mine facilities to be removed are summarised in the following table.

Table 20.3 – Destination of facilities, equipment, and materials to be dismantled

Facility	Action	Destination
Processing Plant	Dismantling and transport	Sale of equipment
Fixed Equipment – crusher, conveyors and silos	Dismantling and transport	Sale of equipment
ROM Pad	Linning removal and soil leveling	Integração Paisagística
Paste-Fill Plant	Dismantling and transport	Sale of equipment
Mobile Equipment	Transport	Sale of equipment
Electrical Substation	Capacity reduction	Sale of equipment or relocation to another location
Fuel Farm	Dismantling and transport	Sale of the equipment / Foundations proceeds to landfill
Social Buildings	Dismantling and transport	Sale of equipment or relocation to another location
Water Treatment	Dismantling and transport	Sale of equipment
Water Storage	Linning removal and soil leveling	Landscape Integration

20.8.3.2

20.8.3.3 ACCESS

The accesses to the mining/industrial complex will be maintained in the end, both for the use of the populations served by these accesses and for the site to be explored from a forestry and agropastoral perspective. Access to the site of the former mining operation will also be maintained for monitoring and for potential industrial archaeology visits.

20.8.3.4 TAILINGS STORAGE FACILITY (TSF)

CLOSURE

The closure plan will be determined by the selected post-closure land use (PCLU) as well as the methodology to create a long-term stable landscape. The PCLU of the TSF has yet to be selected and will need to be determined to proceed with the closure design.

To satisfy regulatory requirements to ensure the long-term safety and stability of the MWF, good practice guidance will be followed as recommended in the ICOLD – sustainable design and post-closure performance of tailings and in the ICMM - Integrated Mine Closure, Good Practice Guide.

The Operating and Maintenance Manual will be updated as the focus of operations turns to closure.

TAILINGS BEACH SURFACE

Rehabilitation of the tailings surface will commence upon termination of tailings deposition. The method of spigotting during the operational life of the TSF will form a tailings surface that slopes at 1V in 50H from the south to the north, with a spillway discharging water to the reclaim pond. This approach to depositional control will avoid mechanical reshaping of the tailings surface at closure and should allow initiation of rehabilitation works soon after the deposition of tailings ceases.

Geochemical tests have indicated that tailings and development rock from the north node are potentially acid-generating (PAG), whereas tailings from the southern node will be non-acid-generating (NAG). During the operational life of the mine measures will be in place to prevent the PAG tailings and development rock drying out. These measures are intended to stop the PAG material from oxidizing by either maintaining a wet surface or covering it with NAG tailings. Geochemical laboratory testing has demonstrated that the NAG tailings are fine-grained and have low permeability making them suitable as a sealing layer over the PAG.

To ensure that the PAG material within the TSF is sealed to prevent oxidation in the long term, tailings deposition during the latter stages of development should be scheduled to ensure that a minimum of 1m of NAG tailings is deposited over the entire TSF.

Drying the tailings sufficiently to allow trafficking is expected to take up to two years. Since geochemical tests have indicated that the tailings from the southern node are non-acid forming, a soil cover is proposed as the most appropriate long-term tailings surface capping solution for the TSF.

Placement of a vegetated cover layer over the tailings surface can commence. The current closure design calls for a soil cover thickness of a minimum of 0.5 m consisting of sand and topsoil. An initial pioneer layer of sand may however also be required to provide access to the consolidating tailings beach.

EMBANKMENT SIDE SLOPES AND BENCHES

The downstream embankments will be shaped to a final downstream slope of 1V:3.H (with 5 m benches every 10 m maximum i.e., a 3H:1V overall slope), covered with 0.5 m of topsoil and vegetated, to minimise infiltration into the embankment and underlying geology.

Rehabilitation trials are recommended during operations to determine the most efficient method of rehabilitating the final embankment slopes and benches.

Post closure and once the embankment slopes are vegetated, these sumps could be either be:

- Drained by locally altering the topography to allow the water from the sumps to flow to the nearest watercourse.
- Removing the liner retaining the water and allowing it to soak away.
- Left in situ as a resource for the landowner.

WATER MANAGEMENT

During the operational life, no ponds will be allowed to form on the tailings surface. Reclaim ponds located outside the active TSF will collect tailings bleed water and run-off water. Temporary ponds will be initially constructed in the footprint of the TSF and ultimately relocated to a permanent pond outside the TSF footprint. At closure, the pond will be located to the north, outside the TSF and no water will be held on the TSF.

After closure, the reclaim pond will change from storing water for the process plant to attenuating flow from the TSF, providing sufficient storage for flood water so it is released at a rate that will not overwhelm water courses downstream of the TSF. However, after the rehabilitation of the TSF surface, there will be a transition period.

Rainfall run-off from the rehabilitated areas is expected to initially have high silt content until vegetation has sufficiently established. Discharging this run-off must therefore be avoided initially and, for this reason, will also be collected and temporarily stored in the reclaim pond.

After the ponded water has been shown to meet discharge water quality standards, the closure spillway can be constructed. The closure spillway will discharge all rainfall run-off (which at this point will comply with discharge water quality standards) into the drainage line towards the north of the facility. Given the depth of the operational pond is 7m, the pond will need to be partially backfilled to make the structure safe.

TESTING OF CLOSURE OPTIONS

Rehabilitation trials are recommended during operations to determine the most efficient method of rehabilitating the final embankment slopes and the tailings surface. The trial results will be used to design the most suitable cover for the tailings surface after decommissioning. The rehabilitation plan should be reviewed every two years. At least two years before decommissioning the cell, a detailed rehabilitation plan should be developed.

20.8.3.5 PHASING OF OPERATIONS

The phasing of deactivating the various structures is primarily related to their period of activity, with priority given, as mentioned before, to deactivating and rehabilitating the site as closely as possible to the ongoing operations to minimise the project's final liabilities.

The underground mining units will be deactivated first since they will be the first operation to cease activity. Therefore, once the underground activity is finished, the supporting structures will be deactivated, including main fans, closure of ventilation chimneys, mine entrance, and paste filling plant.

Once the mining operation is completed and the surface ore stock is depleted, all remaining facilities will be dismantled. It will be necessary to develop a detailed plan for this process, as only at this stage can the logistics required to demolish, dismantle, deactivate, and/or transport the structures and equipment be applied with greater precision.

20.8.4 ENVIRONMENT

Mining activity has as one of its main objectives, in addition to ensuring responsible and efficient mining of mineral resources, a closure plan that aligns with modern social expectations, respecting environmental, social, and human aspects.

The main goal of the closure plan is to leave a positive legacy after the mining activity, ensuring that its benefits endure over time, minimising impacts, and addressing issues that this type of activity affects, such as environmental degradation, soil and water contamination, and landscape.

As mentioned in previous sections, the closure plan should start during the project phase so that everything planned and designed takes into account the future of these activities and how they will be decommissioned. Closure, decommissioning, and environmental rehabilitation activities should take place as early as possible during the operation,

minimising the rehabilitation liabilities for later stages when there is no longer mining activity and capital inflow, allowing for a quicker recovery of the affected areas.

The main measures in the closure plan to minimise environmental impact will be:

- Closure of mine accesses, including ventilation shafts and ramps, to prevent access by people and animals and to seal off contact to avoid groundwater interaction with the natural environment.
- Ensuring designated areas of land during the operational phase to store materials for the closure phase, particularly topsoil.
- Properly managing and isolating the generated waste rocks and rejects from the mineral processing process to prevent the risk of interaction with and contamination of water and soil.
- Gradually conducting landscape rehabilitation and initiating the sealing of the waste disposal site during the filling phase (the installation design has been defined accordingly).

20.8.5 WASTE

Once all the work is completed, an inspection will ensure that all the waste present in the industrial complex has been completely dispatched by accredited entities to appropriate locations. The waste expected during the deactivation phase and its destination are listed in the EIA report.

20.8.6 MONITORING

To ensure that the deactivation phase meets all safety standards, it is necessary to perform quality control of the related activities. Therefore, monitoring the dismantling and demobilising of the structures is crucial, ensuring the absence of mining and non-mining waste, as well as ensuring proper environmental and safety conditions, among other inspection measures.

Process quality control will be by the technical supervisor, whose position will be defined in conjunction with the responsible personnel for deactivation.

The Lagoa Salgada Project will implement monitoring campaigns, considering that extraction, mineral processing, and management of mining waste have a significant impact. Therefore, the use of groundwater resources, waste management, and geotechnical monitoring of urban development plans and infrastructure installation will be monitored.

Monitoring during the deactivation and mine closure phase, in addition to the Monitoring Plans included in the EIA (Environmental Impact Assessment), should reinforce the verification and assessment of the quality of activities related to dismantling facilities and the absence of non-mining waste in the mining area, ensuring proper environmental and

safety conditions. In addition, it is important to highlight monitoring related to the stability and safety of the designed topographic recovery benches.

Overall, monitoring will be the responsibility of the Mine Technical Director and the personnel in charge of deactivation, who will continuously perform this function throughout the deactivation.

21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST ESTIMATES

The capital expenditure (Capex) required for implementing the Lagoa Salgada Project has been estimated to a level of accuracy of +/-10% to 15% using the outcomes of the basic engineering study completed by Quadrante, MinePro, IGAN, SLR and RMS (i.e., equipment selection, sizing, and specification; civil, concrete, mechanical, structural, electrical and instrumentation material takeoffs; construction and management services; taxes, etc.) following both international and Portuguese standards.

Capital cost estimates were developed, for the construction of an underground mining and processing operation capable of treating 1,200,000 tonnes per annum of ore to produce copper, zinc, lead, and tin concentrates, also including credits for gold, and silver. Gold and silver bullion will be also produced.

Total capital expenditures of $255.34 million are estimated for the LOM, $267.24 million with contingency. The initial CAPEX (first eighteen construction months) is $152.45 million, $164.35 million with contingency. The capital cost estimate is divided into ten broad-scope areas:

- Support infrastructure.
- Tailings storage facility.
- Paste-fill plant.
- Mining processing plant.
- Underground capital.
- Mobile equipment.
- External connections.
- Box cut.
- Closure cost.
- Other Capex/studies.

A summary of the estimated LOM capital costs is presented in Table 21.1 and details of the cost breakdowns are summarised in the sub-sections that follow.

The expenditure profile for Capex is illustrated in Figure 21.1. The initial two years of the profile represent the initial capital expenditures, while the increase in year 12 is directly

associated with closure costs. The Capex estimate was incorporated into the project Master Schedule to generate the Capital Expenditure "S-Curve."

The Capex excludes:

- Any additional exploration costs.
- Any financing costs, including capitalised interest and commitment fees.
- Costs incurred to establish the operations and conduct feasibility studies prior to project financing.
- Working capital, which refers to operating costs beyond the scope of Capex that are incurred during the construction and ramp-up of the project, net of revenue generated in the same period.
- bridging costs required to prepare the project for construction, such as detailed engineering FEED, land acquisition, and further geotechnical work.

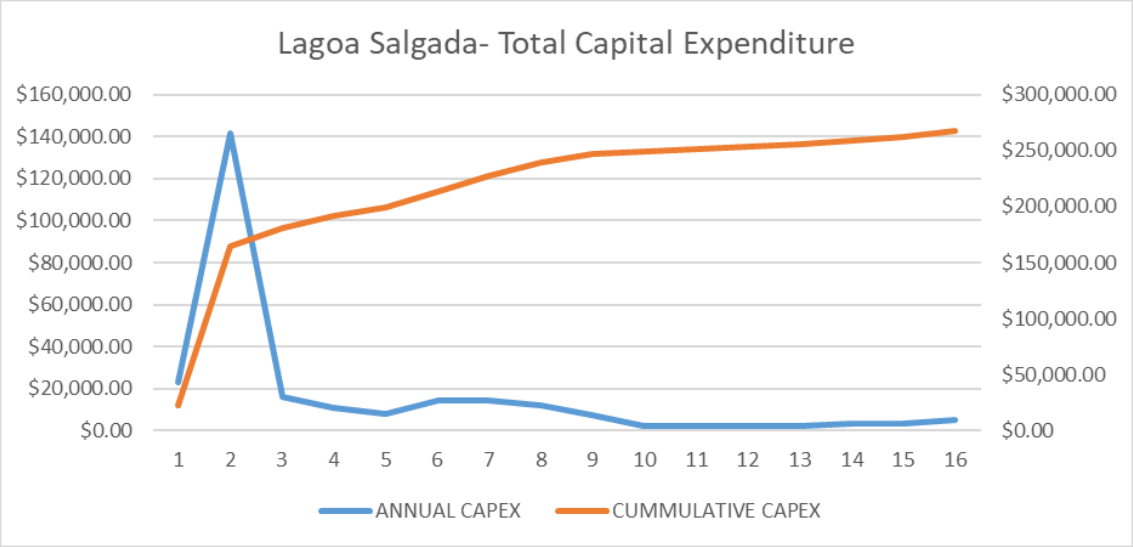


Figure 21.1 - Lagoa Salgada capital expenditure profile

21.1.1 METHODOLOGY

The capital cost estimate meets the criteria to be classified as a Class 3 estimate, as defined by AACE International with an approximate accuracy of ±15%. All costs included in the estimate are reported in American dollars.

The capital cost estimate is derived from a comprehensive bottom-up approach that relies on key engineering deliverables to determine the project's scope. This scope is described and quantified through material takeoffs (MTOs) presented as a series of line items.

The initial capital costs for the project period encompass all expenses incurred over the first eighteen months of construction phase. On the other hand, sustaining capital costs are accrued from that point onwards until the end of the mine's operational lifespan. Pre-production capital costs have been categorised as:

- Support Infrastructure (electromechanical, buildings, internal roads, and hydraulic infrastructures).
- Tailings storage facility (phase 1).
- Paste fill plant (north ore body).
- Processing plant.
- Underground capital (horizontal development, vertical development, UG construction, UG electrical distribution, UG dewatering).
- Mobile equipment.
- External connections (electrical grid connection, and external roads).
- Box cut.

The sustaining capital costs are:

- Support Infrastructure (sustaining).
- Tailings storage facility (expansion – Phase 2 and 3).
- Paste fill plant (expansion to south ore body).
- Processing plant (leaching expansion and spare parts).
- Underground capital (horizontal development, vertical development, main fans, UG construction, UG electrical distribution, UG dewatering).
- Mobile equipment.
- Other CAPEX/studies.

The feasibility study (FS) estimates the initial capital costs of $164.35 million. This estimate includes a contingency of $11.90 million. A 10% contingency was applied to the initial CAPEX, excluding the Mining Equipment.

Table 21.1 summarises the estimated capital costs for the life of the mine (LOM), while the subsequent sub-sections provide further details on the breakdown of these costs.

Table 21.1 – Capital Cost Estimate ($ Millions)

Description	Initial Capital Cost	Sustaining Capital Cost	Total Capital Cost
Mining			
Development	17.85	24.88	42.73
Mining (non-development)	32.41	36.91	69.32
Total Mining	**50.26**	**61.79**	**112.05**
Processing Plant			
Mineral Processing	67.53	0.94	68.47
Paste Plant	5.3	3.9	9.2
Total Processing	**72.83**	**4.84**	**77.67**
Other			
Support Infrastructure	18.51	1.05	19.56
Tailings Storage Facility	5.54	29.2	34.74
External Connections	3.92	0	3.92
Box Cut	1.39	0	1.39
Other CAPEX/Studies	0	0	0
Total Other	**29.36**	**30.25**	**59.61**
Total	**152.45**	**96.89**	**249.33**
Contingency	**11.90**	**0**	**11.91**
Total with Contingency	**164.35**	**96.89**	**261.24**
Closure Costs	0	6	6
Pre-Commercial Production Revenue	-61.86	0	-61.86
Pre-Commercial Production Costs	14.26	0	14.26
Net Capital Costs	**116.77**	**102.89**	**219.65**

1. Pre-production costs are operating costs that occur prior to full mill operating.
2. Pre-commercial production costs include fees for smelting and refining, operating costs over 18 months, and royalties.

21.1.2 MINING CAPITAL COSTS

The capital costs associated with the project's mining activities primarily encompass the development of underground mine workings, both laterally and vertically, as well as the procurement and installation of stationary mine infrastructure. The mobile equipment used in the project is expected to be a combination of leased and purchased assets.

In the direct costs estimation, it is assumed that one contract related to vertical development will commence over the first 18 months of operation as part of the sustaining capital costs.

The UG lateral development and UG of all installations, except vertical development, will be performed by teams of Ascendant's own labour force, including pre-production mining.

No contingency was applied to the Mobile Equipment.

The capital cost estimate for non-development mining activities is provided in Table 21.2, while the estimate for development capital costs is presented in Table 1.7.

Table 21.2 – Mining (Non-Development) Capital Cost Estimate ($ Millions)

Description	Initial Capital Cost	Sustaining Capital Cost	Total Capital Cost
Mining (non-development)			
Main Ventilations Fans	-	0.86	0.86
Mobile Equipment	24.91	24.9	49.81
UG Construction	0.38	2.63	3.00
• *Sumps*	0.04	0.23	0.27
• *Transformer Bays*	0.03	0.18	0.21
• *Refuge Chambers*	0.12	0.54	0.67
• *Vent Doors*	0.17	1.59	1.75
• *Others*	0.02	0.08	0.10
UG Electrical Distribution	0.54	3.79	4.33
• *UG Substation*	0.34	1.20	1.54
• *Distribution Boards*	0.11	0.65	0.76
• *Distribution Cells*	0.09	1.11	1.20
• *Gensets*	0.00	0.83	0.83
UG Dewatering	0.90	3.32	4.22
• *UG Piping Dewatering*	0.43	1.15	1.58
• *UG Main Pumps Dewatering*	0.00	1.71	1.71
• *UG Secondary Water Pumps*	0.20	0.20	0.40
• *Fresh Water Piping*	0.27	0.27	0.53
Other Mine Capital	0.47	1.41	1.88
Mine General	5.22	0	5.22
Total	**32.41**	**36.05**	**68.46**
Contingency	0.75	1.11	1.87
Total with Contingency	**33.16**	**37.16**	**70.33**

Table 21.3 - Mining (Development) Capital Cost Estimate ($ Millions)

Description	Initial Capital Cost	Sustaining Capital Cost	Total Capital Cost
Mining (development)			
Horizontal Development	8.34	21.63	29.97
Vertical Development	1.07	3.25	4.32
Total	**9.41**	**24.88**	**34.29**
Contingency	0.94	-	0.94
Total with Contingency	**10.35**	**24.88**	**35.23**

21.1.3 MINERAL PROCESSING & SURFACE INFRASTRUCTURE CAPITAL COSTS

Mineral processing and surface infrastructure capital costs are summarised in Table 21.4. Project period capital is estimated to be $106.1M. Initial capital is estimated to be $96.5M with $9.7M contingency and sustaining capital is estimated to be $4.8M.

The process plant capital cost was estimated using engineering documents developed during the Feasibility Study, mainly:

- Process design basis:
  - Process Design Criteria (PDC).
  - Mechanical Equipment List (MEL).
  - Process Flow Diagrams (PFD).
- Tender Packages (equipment and contract packages):
  - Site Conditions.
  - Engineering Design Criteria (EDC).
  - Scopes of work and datasheets.
- Drawings:
  - General site layout.
  - Process and non-process building/ equipment layouts.
  - Conveyor profiles.
  - Power distribution and single line diagrams (SLD).

- Load list.

- Material Quantities:
  - Cut, fill, and rock excavation from Civil3D with input from the geotechnical investigation report.
  - Steel and concrete takeoff using major equipment and building loads.
  - Platework takeoff from drawings models.
  - Major piping takeoff from layout drawings.
  - Electrical bulks takeoffs from layout drawings.
  - Valve & instrumentation estimate from PFD.
  - Preliminary I/O count for control system.

These costs encompass various aspects such as site preparation, roads, complete process buildings equipped with machinery, pipework, instrumentation, and controls. They also include the backfill plant, ROM pad, box cut, water management, power supply and distribution, as well as other supporting non-process buildings and facilities mentioned in the infrastructure description section.

Table 21.4 - Process and Surface Infrastructure Capital Cost ($ million)

Description	Initial Capital Cost	Sustaining Capital Cost	Total Capital Cost
Support Infrastructure			
Electromechanical Project	5.00	0.00	5.00
Buildings	4.19	0.00	4.19
Internal Roads	2.79	0.00	2.79
Hydraulic Infrastructures	6.53	0.00	6.53
Paste-Fill Plant			
Backfill Plant	4.60	0	4.6
Piping	0.70	3.9	4.6
Process Plant			
Mechanical Equipment	51.70	0.00	51.70
Electromechanical Project	5.17	0.00	5.17
Exterior Infrastructures	0.19	0.00	0.19
Control Room	0.48	0.00	0.48
Milling	2.06	0.00	2.06
Reagents Storage & Mixing	0.45	0.00	0.45

Description	Initial Capital Cost	Sustaining Capital Cost	Total Capital Cost
Concentrate Handling	1.30	0.00	1.30
Primary Crusher	0.22	0.00	0.22
Flotation Area	1.90	0.00	1.90
Leaching Area	0.00	0.94	0.94
Silo & Bypass Stockpile	0.83	0.00	0.83
Process Water	0.10	0.00	0.10
Air Services	0.12	0.00	0.12
Conveyor Belts	0.77	0.00	0.77
Pipe Racks	1.52	0.00	1.52
Piping	0.54	0.00	0.54
External Connections			
Grid Connection	2.42	0.00	2.42
External Water Supply	0.00	0.00	0.00
External Road Access	1.50	0.00	1.50
Box Cut			
Excavation & Backfill	0.37	0.00	0.37
Temporary Slope Stabilisation	1.02	0.00	1.02
Clay-Based Grounding Cofferdamm	0.00	0.00	0.00
Access Gallery	0.00	0.00	0.00
Other CAPEX Studies	-	-	-
Total	**96.49**	**4.84**	**101.33**
Contingency	9.65	0.00	9.65
Total with Contingency	**106.14**	**4.84**	**110.98**

Sustaining capital costs are mainly associated with expanding the processing plant to install a leaching facility to recover gold and silver from the Gossan domain aligning with the production schedule.

All the details related with disciplines and material costs are in the bill of quantities (BoQ), document T2021-620-Lagoa_Salgada-CAPEX_DFS_10.

21.1.4 TAILINGS STORAGE FACILITY (TSF)

A high-level summary of the estimated CAPEX and sustaining costs for the TSF and reclaim pond is presented in Table 21.5 below.

The cost estimate has been based on the following assumptions:

- TSF required to store 9.3 Mt of tailings and 1.0 Mt of development rock.
- The TSF will be developed within 3 approximately equal-sized cells. Cell 1 will be developed first at the south end of TSF, while Cells 2 and 3 would be developed progressively during operations.
- At startup, water will be managed within Cell 2 by partially excavating and lining the cell. A 3m high separator dyke will divide Cell 1 from the pond in Cell 2.
- Cut required to provide fill for TSF dams and ensure positive drainage to reclaim pond. Soils will either be cut to fill for dams, or stockpiled for future dam raises or closure cover.
- Dam fill quantity for startup CAPEX is the minimum required to store 1 year of tailings (i.e. 1.2 Mt).
- A surplus of soil cut for the startup cell will be generated and may be used to construct more dams rather than temporarily stockpiling.
- Rates are based on other historical projects in which SLR has been involved, adjusted for global inflation up to 2022, and the estimate does not include detailed design, operation, management, surveillance, and closure costs.
- It is recommended that the estimated quantities be further refined during the next design phases before being used for tender purposes.

