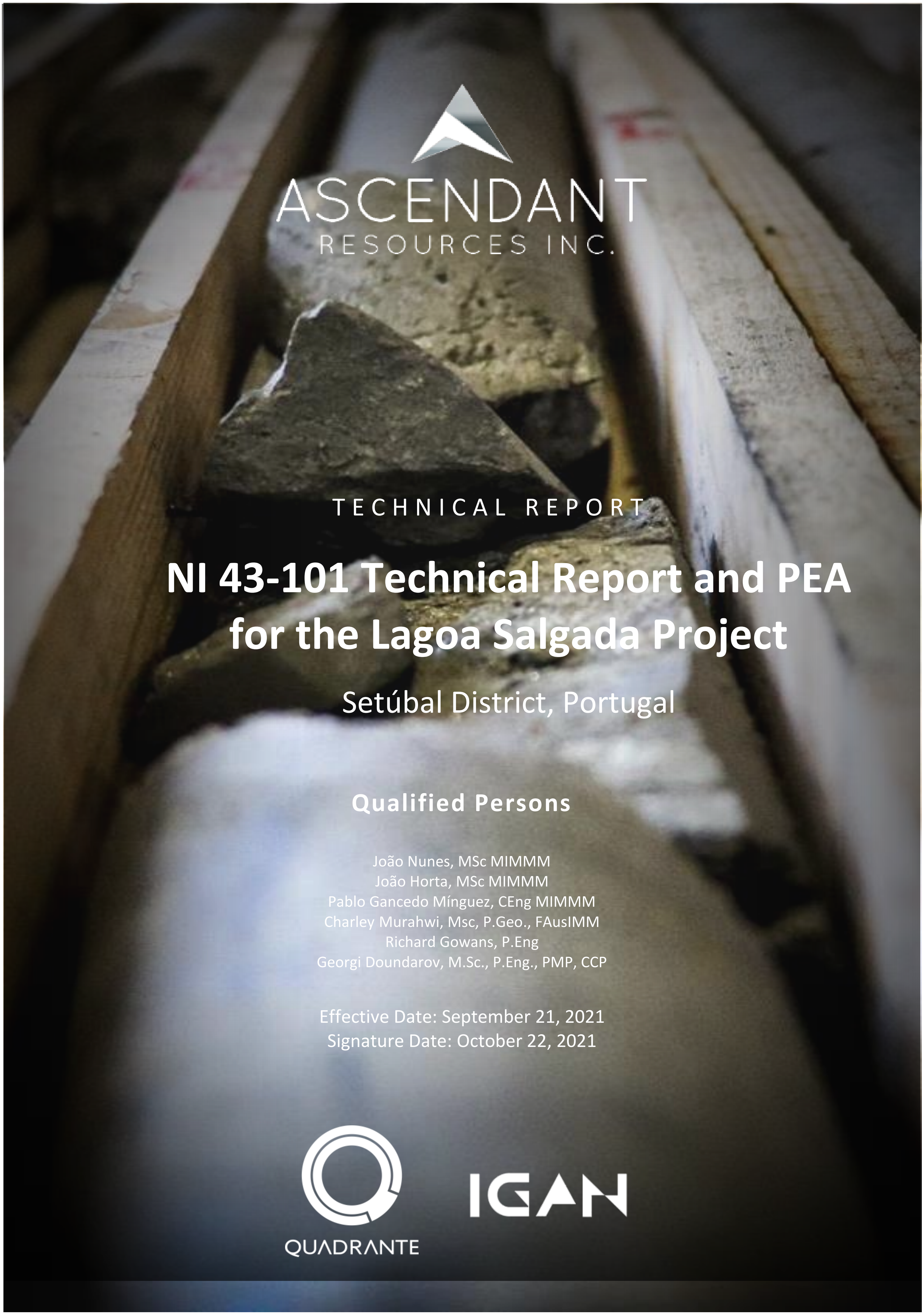

ASCENDANT RESOURCES INC.

TECHNICAL REPORT

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

Setúbal District, Portugal

Qualified Persons

João Nunes, MSc MIMMM
João Horta, MSc MIMMM
Pablo Gancedo Mínguez, CEng MIMMM
Charley Murahwi, Msc, P.Geo., FAusIMM
Richard Gowans, P.Eng
Georgi Doundarov, M.Sc., P.Eng., PMP, CCP

Effective Date: September 21, 2021
Signature Date: October 22, 2021

QUADRANTE

IGAN

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Abbreviations & acronyms Description	Abbreviations & acronyms Description
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
Metre	m
Metre per month	m/month
Metre squared	sqm ; m^2
Cubic metre	m^3
Cubic metre per hour	m^3/hr
Cubic metre per second	m^3/s
Metre above sea level	Masl
Maximum	Max
Methyl isobutyl carbinol	MIBC
Micon International Limited	Micon
Minute; minimum	Min
Mineral Liberation Analysis	MLA
Millilitre	ml
Millimetre	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
Centimetre	cm
Coefficient of variation	Coeff. V / CoV
Certified reference material	CRM
Cleaner	Cl
Concentrate	Conc
Copper	Cu
Copper equivalent	CuEq
Copper sulphate	$CuSO_4$
80% particle size distribution	D_{80}
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

Abbreviations & acronyms Description	Abbreviations & acronyms Description
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 Perfurção 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	ID^2
Inverse distance cubed	ID^3
Intelligent Exploration	IE
Licença ambiental – Environmental license	LA
Million cubic metres	Mm^3
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 Hexameta phosphate	Na6((PO3)6)
National Instrument 43-101	NI 43-101
Nearest neighbour	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	P_{80}
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/m^3
Kilogram per tonne	kg/t
Kilolitre	kL
Kilometre	km
Thousand ounces	koz
Thousand tonnes	kt
Kilovolts	kV
Kilovolt-Ampere	kVA
Kilowatts	kW
Kilowatt-hour	kW-hr
Litre	L
Litres per second	L/s
Pound	lb
Locked-cycle test	LCT
Load-haul-dump	LHD

Abbreviations & acronyms Description	Abbreviations & acronyms Description
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 metre	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	$ZnSO_4$

CERTIFICATE OF QUALIFIED PERSON

I, João Luís Mateus Nunes, 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 (# 44283) and am a Professional Member (MIMMM) of the Institute of Materials, Minerals and Mining (# 679048). I graduated with a M.S. degree from the Lisbon University – Higher Technical Institute in 1997. I have practiced my profession for 24 years.

I have worked in the mining industry in various positions continuously, including mining operations, general management, and mining studies, since my graduation from university. I have worked with various commodities and deposit types, including 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 November 23, 2020 and August 20, 2021.

I am the co-author of this report and responsible for sections 1.1, 1.2, 1.3, 1.5, 1.6, 1.8, 1.9, 1.10, 1.11.2, 1.11.3, 1.11.5, 1.12.2, 1.12.4, 1.12.5, 2, 3, 4, 5, 6, 15, 16.5, 18, 19, 20, 21, 22, 23, 24, 25.1, 25.3, 25.5, 25.6, 26.1, 26.4, 26.5, 26.6, 26.7, 26.8 and 27.

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: October 22, 2021

Signature: "Original signed"

Name: João Luís Mateus Nunes, MIMMM

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 August 20, 2021.

I am the co-author of this report and responsible for sections 1.1, 1.2, 1.3, 1.5, 1.6, 1.8, 1.9, 1.10, 1.11.2, 1.11.3, 1.11.5, 1.12.2, 1.12.4, 1.12.5, 2, 3, 4, 5, 6, 15, 16.5, 18, 19, 20, 21, 22, 23, 24, 25.1, 25.3, 25.5, 25.6, 26.1, 26.4, 26.5, 26.6, 26.7, 26.8 and 27.

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: October 22, 2021

Signature: “Original signed”

Name: João Miguel Cardeal Martins Horta, MIMMM

CERTIFICATE OF QUALIFIED PERSON

I, Mr Pablo Gancedo Mínguez, am employed as a Senior Mining Engineer with IGAN Ingeniería S.L., with an address at Edificio CEEI Asturias, Parque Tecnológico de Asturias, Llanera, (Asturias), Spain.

This certificate applies to the technical report titled "NI 43-101 Technical Report and PEA for the Lagoa Salgada Project, Setúbal District, Portugal", with an effective date of September 10, 2021 (the "technical report"), prepared for Ascendant Resources Inc. (Ascendant).

I am a Chartered Engineer (CEng MIMMM) registered with the Institute of Minerals, Materials and Mining, license # 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 ten years since graduation, including eight 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 have not visited the Lagoa Salgada project area.

I am responsible or co-responsible for Sections 16.1, 16.2, 16.3, 16.4, 16.6, 16.7, 16.8, 16.9, 16.10 and 16.11 of the technical report.

I am independent of Ascendant 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 those sections of the technical report not misleading.

Signed and dated this 22nd day of October 2021 in Llanera, Asturias (Spain).

"original document signed and sealed"

Pablo Gancedo Mínguez CEng MIMMM MEng
Senior Mining Engineer
IGAN Ingeniería S.L.

CERTIFICATE OF QUALIFIED PERSON
Charley Murahwi, P.Geo.

As a co-author of this report for Mineral & Financial Investments Limited, Redcorp – Empreendimentos Mineiros, Lda and Ascendant Resources Inc. entitled "NI 43-101 Technical Report and PEA for the Lagoa Salgada Project, Setúbal District, Portugal" dated October 22, 2021, with effective date of September 10, 2021, I, Charley Murahwi, do hereby certify that:

I am employed as a Senior Economic Geologist by, and carried out this assignment for, Micon International Limited, Suite 900, 390 Bay Street, Toronto, Ontario M5H 2Y2, telephone 416 362 5135, e-mail: cmurahwi@micon-international.com.

I hold the following academic qualifications:

B.Sc. (Geology) University of Rhodesia, Zimbabwe, 1979;
Diplome d´Ingénieur Expert en Techniques Minières, Nancy, France, 1987;
M.Sc. (Economic Geology), Rhodes University, South Africa, 1996.