Table 21.5 - Tailing Capital Cost Summary

Description	Initial Capital Cost	Sustaining Capital Cost	Total Capital Cost
TSF Phase 1			
Strip	0.21	0.00	0.21
Cut	2.35	0.00	2.35
Fill	0.34	0.00	0.34
Liner	2.90	0.00	2.90
TSF Phase 2a			
Strip	0.04	0.00	0.04
Cut	0.56	0.00	0.56
Fill	0.21	0.00	0.21
Liner	1.30	0.00	1.30
TSF Phase 2b			
Strip	0.00	0.13	0.13

Description	Initial Capital Cost	Sustaining Capital Cost	Total Capital Cost
Cut	0.00	1.07	1.07
Fill	0.00	2.16	2.16
Liner	0.00	1.58	1.58
TSF Phase 3			
Strip	0.00	0.12	0.12
Cut	0.00	0.00	0.00
Fill	0.00	15.73	15.73
Liner	0.00	3.04	3.04
Reclaim Pond			
Strip	0.00	0.08	0.08
Cut	0.00	1.18	1.18
Liner	0.00	1.34	1.34
Seepage Collection			
Ditches	0.00	0.32	0.32
Ponds	0.00	0.08	0.08
Total	**7.92**	**26.83**	**34.75**
Contingency	0.79	0.00	0.79
Total with Contingency	**8.71**	**26.83**	**35.54**

21.1.5 CLOSURE COSTS

Closure costs for the site are estimated to be $6.0M. These costs were estimated based on engineering documents and layouts developed during the feasibility design. Closure costs include all surface infrastructure (tailings and the process plant site, anticipated monitoring requirements, and anticipated maintenance).

Materials and quantities for the closure costs are based on the following:

- Neat line estimates
- Conceptual closure details
- Monitoring durations for similar projects

For estimating the direct costs, it has been assumed that all work will be carried out by an independent contractor. Costing inputs to the estimate was derived from the following:

- Earthworks – contractor supply and install unit rates (multiple budget bids).

- Demolition - historic contractor pricing from similar works.
- Monitoring and Maintenance – recent pricing based on similar closure plans.

21.1.6 CONTINGENCIES

Contingencies were incorporated into the estimate by conducting a comprehensive analysis of cost and schedule risks outlined in the capital cost estimate report. Each major component of the CAPEX estimate underwent a separate evaluation to determine its potential for cost increase. These evaluations resulted in assigning a percentage increase to the respective estimated totals for each area.

The total contingency costs were subsequently calculated by considering the weighted factors outlined in the cost summary tables mentioned earlier. Only contingencies for the initial capital costs were considered, excluding the mobile equipments.

21.2 OPERATING COST ESTIMATES

Ascendant, Quadrante, IGAN, and MinePro collaborated to develop operating cost estimates, which serve to determine the annual production costs. These unit costs are presented as total Life of Mine (LOM) expenses and expressed in dollars per tonne processed. All costs included in the estimate are reported in US dollars.

The estimated total operating expenditures over the LOM period amount to $540.32 million. However, certain non-capital expenditures occurring before the processing plant becomes operational over the first eighteen months are categorized as Pre-production operating costs. These costs, totalling $8.5 million, are excluded in the total operating cost expenditure summarised in Table 1.8.

The operating cost estimate is divided into three main areas: Mining, Processing and General and Administrative (G&A). The detailed operating costs presented here are based on design criteria for the mine and process plant, as well as engineering information and unit cost data obtained from various sources.

The operating cost estimate meets the criteria to be classified as a Class 3 estimate, as defined by AACE International, with an approximate accuracy level of ±15%. It is important to note that all costs included in the estimate are reported in US dollars.

The operating cost estimate has been meticulously calculated using a comprehensive bottom-up approach, which relies on crucial engineering deliverables that define the scope of the project. The scope itself was described and quantified through a series of line items within material takeoffs (MTOs).

Table 1.8 provides a summarised overview of the estimated Life of Mine (LOM) operating costs. For a more detailed breakdown of the costs, please refer to the subsequent sub-sections. These sub-sections provide a comprehensive summary of the cost breakdowns, allowing for a closer examination of the various components involved.

Table 21.6 - Operating Cost Estimate

Operating Costs	Cost per unit processed ($/t)	LOM Total ($ Millions)
Mining Costs – Drift & Fill	25.24	269.96
Mining Costs – LHOS	17.72	
Processing Costs – Gossan	15.58	241.47
Processing Costs – TMS	13.71	
Processing Costs – PMS	18.72	
Processing Costs – Stringer	15.37	
Processing Costs – Stockwork	15.37	
G&A Costs	2.00	28.89
Total	**36.45**	**540.32**

21.2.1 MINE OPERATING COSTS

The mine operating costs have been meticulously calculated from the ground up, utilising detailed mine development and production schedules. Various unit costs were taken into account, including those provided by suppliers, benchmark data, and operational experience. These costs were scheduled by specific time periods and categorized by type for each fundamental mining activity, such as lateral development, transverse stoping, maintenance, and general mine/indirect activities.

For a summarised overview of the overall mine operating expenses, please refer to Table 21.7. This table provides a concise summary of the mine OPEX, enabling a clear understanding of the cost breakdowns associated with the mining operations.

Table 21.7 - Summary of Mine Operating Costs

Mine Operating Costs	LHOS ($/t)	D&F ($/t)	Total ($/t)
Supervision	0.41	0.41	0.41
Safety &Training	0.08	0.08	0.08
Ore Development	2.37	0.00	2.06
Waste Development	0.01	0.01	0.01
Waste Handling	0.07	0.07	0.07
D&F Stoping	0.00	13.69	1.76
Longhole Stoping	4.29	0.00	3.73
Ore Handling - UG	2.04	2.04	2.04
Ore Handling - Surface	0.00	0.00	0.00
Backfilling	3.72	3.72	3.72
Mine Rehabilitation	0.31	0.31	0.31
Mine Services	1.48	1.48	1.48
Constr. & Utility Crew	0.65	0.65	0.65

Mine Operating Costs	LHOS ($/t)	D&F ($/t)	Total ($/t)
Maint. Supervision	0.69	0.69	0.69
Maintenance	1.81	1.81	1.81
Technical Services	1.35	1.35	1.35
Total	**19.26**	**26.29**	**20.17**

Table 21.8 - Summary of Mine Direct/Indirect Operating Costs

Mine Operating Costs	LHOS ($/t)	D&F ($/t)	Total ($/t)
Direct Costs	12.80	19.84	13.71
Indirect Costs	6.46	6.46	6.46
Total	19.26	26.29	20.17

Table 21.9 - Mine Operating Costs by Item

Mine Operating Costs	Total ($/t)
Labour	4.50
Equip. Operating	7.19
Supplies	7.55
Power	0.66
Outside Service	0.26
Total	**20.80**

Ascendant assumed a reduction in mine operating costs for LHOS from $19.26/t to $17.72/t and for D&F from $26.29/t to $25.24/t by incorporating BEV vehicles in dump trucks, LHDs, and mixer trucks. This upgrade involved not only the use of electric vehicles but also the implementation of automation. The details of this study can be found in the Automation Study for Lagoa Salgada. Additionally, an increase in mobile equipment CAPEX cost of $10M was taken into account to accommodate this technology.

21.2.2 PROCESSING OPERATING COSTS

Processing operating costs are associated with mineral processing including crushing, ore storage, grinding, flotation, tailings and concentrate dewatering, reagents and utilities.

Process plant operating costs have been estimated from first principles, using mechanical equipment specifications for estimation of power consumption, metallurgical testwork for reagent and grinding media consumption estimates, and labour schedules and salary build-ups for process labour. Maintenance costs were estimated as a percentage of mechanical equipment supply cost.

Quotations for consumables such as reagents, lime, and grinding media were obtained from suppliers. Major consumables for mechanical equipment including crusher liners, SAG/ball mill liners and filter cloths/bags were sourced from vendor equipment quotations.

Annual and life-of-mine operating costs for the mineral processing are shown in Table 21.10.

Table 21.10 - Process Operating Costs Summary

Processing Operating Costs	Gossan ($/t)	TMS ($/t)	PMS ($/t)	Stringer ($/t)	Stockwork ($/t)
Salaries	1.52	1.52	1.52	1.52	1.52
Power	2.45	2.62	3.27	2.62	2.62
Grinding Media	3.88	4.14	5.18	4.14	4.14
Chemicals	5.61	3.32	6.64	4.98	4.98
Fuel	0.12	0.12	0.12	0.12	0.12
Operating Material	0.36	0.36	0.36	0.36	0.36
Operating Spares	0.71	0.71	0.71	0.71	0.71
Repair Material	0.19	0.19	0.19	0.19	0.19
Laboratory	0.73	0.73	0.73	0.73	0.73
Total	**15.58**	**13.71**	**18.72**	**15.37**	**15.37**

22 ECONOMIC ANALYSIS

22.1 INTRODUCTION

An economic model based on various engineering estimates was prepared for the Project to estimate annual cash flows and assess the sensitivity of the Project to certain economic parameters.

The Project indicates an after-tax cash flow of $260.9 million, after-tax NPV of $147.1 million, and after-tax IRR of 39%. The Project is most sensitive to the OPEX costs and the zinc price.

22.2 CAUTIONARY STATEMENT

The economic analysis results rely on forward-looking information, which is influenced by various known and unknown risks, uncertainties, and other factors that could potentially lead to significant deviations from the information presented in this document. This section contains forward-looking statements that encompass, but are not limited to, statements concerning:

- Future commodity prices (copper, zinc, lead, tin, gold, and silver).
- Currency exchange rate fluctuations, primarily the EUR:USD rate.
- Estimation of Mineral Reserves.
- Realisation of Mineral Reserve Estimates.
- Estimated costs and timing of capital and operating expenditures, including labour, materials, consumables, supplies, and services.

22.3 ASSUMPTIONS

The assumptions used in the economic model are grouped in five categories summarized in this chapter. More details can be found in the relevant chapters:

- Financial
- Production.
- Revenue.
- Operating Costs.
- Capital Costs.

The cash flow estimate includes revenue, costs, taxes, and other factors applicable to the Project from the beginning of final design/construction to the end of mine life. Corporate obligations, financing costs, and taxes at the corporate level are excluded.

The model was prepared using mining schedules that were estimated on an annual basis. The cash flow model was based on the following assumptions:

- All costs are reported in US dollars (US$) and referenced as '$', unless otherwise stated.
- 100% equity ownership.
- No provision for inflation, however 8% cost of capital was considered.
- Exploration program costs are deemed outside of the Project.
- Any additional project development costs, including feasibility studies, or permitting activities, are excluded from the analysis.
- Annual gross revenue is determined by applying estimated metal prices to the annual payable metal estimated for each operating year.
- Constant commodity pricing was used for the economic analysis.
- A constant exchange rate assumption of €1.00 = US$1.06 was used in the economic analysis.

22.4 METHODOLOGY USED

The economic analysis for the Project employed the Discounted Cash Flow (DCF) analysis methodology. This involved estimating the total value of future cash flows (both inflows and outflows) and then discounting them by a predetermined cost of capital to determine their present value. The Net Present Value (NPV) is the aggregate of all discounted future cash flows. The Internal Rate of Return (IRR) represents the discount rate at which the NPV becomes zero and serves as an indicator of the project's potential profitability. The payback period is calculated starting from the commencement of mineral processing.

The model was developed using Microsoft Excel spreadsheet software.

22.5 FINANCIAL MODEL PARAMETERS

22.5.1.1 MINERAL RESERVE AND MINE LIFE

The mine plan presented in Section 15.0 of this report was used as the basis for revenue and cost estimates in the DCF, and also represents the current Mineral Reserve estimate. The mine plan excludes all material from the inferred resource category.

The stated Probable Mineral Reserve is 14.8 Mt. The reserve is divided by geological domains such as: Gossan (GOS), Massive Transition Zone (TMS), Stringer (STR), Stockwork (STW) and Fresh Massive (PMS). All the domains will be processed in campaigns and the average grades and tonnes are:

- GOS: 1.20 Mt @ 0,1% Cu, 2,4% Pb, 0,4% Zn, 52 ppm Ag, 1,0 ppm Au, 0,3% Sn
- TMS: 0.88 Mt @ 1,0% Cu, 2,2% Pb, 0,2% Zn, 110 ppm Ag, 1,1 ppm Au, 0,2% Sn
- STR: 0.55 Mt @ 0,2% Cu, 0,2% Pb, 0,5% Zn, 13 ppm Ag, 0,06 ppm Au
- PMS: 4.40 Mt @ 0,3% Cu, 2,7% Pb, 3,6% Zn, 62 ppm Ag, 0,68 ppm Au, 0,1% Sn
- STW: 7.80 Mt @ 0,4% Cu, 0,7% Pb, 1,3% Zn, 15 ppm Ag, 0,06 ppm Au

The mine plan envisaged a 16-year LOM. The mill processing profile is presented in Figure 22.1.

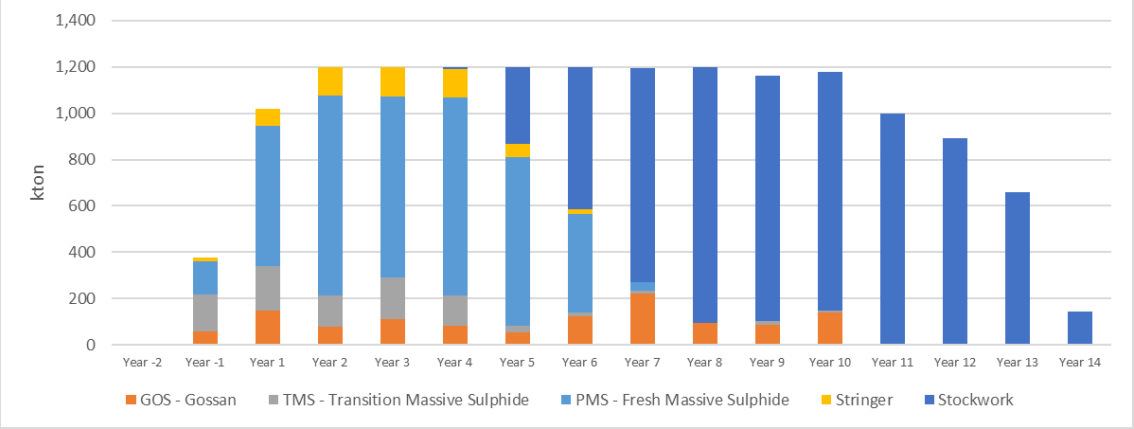


Figure 22.1 - LOM Processing Profile (kTonnes)

The production profile for payable Zn, Cu, Pb and Sn is presented in Figure 22.2. The production profile for payable Au and Ag is presented in Figure 22.3.

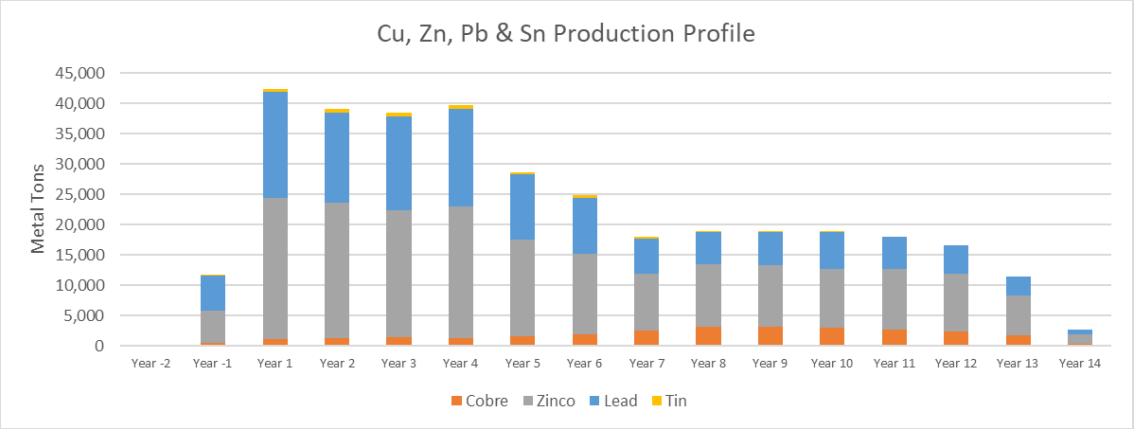


Figure 22.2 – Production profile for Cu, Pb and Zn

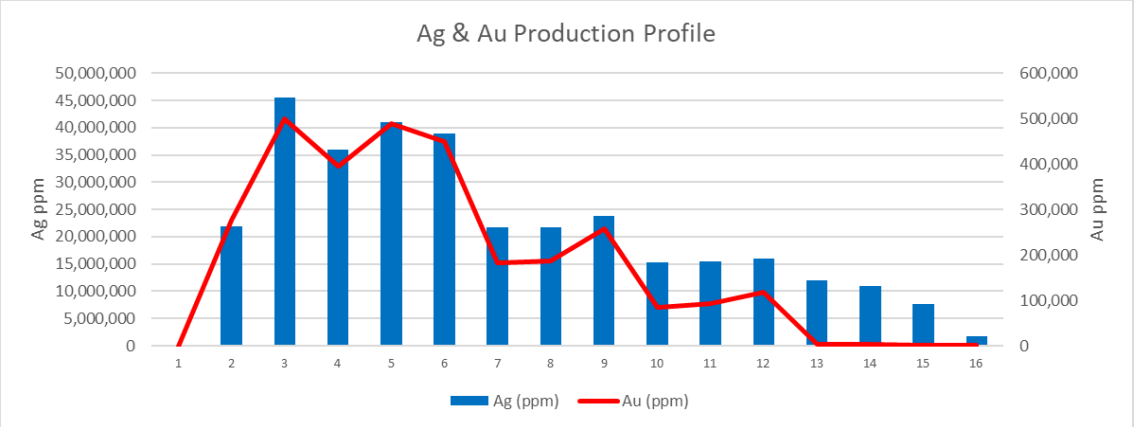


Figure 22.3 – Production profile for Ag and Au

22.5.1.2 METALLURGICAL RECOVERIES

The average metal recoveries that have been used for the economic evaluation are summarised in Table 22.1. Various grade versus recovery formulas were used to calculate recovery metrics on an annual basis within the cashflow model. A detailed review of metallurgical recoveries is provided in Section 13.0 of this report.

Table 22.1 – LOM average metallurgical recoveries

Mine Reserve Domain	Cu	Pb	Zn	Ag	Au	Sn
GOS	-	20%	-	89%	91%	40%
TMS	30%	50%	50%	75%	60%	40%
STR	60%	50%	70%	25%	0.5%	-
PMS	30%	60%	70%	45%	30%	40%
STW	65%	65%	75%	25%	5%	-

22.5.1.3 PAYABLES

The rates used for payables elements in the different concentrates have been applied from smelters consulted during the process. Those rates are stated in the following table.

Table 22.2 – Commercial terms for Cu Concentrate

Payables	Cu	<18% grade, 96.5% payment subject to minimum deduction of 1.2 unit
		<20% grade, 96.5% payment subject to minimum deduction of 1.1 unit
		<30% grade, 96.5% payment subject to minimum deduction of 1.0 unit
		>30% grade, 96.5% payment
	Au	<1 ppm no payment
		1-3 ppm, 90% payment
		3-5 ppm, 92% payment
		5-8 ppm, 93% payment
		8-10 ppm, 94% payment
		>10 ppm, 95% payment
	Ag	<30 ppm no payment
		>30 ppm, 90% payment
Deductions	As	2.5 USD/t for each 100ppm which exceeds 2000ppm, fractions pro rata
	Pb+Zn	2.5 USD/t for each 1% which exceeds 4%, fractions pro rata
	Bi	2 USD/t for each 100ppm which exceeds 300ppm, fractions pro rata
	Sb	2.5USD/t for each 1000ppm which exceeds 1000ppm, fractions pro rata
	Hg	1 USD/t for each 10ppm which exceeds 20ppm

Table 22.3 – Commercial terms for Pb Concentrate

Payables	Pb	95% payment subject to a minimum deduction of 1.5 unit
	Au	<1 ppm no payment
		1-3 ppm, 90% payment
		3-5 ppm, 92% payment
		5-8 ppm, 93% payment
		8-10 ppm, 94% payment
		>10 ppm, 95% payment
	Ag	95% payment subject to a minimum deduction of 25 grams.
Deductions	As	1.5 USD/t for each 0.1% which exceeds 0.5% (until 0,7%).
		3.0 USD/t for each 0.1% which exceeds 0.7%.
	Hg	1.5 USD/t for each 100 ppm which exceeds 100ppm (until 300ppm).
		3 USD/t for each 100 ppm which exceeds 300 ppm.

Table 22.4 – Commercial terms for Zn Concentrate

Zn Concentrate		
Payables	Zn	85% payment subject to a minimum deduction of 5 units.
	Au	>1 ppm no payment
		1-3 ppm, 90% payment
		3-5 ppm, 92% payment
		5-8 ppm, 93% payment
		8-10 ppm, 94% payment
		>10 ppm, 95% payment
	Ag	65% payment subject to a minimum deduction of 1.5 oz.
	Fe	3 USD/t for each 1% the content exceeds 7.5%, fractions pro rata

Table 22.5 – Commercial terms for Sn Concentrate

Sn Concentrate		
Payables	Sn	95% payment
	Pb	75% payment
Deductions	As	20 USD/t
	Zn+Ag+Ni+Co	300 USD/t
	Bi	300 USD/t
	Cu	300 USD/t
	Sb	300 USD/t
	Fe+Mg+WO3	300 USD/t
	FR+Cl	300 USD/t

Table 22.6 – Au and Ag bullion commercial terms

Au/Ag Bullion		
Payables	Au	99.5% payment
	Ag	90% payment

22.5.1.4 FREIGHT, SMELTING AND REFINING COSTS

Industry rates and terms for smelting and refining have been applied from consultations with local smelters and operators when calculating the discounted cash-flow. The cashflow accounts for payable metal rates, toll treatment charges (TC), smelting and refining charges (RC), and freight.

All the rates are stated in the following table.

Table 22.7 – Toll treatment and refining charges

Charge		Cu Conc	Pb Conc	Zn Conc	Sn Conc	Au/Ag Bullion
TC		29.4 USD/t	78 USD/t	172 USD/t	294 USD/t	29.4 $/kg
RC	Cu	0.03 USD/lb	0.03 USD/lb			
	Au	5 USD/oz	5 USD/oz			20 $/kg
	Ag	0.40 USD/oz	0.80 USD/oz			
	Sn				50 USD/t	
Freight		35 USD/t	35 USD/t	60 USD/t	35 USD/t	

The rates used in the model do not reflect or imply that Ascendant has entered into any agreements relating to these contracts.

Freight of concentrate products to market has been estimated using quantities calculated from production schedules, together with quoted rates from logistics companies, assuming sale to Iberian smelters.

22.5.1.5 METAL PRICES

The metal prices used for the economic valuation considers long term quotations considering projections. Metal prices used are summarised in Table 22.8.

Table 22.8 – Metal Prices Assumptions

Metal	Unit	Price
Copper	USD/lb	3.75
Zinc	USD/lb	1.20
Lead	USD/lb	1.05
Gold	USD/oz	1,700
Silver	USD/oz	22.5
Tin	USD/lb	15.00

In Section 19, a detailed review of metal prices can be found.

22.5.1.6 CAPITAL COSTS

A summary of pre-production (CAPEX) and sustaining capital costs (SUSEX) is provided in Table 22.9. A detailed description of the capital costs is provided in previous section of this report (*Capital and Operating Costs*). The initial CAPEX or pre-production CAPEX considers the capital costs to be incurred to construct all mining and processing units, these costs were estimated by Quadrante, SLR, RMS, Minepro and Ascendant according to the scope breakdown. Total construction capital expenditure of the Project is USD 182 million.

Table 22.9 – Capital Cost Summary

Description	CAPEX (M$)	SUSEX (M$)	TOTAL (M$)
Support Infrastructure	18.5	1.1	19.6
Tailings Storage Facility	5.5	29.2	34.7
Paste-Fill Plant	5.3	3.9	9.2
Processing Plant	67.4	0.9	68.3
Underground Capital	25.5	36.9	62.4
Mobile Equipment	24.9	24.9	49.8
External Connections	3.9	0	3.9
Box Cut	1.4	0	1.4
Closure Cost	0	6	6
TOTAL	**152.4**	**102.9**	**255.3**

The sustaining expenditures are related with mobile equipment fleet replacement schedule, the residue disposal facilities expansion, the underground mining capital development, and closure costs. These costs include the rehabilitation and decommissioning the processing plant, support facilities and closing and rehabilitating the tailings storage facility.