I am a registered Professional Geoscientist in Ontario (membership # 1618) and in PEGNL (membership # 05662), a registered Professional Natural Scientist with the South African Council for Natural Scientific Professions (membership # 400133/09) and am a Fellow of the Australasian Institute of Mining & Metallurgy (FAusIMM) (membership number 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 a number of 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 Project from 16 to 19 October 2018, from 13 to 17 November 2018 and from 28 to 31 May 2019.

This is my fourth Technical Report for the mineral properties that are the subject of this Technical Report.

As of the date of this certificate to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

I am independent of. Mineral & Financial Investments Limited, Redcorp – Empreendimentos Mineiros, Lda and Ascendant Resources Inc. and any subsidiaries according to the definition described in NI 43-101 and the Companion Policy 43-101 CP.

I have read NI 43-101 and the Sections of this Technical Report for which I am responsible have been prepared in compliance with this Instrument.

I am responsible for Sections 1.4, 1.11.1, 1.12 1, 6 to 12, 14, 25.2, and 26.2 of this Technical Report.

Effective Date: 10 September 2021
Signing Date: 22 October 2021

"Charley Murahwi" {signed and sealed}

Charley Murahwi, M.Sc., P. Geo. Pr.Sci.Nat., FAusIMM

CERTIFICATE OF QUALIFIED PERSON
Richard Gowans, P.Eng

As a co-author of this report for Mineral & Financial Investments Limited, Redcorp – Empreendimentos Mineiros, Lda and Ascendant Resources Inc. entitled “NI 43-101 Technical Report and PEA for the Lagoa Salgada Project, Setúbal District, Portugal” dated October 22, 2021, with effective date of September 10, 2021, I, Richard Gowans, do hereby certify that:

I am employed as a Principal Metallurgist by, and carried out this assignment for Micon International Limited, Suite 900, 390 Bay Street Toronto, Ontario, M5H 2Y2. tel. (416) 362-5135, e-mail: RGOWANS@MICON-INTERNATIONAL.COM

I hold the following academic qualifications:

B.Sc. (Hons) Minerals Engineering, The University of Birmingham, U.K., 1980

I am a registered Professional Engineer of Ontario (membership number 90529389); as well, I am a member in good standing of the Canadian Institute of Mining, Metallurgy and Petroleum.

I have worked as an extractive metallurgist in the minerals industry for over 35 years. This includes 7 years in operations with Impala Platinum, South Africa; 8 years engineering consulting with LTA Limited, South Africa; 3 Years engineering consulting with SNC Lavalin, Canada and about 20 years consulting with Micon International, my present employer. I have worked with a wide variety of commodities including gold, PGEs, base metals, speciality metals/minerals and industrial minerals. I have worked in a wide range of technical areas as a manager and engineer including mineral processing, hydrometallurgy, pyrometallurgy, logistics and infrastructure design and review, and capital and operating cost estimation.

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 the management of technical studies and design of numerous metallurgical testwork programs and metallurgical processing plants.

I have not visited the Lagoa Salgada Project.

I am responsible for the preparation of Sections 1.1.7, 1.11.4, 1.12.3, 13, 25.4 and 26.3 of this Technical Report.

I am independent of Mineral & Financial Investments Limited, Redcorp – Empreendimentos Mineiros, Lda and Ascendant Resources Inc. and any subsidiaries according to the definition described in NI 43-101 and the Companion Policy 43-101 CP.

This is my second Technical Report for the mineral properties that are the subject of this Technical Report.

I have read NI 43-101 and the Sections of this report for which I am responsible have been prepared in compliance with the instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Effective Date: 10 September 2021
Signing Date: 22 October 2021

“Richard Gowans” {signed and sealed}

Richard Gowans, B.Sc., P.Eng.

CERTIFICATE OF QUALIFIED PERSON
Georgi Doundarov, M.Sc., P.Eng., PMP, CCP

As a co-author of this report for Mineral & Financial Investments Limited, Redcorp – Empreendimentos Mineiros, Lda and Ascendant Resources Inc. entitled "NI 43-101 Technical Report and PEA for the Lagoa Salgada Project, Setúbal District, Portugal" dated October 22, 2021, with effective date of September 10, 2021, I, Georgi Doundarov, do hereby certify that:

I am an associate and independent consultant and carried out this assignment for, Micon International Limited, Suite 900, 390 Bay Street, Toronto, Ontario M5H 2Y2, telephone 647 267 2241, e-mail: GEORGI@MAGEMI.COM.

I hold the following academic qualifications:

B.Sc. (Mineral Processing and Metallurgy) University of Mining and Geology, Bulgaria, 1995;
M.Sc. (Mineral Processing) University of Mining and Geology, Bulgaria, 1996;
M.Sc. (Mineral Processing and Metallurgy), Yokohama National University, Japan, 2005.

I am a registered Professional Engineer in Ontario (Licence Number 100107167). I am also a Project Management Professional (PMP) (Licence Number 1218345) under the Project Management Institute (PMI) and a Certified Cost Professional (CCP) (Licence Number 42319) under the Association for Advancement of Cost Engineering International (AACEI). I am a Member of the Canadian Institute of Mining, Metallurgy and Petroleum (Member Number 141909);

I have worked as a metallurgical engineer in the minerals industry for over 30 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 a number of countries in Canada, USA, Africa, and Europe.

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 over 25 years in precious and base metals, 20 years in iron ore and steel metallurgy and over 10 years in industrial minerals.

I have not visited the Lagoa Salgada Project.

Until now I have not been involved in the mineral properties that are the subject of this Technical Report.

As of the date of this certificate to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

I am independent of Mineral & Financial Investments Limited, Redcorp – Empreendimentos Mineiros, Lda and Ascendant Resources Inc. and any subsidiaries according to the definition described in NI 43-101 and the Companion Policy 43-101 CP.

I have read NI 43-101 and the Sections of this Technical Report for which I am responsible have been prepared in compliance with this Instrument.

I am responsible for Sections 1.7.2., 17, portions of 25.4., and portions 26.3 of this Technical Report.

Effective Date: 10 September 2021
Signing Date: 22 October 2021

"Georgi Doundarov" {signed and sealed}

Georgi Doundarov, M.Sc., P.Eng., PMP, CCP

1 SUMMARY

1.1 TERMS OF REFERENCE

Quadrante and IGAN have been retained by Ascendant Resources Inc. (Ascendant) to complete a Preliminary Economic Assessment (PEA) for the Lagoa Salgada (LS) Project in the Setúbal District of Portugal, and to prepare an independent Technical Report in accordance with the requirements of Canadian National Instrument 43-101 (NI 43-101). The purpose of this Report is to support the public disclosure of a PEA that is based on the Mineral Resource estimate dated 5 September 2019 (for the North Deposit) and dated 10 June 2021 (for the South Deposit) prepared by Micon International Limited (Micon).

The December 2019 PEA completed by AMC Mining Consultants (Canada) Ltd. (AMC) focused on the North deposit only and is superseded by this PEA

1.2 PROPERTY DESCRIPTION AND LOCATION

The LS project is located approximately 80 kilometers (km) south-east of Lisbon, Portugal's capital 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 and approximately three km north of the village of Cilha do Pascoal.

1.3 OWNERSHIP

The LS Project is covered by a single exploration permit with an area of approximately 10,700 hectares (ha). The exploration permit, Contrato MN/PP/009/08, is held by a joint venture between Redcorp Empreendimentos Mineiros, LDA. (Redcorp) and Empresa de Desenvolvimento Mineiro S.A. (EDM) a Portuguese Government owned company for the mining sector. Redcorp holds an 85% interest and EDM holds a 15% interest. A renewal of the exploration permit was granted by Portuguese General Directorate for Energy and Geology (Direção Geral de Energia e Geologia - DGEG), in 2017, and an application for a mining concession was made in April 2019, for which the terms are currently being negotiated. The exploration permit is registered in the Diário da República, Public Register, under Contrato (extrato) n° 377/2015.

Redcorp is a 75% subsidiary of TH Crestgate GmbH (TH Crestgate), a Swiss investment company and a 25% subsidiary of Ascendant, a Canadian company listed on the Toronto Stock Exchange. Under agreements made with TH Crestgate, Ascendant has the right to earn into 80% of the Lagoa Salgada Project.

1.4 GEOLOGY AND MINERALISATION

1.4.1 REGIONAL SETTING

The LS Project is located within the north-western portion of the Iberian Pyrite Belt (IPB). The IPB is one of the most prolific European metallic provinces, hosting one of the largest concentrations of massive sulphides (MS) in the Earth's crust; it contains more than 1,600 million metric tonnes (Mt) of MS mineralisation and about 250 Mt of stockwork mineralisation (Oliveira et al. 2005, 2006; Tornos 2006). The IPB hosts more than 90 MS deposits, 10 of wich are located in Portugal, where currently only Neves Corvo and Aljustrel are being exploited.

1.4.2 PROPERTY GEOLOGY AND MINERALISATION

The entire Property (exploration permit) is covered by a paleo-fluvial fan that ranges in thickness up to 200 metres (m) within the Tertiary Sado Basin and averages 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 Venda Nova North Deposit is comprised of gossan (GO) mineralisation resulting from weathering of the underlying primary massive sulphide (MS) mineralization which in turn is underlain and surrounded by copper-rich stringer / fissure / stockwork mineralisation.

The Venda Nova South Deposit comprises copper-rich stringer / fissure / stockwork mineralisation and isolated gold-rich silicified zones which appear to be structurally controlled. It is hosted in a unit locally known as the iVfr. The iVfr is heterogeneous, being composed of intricately interleaved dacitic-andesitic-basaltic volcanic rocks. The mineralisation has been remobilised into fissures/cracks arising from brittle failure during deformation, hence the name 'fissural mineralisation'. Although the entire iVfr unit is mineralised to some extent, potentially economic grade mineralization occurs in steeply dipping corridors; the steep dip is attributed to post mineralization east-west compressional deformation.