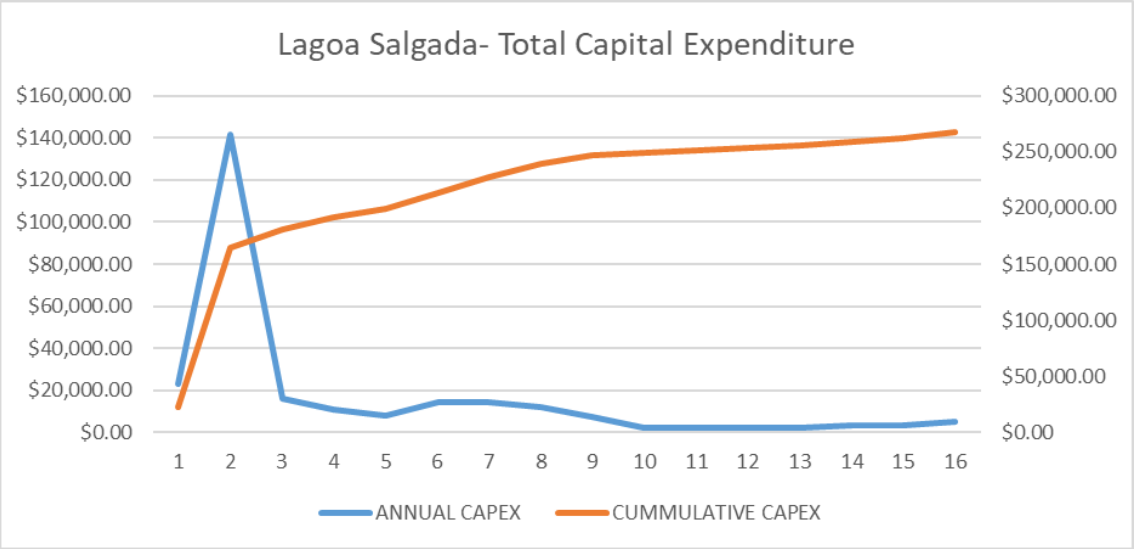


Figure 22.4 – Total capital expenditure (CAPEX) for Lagoa Salgada Project

22.5.1.7 OPERATING COSTS

A summary of LOM operating costs is presented in Table 22.10. A detailed description of operating costs is in previous section of this report (*Capital and Operating Costs*).

Table 22.10 – LOM operating costs summary

Operating Costs		Cost ($/t)
Mining Costs	D&F	25.24
	LHOS	17.72
Processing	Stockwork	15.58
	Fresh MS	13.71
	Transition	18.72
	Gossan	15.37
	Stringer	15.37
G&A		2.00

22.5.1.8 EXCHANGE RATE

Money of origin for each input was converted to US Dollars (USD), the currency on which the economic model is based.

An exchange rate of 1 EUR / 1.06 USD was selected as the average during which both capital and operating expenditure estimates were obtained.

Figure 22.5 – Exchange Rate EUR/USD (source: ECB)

22.5.1.9 ESCALATION RATE

All inputs to the economic model are in real USD. No inflation escalation was applied to the costs or revenues of the Project.

22.5.1.10 GEARING

No debt financing parameters were considered in the economic model.

22.5.1.11 DEPRECIATION

The straight-line depreciation method was used for groups of assets with varying economic life durations determined in line with Portuguese regulations.

22.5.1.12 TAXATION

The following sections describe the taxation included in the capital expenditure, operating expenditure, and economic analysis of the Project.

IRC – Imposto sobre Rendimento das Pessoas Coletivas (Corporate Income Tax)

IRC stands for "Imposto sobre o Rendimento das Pessoas Coletivas" in Portuguese, which translates to "Corporate Income Tax" in English. It is a tax imposed on the income of companies engaged in commercial, industrial, or agricultural activities operating in Portugal or on income earned in Portuguese territory by foreign companies. This tax in Portugal is 21% of the Net Profit.

Derrama Estadual (State Surtax)

This is a taxation on the taxable profit exceeding €1,500,000, which is subject to and not exempt from Corporate Income Tax (IRC). It applies to both resident legal entities operating primarily in Portugal in commercial, industrial, or agricultural activities, and non-resident legal entities that have a permanent establishment in Portugal.

The existence of a state surcharge has made the taxation on taxable profit for legal entities progressive. Generally, there is a 21% tax rate for IRC. However, when the taxable profit exceeds €1,500,000, there is an additional tax that increases as the taxable profit increases.

Therefore, for profits exceeding €1,500,000 up to €7,500,000, the tax rate is 3%. For profits over €7,500,000 up to €35,000,000, the rate is 5%. For profits exceeding €35,000,000, the rate is 9%.

This progressive taxation is structured in brackets, where each bracket or portion of the taxable profit is subject to the corresponding rate. For example, for profits exceeding €7,500,000 up to €35,000,000, a 3% rate is applied to €6,000,000, and a 5% rate is applied to the excess over €7,500,000. When the profit exceeds €35,000,000, a 3% rate is applied to €6,000,000, a 5% rate is applied to €27,500,000, and a 9% rate is applied to the portion exceeding €35,000,000.

Derrama Municipal Grândola (Grândola municipality Local Surtax)

The "derrama" is levied on the taxable profit of legal entities, and its rate is determined annually by municipalities, with a maximum value of 1.5%. The "derrama" is a municipal tax imposed based on the annual resolution of the Municipal Assemblies. The tax applied in the Grândola is 1.4%.

Royalties

A royalty is an amount paid by Ascendant to the Portuguese State to exploit a mineral resource. The terms of royalty payments are laid out in a licensing agreement with the Mining Bureau of Portugal. In the Lagoa Salgada project, the applicable royalties are calculated as follows:

Royalty = Net Sales Value (NSV) x royalty rate

The NSV represents the net sales reported in the company's financial statements.

Royalty rate:

- 3% in the first 3 years of production.
- 4% in the 4th year of production.
- In the 5th year of production and beyond, 5%.

22.6 FINANCIAL RESULTS

A summary of the economic parameters for the Project are illustrated in Table 22.11.

The pre-tax and post-tax cash flows of the Project have been discounted at a rate of 8% to derive the NPVs for the Project. The Project was evaluated excluding debt funding costs and interests.

Table 22.11 – Summary of economic metrics for the Project

Description	Units	Value
Pre-Tax		
NPV @ 8%	$ million	188.8
IRR (Internal Rate of Return)	%	47
Total Cash-Flow	$ million	324.5
Post-Tax		
NPV @ 8%	$ million	147.1
IRR (Internal Rate of Return)	%	39
Payback Period	Years	2
Total Cash-Flow	$ million	260.9

Note: Payback period calculated from start of mineral processing

The post-tax cash-flow is the result of subtracting the total capital expenditure (excluding depreciation) and taxes from the EBITDA (earnings before interest, taxes, depreciation, and amortisation).

The Figure 22.6 shows the annual post-tax cash flow and its cumulative profile.

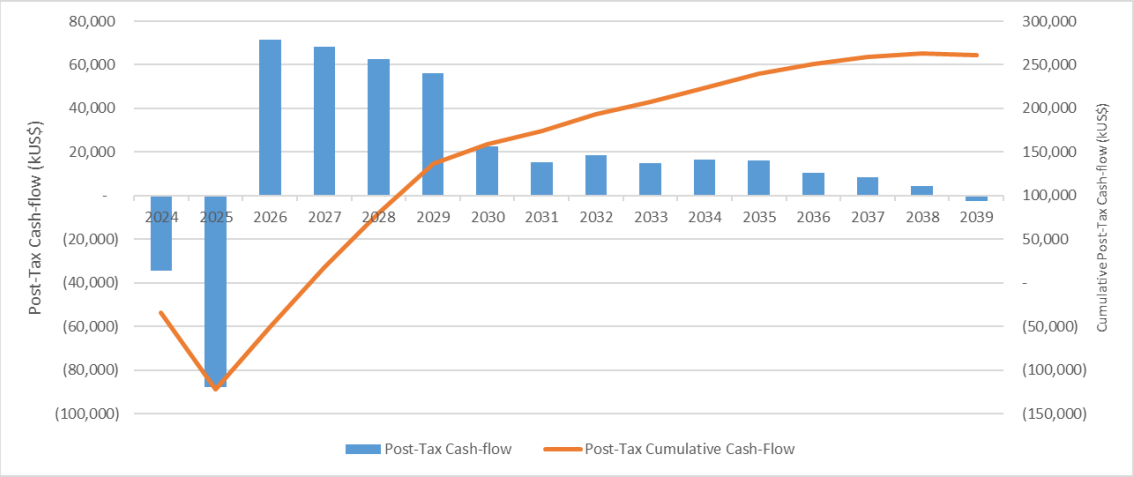


Figure 22.6 - Post-Tax Cash-flow and Cumulative Cash-flow

22.7 PROJECT SCHEDULE

22.7.1 PROJECT CONSTRUCTION PLAN

The Project Construction Plan (PCP) is formulated based on the Construction Strategy, which is developed collaboratively by Quadrante and Ascendant Resources.

The Project Schedule encompasses the planned and scheduled engineering, procurement, construction, commissioning, and ramp-up activities and deliverables necessary to achieve nameplate capacity and the desired product quality.

STRATEGY

The Project Implementation Strategy encompasses three distinct phases and their associated activities.

- Phase 1: Long lead Procurement items for mineral processing major equipment.
- Phase 2: Construction Activities for the Support Infrastructure, Tailings Storage Facility, Mineral Processing Plant, Box-cut, external connections, and underground mining. The phase comprises the following sub-phases:
    - Phase 2.1: Construction activities for Mineral Processing Plant: Primary Crusher, Silo, Milling, Flotation, Concentrates Handling, Reagents, Pebble Crusher, Conveyors, Control Room, Electrical Substation, and E-Rooms.

- Phase 2.2: Construction activities related to support infrastructure: electromechanical project, ancillary buildings, internal roads, and hydraulic infrastructures.

- Phase 2.3: Construction activities related to the tailings storage facility, namely Phase 1. The subsequent phases will be detailed in the next expansion phases (phases 2 and 3).

- Phase 2.4: Construction activities related to underground mining. Mine Portal excavation (box-cut) and initial underground mining development to access first ore production.

- Phase 2.5: Construction activities related to main external connections such as main power grid connections, external water supply pipeline, and external roads/access.

- Phase 3: Commissioning and First Feed to the mineral processing plant. Production Ramp-up.

METHODOLOGY

After gathering all the information, the work methodology followed these steps:

- Selection of productivity indices applicable to the various services of the PQs (Project Queries).
- Organisation of the schedule up to the level aligned with the FEL 3 stage (Viability - Feasibility), using Microsoft Project.
- Generation of the schedule in GANTT chart format.
- Establishing links and dependencies.
- Verification of the entire logical sequence and identification of areas to be adjusted.
- Generation of the project's critical path.

ASSUMPTIONS

The assumptions used to determine the durations (calendar) in the schedule are as follows:

- Work is considered from Monday to Friday, with 8 hours per day.
- All national and state holidays are taken into account.
- No restrictions on weather conditions.

- No restrictions on licenses or permits.
- The project implementation is scheduled to commence on 01/04/2024.
- The acquisition of land for the project is considered as completed before the construction start.
- The duration calculations are based on productivity, and the configuration in this work is Fixed Duration & Units.

22.7.2 PROJECT ANALYTICAL STRUCTURE - PAS

The resume of the project analytical structure is in the figure below (Figure 22.7).

ID	Task Mode	Task Name	Duration	Start	Finish
1		**LAGOA SALGADA PROJECT**	**505 days?**	**Mon 03/06/24**	**Fri 08/05/26**
2		**Project Milestones**	**300 days**	**Fri 06/09/24**	**Fri 31/10/25**
3		Start of Construction	0 days	Fri 06/09/24	Fri 06/09/24
4		Construction Complete	0 days	Fri 31/10/25	Fri 31/10/25
5		First Ore Production	0 days	Fri 24/10/25	Fri 24/10/25
6		**Long Lead Procurement Itens**	**300 days**	**Mon 03/06/24**	**Fri 25/07/25**
12		**Construction**	**505 days?**	**Mon 03/06/24**	**Fri 08/05/26**
13		Mobillzation	30 days	Mon 03/06/24	Fri 12/07/24
14		Clearing site	40 days	Mon 15/07/24	Fri 06/09/24
15		**Support Infrastrucuture**	**200 days**	**Mon 07/10/24**	**Fri 11/07/25**
20		**Tailings Storage Facility**	**240 days**	**Mon 02/12/24**	**Fri 31/10/25**
25		**Primary Crusher**	**170 days**	**Mon 09/09/24**	**Fri 02/05/25**
33		Construction Completion	0 days	Fri 02/05/25	Fri 02/05/25
34		**Silo**	**185 days**	**Mon 09/09/24**	**Fri 23/05/25**
42		Construction Completion	0 days	Fri 23/05/25	Fri 23/05/25
43		**Milling**	**300 days**	**Mon 09/09/24**	**Fri 31/10/25**
51		Construction Completion	0 days	Fri 31/10/25	Fri 31/10/25
52		**Flotation**	**290 days**	**Mon 09/09/24**	**Fri 17/10/25**
60		Construction Completion	0 days	Fri 17/10/25	Fri 17/10/25
61		**Concentrates Handling**	**240 days**	**Mon 09/09/24**	**Fri 08/08/25**
69		Construction Completion	0 days	Fri 08/08/25	Fri 08/08/25
70		**Reagents**	**200 days**	**Mon 09/09/24**	**Fri 13/06/25**
78		Construction Completion	0 days	Fri 13/06/25	Fri 13/06/25
79		**Pebble Crusher**	**95 days**	**Mon 09/09/24**	**Fri 17/01/25**
87		Construction Completion	0 days	Fri 17/01/25	Fri 17/01/25
88		**Conveyor CV01 + Tranfers**	**160 days**	**Mon 09/09/24**	**Fri 18/04/25**
96		Construction Completion	0 days	Fri 18/04/25	Fri 18/04/25
97		**Conveyor CV02**	**210 days**	**Mon 09/09/24**	**Fri 27/06/25**
105		Construction Completion	0 days	Fri 27/06/25	Fri 27/06/25
106		**Conveyor CV03 (silo bypass)**	**160 days**	**Mon 09/09/24**	**Fri 18/04/25**
114		Construction Completion	0 days	Fri 18/04/25	Fri 18/04/25
115		**Conveyor CV04 (mill feed)**	**230 days**	**Mon 09/09/24**	**Fri 25/07/25**
123		Construction Completion	0 days	Fri 25/07/25	Fri 25/07/25
124		**Conveyor CV05 (pebble return)**	**175 days**	**Mon 09/09/24**	**Fri 09/05/25**
132		Construction Completion	0 days	Fri 09/05/25	Fri 09/05/25
133		**Control Room**	**225 days**	**Mon 09/09/24**	**Fri 18/07/25**
140		Construction Completion	0 days	Fri 18/07/25	Fri 18/07/25
141		**Eletrical Substation**	**280 days**	**Mon 09/09/24**	**Fri 03/10/25**
146		Construction Completion	0 days	Fri 03/10/25	Fri 03/10/25
147		**Plant Main E-Room**	**270 days**	**Mon 09/09/24**	**Fri 19/09/25**
153		Construction Completion	0 days	Fri 19/09/25	Fri 19/09/25
154		**Crusher E-Room**	**210 days**	**Mon 09/09/24**	**Fri 27/06/25**
158		Construction Completion	0 days	Fri 27/06/25	Fri 27/06/25
159		**Box Cut**	**185 days**	**Mon 03/06/24**	**Fri 14/02/25**
165		**Underground Mining**	**340 days**	**Mon 20/01/25**	**Fri 08/05/26**
166		Main Ramp	300 days	Mon 20/01/25	Fri 13/03/26
167		Access Levels	160 days	Mon 29/09/25	Fri 08/05/26
168		Vertical Development	60 days	Mon 16/02/26	Fri 08/05/26
169		Fist Ore Production	0 days	Fri 24/10/25	Fri 24/10/25
170		**External Connections**	**1 day?**	**Mon 03/06/24**	**Mon 03/06/24**
174		**Commissioning**	**60 days**	**Mon 03/11/25**	**Fri 23/01/26**

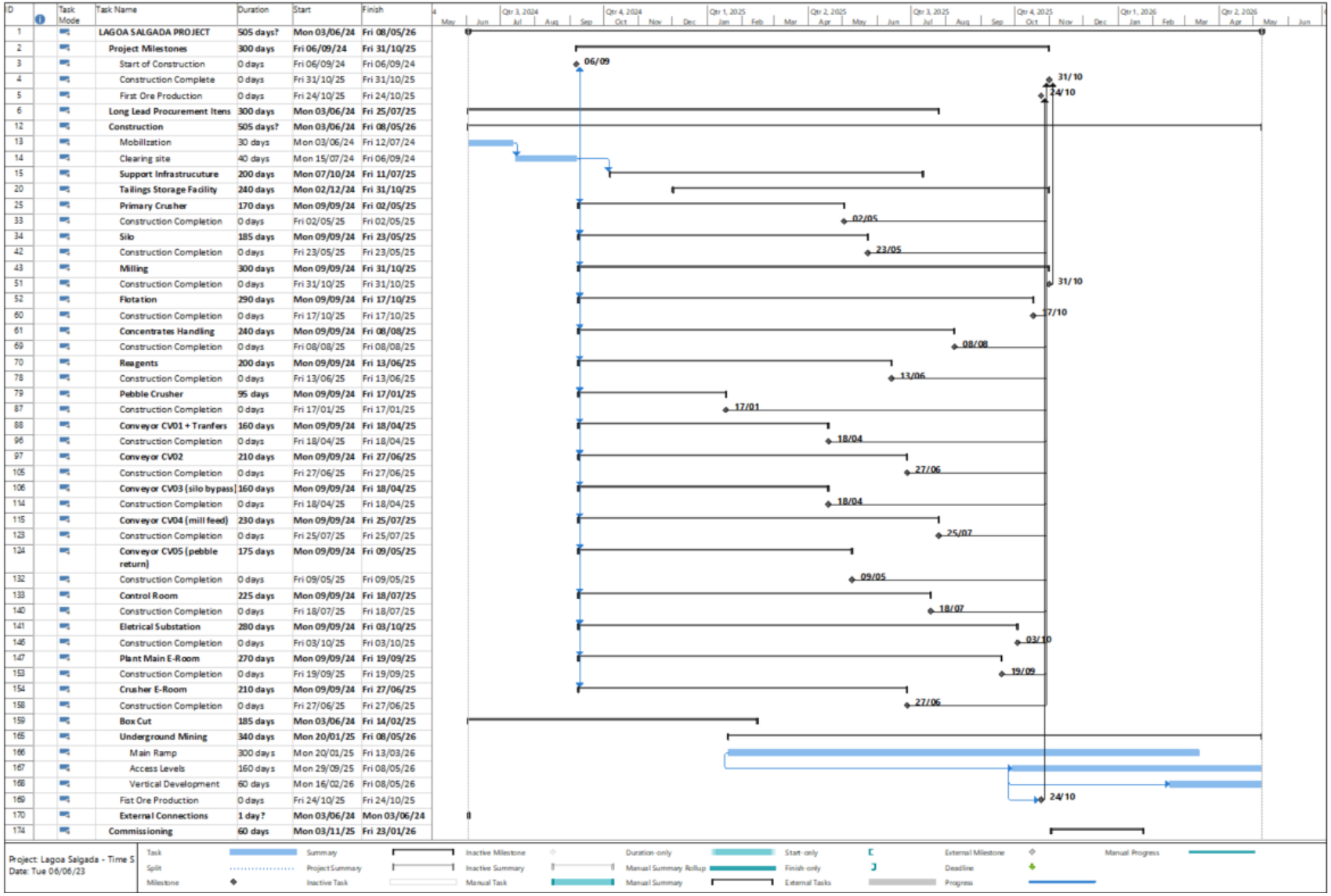


Figure 22.7 – Lagoa Salgada Master Schedule

The overall project schedule is summarised in Figure 22.7 above. This schedule reflects the current status of the project, which may be subject to changes due to ongoing studies and optimisation of engineering solutions, as well as streamlining of construction execution resulting from value engineering studies estimated for the upcoming stages of basic engineering consolidation and detailing.

22.7.3 RECOMMENDATIONS

The recommendations regarding Planning and Schedule for the next stages of basic engineering consolidation and detailing are as follows:

For the basic engineering consolidation stage:

- Identify if there are any impacts on the Project Structure due to Value Engineering studies and basic engineering consolidation and update it if necessary.
- Confirm whether the productivity indices used are consistent with any changes that may occur in basic engineering, especially in Engineering Queries (EQs).
- Confirm with Procurement if the supply deadlines provided are still valid.
- Update the schedule based on Value Engineering studies and basic engineering consolidation. Define the Baseline for control of implementation, reflecting the view of the FEL 3 stage - Consolidated Basic Engineering.

For the detailing stage, the schedule may be revisited and frozen again based on the latest information from service and equipment suppliers.

The frozen schedule may impact the Baseline, which can be adjusted to align with the project's progress or indicate deviations from the FEL 3 stage Baseline. The decision on which Baseline will be adopted for implementation control should be made prior to the detailing phase.

23 ADJACENT PROPERTIES

There are no properties of any significance directly adjacent to the site.

The most relevant current and past mining operations in the IPB are shown in Figure 23.1 below.

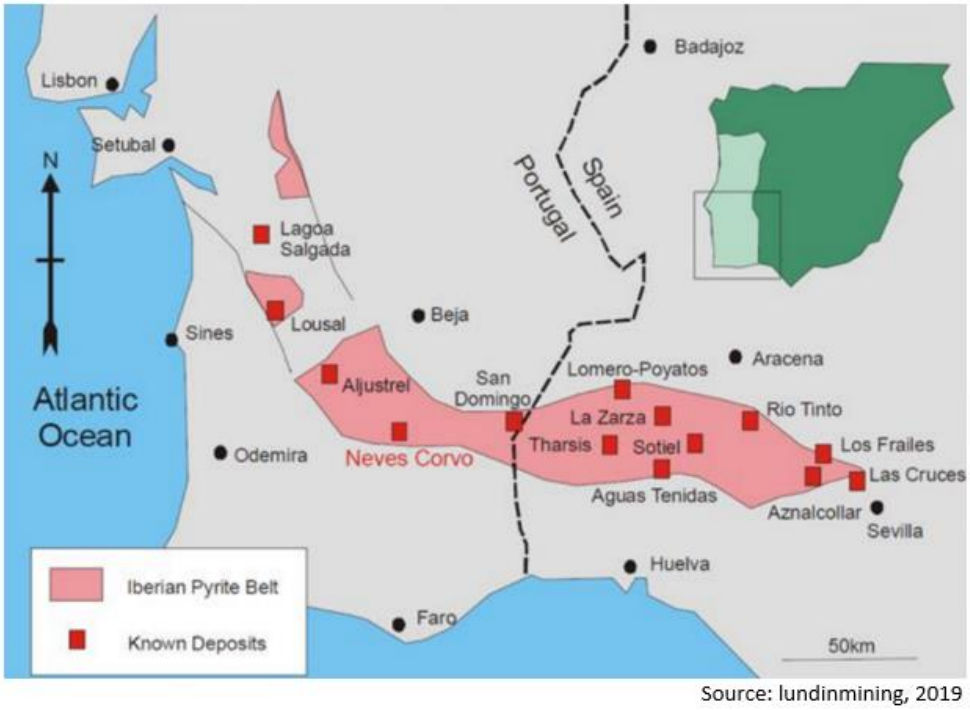


Figure 23.1 – Current and past mining operations in the IPB (source: lundinmining, 2019)

The following mining operations are currently active in the IPB:

- Aljustrel Mine (Almina).
- Neves-Corvo Mine (Somincor – Lundin Mining).
- Aguas Teñidas, Sotiel, and Magdalena Mines (Matsa – Trafigura/Mubadala).
- Rio Tinto Mine (Atalaya Mining).
- Cobre Las Cruces Mine (CLC – First Quantum Minerals) – Currently reprocessing mined tailings.