1.4.3 DEPOSIT TYPES

The LS Property's two deposits are interpreted as components of a single 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.

1.4.4 Status of Exploration

Due to the thick sedimentary cover, previous and current exploration programs have relied heavily on geophysical techniques, complemented by diamond drilling. Recent Induced Polarization (IP) investigations conducted by Intelligent Exploration (IE) of Campbellford, Ontario, Canada, have successfully demonstrated that mineralisation on the LS property remains open in all directions but with a stronger signature on the eastern side of the currently drilled / known linear trend of about 1.7 km.

Step-out drilling combined with geological/structural reinterpretation has culminated in the merging of the former Central Deposit with the southern mineralised envelope to form one continuous deposit, the South Deposit.

1.4.5 Mineral Resource Estimate

The mineral resources for the LS project (Venda Nova North and South Deposits) have been estimated using the Ordinary Kriging technique after completing a geostatistical study 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 ensure that the block grades honour the drill hole data.

The estimated mineral resources for the North Deposit are summarised in Table 1-1 while those for the South deposit are summarised in Table 1-2. All resource parameters are disclosed in Section 14 of this Report. The effective dates of the estimates are shown on the respective tables.

Table 1-1 - LS Property Mineral Resource estimate of the North Deposit as of 5 September 2019 at Cut-off grades Shown in Table

Deposit	Category	Min Zones	Cut-off ZnEq (%)	Tonnes (kt)	Cu (%)	Zn (%)	Pb (%)	Sn (%)	Ag (g/t)	Au (g/t)	ZnEq (%)	AuEq (g/t)	Cu (kt)	Zn (kt)	Pb (kt)	Sn (kt)	Ag (koz)	Au (koz)
North	Measured (M)	GO	2.5	234	0.13	0.70	4.32	0.36	51	1.50	11.38	7.18	0.3	1.6	10.1	0.9	385.2	11.3
	Indicated	GO	2.5	1,462	0.08	0.43	2.55	0.26	37	0.51	6.63	4.18	1.2	6.2	37.3	3.8	1,742.1	23.8
	M & I	**GO**	**2.5**	**1,696**	**0.09**	**0.47**	**2.79**	**0.27**	**39**	**0.64**	**7.28**	**4.60**	**1.5**	**7.9**	**47.4**	**4.6**	**2,127.2**	**35.1**
	Inferred	GO	2.5	831	0.08	0.48	2.62	0.17	27	0.37	5.66	3.57	0.7	4.0	21.8	1.4	727.6	9.9
	Measured	MS	3.0	2,444	0.40	3.12	2.97	0.15	72	0.74	10.95	6.91	9.7	76.3	72.5	3.7	5,623.9	58.4
	Indicated	MS	3.0	5,457	0.45	2.35	2.30	0.13	75	0.67	9.55	6.03	24.5	128.1	125.6	7.3	13,221.5	116.9
	M & I	**MS**	**3.0**	**7,902**	**0.43**	**2.59**	**2.51**	**0.14**	**74**	**0.69**	**9.98**	**6.30**	**34.2**	**204.4**	**198.1**	**10.9**	**18,845.5**	**175.2**
	Inferred	MS	3.0	1,529	0.23	1.96	1.32	0.09	45	0.49	6.36	4.01	3.6	30.0	20.2	1.4	2,219.7	24.0
	Measured	Str	2.5	94	0.37	0.88	0.28	0.05	17	0.12	3.08	1.94	0.3	0.8	0.3	0.0	51.0	0.4
	Indicated	Str	2.5	643	0.34	0.90	0.23	0.09	17	0.06	3.23	2.04	2.2	5.8	1.5	0.6	354.0	1.3
	M & I	**Str**	**2.5**	**737**	**0.34**	**0.90**	**0.24**	**0.09**	**17**	**0.07**	**3.21**	**2.03**	**2.5**	**6.6**	**1.7**	**0.6**	**405.0**	**1.7**
	Inferred	Str	2.5	142	0.24	1.12	0.39	0.04	17	0.09	2.95	1.86	0.3	1.6	0.6	0.1	75.6	0.4
North	**M & I**	**All zones**	**2.9**	**10,334**	**0.37**	**2.12**	**2.39**	**0.16**	**64**	**0.64**	**9.06**	**5.72**	**38.2**	**219.0**	**247.2**	**16.2**	**21,377.7**	**212.0**
North	Inferred	All zones	2.8	2,502	0.18	1.42	1.70	0.12	38	0.43	5.93	3.74	4.6	35.6	42.6	2.9	3,022.8	34.3

Notes:

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).
3. Mineralized Zones: GO=Gossan, MS=Massive, Str=Stringer, Str/Fr=Stockwork.
4. ZnEq% = ((Zn Grade*25.35) + (Pb Grade*23.15) + (Cu Grade * 67.24) + (Au Grade*40.19) + (Ag Grade*0.62) + (Sn Grade*191.75)) / 25.35.
5. AuEq g/t = ((Zn Grade*25.35) + (Pb Grade*23.15) + (Cu Grade * 67.24) + (Au Grade*40.19) + (Ag Grade*0.62) + (Sn Grade*191.75)) / 40.19.
6. CuEq% = ((Cu Grade * 67.24) + (Zn Grade*25.35) + (Pb Grade*23.15) + (Au Grade*40.19) + (Ag Grade*0.62)) / 67.24
7. Metal Prices: Cu $6,724/t, Zn $2,535/t, Pb $2,315/t, Au $1,250/oz, Ag $19.40/oz, Sn $19,175/t.
8. Densities: GO=3.12, MS=4.76, Str=2.88.

Table 1-2 - LS Property South Deposit Resources as of June 14, 2021 at 1.10% CuEq Cut-off Grade

		Average Grade						Contained Metal					
Category	**Ton**	**CuEq**	**Cu**	**Zn**	**Pb**	**Ag**	**Au**	**CuEq**	**Cu**	**Zn**	**Pb**	**Ag**	**Au**
	kt	%	%	%	%	g/t	g/t	kt	kt	kt	kt	k oz	k oz
Indicated	4,044	1.50	0.42	1.55	0.87	17.64	0.06	60.54	16.94	62.53	35.13	2,294	7
Inferred	10,827	1.35	0.31	0.79	0.43	14.56	0.76	145.6	33.12	86.03	46.67	5,068	266

Notes:

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).
3. CuEq % = ((Cu Grade * 67.24) + (Zn Grade*25.35) + (Pb Grade*23.15) + (Au Grade*40.19) + (Ag Grade*0.62)) / 67.24
4. Metal Prices: Cu $6,724/t, Zn $2,535/t, Pb $2,315/t, Au $1,250/oz, Ag $19.40/oz, Sn $19,175/t.
5. Density = 3.

1.4.6 Mineral Resource Comparison

Currently, the contribution of the two deposits to the LS Property mineral resources is split as follows: North Deposit = 48%, and South Deposit = 52%. However, both deposits have the potential to delineate more resources with additional drilling. The stringer/fissure type mineralisation of the South Deposit appears to be more amenable to metallurgical processing than the massive sulphide mineralisation of the North Deposit, and future priority drilling will depend on progress in metallurgical testwork.

1.5 Mining

The mining operations at Lagoa Salgada will be conducted using underground methods. These methods will allow for three independent mining sequences – one in the North Zone and two in the South orebody. The two independent sequences in the South Zone will be achieved by creating a sill pillar at level -255.

The applicability of various mining methods has been studied for the different rock types within the mineralised material – oxides, massive sulphides and stockwork – each of which has distinct geomechanical characteristics. Two main mining methods have been considered – cut&fill for the oxides zone and transverse sublevel stoping for the massive sulphide and stockwork zones. Both methods assume a bottom-to-top mining sequence with pastefill for filling the voids.

The Deswik software has been used for the mine infrastructure design, the stope design and the mine scheduling. The proposed mining methods support the extraction of 2.0 million tonnes of ore per year as shown in Table 1-3.

Table 1-3 – Annual plant feed and average NSR values

Year	Ore Production (kt)	NSR (USD/t)
Year -2	-	-
Year -1	280	81
Year 1	1 800	87
Year 2	2 000	88
Year 3	2 000	75
Year 4	2 000	80
Year 5	2 000	80
Year 6	2 000	61
Year 7	2 000	56
Year 8	2 000	51
Year 9	2 000	53
Year 10	2 000	48
Year 11	2 000	54
Year 12	1 900	51
Year 13	1 500	51
Year 14	590	51

The stopes in the sublevel stoping areas will typically be 15 metres wide by 20 metres high. Footwall accesses will be driven every 25 metres in height. In cut&fill areas, the stopes will be 5 metres wide by 5 metres high. A dilution of 10% and a mining recovery of 95% are planned for all mining methods.

Ore will be transported to a surface ROM pad using underground mine trucks. The infrastructure design allows for the usage of trucks with up to 65 tonnes.

The mine will be equipped with a ventilation system that can move approximately 600 cubic metres of fresh air through the main ramp and the access declines per second. The fresh air will move down the ventilation raises to feed the production levels. At the end of each level, a regulated connection to the ventilation raise will provide for exhaustion of the air, with exhaust fans installed at the surface. This system will be similar for both the North and South Zones.

After Year 2, around 170 people are expected to work in the mining activities.

1.6 INFRASTRUCTURE

The Lagoa Salgada Project infrastructure will be developed on a greenfield site. This site has existing access roads off a public road that will likely require upgrading. The Project will include the support infrastructure for the mine and the processing plant, the tailings storage facility (TSF), internal roads, installations for the supply and distribution of power and a water treatment plant. In addition, a processing plant for the treatment of the different ores and the production of the final products will also be built. A backfill plant will be developed to produce pastefill for filling the voids created by the mining activities.