In addition to these active mining operations, several projects are at the exploration or permitting stages.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT OF NATIONAL INTEREST (PNI)

The Lagoa Salgada project has been granted the PNI (Project of National Interest) status by the Portuguese governmental entities. This status carries certain benefits and recognition for the project, which this chapter aims to elaborate on.

24.1.1 WHAT IS A PROJECT OF NATIONAL INTEREST (PNI) AND WHAT DOES THIS STATUS ENTAIL

The recognition of an investment project with Project of National Interest (PNI) status aims to ensure that the Permanent Commission for Investor Support closely monitors all licensing, authorisations or approvals of the competence of the central and local administration that are necessary for realising the project, allowing any administrative roadblocks to be overcome in order to ensure a rapid response. (Agência para a Modernização Administrativa, 2023)

24.1.1.1 CRITERIA AND OBLIGATIONS FOR THE GRANTING OF PNI STATUS

Recognition with the PNI status is possible for investment projects that cumulatively meet the following eligibility requirements:

- Represent an overall investment of EUR 25 million or more.
- Create several direct jobs equal to or greater than 50.
- Have proven economic viability.
- Are susceptible to adequate environmental and territorial sustainability.
- Are presented by promoters of recognised reputation and credibility.
- Have a positive impact in at least three of the following areas:

  a) Installation of a productive base, with strong national incorporation, creating gross added value.

  b) Production of innovative tradable goods and services that give them a competitive advantage in the global market.

  c) Introduction of innovative technological processes or development of these processes in collaboration with recognised entities of the scientific and technological system.

  d) Insertion in the smart specialisation strategy of the region and or contribution to the dynamization of territories with low economic density.

e) External economic balance, in particular in the increase in exports or in the reduction of imports.

f) Energy efficiency or favouring renewable energy sources.

g) Knock-on effects on upstream or downstream activities, particularly in small and medium-sized enterprises.

Investment projects with an overall value of less than €25 million and/or which create less than 50 direct jobs may also be recognized as PNI projects, provided that they meet two of the following requirements:

- Internal Research and Development (R&D) activity amounting to at least 10% of the company's turnover.
- Strong component of applied innovation, translated into a significant part of its activity anchored in a patent developed by the company.
- Manifest environmental interest.
- Strong export vocation, translated by a minimum of 50% of its turnover directed to the international market.
- Relevant production of tradable goods and services. (Agência para a Modernização Administrativa, 2023).

24.1.1.2 MAIN BENEFITS OF BEING GRANTED PNI STATUS

The recognition as a PNI implies a priority treatment, in terms of licensing procedures, with any entities, bodies or services of the Public Administration. Although the recognition as a PNI does not constitute any rights, as any and all applicable legal and regulatory rules remain valid, it will allow the application of the special regime enshrined in Decree-Law no. 154/2013, of 5 November. (Agência para a Modernização Administrativa, 2023)

24.1.1.3 PROCESS MANAGER

A project classified as a PNI is assigned a Process Manager, who is responsible for closely monitoring the development of the process, relating directly to the promoter within the scope and for the purpose of all legal and regulatory procedures that provide for the issuance of opinions, authorisations, decisions or licensing.

In particular, the Process Manager is responsible for ensuring compliance with the schedule and making the necessary efforts to clarify and agree on positions with a view to the implementation of the investment project, namely through the promotion of meetings with the entities of the Public Administration and with the promoter, as well as with the respective articulation with the local administration.

Among the entities that constitute the Permanent Committee for Investor Support, (CPAI), AICEP, IAPMEI or Turismo de Portugal may be designated as Process Manager, depending on the nature of the project in question and the attributions established in the respective organic diplomas regarding the monitoring of investment projects. (Agência para a Modernização Administrativa, 2023)

24.1.1.4 EXPIRATION

The recognition of an investment project as a PNI expires in the following situations:

- Whenever the schedule is not complied with due to a cause attributable to the promoter
- If, 90 days after the communication of the granting of the PNI status, the promoter does not initiate the subsequent procedure provided for in the project schedule.
- Whenever any legal or regulatory provision is violated by the promoter.
- The follow-up procedure by CPAI ends with the start of the project execution. (Agência para a Modernização Administrativa, 2023).

24.1.2 APPLICATION PROCESS

The application process can be summarised into the following three major steps:

- Gathering of project information and filing the application form issued by AICEP for the candidacy: This first step includes the gathering of technical information developed during the basic design stage of the project, as well as an economic model projection of the future operation to substantiate the economic viability of the project and the development of charters to characterise the project's implementation area, among other information.
- Submitting the application form and responding to any additional requests from CPAI: At this stage, all the information is submitted to *ePortugal's* portal, and the project promoter will await any further contacts or requests that may be made by CPAI whilst analysing the candidacy.
- Approval of the PNI status for the project: CPAI meets on a regular basis to approve the PNI status. Once it is approved, the project promoter receives a communication from CPAI with the considerations of each entity that comprises it.
- Participating in initial meetings with CPAI to approve the Project Timeline proposal.

24.1.3 SCHEDULE/TIMELINE OF THE PROJECT

After being granted the PNI status, AICEP must develop a project timeline, in close cooperation with the promoter, that details all the licensing procedures to be performed,

their interdependencies and the start and finish date of each licensing. This timeline will be approved and then followed by each involved party.

24.1.4 How obtaining PNI status benefits / will benefit the Lagoa Salgada Mine Project

PNI projects, although governed by the applicable legal and regulatory rules due to their nature, benefit from a special regime of the administrative procedure that translates into:

- Simultaneous processing of administrative procedures falling within the competence of the central government
- Reduction and simultaneous course of endo-procedural deadlines
- A single public consultation period for the purposes of the various administrative procedures
- Simplification of procedures for territorial management tools relevant to the project
- Positive tacit opinions and tacit acceptance within the scope of the various applicable procedures
- Simplification of procedures relating to the necessary urban operations. (Agência para a Modernização Administrativa, 2023)

On the other hand, certain general legal regimes are also adapted, always with the purpose of reducing the deadlines or simplifying the procedure, as is the case of the Environmental Impact Assessment, whose decision period is reduced to 80 days. (Agência para a Modernização Administrativa, 2023)

24.1.5 When was the status granted and what activities have elapsed since then

The PNI statute was granted to the Lagoa Salgada Mine Project on the 29th of April, 2022.

Since then, the Promoter and Quadrante, as its Project Manager and Technical Consultant respectively, have participated in several meetings with AICEP and CPAI. These meetings were aimed at tailoring the best path for the project's licensing process, not only to reduce the time elapsed in this process as much as possible, but also to establish open paths of communication between stakeholders for licensing entities to be better informed about the project. In these meetings, the project promoter, and Quadrante showed their availability to exchange all necessary information in due time with all relevant stakeholders, so that they are able to contribute to timelines, and thus extract the most advantage from the inputs of the licensing entities.

24.2 TRANSPORTS AND LOGISTICS

The Transports and Logistics study was carried out by QUADRANTE. The information utilised in this chapter is derived from the *QDE_TRANS _T2022-0588-01-LOG-00-RT_00-01MD* report, which can be accessed in the supporting documentation.

24.2.1 CONCENTRATE LOGISTICS

This section addresses the various development options regarding transport of Lagoa Salgada Mine's concentrates.

The objective of the study is to evaluate modal and shipping port alternatives for the output of the Lagoa Salgada Mine.

The two ports under study are Setúbal and Sines, and the modal alternatives are as follows:

- Exclusively railway, with the construction of a branch line to the processing plant.
- Exclusively road, with a connection to the road network through the EM543.
- Road rail, with a road connection to the future logistics park of Grândola.

24.2.1.1 ASSUMPTIONS

The assumptions made in this study were as follows:

RELATED TO THE MINE

- The planned start date of operation is in the fourth quarter of 2025.
- Mine production: 110,000 tons per year of ore concentrate, with a life span of 16 years.
- The product is a powder with a moisture content of 7% and particle size distribution of d80=12 microns and d100=200 microns. The density is 2.2 for copper concentrate, 3.1 for lead concentrate and 2.1 for zinc concentrate.
- The percentage composition of the products to be transported is as follows:
    - Copper ore concentrate – 9%.
    - Zinc ore concentrate – 54%.
    - Lead ore concentrate – 11%.
    - Copper and lead ore concentrate – 26%.

- The ore concentrates will be ready for export, with no treatment required during transport from the processing plant to the port.

RELATED TO LAND TRANSPORT

- The copper and zinc concentrates will be separately transported in bulk (representing a total of 69,300 t/year).
- The lead and copper/lead concentrates will always be transported in containers to the final destination with an interior lining to avoid any health or environmental risks and to prevent these from coming into contact with water, which could lead to a dangerous increase in temperature.

RELATED TO STORAGE

- Covered storage with capacity for 1.5 or 2 shipments (10,000 t to 14,000 t).
- To estimate costs, it is assumed that there are no additional costs associated with storage at the port due to delays in shipment.

RELATED TO MARITIME TRANSPORT

- The tonnage per shipment for copper concentrate will be between 5,000 t to 7,000 t and for zinc concentrate, 5,000 t.
- The destination of the copper and zinc concentrates will be Northern Europe.
- The destination of the lead and copper/lead concentrates will be the Middle/Far East, taking into account the difficulty of shipping this type of contaminant material to Europe and following the current practice of Somincor.

24.2.2 Analysis Of Shipping Ports

24.2.2.1 Sines Port

INTRODUCTION

The Port of Sines, located south of the city of Sines (Figure 24.1), is a deep-water port, is the national leader in the amount of goods moved and has unique natural conditions on the Portuguese coast to accommodate all types of ships. Equipped with modern specialised terminals, it can handle different types of goods, is open to the sea and has excellent maritime accessibility without constraints.

Figure 24.1 – Location of the Port of Sines

The Port of Sines is the main port on the Iberian-Atlantic coast, and its geophysical characteristics have contributed to its consolidation as a national strategic asset, being, on the one hand, Portugal's main energy supply gate (oil, derivatives, and natural gas) and, on the other hand, positioning itself as an important general/containerised cargo port with high growth potential to become an Iberian, European and global reference.

Owing to its recent construction (1978), the Sines Port has a well-structured layout, free from urban pressures, ensuring long-term expansion capacity. It also has suitable land accessibility for current traffic and a plan for road and railway evolution that will allow the future growth projections of both the port and its area of influence to be met.

The Sines Port and its Back Logistic and Industrial Zone, covering more than 2,000 ha, is already an international logistics platform capable of accommodating the major players in the maritime port, industrial and logistics sectors.

24.2.2.2 SETÚBAL PORT

INTRODUCTION

Located between the city of Setúbal and the Sado Estuary Nature Reserve, the Port of Setúbal is one of the most competitive ports on the Atlantic Coast of Europe, with a privileged location 45 km from Lisbon and a 24-hour open bar, integrating one of the most important industrial and logistics zones in the country (Figure 24.2).

Figure 24.2 – Location of the Port of Setúbal

The strategic location of the Port of Setúbal, combined with excellent port conditions, positions it naturally on the route of Atlantic and Mediterranean cruises.

It is a national leader in the ro-ro cargo segment, which represents significant revenues for both vehicle exports and imports, responding to the automotive sector's needs. Setúbal has been asserting itself with significant growth the containerised cargo segment thanks to the increase in regular lines that call at the port.

Regarding solid bulk, there are five terminals with different characteristics, types of use and equipment (Figure 24.3).

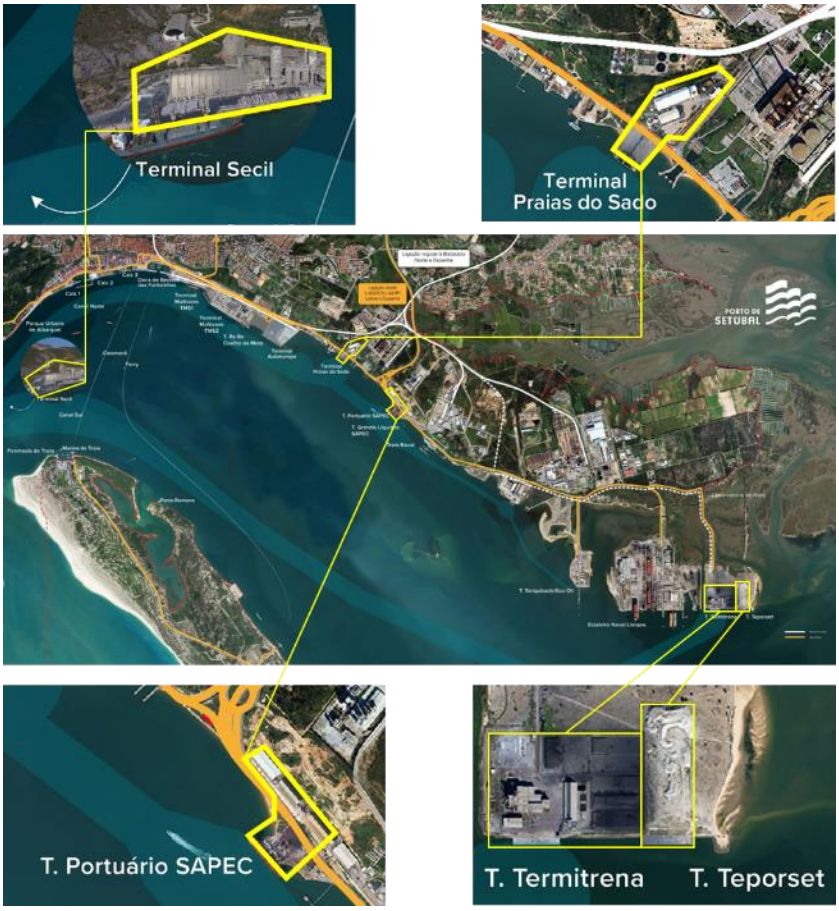


Figure 24.3 – Port of Setúbal | Solid Bulk Terminals

24.2.3 ANALYSIS OF LOGISTICS SOLUTIONS

The following logistic solutions between mine and port were considered:

- Exclusive railway transportation.
- Exclusive road transportation.
- Multimodal transportation, with mining road transport to the future Logistics Park of Grândola and from there to the port.

24.2.3.1 TRANSPORT EXCLUSIVELY BY RAIL

CONNECTION THROUGH THE PORT OF SINES

Railway connections will be provided by the Southern Line between Grândola-Norte station and Ermidas do Sado station, and by the Sines Line between Ermidas do Sado and the Sines port station, on a total railway route of about 85 km (Figure 24.4).

Figure 24.4 – Railway itinerary to the port of Sines

The stretch of the Southern Line between Grândola-N and Ermidas do Sado and the Sines Line are single-track lines equipped with catenary and electronic signalling, with operational control centralised at the Setúbal Operational Command Center (OCC).

In terms of traffic, an average of 11 freight trains/day/direction circulate on the Sines Line, with the following origins/destinations (Table 24.1):

Table 24.1 –Rail Freight Traffic Demand | Linha de Sines

Freight Traffic on the Linha de Sines		
Destination	Trains/day/direction	Goods
Port of Setúbal – SPC	1	Containers
Entroncamento – TVT	2	Containers
Bobadela Terminal	2	Containers
Port of Leixões	3	Containers
Valongo Terminal – SPC	1	Containers
Mérida/Sevilha	1	Containers
Terminal of Loulé	1	JetFuel

Freight Traffic on the Linha de Sines	
TOTAL	11

On the Southern Line (Linha do Sul) between Grândola-Norte and Ermidas do Sado, an average of 10 freight trains/day/direction and 5 passenger trains/day/direction operate, providing interurban connections (*Alfa Pendular* and *Intercidades* services) from Lisbon and Porto to Faro (Table 24.2).

Table 24.2 – Rail Traffic Demand | Linha de Sines e do Sul

Rail Traffic on Linha de Sines e Linha do Sul			
Lines	Trains/day/direction		
	Passengers	Freight	Total
Linha de Sines	0	11	11
Linha do Sul (Água de Moura – Ermidas do Sado)	5	10	15

On this itinerary, there is a characteristic ramp on the Sines Line in the S. Bartolomeu da Serra area with an incline of 19/21% (uphill/downhill), which limits the maximum towed load. For the 4700 series locomotives, which are usually used by Medway in traffic to the port of Sines, this value corresponds to 1040 tonnes.

CONNECTION THROUGH THE PORT OF SETÚBAL

Rail connections will be ensured by the Southern Line between Grândola-Norte station and Praias do Sado station, in a total railway itinerary of about 62 km (Figure 24.5).

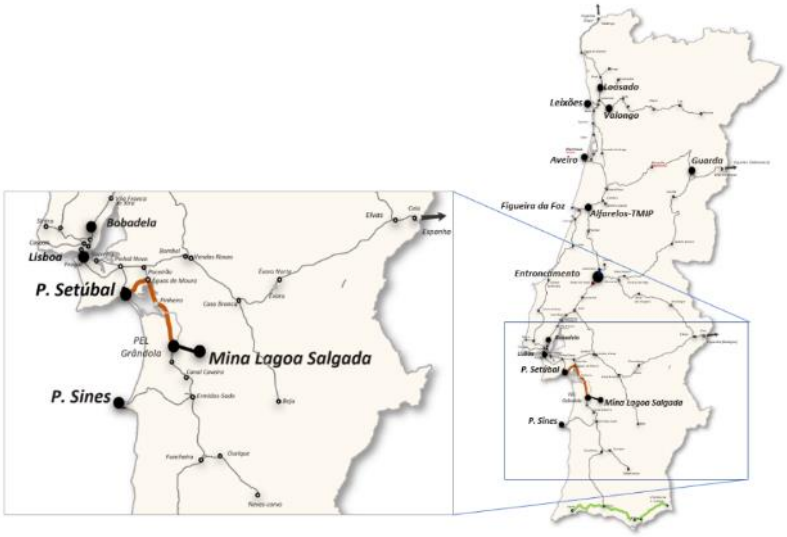


Figure 24.5 – Railway itinerary to the Port of Setúbal

The section of the Southern Line between Grândola-Norte station and Praias do Sado / Port of Setúbal is a single-track equipped with catenary and provided with electronic signalling with its operational command centralised at the Setúbal Operational Command Center (OCC).

In terms of traffic, the itinerary to the Port of Setúbal has two distinct sections, as shown in Table 24.3.

Table 24.3 – Rail Traffic Demand | Linha do Sul

Rail Traffic on Linha do Sul			
Lines	Trains/day/direction		
	Passengers	Freight	Total
Linhas do Sul (Grândola N – Águas de Moura)	5	10	15
Linha do Sul (Água de Moura – Praias do Sado)	33	11	44

The Grândola-Norte - Águas de Moura section, is used by freight trains coming from the port of Sines and heading North, and passenger trains from Alfa Pendular and Intercidades services connecting Lisbon and Porto to Faro.

In the stretch between Águas de Moura and Praias do Sado / Porto de Setúbal all freight trains originating from / heading for the terminals of Praias do Sado / Porto de Setúbal circulate, along with passenger trains from the suburban service Barreiro – Praias do Sado-A, as shown in Table 24.4.

Table 24.4 – Rail Traffic Demand | Port of Setúbal

Freight Traffic on the Port of Setúbal		
Destination	Trains/day/direction	Goods
SPC Terminal	1	Containers
Somincor Terminal	3	Ore in Containers
Navigator Terminal	2	Containers
AutoEuropa Terminal	3	Cars
Sadport Terminal	2	Steel
TOTAL	11	

In this itinerary, the characteristic slopes are around 13%, which correspond to a maximum load towed for the locomotive 4700 of about 1200 tonnes.

The Praias do Sado station provides access to the Somincor/Almina Terminal or the Sapec Terminal for unloading the ore concentrate. Both have access to a maritime dock where the ore concentrate can be shipped.

RAILWAY FUTURE PERSPECTIVES

IP has an ambitious investment plan in development, "Plano Ferrovia 2020", and a National Investment Program until 2030, "PNI 2030", which aims to modernise railway infrastructure by providing conditions to make rail freight transportation more competitive than road transport. These interventions will reduce the length of routes and significantly increase transport capacity, especially on the main international axes, provide access to Setúbal, Sines, Leixões, and Lisbon ports and include the construction of approximately 40 stations with lengths for crossing/train storage of 750m.

There are still investments planned that will allow for, improvements in the homogenisation of operating conditions across the entire national railway network, especially with regard to electrification, the installation of electronic signalling systems with centralised remote control, GSM-R telecommunications systems, and the safety of operations, with the suppression and automation of level crossings.

24.2.3.2 TRANSPORT EXCLUSIVELY BY ROAD

LOCAL ACCESS – MINE TO INTERSECTION BETWEEN IC1 AND IC33

The road connection to the Lagoa Salgada Mine processing plant will be made by constructing direct access to municipal road EM543, which will connect to the José Amaro Circular Road near the Light Industry Zone (ZIL) of Grândola (Figure 24.6).

As can be seen in Figure 24.6, the connection to the IC1 can be made through two routes:

- From the north, through the José Amaro Circular Road (continuation of EM543), following Camilo Castelo Branco Street to the IC1 intersection. It should be noted that the reverse route (Port > Mine) must be made by the southern alternative as the above-mentioned intersection does not allow left turns.

- From the south, through the José Amaro Circular Road, with a direct connection to the IC1 through a grade-separated junction with a more suitable geometry for dealing with the heavy vehicles coming from the mine.

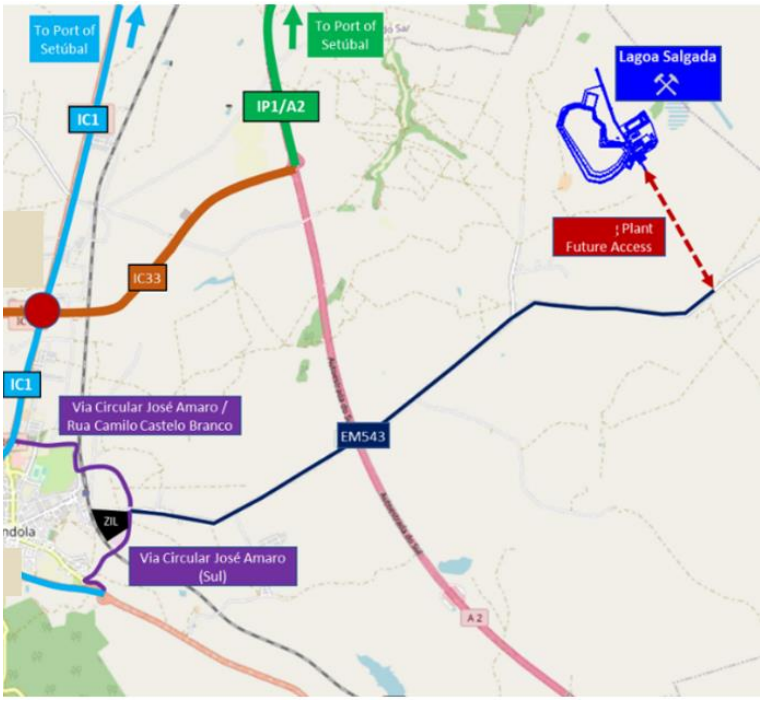


Figure 24.6 – Identification of Local Mine Access Roads

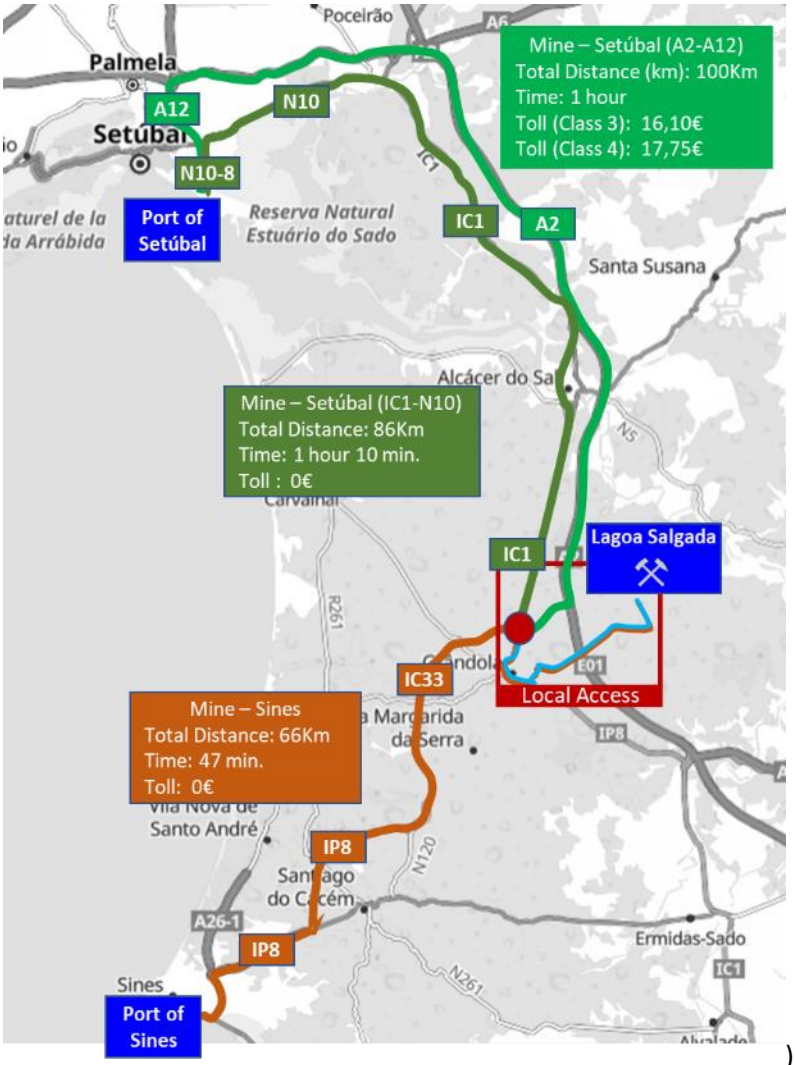
In the end, from the Lagoa Salgada Mine to the intersection of IC1 with IC33, the routes to both ports (Sines and Setúbal) are coincident, with a partial distance of about 14.2 km to the north and 17.7 km to the south.