Ore and temporary waste stockpiles will also be developed on the surface, close to the mine portal. These stockpiles will be used for ore and waste storage before processing or permanent storage

1.7 Mineral Processing

1.7.1 Metallurgical Testing

Several scoping level metallurgical and mineralogical studies have been undertaken for the LS Project by Grinding Solutions (GSL), a metallurgical testing laboratory located in Cornwall, UK.

The aim of the studies undertaken by GSL in 2021 was to confirm results previously achieved on samples with head grades more representative of potential ROM material.

The -2 mm crushed samples selected by Ascendant Resources for testing had been stored without refrigeration for approximately two years and, there was concern that they may have oxidised which would have a significant detrimental effect on the metallurgy. As such, preliminary testing was conducted on samples from the two metallurgical composites, massive sulphide (MS) and stockwork (SW), to assess the extent of the oxidation.

The main testing that followed the preliminary assessment of the samples was split into 6 phases

1. Feed characterisation and sample preparation.
    - Head assay.
    - Comminution (rod mill grind curve and SMD Signature plot) to target primary grind sizes.
2. Gravity separation trials on Massive Sulphide (MS) sample.
3. Flotation evaluation on both MS and Stockwork (SW) samples.
    - Scoping trials.
    - Cleaner float tests.
    - Locked Cycle Testing.
4. Leach trials on a bulk pyrite concentrate on the MS.
5. Evaluation of potential tin (Sn) extraction post pyrite concentrate.
6. Flotation testing to evaluate the potential for processing a blend of MS and SW.

The preliminary oxidation assessment suggested that the samples were oxidized and not truly representative of fresh mineralization that would feed a mineral processing facility.

Therefore, the flotation recoveries and concentrate grades produced from the 2021 GSL programme were not optimal. Therefore, the flotation recoveries and concentrate grades produced from the 2021 GSL programme were not optimal.

Gravity separation tests on the MS sample gave no selectivity, recoveries were proportional to mass pull.

Intensive cyanide leach tests on flotation pyrite concentrates gave metal extractions after 48 hours of between 9% and 25% for gold, between 40% and 65% for silver, between 62% to 82% for copper, and between 13% and 24% for zinc.

The flotation locked cycle test (LCT) completed using the MS composite sample resulted in Pb concentrate grade of 22% with 43% recovery and a Zn concentrate grade of 35% at 66% recovery. Separate Cu and Pb concentrates were not recovered for the MS tests.

The LCT completed for the SW sample resulted in Cu concentrate grade of 25% at 69% recovery, Pb concentrate grade of 28% at 16% recovery and Zn grade of 44% at 54% recovery

A LCT using a blend of MS and SW resulted in Cu concentrate grade of 24% at 53% recovery, Pb concentrate grade 12% at 5% recovery Zn concentrate grade 28% at 64% recovery.

1.7.2 RECOVERY METHODS

The process design criteria, flowsheet with major process equipment and a description of the process facilities are developed based on the existing metallurgical testwork as described in detail in Chapter 13 and in benchmark and author experience previously at Aljustrel and Neves-Corvo operations. Both operations are located in the Iberian Pyrite Belt as the Lagoa Salgada Project.

The plant is designed for a total of 2Mtpa or 250tpa throughput to produce two concentrates. Based on this, the plant LOM is expected to be 14 years. The crushing plant is designed at 65% availability, while the main plant will operate at 92% availability. Based on head grades of 0.31% for Cu, 1.44% for Zn, and 1.22% for Pb the designed commodity recoveries are 80% for the Cu SW and 25% for the Cu MS, 80% for the Zn, and 75% for the Pb respectively.

It is to be noted that due to fluctuations of the head grades of the different minerals from the initial years of operation through the LOM, the plant has been designed for the range of head grades of each mineral, rather than on the average grades presented in Table 17-1. Same logic has been used as part of the equipment selection and sizing

The conceptual methodology process flowsheet envisioned for this deposit consists of:

- Primary, Secondary, and Tertiary crushing.
- Coarse material stockpiling.
- Primary ball mill grinding.

- Secondary ball mill grinding and cyclone classification.
- Lead Flotation.
- Zinc Flotation.
- Concentrates Thickening.
- Concentrates Filtration.
- Final Concentrates Stockpiling.
- Tailings Disposal.

The process plant equipment consists of:

- Grinding and regrind equipment including ball and SMD mills and cyclones
- Pb and Zn flotation conditioning tanks and cells
- Concentrates thickeners and pressure filters.

Process plant labour includes plant management, technical staff, operations, and maintenance personnel, and administrative staff. The overall personnel peaks at 73 persons in the first two years of operation.

1.8 ENVIRONMENTAL

The PEA approach consisted of the following environmental analysis:

- Environmental permitting of the Lagoa Salgada Project.
- Environmental approach and relevant potential areas of impact.
- Community engagement and the status of local agreements.
- Mine closure.

The Lagoa Salgada Project has been developed within the framework of the Portuguese legal regime for environmental impact assessments (RJAIA), and the following stages of environmental permitting have been identified:

- PDA – The scope definition proposal (Proposta de Definição de Âmbito – PDA). This document defines the proposed scope of the environmental impact study, establishes the importance of each factor for the environmental assessment of the Project and sets out the extent to which each environmental component will be developed.
- EIA – The environmental impact study (Estudo de Impacte Ambiental – EIA) will assess the early stages of the Project (such as the pre-feasibility study, the feasibility study and the base project); the environmental statement (Declaração de Impacte Ambiental – DIA) issued by the Portuguese Environmental Authority (APA) after the EIA analysis approves the location of the Project and its main characteristics and establishes some constraints that need to be considered in the project to be executed.
- RECAPE – The environmental conformity of the project to be executed (Relatório de Conformidade Ambiental do Projeto de Execução – RECAPE) will provide an environmental assessment of the Project at the detailed design stage to verify its conformity with the DIA measures; after the RECAPE analysis, APA will issue the environmental conformity declaration (Declaração de

Conformidade do Projeto de Execução – DCAPE), which will allow for the implementation of Project at the location evaluated and approved in the EIA; the DCAPE may lay down some constraints during the construction and/or operation phases as well as a monitoring plan.

- LA – The environmental license (Licença Ambiental – LA) that is required for projects in the framework of the regime of integrated prevention and pollution control of industrial emissions (Regime de Emissões Industriais – REI); as part of the REI application, the applicant must apply for a single environmental title (Título Único Ambiental – TUA) for the project on the APA's electronic platform. Together with the result of the LA approval, this TUA will allow for the operation of the mine.

To produce these environmental documents, some technical information about the Project is required, including the following documents:

- Mine Plan – This plan should include all the activities that will be executed within the concession area and across all mining annexes ("anexos mineiros"), whether they are located inside or outside of the concession area. These must be compliant with the best available techniques (BAT), ensure adequate operational economy and respect all the applicable safety and environmental protection measures
- Waste management plan – This plan should be developed in accordance with the rules laid down in the Legal Regime for Mining Waste Management.
- Health and safety plan – This plan should be developed in accordance with the Portuguese legislation in force.
- Environmental and landscape recovery plan – This is a dynamic plan that will follow the evolution of the operational activities and must be reviewed every five years. Failure to comply with the plan constitutes a serious environmental infraction.
- Mine closure plan – This plan establishes the technical measures regarding the closure of the mine and the mitigation measures to reduce the social, economic and environmental impacts of the end of operation on the mining site and its surroundings, and it must be reviewed every five years. Failure to comply with the plan constitutes a serious environmental infraction.

The potential areas of impact of the Lagoa Salgada Project and sustainable measures for mitigating them have also been addressed in the PEA.

1.9 CAPITAL AND OPERATING COSTS

The capital costs of the Lagoa Salgada Project are divided into three categories. The initial CAPEX – the investment of capital needed to build the project infrastructure until commercial production of concentrates can commence; the sustaining CAPEX – the investment of capital needed to support the projected schedule throughout the LOM; and the closure costs – the estimated capital required for closing the operations as planned.

The estimates indicate that the Project requires an initial capital of US$110 million and a sustaining capital of US$103 million to support the projected infrastructure and production schedule. The cost for the closure activities is estimated at US$6 million.

A contingency of US$12 million is foreseen for the Lagoa Salgada Project.

The capital costs are summarised in Table 1-4.

Table 1-4 – Breakdown of capital costs

Capex			
Initial Capex	kUS$	$	109 570
Sustaining Capex	kUS$	$	102 583
Closure Costs	kUS$	$	6 000
Contingency	kUS$	$	12 028
Total Capex	**kUS$**	**$**	**230 180**

Quadrante and Ascendant have prepared an operating cost estimate for the production schedule outlined in the PEA. These costs have been subdivided into the following categories:

- Mining Operating Expenditure.
- Processing Operating Expenditure.
- Site G&A Operating Expenditure.

The cost estimate is presented in Table 1-5.

Table 1-5 – Breakdown of LOM operating costs

Opex			
Mining Costs	kUS$	$	498 805
Processing Costs	kUS$	$	414 286
G&A Costs	kUS$	$	91 245
Total Opex	**kUS$**	**$**	**1 004 336**

1.10 ECONOMIC ANALYSIS

The valuation results of the Lagoa Salgada Project indicate that it has a post-tax net present value (NPV) of approximately US$247 million, based on an 8% discount rate. The operation is expected to have negative cashflows during the two years of construction (Years -2 and -1), with payback expected by Year 2. The annual cashflow profile is presented in Figure 1-1.