For the total road distance from mine to ports, the south option was considered due to the more suitable geometric characteristics for heavy vehicle circulation.

From the referred intersection to their respective ports, vehicles will take different routes that will be analysed in the following subchapters.

LONG-DISTANCE ITINERARY TO THE PORTS

The route from the intersection of IC33 and IC1 to the port will be as follows (Figure 24.7).


Poceirão
Palmela
A12
N10
Setúbal
N10-8
Port of Setúbal
Reserva Natural Estuário do Sado
IC1
A2
Santa Susana
Alcácer do Sal
Mine – Setúbal (A2-A12)
Total Distance (km): 100Km
Time: 1 hour
Toll (Class 3): 16,10€
Toll (Class 4): 17,75€
Mine – Setúbal (IC1-N10)
Total Distance: 86Km
Time: 1 hour 10 min.
Toll : 0€
Lagoa Salgada
IC1
Local Access
IC33
Mine – Sines
Total Distance: 66Km
Time: 47 min.
Toll: 0€
IP8
Santo André
IP8
A26-1
IP8
Ermidas-Sado
Sines
Port of Sines
N261
IC1
):

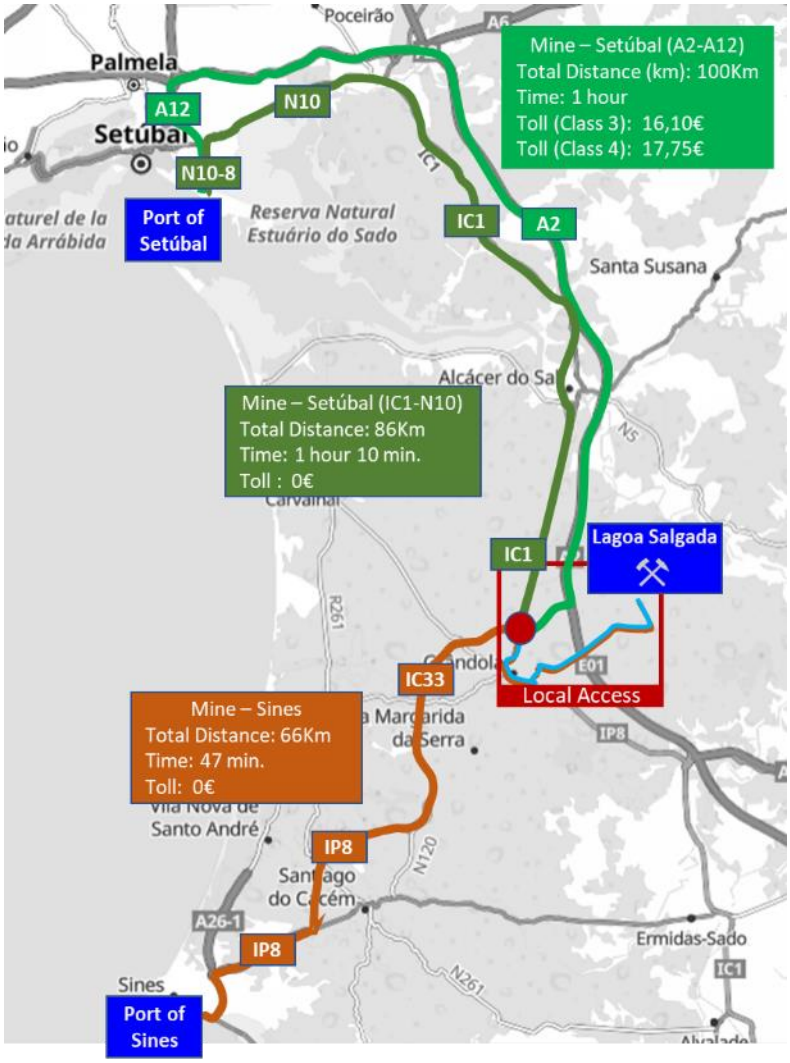


Figure 24.7 – Long Distance Itineraries

24.2.3.3 ROAD-RAIL TRANSPORT

This scenario considers a multimodal itinerary for transporting the mine's concentrate to the port of shipment.

The itinerary will consist of two sections (Figure 24.8):

- Section 1: Lagoa Salgada Mine – Grândola Business and Logistics Park (PEL) will be carried out by road transport – 24 km
- Section 2: between the Grândola Business and Logistics Park (PEL) and the port of Sines (85 km) or Setúbal (61 km), will be carried out by rail transport.

The transfer of concentrate from road transport to rail transport will take place at the Grândola Business and Logistics Park (PEL).

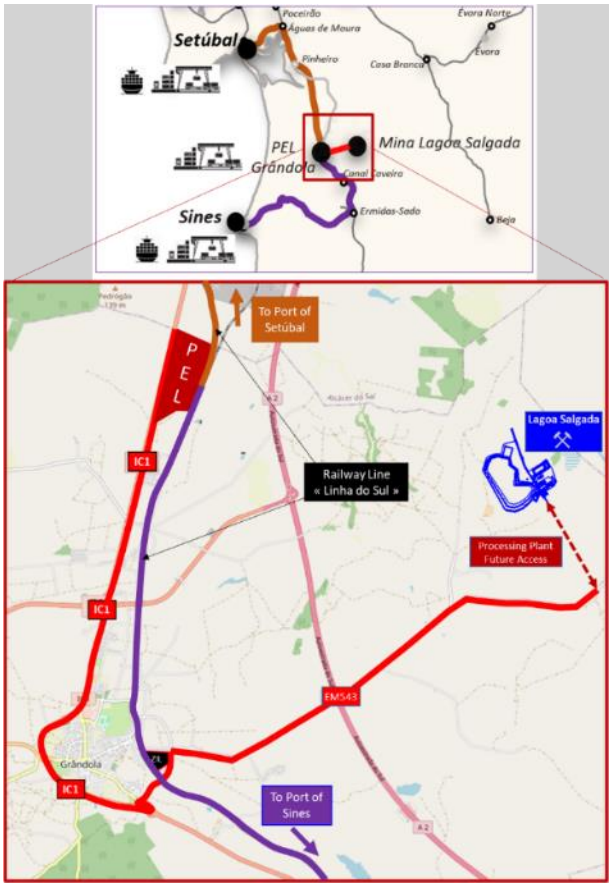


Figure 24.8 – Multimodal Route Scheme

24.2.4 COST ANALYSIS OF LOGISTIC ALTERNATIVES

24.2.4.1 INTRODUCTION

The comparative analysis of costs for the different alternatives will be done separately for copper and zinc ore concentrates that will be transported in bulk (although "open top" containers will be used for transportation between the mine and the port), and for lead and copper/lead concentrates that will be transported in containers with interior lining ("liner") to the destination.

These values were calculated based on information obtained in meetings and the best commercial proposals from some of the main stakeholders in the logistics and port market:

- APSS – Port Administration of Setúbal and Sesimbra ports.

- Sapec - SPC – Logistic Operator and Solid Bulk Dock Concessionaire at the port of Setúbal.
- Somincor Setúbal – Concessionaire of the Neves Corvo Mine and Solid Bulk Dock Concessionaire at the port of Setúbal.
- PortSines – Concessionaire of the multipurpose terminal at the port of Sines.
- PSA - Port of Singapore Administration – Concessionaire of Terminal XXI at the port of Sines.
- Medway – Railway-based logistic operator in Portugal, Spain, Italy, Belgium, the Netherlands and Germany.
- IP – Infrastructures of Portugal, manager of the national railway and road networks.

24.2.4.2 TRANSPORT OF COPPER AND ZINC ORE CONCENTRATE

SUMMARY OF THE COMPARATIVE ANALYSIS

The synthesis of the cost estimates for transporting copper and zinc ore concentrate from the mine to the shipping port for each mode of transport and each port is presented in Table 24.5.

Table 24.5 – Summary of Bulk Transport Costs

Transport Costs [€/net tonne]	To/From	
Stages	Port of Sines	Port of Setúbal
Exclusive by Road	20.24	21.81
Exclusive by Rail	39.40	41.71
Road-Rail Transport	40.48	42.85

Analysing the values, the road transport solution has the lowest cost, as has the Sines port alternative when compared with the Setúbal port.

The railway solution has higher costs than the road solution but lower costs than the road-rail solution. Despite the initial investment needed to build a new railway line between the mine and the Southern Line of the national railway network, estimated at €20M, this cost is less penalising than the transfer costs at the Grândola PEL, which is necessary in the road-rail solution.

The road-rail and railway solutions are both penalised in relation to the road solution at the terminals of both ports. In the Setúbal port (Somincor terminal), an additional investment of €8M is needed for the unloading machine, and in the Sines port (PortSines terminal), there is an additional cost of a "last mile" road due to the railway line being about 300 m away from the storage area.

The road-rail solution has the highest costs because, in addition to the railway surcharges at the port terminals, it entails a significant additional cost for transferring the ore concentrate at the Grândola PEL.

24.2.4.3 TRANSPORT OF PLUMB AND PLUMB/COPPER ORE CONCENTRATE

The synthesis of the cost estimates for transporting lead and copper-lead concentrates in a lined container, from mine to the Sines port (Terminal XXI), for each mode of transport is presented in Table 24.6.

Table 24.6 – Summary of Container Transport Costs

Transport Costs [€/20' Container]	To/From
	Port of Sines
Exclusive by Road	654.29
Exclusive by Rail	1079.63
Road-Rail Transport	1107.37

These values were calculated based on information obtained in meetings and the best commercial proposals from some of the main stakeholders in the logistics and port market:

- Somincor Setúbal – Concessionaire of the Neves Corvo mine and solid bulk port terminal in the port of Setúbal.
- PSA - Port of Singapore Authority – Concessionaire of Terminal XXI in the port of Sines.
- Medway – Rail-based logistics operator, in Portugal, Spain, Italy, Belgium, the Netherlands and Germany.
- IP – Infrastructures of Portugal, manager of the national railway and road networks.

The road transport solution has the lowest costs and the road-rail solution has the highest costs.

About 37% of the ore concentrate extracted from the mine, approximately 40700 tonnes of lead concentrate (11%) and copper-lead concentrate (26%), will be transported in lined containers for safety reasons to prevent any risk to health or the environment, and to avoid contact with water, which could lead to a dangerous increase in its temperature.

The destination for the lead and copper-lead concentrates will be the Middle/Far East, considering the difficulty of disposing of this type of contaminating material in Europe and following what Somincor currently practices.

Therefore, it was considered that the transport of this ore concentrate would be done through the Terminal XXI of the Sines port, where major shipping companies such as MSC, CGM-CMA, MAERSK call, and their shipping routes provide those destinations.

The costs of transporting the concentrate containers were obtained by considering the road or rail freight costs between the mine and each port presented in Medway's commercial proposal.

The container transport costs for lead concentrates were obtained by considering the road or railway freight costs between the mine and each port, as presented in the commercial proposal from Medway.

The weighing, storage, and shipment costs for each terminal, provided by PSA, Terminal XXI concessionaire were added to these costs.

Given the impossibility of obtaining commercial information from Qantara Capital, promoter of the Grândola PEL, the costs were estimated based on the handling costs presented in the PSA-Terminal XXI commercial proposal. These costs include storage for the first three days and fees to PEL.

It should be noted that these values may be overestimated, as we considered that PEL, not being specialised in mineral traffic, might penalise it. However, we believe that in this case, where the ore concentrate is transported in containers, it will be possible to obtain a more favourable commercial proposal that might close the gap between both options, or even invert it. In short, we believe that the transhipment cost at PEL may be equivalent to or lower than the cost of building the railway line to the mine.

24.2.5 Conclusion and Definition of Logistics Strategies

The present section consisted of collecting information from key stakeholders in the transportation and logistics market, as well as from the manager of the national railway and road networks through various meetings and commercial proposals.

Based on the study of various logistical solutions, objective analyses were carried out for each of the transport modes and shipping port options, indicating their respective advantages, disadvantages, opportunities and threats.

It was found that the road mode is clearly the one that presents a lower cost, with values of €20.24/tonne and €21.81/tonne for the ports of Sines and Setúbal, respectively. The railway and rail-road costs are comparable to other but with values about twice those of the road mode. In each of the cases, the main reasons for the higher cost are as follows:

- Rail mode – requires significant investments in the access line to the mine processing plant as in the port.
- Road-rail mode – costs associated with the transfer of ore concentrate and storage at the Grândola Business and Logistics Platform (PEL) as in the port.

Therefore, should the decision be solely based on price, the recommended logistics solution to implement would be road-based with shipping via the Sines port, as it has lower operational costs, greater flexibility and faster implementation.

However, despite the road solution being the most attractive from a financial point of view, it should be noted that, considering the environmental and reputational costs involved, the possibility of integrating the railway mode into the logistics solution should be considered. It is suggested that the possibility of using financing funds to build the railway line between the Lagoa Salgada mine and the Southern Line of the national railway network should be further explored in order to have a competitive alternative logistics solution to the road-based one.

It is also recommended, for the same reasons, that the commercial relationship with the Grândola Business and Logistics Platform's promoters be deepened, as there might be conditions that ensure a road-rail solution as a fallback or complement to the purely road-based solution.

Regarding the lead and copper/lead ore concentrate, the information provided by Somincor in accordance with its current practice is that its most likely destination will be the Middle/Far East, considering the difficulty of selling this type of contaminating material to Europe.

Therefore, given that the Sines port is a transhipment port, it is recommended that the containers with this type of concentrate be shipped through the XXI Terminal of the Sines port, where the major shipping lines such as MSC, CGM-CMA, and MAERSK ensure those destinations through their Short Sea Shipping lines, benefiting from economies of scale and more advantageous commercial conditions.

Even though some destinations may be in northern Europe and given that the port of Sines is a transhipment port, it is recommended that the containers with this type of concentrate be shipped through the XXI Terminal of the Sines port, where the major shipping lines such as MSC, CGM-CMA, and MAERSK provide both Far East destinations via long-distance routes and closer European destinations through their short sea shipping lines. This way, the company can benefit from economies of scale and more advantageous commercial conditions.

24.3 RISK MANAGEMENT

24.3.1 INTRODUCTION

Risk management is a critical aspect of project success, encompassing technical, commercial, and corporate risks. Ascendant Resources recognises the importance of effective risk management for the Lagoa Salgada Project, considering it an industry best practice and a key evaluation criterion for independent reviewers.

Establishing a strong foundation for risk management requires setting the context within which the project operates. This includes three key components: the external context, the internal context and the risk management context. These components provide the necessary structure and alignment with project objectives. Establishing the context involves defining the purpose of the risk review, determining scope and objectives, examining the external environment, identifying stakeholders and objectives, specifying criteria for analysis and defining key elements.

Continuously identifying and assessing risks is integral to the risk management process of the Lagoa Salgada Project. This proactive approach occurs throughout all project phases, aiming to anticipate potential events, changes, and circumstances. By considering factors such as timing, location and impact, the project team can effectively manage uncertainties and optimise opportunities while minimizing threats. Risk identification is an ongoing and dynamic process, enabling the team to adapt strategies as needed.

For the Lagoa Salgada Project, the risk management approach begins with an initial assessment as part of the project's involvement. The project team follows a systematic and methodical process to identify, assess, score and prioritise risks. By proactively identifying potential risks from the project's inception, effective mitigation strategies can be implemented to address them.

By emphasizing the establishment of context and continuous risk identification, the Lagoa Salgada Project ensures a comprehensive and proactive approach to risk management. This approach enhances the project's ability to navigate uncertainties and achieve desired outcomes while minimising potential disruptions.

24.3.2 RISK ASSESSMENT

Risk assessment is a vital component of the Lagoa Salgada Project and involves the categorisation of risks based on their potential impact. In addition to evaluating risk levels and their probabilities, it is essential to establish the context within which the assessment takes place. This context provides the structure for risk management and ensures its effectiveness. The context establishment process involves identifying the purpose of the risk review, defining the scope and objectives, examining the external environment, identifying stakeholders and objectives, specifying criteria for analysis and defining key elements.

Risk assessment for the Lagoa Salgada Project involves categorizing risks based on their potential impact. The following risk levels have been defined:

- Low-Level Risk: Risks in this category are expected to have a relatively insignificant impact on the project's deposit characteristics and overall economics. These risks can be effectively managed through standard management processes, requiring minor cost adjustments or schedule allowances.
- Moderate-Level Risk: Risks in this are considered average or typical for projects of a similar nature. By employing proper planning and adhering to technical best practices, these risks can be minimized to ensure manageable impacts on the deposit and project economics.
- High-Level Risk: Risks in this category are characterized by their uncontrollable, unpredictable, or atypical nature for projects of this type. Despite implementing good technical practices and quality planning, successful mitigation is not guaranteed. These risks can significantly impact the project's economics, resulting in schedule disruptions, substantial cost increases and compromised physical performance. Additionally, risks related to environmental and social noncompliance, in accordance with relevant standards and principles, may also be categorized as high-level risks.

Alongside evaluating risk levels, due consideration has been given to the likelihood of each risk materializing during the project. To this end, the following probabilities have been established:

- Low Probability: Risks assigned a low probability are unlikely to occur during the project.
- Moderate Probability: Risks with a moderate probability have an increased likelihood of occurrence during the project.
- High Probability: Risks with a high probability are likely to occur at some point during the project's timeline.

Table 24.7 – Risk Management: Severity Factors

Severity Factor	Consequences				
	Health & Safety	**Environment**	**Social & Community**	**Government, Media & Legal**	**Financials**
High	Fatality. Severe irreversible disability.	Major impact, long term effects to the environment in the project surroundings. Significant remediation costs.	Extensive social impacts involving very serious accusations. Irreparable damage to the community.	Prosecution and fines exposed to international media coverage. Difficulties renewing or getting new licenses or permits to operate.	$25M to $100M

Severity Factor	Consequences				
	Health & Safety	**Environment**	**Social & Community**	**Government, Media & Legal**	**Financials**
Moderate	Lost time. Injury. Moderate irreversible disability.	Moderate, short-term effects to the environment in the project surroundings.	Continuous social and local community issues impacting normal project activities.	Ongoing criticism by local and national media coverage. Serious breach of applicable regulations resulting in active investigations and continuous external audits.	$500k to $5M
Low	Medical Treatment. Restricted Work. First Aid.	Minor effects on the environment within project area.	Minor social impacts.	Minor, adverse local media complaints.	$50k to $500k

By categorising risks and evaluating their probability, the Lagoa Salgada Project can effectively analyse and manage potential risks. This approach allows appropriate mitigation strategies to be implemented, ensuring the successful execution of the project while addressing its unique challenges and requirements.

24.3.3 PROJECT RISK EVALUATION

Alongside evaluating risk levels, due consideration has been given to the likelihood of each risk materialising during the project. To this end, the following probabilities have been established:

- Low Probability: Risks assigned a low probability are unlikely to occur during the project.
- Moderate Probability: Risks with a moderate probability have an increased likelihood of occurrence during the project.
- High Probability: Risks with a high probability are likely to occur at some point during the project's timeline.

Table 24.8 presents a comprehensive overview of the risks identified in connection with the Project. It provides a detailed breakdown of each risk, including the project component, risk description, severity factor, risk probability and risk mitigation. It serves as a valuable reference tool, offering a concise summary of the identified risks, their classifications and the corresponding strategies and actions to effectively manage and mitigate these risks.

The evaluation does not include the presentation of risks associated with environmental impact, as they are discussed in Chapter 20 – Environmental Studies, Permitting, and Social

or Community Impact. Similarly, health and safety risks are addressed in a separate document, namely the support information document T2021-620-01-PSS-00-LagoaSalgada (Health and Safety Plan).

In this evaluation, the risks related to environmental impact are not presented once they are identified in the respective chapter (Chapter 20 – Environmental Studies, Permitting and Social or Community Impact). The same applies to health and safety risks, which are presented in a separate document, namely document T2021-620-01-PSS -00-LagoaSalgada (Health and Safety Plan).

Table 24.8 – Project Risk Evaluation - Resume

Nr	Project Component	Risk Description	Severity Factor	Risk Probability	Risk Mitigation
01	Construction & Operations	Inadequate water supply for construction activities or operations.	High	Moderate	Ensure accurate calculation of construction water needs and assess the availability of sufficient water resources in terms of quality and volume, while exploring alternative solutions. Currently it is not foreseen any CAPEX to construct an external water supply.
02	Construction & Operations	Lack of qualified manpower for construction and execution.	High	Moderate	Ensure thorough definition of requirements and early implementation, while providing training for professionals in the different roles.
03	Geotechnical & Operations	Failure to implement a waterproof curtain in the Boxcut could have a significant impact on operations due to the heightened productivity of the superficial aquifer	High	Moderate	In the next phase of the project, a trade-off study must be conducted to determine whether to implement the curtain or establish an appropriate pumping system.
04	Cost Estimation	Underestimation of capital costs.	High	Low	Continue to refine cost estimates and identify key vendors / line up additional sources of finance in advance as a contingency plan.
05	Cost Estimation	Underestimation of dewatering needs	Moderate	Moderate	In the detailed CAPEX, one main pump (GEHO) was included. However, in the subsequent project phase, it is imperative to conduct a trade-off study to assess the risks associated with relying solely on a single pumping station without a spare pump.
06	Cost Estimation	Delays in equipment manufacturing and delivery.	Moderate	Low	Identify critical equipment and materials, both domestic and imported, and place orders promptly. Ensure effective supplier qualification and include contract deadlines.
07	Cost Estimation	Underestimation of operating costs.	High	Low	Conduct thorough cost analysis, incorporate contingency provisions and regularly review and update cost projections.
08	Cost Estimation	High fuel prices.	Moderate	Low	Look at the electrification of vehicles, ESS and solar/wind.