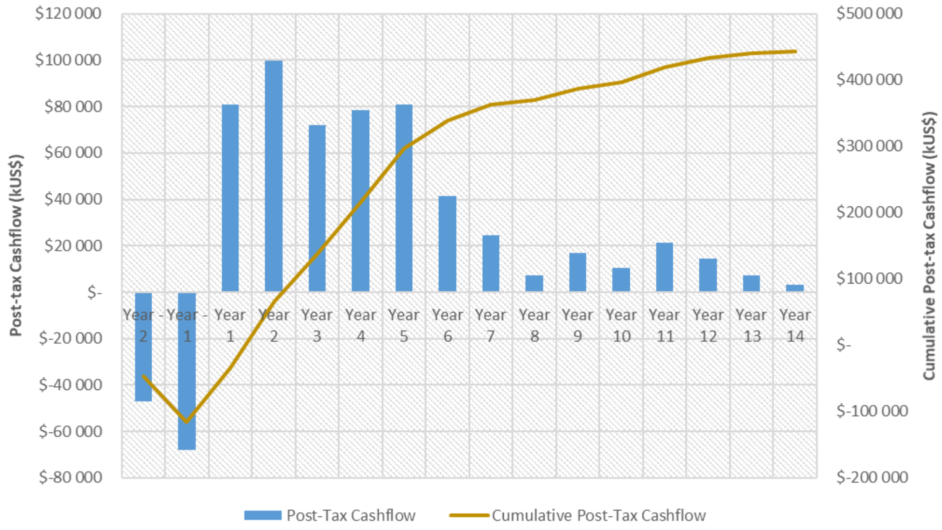


Figure 1-1 – Post-tax cashflow for the Lagoa Salgada Project

The Lagoa Salgada Project will be a polymetallic operation where revenues will be generated by six different metals. Although zinc will provide the highest percentage of revenues (29% of the total), lead, silver, copper and gold will also contribute significantly. Tin will be the least significant metal, contributing about 5% of the total revenue. Variations in metal prices may significantly change this revenue distribution. The projected distribution of revenues is presented in Figure 1-4.

	Unit		TOTAL
Net Smelter Return			
Revenue	kUS$	$	1 958 335
Commercialization Costs	kUS$	$	-
Concentrate Freight Costs	kUS$	$	78 972
NSR	**kUS$**	**$**	**1 879 363**
Royalties			
Royalties	kUS$	$	46 984
Opex			
Mining Costs	kUS$	$	498 805
Processing Costs	kUS$	$	414 286
G&A Costs	kUS$	$	91 245
Total Opex	**kUS$**	**$**	**1 004 336**
EBITDA	**kUS$**	**$**	**828 043**
Capex			
Initial Capex	kUS$	$	109 570
Sustaining Capex	kUS$	$	102 583
Closure Costs	kUS$	$	6 000
Contingency	kUS$	$	12 028
Total Capex	**kUS$**	**$**	**230 180**

Figure 1-2 – Economic indicators of the Project

Pre-Tax NPV @ 8%	**kUS$**	**$**	**341 578**
Pre-Tax IRR			**68%**
Pre-Tax Payback Period	**Years**		**1,3**
Post-Tax NPV @ 8%	**kUS$**	**$**	**246 702**
Post-Tax IRR			**55%**
Post-Tax Payback Period	**Years**		**1,5**

Figure 1-3 – Main economic indicators of the Project

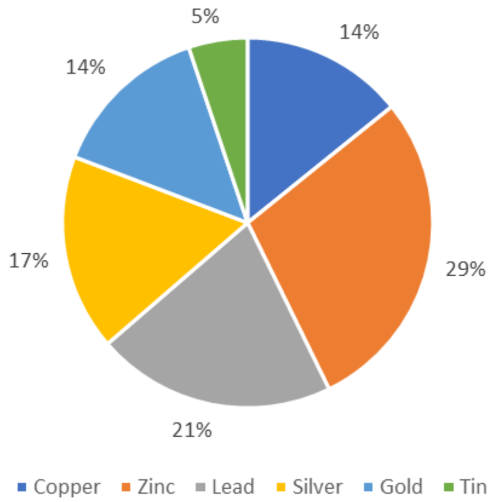


Figure 1-4 – Distribution of revenues per metal

1.11 INTERPRETATION AND CONCLUSIONS

1.11.1 GEOLOGY/RESOURCES

The subdivision of the LS property into the North and South deposits is arbitrary, being based on the existing drill pattern. With further concerted systematic drilling, the two deposits are likely to coalesce into a single zinc-lead-copper VMS system, manifesting / displaying its macro-genetic features from secondary GO to primary MS and ending with peripheral primary / secondary stringer / fissure type mineralization in the waning phases of volcanic activity. This interpretation is backed by geophysics which shows that all two deposits lie on a continuous coincidental IP chargeability anomaly with an estimated geological strike length of 1.7 km in an SSE to NNW direction from the South deposit to beyond the North deposit and terminating against the Alpine fault. Thus, the LS property's two deposits are components of one polymetallic, volcanogenic massive sulphide (VMS) deposit.

Potential to expand the resource is realistic as the LS property mineralization remains open in all directions although with a stronger signature on the eastern side of the currently drilled / known linear trend of about 1.7 km. The geometry of the MS domain of the North deposit appears to suggest that the main vent of the volcanic activity that gave rise to the LS property deposit may be located at the north-western end where the plunge swings westwards. However, this remains speculative until proven by additional drilling.

1.11.2 MINING

Following the completion of geotechnical preliminary studies and mine design and scheduling, conclusions can be made regarding the positive indications that the orebody geometry and geomechanical characteristics support a sustainable mining plan of 2.0

Mtpa. Mining recoveries of 95% can be achieved with the usage of backfill to fill the voids generated by mining activities.

1.11.3 Infrastructure

The site can be easily accessed, and no major restrictions for the construction of support infrastructure have been identified.

1.11.4 Processing

The two composite samples selected by Ascendant Resources originated from the 2019/2020 mineral resources definition drilling of the South Zone that intersected the mineral resources and the average grades of these two composite samples were similar to the average mineral resource grades presented in Section 14 of this Technical Report.

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. Further 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

Based on a sample oxidation evaluation by GSL, the samples used for this program of testwork, which is the basis for this PEA, were not truly representative of fresh mineralization that would feed a mineral processing facility. 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. Grinding Solutions Ltd. is 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.

In addition, the oxidised nature of the sample does not fully represent the whole orebody, but some specific sections of the North and South Sectors, that was important to evaluate. Based on the results from the two phases of testwork undertaken at Grinding Solutions, Grinding Solutions are confident that comparative performance could be achieved on projects working on similar ore bodies in the IPB, thus supporting recovery and concentrate grade assumptions used in the PEA. Further and continuous testing on fresh representative samples from drilling will be required to confirm this.

The requirement for a very fine primary grind will result in relatively high comminution capital costs for multiple crushing and grinding stages and high operating costs for grinding power and grinding media.

Gravity separation tests on the MS sample gave no selectivity, recoveries were proportional to mass pull.

Intensive cyanide leach tests on flotation pyrite concentrates gave metal extractions after 48 hours of between 9% and 25% for gold, between 40% and 65% for silver, between 62% to 82% for copper and between 13% and 24% for zinc.

The process flowsheet used for the locked cycle tests comprised the following:

- Primary grind to target P_{80} sizes of29 µm for the MS sample and 37 µm for the SW sample.
- Rougher copper/lead flotation followed by zinc flotation.
- Regrinding of Cu/Pb and zinc rougher concentrates to target P_{80} size of 15 to 20 µm.
- Three stages of copper, lead and zinc flotation cleaning.
- Dewatering of separate Cu, Pb and Zn concentrates.
- Tailings dewatering and disposal.

The flotation locked cycle test (LCT) completed using the MS sample resulted in Pb concentrate grade of 22% with 43% recovery and a Zn concentrate grade of 35% at 66% recovery. Separate Cu and Pb concentrates were not recovered for the MS tests.

The LCT completed for the SW sample resulted in Cu concentrate grade of 25% at 69% recovery, Pb concentrate grade of 28% at 16% recovery and Zn grade of 44% at 54% recovery.

A LCT using a blend of MS and SW resulted in Cu concentrate grade of 24% at 53% recovery, Pb concentrate grade 12% at 5% recovery Zn concentrate grade 28% at 64% recovery.

1.11.5 Economics

The economic indicators for the Project are positive. The sensitivity analysis indicates that these economic indicators are robust.

1.12 Recommendations

1.12.1 Geology/Resources

A systematic program to upgrade and expand the resources is recommended as follows:

- The immediate priority should be drilling the gap separating the North and South deposits.
- The second priority should be infill drilling to upgrade the Inferred resources for the North and South deposits. This is necessary as Inferred resources cannot be included in advanced economic studies, i.e., prefeasibility/feasibility studies.
- The third priority should be drilling directed at the north-west end of the North deposit to define the geometry / extent of the plunge and at the same time increase the resource.
- Models of the deposits should continue to be refined / updated as more information becomes available.

- Geophysical investigations of the LS deposits should be sustained and continued to cover the area to the north of the North deposit, targeting the area immediately beyond the major east-west Alpine fault.

1.12.2 Mining

Quadrante recommends a review of the mining parameters for the Project to optimise the mining plan for the deposit. Additional work on the ventilation, geotechnical and hydrogeological aspects of the project is required for completion of a PFS/FS. This information will provide the basis for a PFS/FS-level mine design infrastructure.

Trade-off studies should be developed to optimise several aspects of the mine design, such as the hoisting/trucking systems, the ventilation design and the automation of mining equipment.

1.12.3 Processing

A review of the available metallurgical testwork results to date suggests that there is insufficient metallurgical data available to allow an accurate forecast of metallurgical performance for the deposit. The process design criteria for this PEA should include the comminution and flotation requirements developed in this testwork program although any additional recovery of gold and tin using gravity separation or leaching technologies cannot be justified at this time due to lack of any supporting data.

Metallurgical recoveries and concentrate grades assumed for the PEA should take into account the results presented in Section 13.0. Although the samples used were oxidized, these flotation test results, together with other historical work, suggest that the poly-metallic mineralization Lagoa Salgada is complex, and it will be challenging to produce high quality concentrate products with high recoveries. 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. Grinding Solutions Ltd. is 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.

Although detailed final concentrate characterization was not included in the most recent program of metallurgical testwork, historical results suggest that the concentrates produced may include deleterious elements (to be confirmed with additional test work). Depending on the additional test work results, penalties may need to be applied in future techno-economic studies.

A new program of testwork is recommended using fresh drill core samples that represent the different lithologies found within the mineral resources. This program should use the recent and historical testwork as a basis and include the following

- Detailed mineralogical characterization including particle liberation analyses and valuable metal deportment.
- Detailed multi-element analyses of the samples.

- Comminution tests to develop design criteria to support the crushing, primary coarse and fine grinding and concentrate regrinding circuits.
- Flotation tests to provide detailed design information and to quantify final concentrate recoveries and grades that can be used to support a preliminary technical study.
- Detailed characterization of flotation concentrates and preliminary concentrate marketing studies.
- Preliminary thickening and filtration tests to support the design of dewatering equipment included in the process flowsheet.
- Preliminary geochemical studies of tailings, waste rock and economic mineralization.

1.12.4 Economics

A PFS/FS-level estimate of the capital and operating costs will be undertaken in the next phase of work. In addition, a programme of trade-off studies should be completed in order to optimise the economic results of the project.

1.12.5 Additional Work

An exploration/development programme and budget should be prepared in order to:

- Conduct follow-up work to confirm the favourable geophysics results.
- Prepare an exploration programme.
- Undertake ground and drillhole IP surveys.
- Undertake diamond drilling (infill, step-out and metallurgical testwork drillholes).
- Undertake detailed metallurgical testwork.
- Complete geotechnical work to support the mine design.
- Implement a hydrogeological study to better define and predict groundwater inflow.
- Complete a site-based climate and water balance.
- On completion of the above, consider undertaking a pre-feasibility study (PFS) or feasibility study (FS).
- Continue the environmental permitting.

To complete the planned exploration/development work, QUADRANTE has proposed a budget of approximately €6.5 million, broken down as summarised in Table 1-6.

Table 1-6 - Proposed work program and budget for the LS Project

Program	Activity	Cost (€)
Drillhole Survey (North, Central & South Deposits)	Exploration Drilling (10,000 m) - Increase the M&I resource - 30 drillholes	1.500.000
Drillhole IP Survey (North, Central & South Deposits)	Interpretation / Modeling	50.000
Detailed Metallurgical Teswork	Optimizing Recoveries & Concentrate Grades	260.000
Backfill laboratory testwork	Define Paste Fill final recipe (requires tailing samples)	10.000
Tailings & Waste laboratory testwork - soil and tailings	Fied testwork and measurements, modelling	60.000
Drilling for Hydrogeological Study	Drilling representative of the lithological and structural sequence - 4 drillholes	150.000
Hydrogeological study	Fied testwork and measurements, modelling	150.000
Drilling for Geotechnical Tests	Directional drilling - 3 directional drillholes	270.000
Geotechnical testwork including Boxcut	Laboratory testwork with representative samples, modelling	15.000
Geotechnical work to support FS level design	Stope modelling empirical modelling, design criteria, ground support recommendations	25.000
Feasibility Study	Complete study	2.350.000
EIS	Environmental Impact Study ("Estudo Prévio)	300.000
PDA	Scope Definition Proposal	20.000
RECAPE	Checking of the conformity of the Execution Project with the environmental statement (DIA) that approved the EIS	350.000
Permit Design/Tender Design	Design for Permit and Tender	1.000.000
All Activities	**Grand Total**	**6.510.000**

2 INTRODUCTION

2.1 INTRODUCTION

Ascendant Resources Inc. (Ascendant) requested that Quadrante Engenharia (Quadrante) and IGAN Ingeniería, S.L. (IGAN) compile a technical report (the Report) on a preliminary economic assessment (PEA) study for the Lagoa Salgada Project (the Project) located in the Grândola municipality in southern Portugal.

This PEA is based on the Mineral Resource estimate dated September 5, 2019 and updated on June 14, 2021 by Micon International Limited (Micon).

2.2 TERMS OF REFERENCE

The Property is held in a joint venture between Redcorp Empreendimentos Mineiros, Lda. (Redcorp) and Empresa de Desenvolvimento Mineiro S.A. (EDM), a Portuguese Government owned company for the mining sector. Redcorp holds an 85% interest and EDM holds a 15% interest. Redcorp is a 75% subsidiary of TH Crestgate, a Swiss investment company, and a 25% subsidiary of Ascendant, a Canadian company listed on the Toronto Stock Exchange.

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 QUALIFIED PERSONS

This Report was prepared by the following Qualified Persons (QPs):

- Mr. João Nunes, M.Sc., MIMMM, Project Director, Quadrante.
- Mr. João Horta, M.Sc., MIMMM, Project Director, Quadrante.
- Mr. Pablo Gancedo, CEng MIMMM, Senior Mining Engineer, IGAN.
- Mr. Charley Murahwi, P.Geo. Pr.Sci.Nat. FAusIMM, Senior Geologist, Micon.
- Mr. R Gowans, PEng, Principal Metallurgist, Micon.
- Mr. Georgi Doundarov, M.Sc., P.Eng., PMP, CCP, Independent Associated Consultant, Micon.

2.4 SITE VISITS

Mr. João Nunes visited the site on two occasions, the first on 23rd November 2020, and the second on 20th August 2021. Mr. João Horta visited the site on the 20 August 2021. During these site visits, they reviewed the following aspects of the Project:

- Access routes to the area from Lisbon, Setúbal, and Sines, which are the main cities for concentrate export and the import of materials.

- Access conditions from nearby towns that could potentially provide services and labour to the mine.
- Inspected the current topography with the purpose of identifying potential sites for major surface infrastructure including mine portals, temporary rock storage facilities and tailings disposal facility.
- Inspected the drill core for general mineralisation characteristics and geotechnical information.
- Identified potential water sources.

Mr. Charley Murahwi visited the site from 16 to 19 October 2018, from 13 to 17 November 2018 and from 28 to 31 May 2019. During these visits, he reviewed the following aspects of the Project:

- Discussed the geological model.
- Verified some of the drillhole collar positions.
- Witnessed downhole survey measurements.
- Examined the drill cores.
- Reviewed the drillhole logs.
- Reviewed the mineralisation types.
- Reviewed/discussed the Quality Assurance/Quality Control (QA/QC) protocols / results of the on-going drilling programmes.

The main driving factor in updating the South Deposit resource is a re-interpretation of the structure/geology/mineralisation trends of the deposit in which Micon has played a continuous role. The new interpretation conforms to the genetic model and has been confirmed by six new diamond drill holes, as recommended by Micon at the time of its last site visit.

2.5 INFORMATION SOURCES AND REFERENCES

This Report is based in part on internal company reports, maps, published government reports, and public information, as listed in Chapter 27. It is also based on the information cited in Chapter 3.

2.6 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).

3 RELIANCE ON OTHER EXPERTS

The authors are not experts in legal matters, such as the assessment of the legal validity of mining claims, private lands, mineral rights, property agreements, and taxation and royalty aspects. The authors did not conduct any investigations of the environmental or social-economic issues associated with the Lagoa Salgada Project, and the authors are not experts with respect to these issues.

The QPs have relied, in respect of the aspects described above, upon the work of the issuer's Expert listed below. To the extent permitted under NI 43-101, the QPs disclaim responsibility for the relevant chapters of the Report:

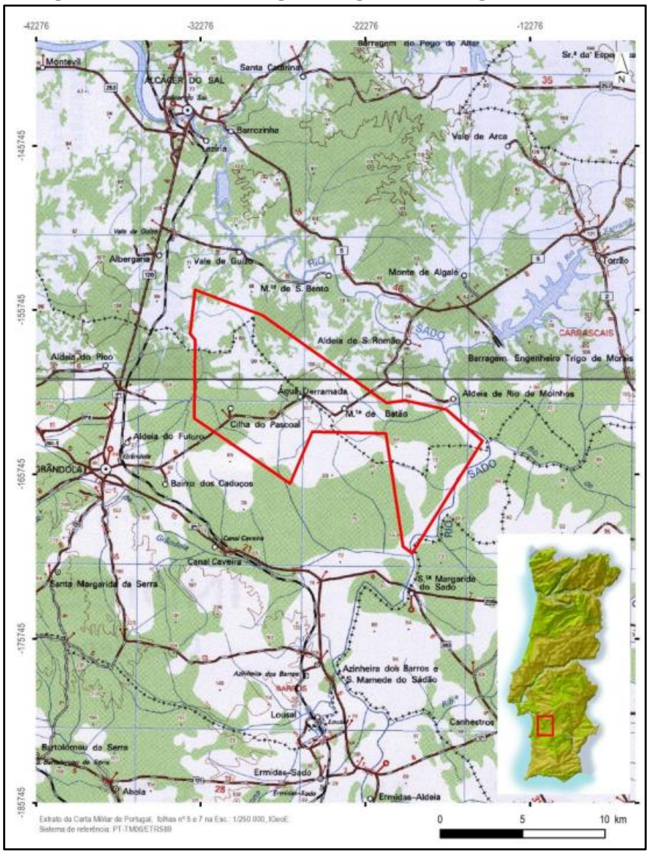
- Mr João Barros, President, Ascendant.
- Report, opinion or statement relied upon information on mineral tenure and status, title issues, and mining concessions.
- Report, opinion or statement relied upon information on environmental studies and permitting.
- Report, opinion or statement relied upon information on taxation and royalty aspects.
- Extent of reliance: full reliance following a review by the QPs.
- Portion of Technical Report to which disclaimer applies: Chapter 4, Chapter 20, and Chapter 22.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 DESCRIPTION AND LOCATION

The Project is located in Portugal, approximately 80 km south-east 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 and approximately 3 km north of the village of Cilha do Pascoal.

Figure 4-1 - Area of Lagoa Salgada mining concession



Source: Resumo Não Técnico do Plano de Lavra, September 2019

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 Prospecção e Pesquisa (exploration permit), which originally covered a total area of approximately 13,400 ha. However, when Redcorp renewed the permit in 2017, the exploration permit was reduced by 20% to 10,700 ha, in accordance with Portuguese law.