Nr	Project Component	Risk Description	Severity Factor	Risk Probability	Risk Mitigation
09	Cost Estimation	High electricity prices.	Moderate	Moderate	Implement PV Plant project.
10	Cost Estimation	High construction cost	Moderate	Moderate	Encourage competitive bidding from multiple contractors or suppliers. Explore alternative construction materials, methods and technologies that could potentially lower costs without compromising the project's quality and objectives.
11	Engineering	Incorrect water balance assessment.	Moderate	Moderate	Employ robust data collection, utilise hydrological modelling, engage expert review, consider site-specific factors, implement continuous monitoring, develop contingency plans and involve stakeholders.
12	Engineering	Inadequate or lack of power supply.	Moderate	Low	Prepare an electrical demand profile for project execution.
13	Engineering	Poor definition of process design criteria.	High	Low	Ensure comprehensive analysis of design requirements, engage expert input for criteria definition, conduct thorough reviews and incorporate feedback loops for continuous improvement.
14	Engineering	Late delivery of engineering studies.	Moderate	Low	Establish clear project timelines and milestones, closely monitor progress throughout the study, allocate adequate resources, maintain open communication channels with the engineering team and implement contingency plans to address potential delays.
15	Engineering	Design flaws in piping system.	Moderate	Low	Ensure expert designers, rigorous reviews, thorough testing and adherence to industry standards.
16	Infrastructure and Facilities	Insufficient local infrastructure to support project execution (e.g. hotels, rental accommodations, restaurants, services).	Moderate	Moderate	Conduct thorough infrastructure assessments, engage with local authorities and communities, explore options for temporary or expanded infrastructure, establish partnerships with local businesses and develop contingency plans to address potential shortfalls.
17	Licensing	Delayed or conditional award of construction permit.	High	Moderate	Continuously analyse licenses and scheduled meetings with technical analysts from the environmental agency. Establish a relationship of trust to enable prompt attention to any pending issues. Maintain ongoing contacts with authorities and PNI project regulators to ensure effective communication and collaboration throughout the project.
18	Mineral Resources	Resource tonnes and grade estimates lower than expected.	Moderate	Low	The core area of the deposit is drilled to the Indicated Mineral Resource category.

Nr	Project Component	Risk Description	Severity Factor	Risk Probability	Risk Mitigation
19	Operations	Ore feed not aligning with process design criteria during project ramp-up and future operation.	Moderate	Moderate	Closely monitor ore characteristics, establish control measures and maintain collaboration between mining and processing teams.
20	Operations	Underperformance of material handling system.	High	Low	Ensure proper system design, conduct thorough testing and commissioning, implement regular maintenance and inspection protocols, provide training for operators and establish performance monitoring mechanisms.
21	Operations	Higher or lower than expected ground water inflows from surface and intermediate aquifers.	Moderate	Moderate	Update the hydrogeological model based on portal grouting program results and prepare an alternative water supply source.
22	Processing	Consistently lower than expected recoveries and grades.	High	Low	Additional testwork to refine conditions.
23	Procurement	Poor performance of contractors or suppliers.	Moderate	Moderate	Implement a thorough pre-qualification and contract award process that considers both performance capability and price, ensuring the selection of qualified suppliers. Additionally, establish a robust procurement organisation that aligns with the project's specific requirements.
24	Procurement	Delays in equipment deliveries on the critical path.	Moderate	Low	Detailed engineering to address this risk.
25	Procurement	Variability in lead times for certain equipment.	Moderate	Low	Prioritise ongoing lead-time awareness for key equipment, including the development of a long-lead register and other procurement initiatives.
26	Land Access	Challenges in obtaining timely land access.	High	Low	Ensure adequate funds are available for low-risk strategy.
27	Stakeholder Engagement	Inadequate stakeholder identification causing continuous issues affecting project development plans.	Moderate	Low	Enhance stakeholder analysis to include newly identified national and international stakeholders.
28	Stakeholder Engagement	Lack of community perception of project benefits and poor work relationship with local authorities.	Moderate	Moderate	Engage a communications company specialised in improving communication within the plant and surrounding communities through effective strategies and practices.

Nr	Project Component	Risk Description	Severity Factor	Risk Probability	Risk Mitigation
29	Stakeholder Engagement	Failure to meet local expectations of job creation and income generation from full-scale implementation.	Moderate	Low	Develop matrix to analyse the major problems that impact the communities, allowing resources to be allocated towards actions that yield maximum impact with minimum effort. Define and hire a communications company to enhance communication within the plant and the surrounding communities.
30	Mining	Actual ground conditions in the mining area differ from those studied.	Moderate	Low	397 core samples were analysed, characterising 8 different domains, in order to determine all possible ground conditions.
31	Mining	Mine design inconsistent with reserve estimates	Moderate	Low	Mitigate inconsistent mine design with reserve estimates through regular updates, detailed planning, proactive monitoring, expert consultation and effective communication.

The risk assessment for the project TSF, conducted by SLR Consulting (*Lagoa Salgada TSF Feasibility Study Report, SLR 2023*) is summarised in the table below:

Table 24.9 – Project Risk Evaluation – TSF (resumed)

(Source: Lagoa Salgada Tailings Storage Facility Feasibility Study Report, SLR 2023)

Nr	Risk Factor	• Impact	Severity Factor	• Risk Probability	• Risk Mitigation
01	Seismic Activity	Earthquakes in the region could cause liquefaction of tailings stored within TSF, leading to potential overtopping failure and increased seepage through cracks in the lining.	High	• Low	Downstream construction ensures stable containment of tailings in case of liquefaction, with water stored in excavated reclaim ponds.
02	Extreme Rainfall Event	Excessive rainfall can result in overtopping, downstream flooding and contamination. May compromise embankment stability and cause erosion around the reclaim pond and downstream face of the TSF.	High	• Low	Reclaim ponds are designed with sufficient freeboard for extreme flood events, supported by a sized spillway. Additional pumping capacity and erosion control measures are in place. Regular updates to water balance and design storm events account for climate change.
03	Spillway failure during design storm event conveyance	Spillway failure during a design storm event may result in erosion and compromise the facility's stability.	High	• Low	Spillway designed by qualified engineer, positioned shallowly to prevent erosion, and minimize tailings release. Regular inspections maintain its condition.

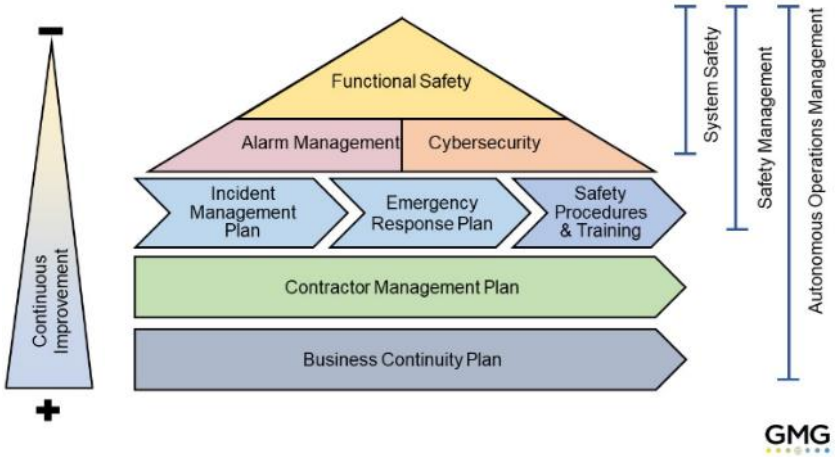
Nr	Risk Factor	• Impact	Severity Factor	• Risk Probability	• Risk Mitigation
04	Elevated Phreatic Surface	Elevated phreatic surface within the facility weakens construction materials and affects stability.	High	• Low	Downstream construction reduces reliance on tailings strength, with stability dependent on the engineered embankment. Phreatic surface in embankment monitored using vibrating wire piezometers.
05	Tailings In-Situ Density	In-situ tailings density below 2 t/m3 reduces TSF tonnage capacity.	Medium	• Low	Annual assessment of in-situ volume and density of tailings deposited to evaluate design assumptions and fast-track storage alternatives if needed.
06	Beach Slopes	Deviations from the design beach slopes of 1V in 50H can reduce TSF capacity.	Medium	• Low	Regular surveys monitor beach slopes, adhering to deposition plan for design alignment.
07	Ground Contamination	Higher-than-anticipated seepage through the liner system can lead to significant contamination of groundwater and nearby rivers.	High	• Low	Permeability testing and quality assurance ensure effective containment, minimising leakage. Water quality tested regularly, with appropriate mitigation measures for contamination.
08	Availability of Construction Materials	Limited availability of construction materials at the required grading or specification.	Medium	• Low	Close monitoring of material volumes prevents shortfalls, maintaining an earthworks balance in the facility design.

24.4 HEALTH AND SAFETY

24.4.1 INTRODUCTION

The health and safety of employees, contractors, and visitors is of paramount importance to Ascendant Resources Inc. The company is committed to maintaining a safe and healthy work environment at its Lagoa Salgada DFS project. Health, safety, and environmental (HSE) performance are managed through robust corporate HSE management systems that align with internationally accepted standards.

At Ascendant Resources, HSE management systems are designed to meet the requirements of recognised international standards, including ISO 45001 for Health and Safety and ISO 14001 for the Environment. These systems form the foundation for managing health, safety, and environmental aspects at the Lagoa Salgada DFS project.



Source: Global Mining Guidelines Group

Figure 24.9 – HSE for Mine Management

24.4.1.1 KEY COMPONENTS OF THE HSE MANAGEMENT SYSTEMS

POLICIES AND PROCEDURES

Clearly defined policies and procedures are established to guide health, safety, and environmental practices at the project site. These policies and procedures align with corporate HSE standards and applicable regulations.

RISK ASSESSMENT AND MANAGEMENT

Comprehensive risk assessments are conducted to identify potential hazards and assess associated risks. Mitigation measures are implemented to eliminate or minimise these risks, ensuring a safe working environment.

TRAINING AND COMPETENCE

Ascendant Resources recognises the importance of providing appropriate training and ensuring the competence of all personnel involved in the Lagoa Salgada DFS project. Training programs are developed to equip employees and contractors with the necessary skills and knowledge to perform their tasks safely.

INCIDENT REPORTING AND INVESTIGATION

A robust incident reporting system is in place to promptly report and investigate all incidents, accidents, near misses and hazardous conditions. Lessons learned from these incidents are utilised to enhance safety measures and prevent similar occurrences.

EMERGENCY PREPAREDNESS AND RESPONSE

Adequate measures are implemented to ensure preparedness for emergency situations. Emergency response plans are developed, tested and communicated to all personnel, enabling a swift and effective response in the event of an emergency.

AUDITING AND PERFORMANCE MONITORING

Regular audits and performance assessments are conducted to evaluate the effectiveness of health, safety and environmental measures. These assessments provide valuable insights to continually improve HSE performance at the Lagoa Salgada Project.

24.4.2 Safety, Environmental, and Social Responsibility Policy

The Safety, Environmental and Social Responsibility Policy of Ascendant Resources Inc. outlines the company's commitment to conducting business in a safe, environmentally friendly manner while upholding the highest standards of corporate social responsibility.

24.4.2.1 Objective of the Policy

The objective of this policy is to ensure that Ascendant, together with its directors, officers, employees, consultants and contractors, operates in a manner that prioritises safety, environmental protection and social responsibility.

24.4.2.2 Application of the Policy

This policy applies to all individuals associated with Ascendant, including directors, officers, employees, consultants and contractors. It becomes binding upon notification of the most recent copy.

24.4.2.3 Communication of the Policy

To ensure widespread awareness and understanding, a copy of the policy will be provided to all stakeholders. It will also be made available on Ascendant's website. Any significant changes to the policy will be promptly communicated to all relevant parties. New personnel will receive a copy and be educated on the policy's importance. Furthermore, an annual acceptance and compliance verification process will be conducted.

COMPLIANCE

All directors, officers, employees, contractors (and their sub-contractors) and consultants must adhere to the laws, rules and regulations of the jurisdictions in which Ascendant operates. In cases of uncertainty or ambiguity, competent legal advice should be sought.

24.4.2.4 VISION FOR SUSTAINABLE DEVELOPMENT

Ascendant is committed to sustainable development, which includes a safe, environmentally friendly, and socially responsible culture. The company aims to minimise its impact on the environment, support local employment and entrepreneurship and collaborate with the community to develop infrastructure, health, education, training and cultural activities.

SUSTAINABLE DEVELOPMENT POLICY

Ascendant strives for responsible mining and sustainable development in the communities where it operates. The company aims to exceed industry standards and comply with relevant legislation and international norms.

OBJECTIVES

- Transparency, honesty, accountability, integrity and legality in corporate governance and dealings with stakeholders.
- Adherence to best practices and responsible behaviour.
- Respect for the human rights, culture, customs and values of the host community.
- Identification, assessment, management and mitigation of risks to the community and environment.
- Continual improvement of environmental performance beyond legal requirements.
- Minimisation of the impact of operations on surrounding habitats, including endangered species.
- Implementation of an environmental and social management system.
- Comprehensive monitoring, audits, and reporting to ensure compliance with relevant laws, guidelines and standards.
- Promotion and implementation of resource efficiency, waste minimisation, reuse and recycling.
- Emphasis on local workforce employment and provision of education and training.
- Capacity building of local small and medium enterprises (SMEs) to supply goods and services.
- Allocation of sufficient financial resources to fulfil all commitments, including those beyond mine closure.
- Proactive collaboration with local authorities and the host community for post-mining planning.

24.4.2.5 ENVIRONMENT POLICY

OBJECTIVE

Ascendant acknowledges that the long-term sustainability of its activities depends on good environmental management. The company is committed to integrating environmental considerations into all aspects of its operations.

IMPLEMENTATION

- Compliance with laws and applicable regulations in Canada, the U.S. and other jurisdictions.
- Adoption of the best available practices in all activities.
- Establishment and maintenance of a management system to identify, monitor, control and improve environmental performance.
- Annual reporting on environmental performance.
- Transparent communication and dialogue with stakeholders regarding environmental aspects.
- Collaboration with local, national and international institutions and organisations to protect the environment.

24.4.2.6 HUMAN RIGHTS POLICY:

OBJECTIVE

Ascendant acknowledges its responsibility to respect human rights at every level of its operations. The company is committed to integrating human rights considerations into all aspects of its activities.

IMPLEMENTATION

- Compliance with applicable laws and regulations related to human rights.
- Adoption of best available practices to respect and promote human rights.
- Establishment and maintenance of a management system to identify, monitor, control and improve human rights performance.
- Periodic evaluation of human rights performance.
- Allocation of sufficient financial resources to fulfil human rights obligations.

- Transparent communication and engagement with stakeholders on human rights matters.
- Collaboration with local, national, and international institutions to protect and promote human rights.

24.4.2.7 OCCUPATIONAL HEALTH AND SAFETY POLICY

OBJECTIVE

Ascendant provides a safe and healthy working environment for all its employees, contractors and visitors. The company is committed to preventing accidents, injuries and occupational illnesses.

IMPLEMENTATION

- Identification, assessment and management of health and safety risks.
- Compliance with applicable laws, regulations and international standards.
- Integration of health and safety strategies into all business processes.
- Provision of training, resources, and equipment necessary for the health and safety of employees and contractors.
- Promotion of initiatives to reduce occupational health and safety risks.
- Provision of adequate resources for emergency response.
- Diligent and correct use of work/access permit systems.
- Induction of new personnel regarding occupational health and safety policies and requirements.
- High levels of workplace hygiene and sanitation.
- Clear communication and enforcement of occupational health and safety standards with contractors.

24.4.3 HEALTH, SAFETY, TECHNICAL AND OPERATIONS COMMITTEE CHARTER

All Lagoa Salgada Project and associated contractor activities shall adhere to the guidelines and principles set forth in the Corporate Health, Safety, Technical and Operations Committee Charter.

The Corporate Health, Safety, Technical and Operations Committee Charter underscores the commitment to upholding the highest standards of health, safety, environmental

protection, technical oversight, risk management, social responsibility and human rights within the context of the Lagoa Salgada Project and its associated contractor activities.

This charter serves to outline the purpose, mandate and responsibilities of the Committee, with a primary objective of ensuring that life, health and the environment are safeguarded throughout mining operations. Key areas of focus for the Committee include environmental management, health and safety, technical oversight, enterprise risk management, social responsibility and human rights, all of which are integral components of maintaining the utmost integrity and adherence to industry best practices.

24.4.3.1 ENVIRONMENTAL CHARTER

The Committee is responsible for reviewing and making recommendations regarding the Corporation's environmental management program, compliance issues, incident reports and potential environmental liabilities. It ensures that the Corporation's policies and procedures align with best practices and that environmental risks are adequately managed.

24.4.3.2 HEALTH & SAFETY CHARTER

The Committee is dedicated to the overall health and safety of stakeholders, particularly employees, and their families. It reviews and recommends improvements to the Corporation's health and safety program, compliance issues and performance. The Committee monitors emerging health and safety issues and ensures that effective measures to maintain a safe and healthy workplace are implemented.

24.4.3.3 TECHNICAL CHARTER

The Committee oversees the exploration, development and operational activities of the Corporation. It reviews construction, development and operating plans, assesses risks, and provides support and guidance to management. The Committee assists in enterprise risk management by identifying and mitigating technical risks. It also ensures the integrity of the Corporation's reported mineral reserves and resources.

24.4.3.4 SOCIAL RESPONSIBILITY AND HUMAN RIGHTS CHARTER

The Committee recommends actions to develop social policies, programs and activities in communities where the Corporation operates. It promotes transparent engagement and communication with stakeholders, upholds fundamental human rights and ensures compliance with applicable laws and regulations. The Committee integrates corporate social responsibility considerations into the decision-making process.

24.4.3.5 SPECIFIC DUTIES

The Committee reviews and formulates the Corporation's goals, policies and programs related to technical, environmental, community, health and safety issues. It monitors

compliance, assesses risk exposure and evaluates the Corporation's performance. The Committee informs the Audit Committee of significant financial risks and potential disclosure issues related to its areas of responsibility.

24.4.3.6 PROCEDURES AND ORGANISATION

The Committee consists of at least three Board members, including independent directors. It appoints a Chair and members annually and is supported by the Corporation's Corporate Secretary. Meetings are held regularly, and quorum requires a majority of members. The Committee has access to necessary resources, invites management representatives and reports its activities and recommendations to the Board.

24.4.3.7 LIMITATIONS ON THE COMMITTEE'S DUTIES

Members of the Committee are expected to exercise care, diligence and skill in line with applicable standards. The Committee is a committee of the Board and not an agent of the Corporation's security holders. The Board may permit departures from the Charter's terms, and the Charter does not impose additional legal liabilities on Committee members.

25 INTERPRETATION AND CONCLUSIONS

25.1 GEOLOGY AND MINERALIZATION

25.1.1 GENERAL NOTES

Micon developed and updated the LS project mineral resources integrating the recent drilling campaign, being the most updated and final estimation, in May 2023. In 2021-2022, Ascendant/Redcorp drilled 25 step-out and infill drill holes on the north deposit and south deposit. The north deposit resource underwent more detailed wireframing upon additional drilling and reinterpretation. The estimation protocols/methodologies used in 2019 for the north deposit and in 2021 for the south deposit were used as base, but they were refined as the drilling information became denser. The former central deposit junction with the southern mineralized envelope proved to form one continuous deposit, the south deposit. The south deposit resource estimation was improved with sub-domaining into 5 different corridors.

25.1.2 SUPPORTING DATABASE

The LS property deposits have been tested by diamond drilling over a cumulative strike length of approximately 1.7 km and down to a maximum vertical depth of approximately 800 m. The resource database is derived from 132 surface diamond drillholes, all of which were utilized at the resource estimation. Original assay certificates from the laboratory were provided as csv and pdf documents. A detailed DTM and overburden depth model were provided as dxf surfaces.

The average drillhole spacing in the densest drilled areas of the project is about 20 m; the spacing in the less densely drilled areas is between 40 and 150 m.

25.1.3 SELECTION AND MODELLING OF ESTIMATION DOMAINS

25.1.3.1 NORTH DEPOSIT

Micon's estimation domain selection criteria are based on geology for the Gos and MS domains and on the zinc equivalent (ZnEq) "grade-shell" value for the stringer. To draw the "grade shells" the ZnEq% value for modeling was used, via the following formula:

$$ZnEq\% = ((Zn\ Grade \times 26.46) + (Pb\ Grade \times 22.05) + (Cu\ Grade \times 77.16) + (Au\ Grade \times 54.66) + (Ag\ Grade \times 0.71) + (Sn\ Grade \times 264.55))/26.46$$

Some more detailed re-logging from previous holes was carried out and specific intervals were included at the Gos domain that previously was identified as oxidized volcanic at the surroundings of the oxidized massive sulfides, but still carry potential economic grades.

In summary, the selected domains are as follows:

- Gos - Gossan
- tMS – transition massive sulphides
- pMS – primary massive sulphides
- str - stringer

Boundaries between the domains are predominantly hard as dictated by geology and fine-tuned with geochemical input.

25.1.3.2 SOUTH DEPOSIT

With the infill and step-out drilling after a detailed *section-by-section* interpretation of CuEq% grades, the best coherency was obtained by modeling 5 main corridors that can be traced within the main host rocks of mineralization: esV and iV rocks.

The corridors were outlined with hanging wall and footwall contact points definition with the *on-section* study of CuEq% main intervals as *grade shells*.

The modeling involved the following steps:

- Defining the boundary/outer limit of the iV and esV units using geological logs to split sMS and FR corridors sub-domains. FR corridors are exclusively included at iV lithology and sMS corridors at esV.
- Assigning each sample a copper equivalent value (CuEq%) calculated with the formula below:

$$CuEq\% = ((Zn\ Grade \times 26.46) + (Pb\ Grade \times 22.05) + (Cu\ Grade \times 77.16) + (Au\ Grade \times 54.66) + (Ag\ Grade \times 0.71) + (Sn\ Grade \times 264.55))/77.16$$

- Wireframing/modeling the main corridors of mineralization using the interpretation of each section *grade-shells* (average of 20m *step*).

In summary, the selected domains are as follows:

- C1 FR – Fissural Remobilizations Corridor 1.
- C2 FR – Fissural Remobilizations Corridor 2.
- C3 FR – Fissural Remobilizations Corridor 3.
- C1 sMS – semi-massive sulfides Corridor 1.
- C2 sMS – semi-massive sulfides Corridor 2.

25.1.4 GRADE CAPPING, COMPOSITING, STATISTICS AND VARIOGRAPHY

In previous estimations Micon examined the relationship between sample length and grade and established that a considerable number of high grades were associated with lengths greater than the model of the sample lengths of 1 m, therefore a 2 m composite was used. Following that, all the recent drilling used a sampling length of 2m. All domains were composited into 2 m for the determination of grade capping threshold value samples using population histograms and probability / log probability plots.

Some older assays under the detection limit were inputted at the database as “0” (zero) and in other cases, at the creation of the composite, some of the lower values (half of the detection limit) were broken into lower values at the residual lengths at the edges of the wireframes. For that reason, a lower cap equivalent to half of the detection limit of each metal was added. The capping was applied at the 2 m composites prior to variography

Micon completed a geostatistical analysis of all domains to determine the optimum grade interpolation parameters. Conventional variograms were applied at Gos, tMS, and pMS and correlograms were computed to str and all the south sector domains. Note that the more economically important metals per domain were the drivers of the chosen ranges. Nuggets were calculated from the Downhole variogram and sill locked at 1 for variogram and 0 for the correlogram.

The similar ranges of influence along the major and semi-major directions confirm the isotropic nature of the tMS, pMS and str domains.

25.1.5 ESTIMATION

25.1.5.1 BLOCK SIZE ANALYSIS

The results indicate an optimum block size between 5 x 10 x 5 m or 10 x 20 x 10 m. The 5 x 10 x 5 m was chosen as it honors better the local variability of the composites and mining shapes coherency.

25.1.5.2 DENSITY

Bulk density measurements were conducted on a regular way by the company on site in previous campaigns. However, at the recent drilling, systematic certified Specific Gravity was carried out at ALS Lab in a total of 358 samples at the North Sector and 914 samples at the South Sector. The density was estimated using ID3 for all the domains except for the Gossan domain where a R-sq of 89.5% regression was achieved only using Fe.