Exploration permits are granted for an initial period of three years. Upon completion of the first three years, the company may apply for a renewal of the permit for an additional period of two years and submit a reduction of the permit area of up to 20%. The exploration permit may be renewed for a maximum of two times. During this time, the company is obliged to carry out exploration activities that include drilling, geophysical and geochemical surveys.

The exploration permit, Contrato MN/PP/009/08, 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 exploration permit was granted by the Direção Geral de Energia e Geologia (the Portuguese General Directorate for Energy and Geology – DGEG). The exploration permit is registered in the Diário da República, Public Register, under Contrato (extrato) nº 377/2015.

The original exploration permit had an effective expiry date of 20 June 2017, but it was extended to 20 June 2019. Table 4-1 summarises the information regarding the original exploration permit along with information on the renewal. Figure 4-2 shows the outline of the reduced exploration permit after its renewal in 2017. Superimposed on the exploration permit is the mining concession, which is discussed below. Note that this permit also encompasses the Rio de Moinhos Project, 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 Exploration Permits

Name	Exploration Permit	Expiry Date	Area (ha)
Lagoa Salgada	Contrato MN/PP/009/08	20 June 2017	13,333.9
Lagoa Salgada	Contrato MN/PP/009/08	20 June 2019	10,700.0

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 7,500 ha (Figure 4-2). 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).

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

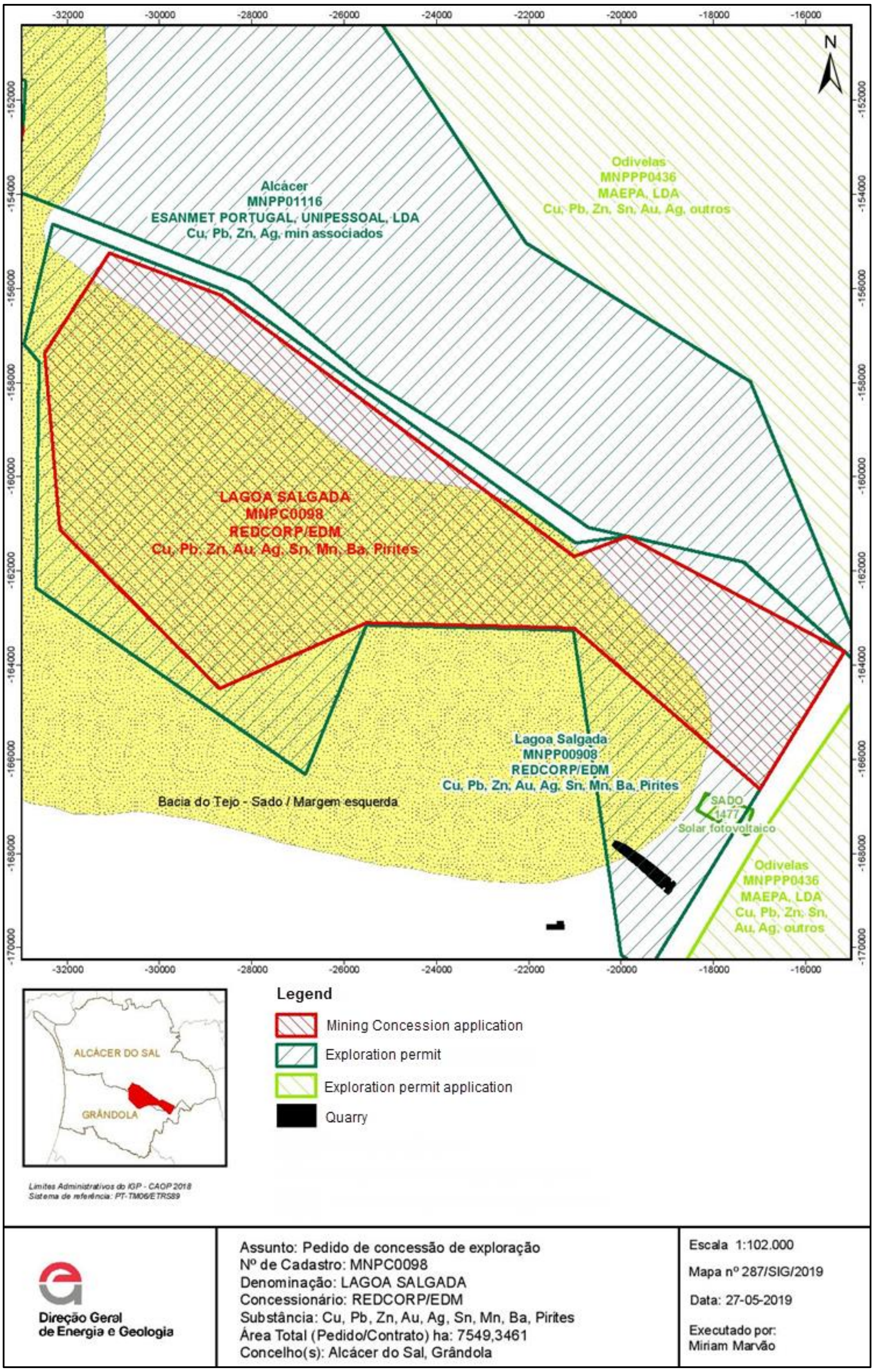
4.3 Property Ownership and Agreements

In July 2015, TH Crestgate GmbH (TH Crestgate) acquired a 100% stake in Redcorp. Redcorp and EDM hold respectively 85% and 15% interests in the exploration permit for the LS Property and Redcorp remains the operator of the Project.

On 1 August 2018, Ascendant announced in a press release that it acquired from TH Crestgate a 25% interest in Redcorp, which holds an 85% interest in the polymetallic Project and that Ascendant has an additional option to earn up to an 80% interest in Redcorp upon completion of the milestones highlighted below. Under subsequent agreements made with TH Crestgate, Ascendant has the right to earn into 80% of the Lagoa Salgada Project

Redcorp and EDM are now negotiating the terms of the concession contract with the Portuguese mining bureau: DGEG. The negotiation is a conventional procedure which essentially confirms the type of royalties (precious and base metals vs. coal etc.) applicable per established Portuguese law.

Figure 4-2 - Plan of Property



Source: DGEG, but legend has been translated into English, December 2019

4.3.1 TRANSACTION SUMMARY – KEY OPTION TERMS

- Ascendant acquired an initial effective 25% interest in Redcorp for an upfront payment of $2.45 million (M), composed of $0.8M in cash ($400,000 on closing of the transaction and $400,000 on 15 July 2018) and $1.65M in Ascendant shares.
- Ascendant holds the right to acquire a further effective 25% interest via the staged payments and funding obligations outlined below:
  - By investing a minimum of $9.0M directly in the operating company, Redcorp, within 48 months of the closing date, to fund exploration drilling, metallurgical testwork, economic studies, and other customary activities for exploration and development.
  - By making payments totalling $3.5M to TH Crestgate according to the following schedule or earlier:
    - 6 months after the closing date: $0.25M.
    - 12 months after the closing date: $0.25M.
    - 18 months after the closing date: $0.5M.
    - 24 months after the closing date: $0.5M.
    - 36 months after the closing date: $1.0M.
    - 48 months after the closing date: $1.0M.
- Ascendant then has the option to acquire an additional 30%, thereby increasing its total interest in Redcorp, the operating subsidiary, to 80%, by completing a feasibility study within 54 months and making a further payment of $2.5 M to TH Crestgate.
- Ascendant will fund all development and future construction costs and recoup TH Crestgate's share of the investment through cash flow until it has been repaid.
- Ascendant will retain a right of first offer on the remaining equity held by TH Crestgate.
- Under subsequent agreements made with TH Crestgate, Ascendant holds the right to acquire 80% of the Lagoa Salgada Project.

4.4 SURFACE RIGHTS, PERMITTING, AND ENVIRONMENTAL LIABILITIES

4.4.1 SURFACE RIGHTS

The surface rights covering the Property are held by two main landowners: Mr Manuel Rocha and Mr Carlos Caiado. The LS Property is situated within the surface rights of Mr Rocha. Relations with Mr Rocha are favourable with an agreement made to conduct exploration activities on the Property.

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

4.4.2 Permitting

To the QPs 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 is unaware of any environmental liabilities that would prevent Redcorp from conducting exploration activities on the property.

4.5 Royalties

Royalties are the highest value of the following:

- 10% of the net profit.
- 2.5% of the NSR.

5 ACESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURES AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Property is located approximately 120 km south-east of Lisbon, Portugal’s capital, and is accessible by road (highway). The Property is easily accessible by national highways and roads, and the drive from Lisbon takes about 1:30 hours.

Highway connections also exist to the Setúbal port facilities and the Sines industrial area and port facilities (deep-water port). The nearest town is Grândola, located at a distance of 12 km from the Property.

All parts of the Property can be accessed by truck/utility vehicle or 4 x 4 vehicle via unpaved roads (with a length of approximately 4 km). The roads are maintained, and some may be accessed by car.

5.2 PHYSIOGRAPHY

The Property is flat, with an elevation ranging between 20 m to 100 m above sea level.

The vegetation is typical of dry Mediterranean climates, consisting of scrub brush, tall grass and pine trees. The land of the Property is privately owned and used primarily for the cultivation of pine trees, eucalyptus and occasional cork trees. The soil in the area of the Project is sandy, with limited exposure of the Tertiary sedimentary bedrock.

Figure 5-1 - View of Lagoa Salgada Property

Source: Photo taken by Quadrante, August 2021

5.3 CLIMATE

The Property is located in a Mediterranean climate zone (Csa; Köppen climate classification), featuring a temperate climate with hot and dry summers, while winters are moderately cool with changeable rainy weather.

The July average minimum and maximum temperatures are 15.8°C and 29.3°C, respectively, while the January average minimum and maximum temperatures are 4.7°C and 15.1°C, respectively (source: IPMA, Portugal). The average annual precipitation is roughly 700 millimetres (mm) (source: World Climate), with very little or no precipitation during the summer months.