25.2 MINERAL RESOURCE ESTIMATE

25.2.1 CLASSIFICATION OF THE MINERAL RESOURCE

25.2.1.1 NORTH DEPOSIT

Micon has classified the north deposit mineral resource estimate in the Measured, Indicated, and Inferred categories.

Some more isolated Measured were identified, but for the feasibility study purpose, Measured and Indicated are suitable for the LOM and this way the measured *gaps* are well recognized and can be easily targeted with drilling. No Inferred inclusions were detected within the Measured plus Indicated backbone.

25.2.1.2 SOUTH DEPOSIT

Micon has classified the south deposit mineral resource estimate in the Indicated and Inferred categories.

25.2.2 MINERAL RESOURCE STATEMENT

The overall LS project resource estimate is displayed at Table 25.1. The QP considers that the resource estimate for the LS project has been reasonably prepared and conforms to the current CIM standards and definitions for estimating Mineral Resources.

Table 25.1 - LS Property Overall North and South Deposit Resources as of May 23, 2023

Deposit	Domain	Category	Density g/cm³	Mass Mt	NSR-MRE $/t	Zn %	Pb %	Cu %	Sn %	Ag g/t	Au g/t	ZnEq %	NSR-MRE M$	Zn kt	Pb kt	Cu kt	Sn kt	Ag koz	Au koz
North	Gos	**Measured + Indicated**	**2.96**	**1.89**	**109.0**	**0.46**	**2.51**	**0.10**	**0.26**	**44.4**	**0.86**	**8.38**	**206.2**	**8.8**	**47.4**	**1.9**	**4.8**	**2 700.1**	**52.5**
		Inferred	2.56	0.35	81.1	0.42	1.67	0.10	0.10	36.9	0.79	5.77	28.6	1.5	5.9	0.4	0.4	418.2	9.0
	tMS	**Measured + Indicated**	**4.06**	**0.92**	**107.7**	**0.25**	**2.47**	**1.08**	**0.20**	**121.6**	**1.24**	**13.27**	**98.9**	**2.3**	**22.7**	**9.9**	**1.8**	**3 591.2**	**36.5**
		Inferred	3.74	0.07	75.6	0.27	2.12	0.84	0.14	74.3	1.41	10.74	5.6	0.2	1.6	0.6	0.1	177.7	3.4
	pMS	**Measured + Indicated**	**4.44**	**5.46**	**143.7**	**3.79**	**2.90**	**0.28**	**0.13**	**66.5**	**0.72**	**11.58**	**784.1**	**206.8**	**158.3**	**15.0**	**7.1**	**11 677.1**	**125.6**
		Inferred	4.81	0.06	88.7	2.78	1.47	0.11	0.07	33.6	0.40	6.73	5.2	1.6	0.9	0.1	0.0	62.9	0.7
	str	**Measured + Indicated**	**2.91**	**0.63**	**45.0**	**0.75**	**0.24**	**0.48**	**0.08**	**22.1**	**0.06**	**3.82**	**28.5**	**4.8**	**1.5**	**3.0**	**0.5**	**450.8**	**1.2**
		Inferred	2.83	0.01	44.8	0.58	0.38	0.46	0.13	29.4	0.20	4.75	0.3	0.0	0.0	0.0	0.0	7.2	0.0
North	**Total**	**Measured + Indicated**	**3.85**	**8.90**	**125.6**	**2.50**	**2.58**	**0.34**	**0.16**	**64.4**	**0.75**	**10.52**	**1 117.7**	**222.7**	**229.8**	**29.9**	**14.3**	**18 419.3**	**215.8**
		Inferred	2.86	0.49	80.6	0.68	1.70	0.22	0.11	42.0	0.83	6.62	39.7	3.3	8.4	1.1	0.5	666.0	13.1

Deposit	Domain	Category	Density g/cm³	Mass Mt	NSR-MRE $/t	Cu %	Zn %	Pb %	Ag g/t	Au g/t	CuEq %	NSR-MRE M$	Cu kt	Zn kt	Pb kt	Ag koz	Au koz
South	C1 FR	**Indicated**	**2.89**	**6.43**	**63.3**	**0.40**	**1.26**	**0.75**	**13.13**	**0.07**	**1.21**	**407.0**	**25.5**	**81.2**	**48.2**	**2 715.4**	**13.7**
		Inferred	3.06	0.83	50.4	0.61	0.47	0.23	12.26	0.13	1.04	42.1	5.1	3.9	1.9	329.0	3.4
	C2 FR	**Indicated**	**2.86**	**2.50**	**67.5**	**0.50**	**1.23**	**0.64**	**15.58**	**0.04**	**1.28**	**168.5**	**12.4**	**30.8**	**16.0**	**1 251.4**	**3.6**
		Inferred	3.01	3.66	62.0	0.54	0.96	0.42	17.32	0.08	1.20	226.6	19.6	35.0	15.5	2 036.7	9.6
	C3 FR	**Indicated**	**2.85**	**1.11**	**62.5**	**0.39**	**1.17**	**0.54**	**20.77**	**0.04**	**1.16**	**69.2**	**4.3**	**13.0**	**6.0**	**740.0**	**1.3**
		Inferred	3.07	3.64	60.1	0.67	0.69	0.22	16.62	0.04	1.16	218.7	24.5	25.1	7.9	1 943.9	5.3
South	**Total**	**Indicated**	**2.88**	**10.04**	**64.2**	**0.42**	**1.24**	**0.70**	**14.58**	**0.06**	**1.22**	**644.7**	**42.2**	**124.9**	**70.2**	**4 706.8**	**18.6**
		Inferred	3.04	8.13	59.9	0.60	0.79	0.31	16.49	0.07	1.16	487.4	49.1	64.0	25.3	4 309.5	18.3

Differences may occur in totals due to rounding.

1. Mineral resources unlike mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
2. The mineral resources have been estimated in accordance with the CIM Best Practice Guidelines (2019) and the CIM Definition Standards (2014). Coherent and contiguous mining shapes backing reasonable prospect for economic extraction were considered.
3. Mineralized Zones: Gos=Gossan, tMS= transition massive sulphides pMS= primary massive sulphides, Str=Stringer, FR= -Fissural Remobilizations.
4. ZnEq% = ((Zn Grade*26.46)+(Pb Grade*22.05)+(Cu Grade * 77.16)+(Au Grade*54.66)+(Ag Grade*0.71)+(Sn Grade * 264.55))/26.46
5. CuEq% = ((Zn Grade*26.46)+(Pb Grade*22.05)+(Cu Grade * 77.16)+(Au Grade*54.66)+(Ag Grade*0.71)+(Sn Grade * 264.55))/77.16
6. Metal Prices: Cu $7,716/t, Zn $2,646/t, Pb $2,205/t, Au $1,700/oz, Ag $22.00/oz, Sn $26,455/t.
7. NSR-MRE: Gos= ((Sn%*0.40)*Sn$)+((Pb%*0.20)*Pb$)+((Au ppm*0.91)*Au$)+((Ag ppm*0.89)* Ag$)
   tMS= ((Sn%*0.40)*Sn$)+((Pb%*0.50)*Pb$)+((Ag ppm*0.65)* Ag$)+((Zn% *0.5)*Zn$)
   pMS= ((Sn%*0.40)*Sn$)+((Pb%*0.60)*Pb$)+((Ag ppm*0.45)* Ag$)+((Zn%*0.70)*Zn$)
   str= ((Cu%*0.60)*Cu$)+((Pb%*0.5)*Pb$)+((Ag ppm*0.4)* Ag$)+((Zn%*0.70)*Zn$)
   South Deposit= ((Cu%*0.65)*Cu$)+((Zn%*0.75)*Zn$)+((Pb%*0.65)*Pb$)+((Au ppm*0.05)*Au$)+((Ag ppm*0.80)*Ag$)
8. NSR-MRE cut-off: Gos= $48.24, tMS= $46.54, pMS= $41.06, str= $37.83, South Deposit= $37.83

25.3 MINERAL PROCESSING AND METALLURGICAL TESTING

All metallurgical and mineralogical tests results were used to develop the best estimates of metallurgical projections.

25.3.1 PRIMARY MASSIVE SULPHIDE (PMS)

- Lead concentrate: 30% Pb grade at 60% recovery (30% Ag and 10% Au associated recoveries), supported by LCTs completed by GSL and Maelgwyn laboratories.
- Zn concentrate: 35% Zn grade and 70% recovery (15% Ag recovery), supported by LCTs completed by GSL and Maelgwyn laboratories.
- Sn concentrate: 50% Sn grade and 40% recovery, estimated with the theoretical recovery/grade curve built with the mineralogical analysis completed by Petrolab. Open circuit test works with sulphide flotation, tin flotation and tin gravity completed by GSL, anticipating good tin mineral concentration. Additional work is required to confirm final tin grade and recovery, as well as auxiliary elements in the final tin concentrate.

25.3.2 STOCKWORK (STW)

- Cu concentrate: 20% Cu grade at 65% recovery (15% Ag and 5% Au associated recovery), supported by LCTs completed by GSL laboratory.
- Pb concentrate: 35% Pb grade at 65% recovery (50% Ag and 5% Au associated recovery), supported by LCTs completed by GSL laboratory.
- Zn concentrate: 45% Zn grade at 75% recovery (15% Ag and 5% Au associated recovery), estimate based on open circuit tests developed by GSL laboratory.

25.3.3 GOSSAN (GOS)

- Au/Ag bullion: 91% Au and 89% Ag extraction, supported by one bulk sample performance leaching test work. Very erratic behaviour of variability samples, where performances varied from 90% to 25% extraction for the two metals. The proposed numbers come from bulk sample but, further test work is required to confirm the data.
- Pb/Sn concentrate: 20% Pb and 40% Sn recoveries to a 30% Pb and 10% Sn grade concentrate. Estimate based on the gravity test work completed by GSL, demonstrating that both metals can be easily concentrated and could potentially bring some additional Ag and Au credits. Further test work to simulate the complete gravity circuit and determine the final grades and recoveries.

25.3.4 TRANSITION MASSIVE SULPHIDE (TMS)

- Pb concentrate: 30% Pb grade at 50% recovery (65% Ag and 50% Au associated recovery), estimated with open circuit tests developed by GSL and estimated with mineralogical analysis completed by Petrolab. Additional work is required to confirm the Pb upgrade levels, as well as Pb, Au and Ag recoveries.

  Alternatively, leaching could be applied to this domain with 45% Au and 3% Ag extraction, according to the cyanidation testing completed in GSL. With the Cu head grade and speciation, it cannot be estimated, anticipated, or justified, at this stage, that better performance of a leaching process will be achieved.

- Zn concentrate: 40% Zn grade at 50% recovery, estimated with mineralogical analysis completed by Petrolab. Additional work required.

- Sn concentrate: 50% grade and 40% recovery, estimated with the theoretical recovery/grade curve built with the mineralogical analysis completed by Petrolab. Metallurgical test works required to demonstrate the possibility of recovering this metal.

25.3.5 STRINGER (STR)

- Cu concentrate: 20% grade at 60% recovery (10% Ag associated recovery), supported by LCTs completed by GSL laboratory.

- Pb concentrate: 30% grade at 50% recovery (15% Ag associated recovery), estimate based on open circuit tests developed by GSL and supported by the mineralogical analysis completed by Petrolab. Additional work is required to confirm the Pb upgrade level and recovery.

Zn concentrate: 40% grade at 70% recovery (15% Ag associated recovery), supported by open circuits completed by GSL laboratories. Additional works required to confirm the proposed numbers.

25.4 MINERAL RESERVE ESTIMATE

The mine design and Mineral Reserve estimation have been completed to a level appropriate for feasibility studies. The Mineral Reserve estimate stated herein is consistent with the National Instrument 43-101: Standards of Disclosure for Mineral Projects and is suitable for public reporting. As such, the Mineral Reserves are based on Measured and Indicated Resources, and do not include any Inferred Resources.

The Net Smelter Return (NSR) model intends to simulate the geo-metallurgical behavior of each mineable shape in isolation, that is, without any blending with other shapes. Effectively, each shape ought to produce within specification products on its own.

IGAN coded the NSR values for all the mineralisation defined in the Mineral Resource block models and with consideration to the effects of deleterious elements in the concentrate net value. To do so, costing, pricing, processing recoveries and concentrate commercial terms provided by both Ascendant and Minepro were used.

NSR cut-off grades were established following Mortimer's approach to cut-off grades (Hall, 2014), which identifies rock as economically mineable when the following criteria are met:

- The lowest grade of rock must pay for itself.
- The average grade of rock must provide a minimum average profit per ton.

The former, also known as "boundary cut-off", guarantees that no losses are incurred by mining and treating a volume of rock. The latter or "volume cut-off" assures that the rock volume yields a certain average level of profit per treated ton, so shared costs such as general & administration are covered by the average production grade.

In the estimate of the Mineral Reserves, modifying factors were applied to the tonnages and grade of all mining shapes (both stoping and cut & fill) to account for unplanned dilution and ore losses that are anticipated in this project.

Ore dilution includes overbreak into the design hanging wall, footwall, and adjacent backfilled stopes. Diluting materials are assumed to carry no metal values in the estimation of inventory grades. Ore losses (recovery factor) are related to the practicalities of extracting ore under varying conditions, including difficult mining geometry, problematic rock conditions, blasting issues, and other aspects.

The dilution has been estimated by IGAN considering its own experience and benchmarking against similar long-hole open-stope operations in the Iberian Pyritic Belt. The recovery has been estimated based on experience in operations within the region with similar mining methods, equipment, and operational drilling practices to those anticipated to be used in this project. As no paste strength tests are available at the time of the preparation of this report, an average value of dilution and recoveries has been assumed for both primary and secondary stopes.

IGAN employed Deswik.SO ("Stope Optimizer") optimisation software to generate mineable stope and cut & fill shapes for each orebody meeting meet both cut-off grade and operational design criteria. These preliminary shapes were then individually refined to minimize the amount of sub-economic material within the shape volume that cannot be selectively mined.

Table 25.2 presents the Mineral Reserve estimate, broken down by mine zone and reserve category. The values presented herein are reported in a diluted and recovered mass and grade basis and have been rounded to an appropriate number of significant figures that reflect the uncertainty in the estimate; thus, totals may reflect small rounding errors.

Table 25.2 - Mineral Reserve estimate per mine area

Zone	Reserve category	Mass (kt)	Cu (%)	Pb (%)	Zn (%)	Sn (%)	Ag (ppm)	Au (ppm)
North	Proven	2 100	0.26	2.8	3.2	0.15	62	0.75
	Probable	4 900	0.34	2.3	2.1	0.14	63	0.054
	Total							
South	Proven	-	-	-	-	-	-	-
	Probable	7 600	0.41	0.72	1.3	0	15	0.054
	Total					-		
Total	Proven	2 100	0.26	2.8	3.2	0.15	62	0.75
	Probable	13 000	0.38	1.3	1.6	0.056	33	0.32
	Total	15 000	0.37	1.5	1.8	0.069	37	0.38

The Mineral Reserve estimate per mine zone and lithology is detailed in Table 25.3. As with Table 25.2, the values are reported to a relevant number of significant figures that reflect the uncertainty in the estimate. The totals may reflect rounding errors as well.

Table 25.3 - Mineral Reserve estimate per lithology

Zone	Lithology	Mass (kt)	Cu (%)	Pb (%)	Zn (%)	Sn (%)	Ag (ppm)	Au (ppm)
North	Gossan	1 200	0.1	2.4	0.41	0.25	52	1.0
	Transition	880	0.98	2.2	0.23	0.18	110	1.1
	Massive sulphides	4 400	0.25	2.7	3.6	0.12	62	0.68
	Stringer	550	0.23	0.18	0.54	0.056	13	0.063
	Total	7 100	0.31	2.4	2.4	0.15	63	0.75
South	Stockwork	7 800	0.41	0.71	1.3	-	15	0.06
	Total	7 800	0.41	0.71	1.3	-	15	0.06

25.5 MINING

The proposed underground mine design supports the annual extraction of 1.2 million ore tonnes ("Mtpa") through a combination of transverse sublevel stoping and cut&fill. Cemented paste is ought to be employed as backfill material of the mining voids, so ore recovery and productivity are maximised.

A fleet of LHDs (Load-Haul-Dump) units will be used for both material loading and tramming between production areas and dedicated level stockpiles. From the level stockpiles, ore will be loaded into 65-tonne mine trucks and hauled to a Run of Mine (ROM) pad, located on the surface. Waste is also transported to the surface with 65-tonne mine trucks.

The ventilation system is designed as a two parallel branch system. Fresh airflow ingresses through the main ramp and a central ventilation shaft. It is then distributed into two branches, one serving a mine area (i.e., North and South). Polluted air is exhausted through two exhaust shafts, located at the far end of each area. Two main ventilation fans are installed on the surface, one per ventilation exhaust shaft.

A pre-production development program will be required to provide access to the initial stoping levels in the North zone during the first twenty-four months. Production will start in the second year, reaching the nominal plant feed in the second year.

A main decline, starting from the surface portal, placed close to the processing plant, will be used to access the mine. The main decline bifurcates into two independent ramps, one per mine area (North and South orebodies), at the 15 Level. The zone ramps connect all the mine levels within.

The use of an independent ramp per zone instead of one providing access to both zones was preferred as it reduces the initial Capital Cost (CAPEX). Production starts earlier in the North zone and the mine economics favours the ramp design option providing faster access to it, which in this case is achieved through a dedicated zone ramp.

All ramps and main decline were designed with a 25 m turning radius and a maximum 12.5% gradient. As a rule, declines and ramps are designed with a minimum 75 m standoff distance from the ore contacts.

Sublevel stoping extraction levels in both zones were designed with a 25 m vertical spacing, as determined by the stope height; all of them are accessed through the zone ramp. Footwall drives, as a rule, are designed with a minimum 35 m standoff distance from the ore contact. Fresh and return airways are located at each of the sublevel ends, with the level access located at the approximate centre of gravity of the footwall drive. Ore drives, spaced 15 m from each other, provide access to stopes.

Cut&fill is the mining method employed for the extraction of the Gossan and Transition lithologies in the North orebody.

The deposit is divided into four mining blocks, defined by mining method, level and mine area. The number and geometry were determined in a way so they sustain the nominal annual target production of 1.2 Mt.

According to the work completed by RMS, the primary means of backfilling at the Property will be cemented pastefill, generated from thickened mill tailings and mixed with a blend of sand and binding agents to a defined recipe.

Two paste fill mixes, one for stockwork lithology and other for the North (i.e., massive sulphide, transition zone, gossan and stringer lithologies), will be used in the backfill of mining voids.

The so-called stockwork mix will contain 5% binder and is anticipated to be used in the South sector; thus, its strength ought to be enough so the stability of voids is guaranteed. The North mix will contain 2.5% binder and will be used in stopes as well as in cut&fill drifts. This mix will contain sufficient binder to remove any risk of future liquefaction.

An extra high-strength paste fill will be required in bottom cut&fill lifts and that will be undercut by sill extraction from below.

Waste rock will need to be disposed of on an ongoing basis throughout the mine life. Waste will be hauled to surface for disposal at a designated waste dump. Some of the waste generated during the pre-production phase of the Project is anticipated to be used as construction material for the TSF dam and the RoM pad ramp.

Mine dewatering was designed to accommodate groundwater inflows from the Lagoa Salgada workings and inflows from drill, dust suppression and other operating equipment.

Using the Monte-Carlo method, the freshwater flow rate required to supply 95% of the possible water consumption scenarios was estimated. The required flow, including leakages, is 9.2 l/s (or 33 m^3/h) with a peak demand of 16 l/s.

The average groundwater inflow was estimated, considering the available studies at the date of this report ("*Lagoa Salgada_ relatoriofinal_versao_draft.pdf*"), to be 30 l/s in the North and 15 l/s in the South. Thus, the total inflows are estimated to be approximately 50 l/s in the North and 40 l/s in the South. These pumping flow rates ensure that the dewatering system can pump the peak demand and has an allowance to accommodate for uncertainty in the groundwater inflow.

25.5.1 GEOTECHNICAL

Geotechnical designs and recommendations are based on the results of available data and geotechnical assessments, which included intact rock assays, rock mass characterisation, structural geology, excavation and pillar stability analyses, and ground support design.

Geotechnical site investigations completed to support the underground rock mechanics assessments included: geotechnical drilling and logging, oriented drill core measurements, borehole televiewer surveys, laboratory testing of rock core samplesy.

The geotechnical survey campaign includes nine geotechnical boreholes totalling 3 310 drilled metres. To intersect the orebodies perpendicularly, a 60º angle from the horizontal was used on the boreholes.

Samples obtained from the geotechnical drilling campaign were sent to the Centro de Recursos Naturais e Ambiente ("Cerena") geotechnical laboratory in Lisbon to perform 396 assays.

Although all the units are defined as a homogeneous rock massif, they may be the result of a combination of several lithologies. Assay values from laboratory test work are listed in Table 25.4.

Table 25.4 – Geotechnical properties

Unit	σ_C (MPa)	E (MPa)	v	E/σ_C	C (MPa)	Friction (º)
MS	129.74	118 883	0.22	1 244.01	19.93	49.17
dtV/aV	20.85	43 360	0.36	1 955.2	3.31	37.39
Gos	69.11	77 575	0.36	0.36	8.91	53.34
Ter	33.06	16 900	0.26	555.94	6.99	33.04
iV/aV	33.6	83 900	0.23	2 750.81	10.12	25.96

Rock mass characterization was made considering the intact rock characterization and data available from the geotechnical drilling campaign. Long and transverse sections were made to identify the distribution of the Q values within the orebodies.

Considering both the assayed Q values and the spatial distribution of the drillhole data, a total of five geotechnical domains were defined as shown in Table 25.5, dividing the mine into the different domains.

Table 25.5 – Geotechnical domain definition

Domain	Type	Q
DG I	Very poor	0.01 – 0.1
DG II	Poor quality	0.1 - 4
DGIII	Fair quality	4 - 10
DG IV	Good quality	10 - 40
DG V	Very Good quality	>40

For each geotechnical domain a support standard was defined to support mine planning.

25.6 HYDROGEOLOGICAL

A hydrogeological study (BHS) was prepared in the FS. The BHS intended to evaluate, understand and characterise the dynamics of the hydrogeological systems, including the cover formations of the Cenozoic and the underlying formations of the Palaeozoic, where the mineral deposits (mineralization in polymetallic sulphides of Venda Nova North and Venda Nova South) are located, considering the role of the various hydrogeological units present, their complexities and influences, and the interrelationship with surface waters, natural and industrial ones produced by the Project, within the scope of the design of the future mining operation.

To carry out this hydrogeological characterisation study of the Project area, it was necessary:

- To establish, based on available data, the hydroclimatic parameters required to estimate the water balances under different climatic conditions and to determine the probable likely maximum rainfall.
- To characterise the conditions of the surface water runoff, in the Project area and its surroundings, to define the infiltration rate into the soil, the water surplus and/or water deficit, and the floodable areas.
- To define the hydrogeological characteristics of the Cenozoic and Palaeozoic lithological materials and the predictable impacts on the groundwater aquifer systems.

The specific objectives being:

- The identification of the geological formations and their structural characteristics, which contribute to groundwater storage and flow (aquifers, aquifuges, and aquitards).
- The determination of the hydrodynamic parameters that characterise the different hydrogeological systems.

- The surface water and the groundwater physicochemical characterisation, as an aid in the establishment of flow dynamics, and the relationship between the different hydrogeological systems and surface waters.

- The determination of recharge rates in the underground systems, considering heterogeneities and the presence of sedimentary coverings (multi-layered hydrogeological system).

- The establishment of the conceptual hydrogeological model for the area of potential mining affectation/influence.

For the hydrometeorological setting, the temporal distribution of precipitation and temperature (monthly) were defined, and the characterization of dry, average, and wet years, and the water balance were carried out according to the Thorntwaite and Mather formulas, establishing the useful rainfall, surface runoff, and infiltration.

Climate change was considered for several future scenarios, in short, and intermediate time, considering the less restrictive level RCP 4.5 scenario, and the most restrictive RCP 8.5 scenario, according to information made available by the Climate Portal (http://portaldoclima.pt).

The streams and hydrographic basins in the Project implementation area and their vulnerability to the in-depth aquifer recharge, which could be felt throughout the Project area, have been characterised.

The hydrogeology of the aquifer systems was characterised, i.e., it has been established the general and detailed hydrogeological characteristics of the study area, based on 12 core drillings intersecting the Tertiary formations that including the 5 initials, mentioned above, plus a new set of 7 holes meanwhile carried out by REDCORP during 2022 and, even more, data from old drillings carried out by LNEG and REDCORP.

Four aquifer systems have been defined in the Project area, with variable thickness and productivity, namely:

- **Cutaneous Aquifer System**. The upper young formations (Holocene and Plio-Pliocene); contain 2 free aquifers in unconsolidated fine sandstones separated by a thin impervious clayey metric layer intercalated between the top of the sandstones and the surface aeolic sands, at a depth of 0-20 m. It is an aquifer system of great environmental importance as it supports the plants habitat dominated by cork oaks trees and is of social importance of its use by local inhabitants and landowners.

    - **Intermediate Aquifer System**. In the Miocene formations; porous sandstones, conglomerates and calcarenites; multi-layered semi-confined aquifer system (up to 4-5 m thick each), the average depth between 30 and 80 m. It includes the top of the Palaeogene formation (top of the Vale do Guizo Formation) when it is sandstone type within argillites (80-90m). It contains at least two important and identified aquifers each one with its own characteristics, but there may be other multi-layered aquifers of lesser

expression and importance throughout this formational sequence; it is the main groundwater catchment and supply aquifer system for various public and private users.

- **Deep Aquifer System**. The two lower systems merge into a single deep aquifer system because, although with their own specificities, they are contiguous and interfacing, and can only be easily identified where there is no overlap with mineralization.
    - **Basal System**. Paleogene/Palaeozoic contact corresponding to the important stratigraphic hiatus that emerged; heterogeneous aquifer, at variable depths between 120 and 150 m in the area of the deposits.
- **Deep System *ss***. Palaeozoic within the Volcano-Sedimentary Complex and massive, disseminated, or venular sulphides; corresponds to fractured and/or mineralization-related rocks: polymetallic sulphides, stockwork and *gossan*; mixed heterogeneous porous/fissured confined to semi-confined system; indefinite depth.

Geochemical techniques facilitate the interpretation of the relations between surface water and groundwater to elaborate the conceptual hydrological-hydrogeological model and, based on it, to be able to predict changes that may cause anthropic impacts in the hydrological systems *sensu lato*.

A study was made for each parameter recorded, both *in situ* and laboratory (ALS), and different hydrochemical conclusions based on the analytical results are presented in the report.

The chemical facies and the temporal and spatial variations of the parameters between the sampling dates were studied.

The results are presented for the main cations (calcium, magnesium, sodium, and potassium) and for the main anions (chlorides, hydrogen carbonates (bicarbonates), sulphates, nitrates, and boron). As well as other parameters of interest (free CO2, nitrites, aluminium, antimony, arsenic, barium, beryllium, lead, copper, chromium, iron, dissolved iron, bivalent iron, trivalent iron, phosphorus, phosphorus as P2O5, fluorides, lithium, manganese, mercury, molybdenum, nickel, vanadium, zinc, cadmium, cobalt, silver, selenium, and thallium.

Regarding the radiological parameters, for tritium, no conclusion was obtained due to the detection limit which needs to be much lower than that performed by the laboratory. It is recommended that a new dating with a lower limit be performed and that the isotopes oxygen 18 (δ18O) and deuterium (2H) be studied to be incorporated in the determinations in future campaigns to obtain a better understanding of the hydrodynamic relationships.

A conceptual hydrogeological model of the study area is presented, focusing on the following items:

- Interpretation of the model of flows in the cutaneous system.
- Flows in the intermediate aquifer system.
- Flows in the deep aquifer system.

Numerical modelling was not carried out. A numerical model cannot be made without historical piezometric data, nor with the pumping and testing carried out, which will contribute to it, in the future, not without first having the coverage of a whole hydrological year, with both dry and wet seasons, and long-lasting tests, and as such already the subject of the recommendation.

The BHS is an approach as a first step towards establishing the relationships between the different aquifer systems and the Lagoa Salgada Mining Project itself, to be implemented as the Project develops and progresses over time.

25.7 RECOVERY METHODS

The metallurgical test work results have been interpreted to establish comprehensive process design criteria for the plant. This information, along with mass and water balances, has been utilized to create plant specifications and equipment listings, which serve as the foundation for estimating capital costs. A conceptual plant layout has been devised, incorporating enough detail to accurately estimate capital costs related to steel, civil work, and earthwork.

The mineral treatment plant will mainly consist of 10 different process areas where the mineral will be crushed, grinded and then fed to several flotation processes where Cu, Pb, Zn and Sn concentrates and Au/Ag dore bars will be produced. Other than crushing, grinding, dewatering and auxiliary services, common for all mineral domains, the remaining process areas will be configured depending on the mineral domain being processed, briefly summarized below:

- Common process areas for all mineral domains:
  - Area 100, Crushing.
  - Area 200, Grinding.
  - Area 800, Dewatering.
  - Area 900, Reagents.
  - Area 1000, Auxiliary Services.
- Specific process areas for Fresh Massive Sulphide (PMS):
  - Area 300, Copper, and Lead flotation (bulk Cu/Pb concentrate will be produced).

- Area 400, Zinc flotation.
- Area 500, Sulphide flotation.
- Area 600, Tin flotation.

- Specific process areas for Transition Sulphide (TMS):
  - Area 300, Copper, and Lead flotation (bulk Cu/Pb concentrate will be produced).
  - Area 400, Zinc flotation.
  - Area 500, Sulphide flotation.
  - Area 600, Tin flotation.
  - Area 700, Leaching.
- Specific process areas for Stockwork (STW) and Stringer (STR):
  - Area 300, Copper, and Lead flotation (separate Cu and Pb concentrates will be produced, using the Cu/Pb rougher circuit for Cu and the Cu/Pb scavenger circuit for Pb).
  - Area 400, Zinc flotation.
- Specific process areas for Gossan (GOS):
  - Area 600, Tin flotation.
  - Area 700, Leaching.

The concentrates grades and recoveries for the different domains are stated in the following table.

Table 25.6 – Metallurgical performance projections

	PMS	STW	GOS	TMS	STR
Cu Conc	-	20% Cu @ 65% Rec. (15% Ag / 5% Au Rec.)			20% Cu @ 60% Rec. (10% Ag Rec.)
Zn Conc	35% Zn @ 70% Rec. (15% Ag Rec.)	45% Zn @ 75% Rec. (15% Ag / 5% Au Rec.)		40% Zn @ 50% Rec.	40% Zn @ 70% Rec. (15% Ag Rec.)

	PMS	STW	GOS	TMS	STR
Pb Conc	30% Pb @ 60% Rec (30% Ag / 10% Au Rec.)	35% Pb @ 65% Rec. (50% Ag / 5% Au Rec.)		30% Pb @ 60% Rec. (65% Ag / 50% Au Rec.)	30% Pb @ 50% Rec. (15% Ag Rec.)
Au/Ag Buillion	-		91% Au / 89% Ag extraction		
Sn Conc	50% Sn @ 40% Rec		30% Pb @ 20% Rec. 10% Sn @ 40% Rec.)	50% Sn @ 40% Rec	

Additional work is required to confirm final concentrates grades and recoveries, as well as auxiliary elements in the final concentrates. Ascendant will continue the program test work at Maelgwyn laboratories in South Africa.

Regarding the water process quality, depending on the evolution of the quality of process water, the installation of a water treatment plant will be required to reduce the thiosalt content and allow the water to be used in the flotation processes. This facility will not be implemented from the beginning of the operation, but a reserved area for its future implementation must be considered.

25.8 PROJECT INFRASTRUCTURE

The Lagoa Salgada Project will be developed on a greenfield site located in close proximity to Grândola, Setúbal district, which benefits from well-established infrastructure including road and rail transport, power, and water supply services. The site will be accessed via an upgraded tarmac road connected to the municipal road EM-543. Three access routes have been studied, with one preferred access for light vehicles (Secondary Road access) and two alternative access routes for heavy vehicles. Transportation of supplies will be facilitated by trucks from Portugal or Spanish locations, while concentrate products will initially be shipped to the Sines port by road and subsequently by ship to final destinations.

The project site will have a compact layout that incorporates essential components such as the tailings storage facility, ore and waste dumps, water treatment infrastructure, and various buildings including administration, warehouse, laboratory, gatehouse, and mobile equipment workshop. The processing facilities will consist of a primary crusher building, ore stockpiles on the ROM pad, a mill building, and a paste plant building.

The mine access, known as the mine portal, will be implemented using a Cut & Cover solution. The ore will be brought to the surface and stored as stockpiles on the ROM pad near the Mine Portal, while waste stockpiles will be utilized for constructing the embankments of the Tailings Storage Facility (TSF). These stockpiles will serve for ore processing or permanent storage, with the material being crushed at the surface by the Primary Crusher and transported to the processing plant via conveyor belts.

During the construction phase, power supply for the site will be obtained by connecting to the existing grid, while an overhead transmission line will be installed during operations to replace the grid connection. The primary source of water supply for the project will be the underground mine dewatering system.

The mine plan outlines the processing of 14.8 million tonnes (Mt) of ore and the generation of 1.9 Mt of waste rock. After accounting for concentrate and underground backfill, a total of 11.3 Mt of tailings, along with 1.0 Mt of development rock, will be deposited in the TSF. The TSF design provides an estimated storage capacity of 5.9 million cubic meters (Mm3) for tailings and development rock.

25.9 MARKET STUDIES AND CONTRACTS

The Lagoa Salgada Project will produce a gold/silver dore and four (4) saleable metal concentrates and, including:

- Zinc concentrates.
- Copper concentrates.
- Lead concentrates.
- Tin concentrate.

Annual production of each concentrate produced will represent a small portion of supply in a large and liquid global market with transparent daily market pricing. As such, production from Lagoa Salgada is expected to be freely delivered into the market with minimal impact on overall supply/demand dynamics for their respective global markets.

The concentrates are expected to be delivered to smelters and refineries within Europe and transportation costs have been assumed on this basis. However, the concentrates are expected to be saleable into any of the major markets or smelters globally and are not dissimilar to products being delivered by currently producing mines in the Iberian Pyrite Belt.

Currently, no contracts for offtake for any products have been executed. It is expected that these will be negotiated as part of future financing decisions as is normal for a project at this stage of development. However, the Company has confirmed the saleability of the products with several large commodity traders and smelters within Europe. The results of these discussions have been incorporated into this study, specifically as it relates to market terms for payability, by-product credits, treatment and refining charges and penalty rates for any deleterious elements.

25.10 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL IMPACT

To comply with the established in Canadian Norm NI-43-101, an environmental section was considered in the present DFS, however it should be noticed that, under the Portuguese

Legal Environmental Regime applicable, the Environmental Impact Assessment is still ongoing and its main report can only be delivered to the authorities after the DFS (that, in this particular case, is the Project that will be evaluated during the Environmental Impact Assessment Procedure) is concluded.

In this context, it is presented the environmental applicable information that is available at this stage of the Portuguese environmental process of Lagoa Salgada Project, namely: site setting, environmental and heritage baseline studies, environmental risks, environmental assessment process & permitting, considerations of social and community impacts.

25.11 CAPITAL AND OPERATING COSTS

The capital expenditure (Capex) required for the implementation of the Lagoa Salgada Project has been estimated to a level of accuracy of +/-10% to 15% using the outcomes of the basic engineering study completed by Quadrante, Minepro, IGAN, SLR and RMS (i.e., equipment selection, sizing, and specification; civil, concrete, mechanical, structural, electrical and instrumentation material take offs; construction and management services; taxes, etc.) following both international and Portuguese standards.

Capital cost estimates were developed for the construction of an underground mining and processing operation capable of treating 1,200,000 tonnes per annum of ore to produce copper, zinc, lead, and tin concentrates, also including credits for gold, silver. Gold and silver bullion will be also produced.

Total capital expenditures of $255.34 million are estimated for the LOM, $267.24 million with contingency. The initial CAPEX (first eighteen construction months) is $152.45 million, $164.35 million with contingency. The capital cost estimate is divided into ten broad-scope areas:

- Support infrastructure.
- Tailings storage facility.
- Paste-fill plant.
- Mining Processing Plant.
- Underground capital.
- Mobile Equipment.
- External connections.
- Box cut.
- Closure cost.
- Other Capex/studies.

The Capex excludes:

- Any additional exploration costs.
- Any financing costs, including capitalized interest and commitment fees.
- The costs incurred to establish the operations and conduct feasibility studies prior to project financing.
- The working capital, which refers to operating costs beyond the scope of Capex that are incurred during the construction and ramp-up of the project, net of the revenue generated in the same period.
- The bridging costs required to prepare the project for construction include detailed engineering FEED, land acquisition, and further geotechnical work.

The operating costs estimates resulted from collaborative work between Ascendant, Quadrante, IGAN, and Minepro, which serve to determine the annual production costs. These unit costs are presented as total Life of Mine (LOM) expenses and expressed in dollars per tonne processed. All costs included in the estimate are reported in US dollars.

The estimated total operating expenditures over the LOM period amount to $540.32 million. However, certain non-capital expenditures occurring before the processing plant becomes operational over the first eighteen months are categorized as Pre-production operating costs. These costs, totalling $8.35 million, are included in the total operating cost expenditure.

The operating cost estimate is divided into three main areas: Mining, Processing, and General and Administrative (G&A). The detailed operating costs presented here are based on design criteria for the mine and process plant, as well as engineering information and unit cost data obtained from various sources.

The operating cost estimate meets the criteria to be classified as a Class 3 estimate, as defined by AACE International, with an approximate accuracy level of ±15%. It is important to note that all costs included in the estimate are reported in US dollars.

The operating cost estimate has been meticulously calculated using a comprehensive bottom-up approach, which relies on crucial engineering deliverables that define the scope of the project. The scope itself was described and quantified through a series of line items within material takeoffs (MTOs).

25.12 ECONOMIC ANALYSIS

A discounted cash flow analysis model has been developed for the project using Microsoft Excel. Revenues and costs are input on an annual basis, starting from a specific date that corresponds to the continuation of detailed design and construction.

According to the model, the project described in this feasibility study, along with all the stated assumptions, is economically viable. The post-tax net present value (discounted at a rate of 8.0%) is calculated to be $147.1 million, with an internal rate of return (IRR) of 39%.

The total pre-production capital expenditures amount to $164.4 million, which includes a contingency allowance of $11.9 million.

The economic viability of the project is influenced by commodity prices, specifically zinc, as well as the OPEX costs.

25.13 ADJACENT PROPERTIES

There are no properties of any significance directly adjacent to the site.

The most relevant current and past mining operations in the IPB. The following mining operations are currently active in the IPB:

- Aljustrel Mine (Almina).
- Neves-Corvo Mine (Somincor – Lundin Mining).
- Aguas Teñidas, Sotiel, and Magdalena Mines (Matsa – Trafigura/Mubadala).
- Rio Tinto Mine (Atalaya Mining).
- Cobre Las Cruces Mine (CLC – First Quantum Minerals) – Currently reprocessing mined tailings.

In addition to these active mining operations, several projects are at the exploration or permitting stages.

26 RECOMMENDATION

26.1 GENERAL

The work carried out during the Feasibility Study and presented in this Technical Report outlines an economically feasible project based on the provided inputs and assumptions.

In recent months, extensive analysis have been conducted to compare different operational scenarios for the project, including variations in ore and waste handling methods, diesel/electrical equipment, automation, mineral processing alternatives, concentrate commercial terms, and more.

Based on our findings, Quadrante recommends further in-depth investigation of the preferred project configuration described in this report through Detailed Engineering, namely the mineral processing investigation and exploration drilling.

26.1.1 GEOLOGY/RESOURCES

A methodical program to upgrade and expand the resources and lead into the discovery of new VMS deposits is suggested as follows:

- Continue the drilling between the gap that still exists between north and south deposits.
- Systematic infill and step-out drilling at the south deposit to convert the Indicated resources into Measured, the Inferred into Indicated and increase the Inferred, as the north dip plunge at this sector stills open.
- Strategic Infill in some portions of the north deposit to confirm some edges of the wireframes and convert the left Inferred at Gos, tMS and pMS.
- Systematic drilling at the stringer from the north sector to build the Indicated resources and allow the proper geostatistical representation of the Cu high grade spots.
- Step-out and exploration drilling at the possible northwest plunge of the north edge and eventual offset of the south edge of the north deposit.
- Geophysical investigation is also recommended paired methodically with drilling and downhole surveys as many logical targets are defined between the different surveys carried out.

26.1.2 MINING

26.1.2.1 HYDROGEOLOGY

Consmaga QP and Quadrante suggest the following key aspects:

- Plan the mining infrastructure in such a way that the cutaneous and intermediate aquifer systems are not affected. The environmental and social advantages of keeping these aquifer systems untouched are very obvious in the project's relationship with the region.
- Recommend the installation of a system to pre-drain the mineral deposits, in the deep aquifer system, from pumping and bleeding boreholes with eventual aquifer re-injection (recharge) both installed in the direction of predetermined flows from South southwest (pumping) to north-west (re-injection), to be replaced by the drainage system to be installed later in the mining works.
- Plan the mine drainage values, with sufficient pumping capacity to pump the drainage flows established above, for the deep aquifer system of the order between 20 and 40 l/s, as established above, and taking into consideration the values for each mineral deposit, separately.
- To elaborate, in the near future, as soon as the mining decision is taken, a PVCASS - Plan for Surveillance and Control of the Project's Surface water and Groundwater, with continuous, integrative monitoring criteria, established as a real Water Management Plan, also integrating industrial, waste, and rainwater, its reuse and utilization, as a way to control the Project's relationship, in the underground and on the surface, in the aquifer protection, but also the hydrological and soil protection and related geological formations.

26.1.2.2 UG MINING

Quadrante QP has identified various opportunities that have the potential to further improve project economics:

- Further optimization of the mine plan and stope sequence to identify and realize higher NPV.
- Further optimization of the cut-off NSR, based on the final FS costs, to increase resource conversion and extend the mine life.
- Considering the findings of the hydrogeological program, it is advisable to review the methodology for sinking and supporting the collars and initial meters of the ventilation raises. The current approach in the feasibility study relies on an extensive grouting program. However, conducting further investigations could potentially lead to significant cost reductions.

- Assess the possibility of contracting mining development or production services as a part of risk mitigation strategies to address the current market limitations in skilled employees, specifically in maintenance and mechanics.
- Further investigate expanding electrification of underground mining equipment to include major other underground mobile equipment. BEV equipment was determined to be economically beneficial to the project (trucks, loaders and mixer trucks), but as the BEV technology improves, the cost of the equipment is becoming more competitive with the standard diesel equipment with potential project benefits.
- Maintain ongoing evaluation of emerging technologies aimed at enhancing automation in operations to enhance efficiency and safety. This assessment may involve considering the integration of autonomous production equipment and support equipment.
- Continue to investigate reducing the addition of sand in the backfill to reduce cost. Further test work is required.
- Increase geotechnical knowledge. The design of C&F, stopes, and infrastructure are based on preliminary geotechnical data, and in IGAN´s opinion, more data is needed to decrease uncertainty in some design areas. Specifically, IGAN recommends increasing knowledge in:
    - Stope sizing and geometry
    - Top 100 meters from the surface where the two aquifers are located. The development design needs to be customized for the specific conditions of that area.
    - Area between the north and south orebodies. The ramp is designed in that area and there is little knowledge of how the north orebody transitions into the south with a completely different mineralogy. This could be an indication of a geological structure, yet to be detected, between the north and the south.
- Structural model: to IGAN´s knowledge, no formal geology structural model is available. IGAN recommends that one is developed as soon as possible since it is the founding block to understand the orebody and it is an important guiding piece of information on the design phase.
- Building method for the shafts. Ventilation shafts go through the soft and saturated ground present on the upper part of the property. A detailed building schedule needs to be developed to make it match the mine development needs as well as making sure the capital estimate is correct.
- IGAN recommends checking the plant feed in a smaller time window than the one used for the LOMP. This will ensure that mine output, ROM capacity, and plant changes between processes are correctly aligned to process all the ore produced by the mine.

26.1.3 METALLURGY AND PROCESS

Some of the metallurgical performances described will require additional work in the laboratory. Key investigation works to be performed are described below.

26.1.3.1 PRIMARY MASSIVE SULPHIDE (PMS)

Grades and recoveries are well supported by LCT performed in the laboratory, however additional work to improve the Zn concentrate grade should be completed. Sn concentration has shown good potential but will require more work.

26.1.3.2 STOCKWORK (STW)

Grades and recoveries are well supported by LCT performed in the laboratory, except for the Zn recovery which would require more testing to achieve the target values determined by open circuit test works.

26.1.3.3 GOSSAN (GOS)

Leaching efficiencies come from one individual sample and the performance of variability samples was not consistent with these values, so further works is required to determine the difference between materials and the optimal route to achieve the desirable performances. Pb and Sn concentration have shown good potential but will require more investigation.

26.1.3.4 TRANSITION MASSIVE SULPHIDE (TMS)

The performance of open circuit testwork has shown a good potential to produce Pb and Zn concentrates, however the performance of cleaners will require more investigation, as well as LCTs to confirm the final values. Sn concentration looks feasible based on mineralogical analysis but will require laboratory testing.

26.1.3.5 STRINGER (STR)

Open circuit and LCT completed have shown a good potential for the three Cu, Pb, and Zn concentrates but additional works are required to confirm target values established with open circuit tests.

26.1.3.6 BIO-LEACHING

Bio-leaching of the TMS test results achieved following (needs to be pursued in more detail moving forward):

- Mass reduction of tailings by 70%/ reduction of CN leach feed by 70%;

- Generation of H2SO4 from bio-leach;

26.1.4 INFRASTRUCTURE

The following potential opportunities related to surface infrastructure should be investigated as new technology or information becomes available.

- With the deepening of the hydrogeological study, review the water management system to minimize water consumption, treatment, and discharge needs into the environment.
- With the constant advancement of technology and the consequent reduction in costs, continue to investigate the use of batteries for energy storage in the PV plant to maximize the utilization of solar energy and reduce energy costs.
- The ground investigation data for the TSF location is limited currently and further investigation is needed to address issues relating to the clay layer which is believed to underlie the entire TSF.
- Develop a ground investigation program to the TSF; this investigation is intended to provide both hydrogeological and geotechnical data for the TSF area. Additional investigation elements may be required based on the conditions encountered. Hence, a multi-technique and phased approach is recommended, this will allow the efficient coverage of the footprint and follow up targeting of areas of ground that are of interest. Subsequent to the ground investigation monitoring of the instruments installed will be required.
- For the purpose selecting design criteria to be adopted, the preliminary consequence classifications indicated that the TSF is classified as Very High, under both GISTM and CAD guidelines. It is recommended a quantitative dam break assessment to confirm or revise the preliminary consequence classification presented in this study. The DBA should be undertaken in line with current best practice guidelines presented in CDA's Technical Bulletin (CDA, 2021).
- A design for the TSF has been developed. This design will need to be updated on the basis of the findings of the studies recommended.

26.1.5 FURTHER STUDIES

It is recommended that Ascendant proceeds with detailed engineering, which will serve as the foundation for procurement, construction, and commissioning activities. The scope of work will encompass the following:

- Complete engineering tasks to achieve a 40-45% level of completion, in accordance with the AACE Class 2 estimate requirements.
- Prepare Request for Proposal (RFP) packages for construction quotations.

- Clearly define long-lead procurement items and initiate the procurement process for critical path items.
- Create a Level 4 Implementation Schedule for the project.

It is important to note that the feasibility study does not include expenses related to pre-project approval, such as those associated with the previously announced pre-development program (surface preparation and land acquisition), as well as early engineering costs.

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