All activities can be conducted all year round, with occasional interruptions due to extreme weather conditions.

5.4 LOCAL RESOURCES AND INFRASTRUCTURE

The closest town to the Project is Grândola, population 14,000, whose main economic activity is agriculture. Grândola belongs to the Setúbal district, which has a population of approximately 90,000 and is located midway between LS and Lisbon. Setúbal has port facilities used by the Neves-Corvo Mine and Autoeuropa (the biggest car producer in Portugal and a leading exporter). The local economy is based on the cellulose, paper and cement industries. The Project is close to another industrial town, Sines, located at a distance of 60 km and accessed by highway. Sines is host to heavy and chemical industry and the main port in Portugal (deep-water port). Both ports (Setúbal and Sines) are connected to the national rail network. Basic services and supplies may be sourced from either of these towns.

The LS Property is located 50 km from the Aljustrel Mine (zinc/lead) and 85 km from the Neves Corvo Mine (zinc/copper), so that experienced mining personnel may be locally available. Labour may also be sourced from the nearby towns and villages, from other parts of the country or even from Estremadura in Spain, where there are many mines operating in the continuation of the Iberian Pyrite Belt.

The Project has sufficient land holdings for exploration and development purposes.

As described in the previous paragraphs, the Property is well connected to Portugal's main infrastructure installations, including roads, railways, electric power lines, ports and airports.

Given the power supply required for the Lagoa Salgada Project, the grid connection should be done via a high-voltage overhead line, at an intersection point located approximately 11.6 km from the Project. More details can be found in Chapter 18.9 of this Report.

Water sources are available on the Property, with current drill operations drawing water from a refurbished well.

6 HISTORY

6.1 PRIOR OWNERSHIP

The prior ownership and ownership changes of the LS Property are summarized 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 was able to maintain the property in good standing through office work and marketing to find a new partner.
- 2015-2017: In July 2015, TH Crestgate acquired a 100% stake in Redcorp. Redcorp then signed an addendum to the current contract for a period of 5 years in a joint venture with EDM (85% Redcorp and 15% EDM).
- 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 an 85% 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. Under subsequent agreements made with TH Crestgate, Ascendant has the right to earn into 80% of the Lagoa Salgada Project.

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 was incorporated into the Laboratório Nacional de Energia e Geologia (LNEG). Between 1992 and 1993 were completed 17 drillholes 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, averaging 135 m thick; 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, electro-magnetic 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 were successful in defining the broad outlines of the North (formerly LS-1), Central, and South (formerly LS-1 Central) deposits.

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

In Portugal, two years from 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. 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 core 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 program 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 a 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 program 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 (see Figure 4-2**Error! Reference source not found.**) 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 program consisted of six holes totalling of 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 program and a downhole geophysical survey program.

6.2.4.1 DRILLING PROGRAM

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 programs 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 rough 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 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 anomaly 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.

6.3 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.

Other than the January 2018 and February and November 2019 reports, the prior Mineral Resources 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 14 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.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 PREAMBLE

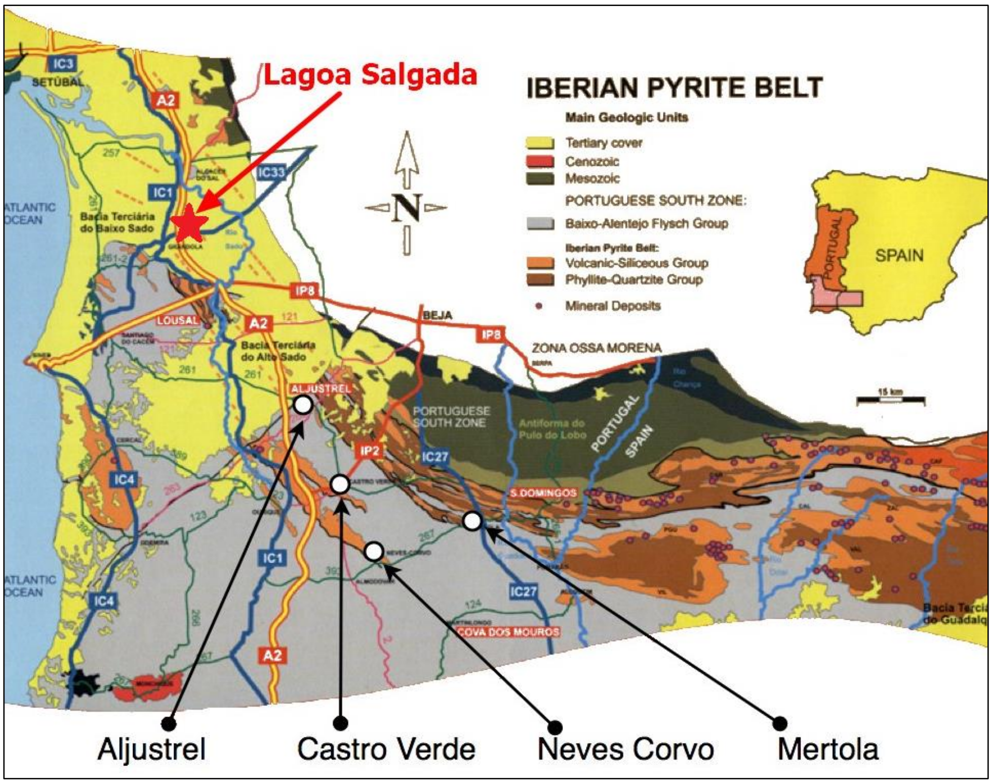
The LS project is located within the north-western portion of the Iberian Pyrite Belt (IPB). The IPB is one of the most prolific European ore provinces, hosting one of the largest concentrations of MS in the Earth's crust; it contains more than 1,600 million tonnes (Mt) of MS ore and about 250 Mt of 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 only 11 deposits can be considered large regarding their Cu Zn Pb contents. Ten deposits are in Portugal where currently only Neves Corvo and Aljustrel are being exploited.

There have been a few reports written on LS (Oliveira et al. 2009, 2011; Barros 2013) and several more written on the IPB and the other deposits (Clarke, et al. 2004; Oliveira el al. 2005, 2006; Tornos 2006; Laznicka 1999). This report will summarize the extensive work by others in the sections below.

7.2 REGIONAL GEOLOGY

The Property is located within the north-western portion of the IPB which stretches from southern Spain into Portugal (Figure 7-1). This belt is one of the three domains of the south Portuguese zone, the southernmost terrane of the Variscan orogen in the Iberian Peninsula. This terrane collided obliquely with the Ossa Morena terrane during the Variscan orogeny, leading to strike-slip tectonism (Oliveira et al. 2006). The result of the collision was opening of pull-apart basins within the continental crust of the south Portuguese terrane, triggering submarine volcanism in the IPB (Silva et al. 1990; Quesada 1991; Tornos et al. 2002). The IPB has a relatively simple geologic record (Schermerhorn 1971), with a sequence that includes about 1,000 to 5,000 m of late Paleozoic rocks. The oldest rocks found are grouped in the Phyllite-Quartzite Group Late Famennian; (Oliveira et al. 2005, 2006) that consists of a monotonous detrital sequence of alternating dark gray shales and quartz sandstone.

Figure 7-1– Regional Geologic Setting of the LS Deposit in the North-western Region of the IPB



Source: Ascendant.

The Volcano Sedimentary Complex overlies the Phyllite-Quartzite Group and hosts the VMS. This belt is a thrust faulted sequence of sedimentary rocks spatially related to local sub-aqueous volcanic centres which host the VMS deposits. The stratigraphic sequence of the Volcano Sedimentary Complex was defined in the Pomarão area of Portugal and grouped into three felsic volcanic cycles separated by two mafic ones (van den Boogard 1967). The volcanic sequence can reach a thickness of up to 1,300 m (true thickness) near the volcanic centres according to Tornos (2006) and is characterized by a large diversity of volcanic and sedimentary facies. The Volcano Sedimentary Complex includes a felsic-mafic volcanic sequence interbedded with shale (~75% shale and ~25% felsic and mafic volcanic rocks) and some chemical sediments (Oliveira 1990; Barrie et al. 2002; Oliveira et al. 2005, 2006). The VMS deposits are generally interpreted to be syngenetic in origin; however, mineralization ranges from sulphide precipitates to re-worked sulphide / silicate sediments and local sulphide replacement mineralization located near the felsic submarine volcanic centres. The MS deposits are hosted by the felsic volcanic units and / or black shales. Recent detailed physical volcanology studies (Rosa 2007; Rosa et al. 2008, 2010) show that the felsic volcanic centres of the Volcano Sedimentary Complex were built up by a variable number of effusive and explosive volcanic episodes. The volcanic centres consist mainly of felsic lavas and domes and may have intercalated thick pyroclastic units that were sourced from the lavas and / or domes (Rosa 2007; Rosa et al. 2008, 2010). Quartz and feldspar-phyric rhyolitic and dacitic compositions are dominant. The volcanic centres have marginal aprons of abundant bedded volcaniclastic units that gradually develop into shales with nonvolcanic origin that are the dominant rock type of the Volcano Sedimentary Complex. Regionally, the IPB can be divided in northern and southern branches that are distinguishable by different tectonic styles (Oliveira et al. 2005, 2006) and by distinct characteristics of the MS deposits (Sáez et al. 1999; Tornos 2006).

The Volcano Sedimentary Complex is overlain by the Baixo Alentejo Flysch Group, a turbiditic sequence that comprises shales, litharenites, and rare conglomerates (Oliveira 1990). The Baixo Alentejo Flysch Group is up to 3,000 m thick, ranges in age from Late Visean to Middle-Upper Pennsylvanian (Oliveira et al. 2005, 2006; Tornos 2006), and represents the synorogenic foreland flysch associated with Variscan collision and tectonic inversion (Moreno 1993).

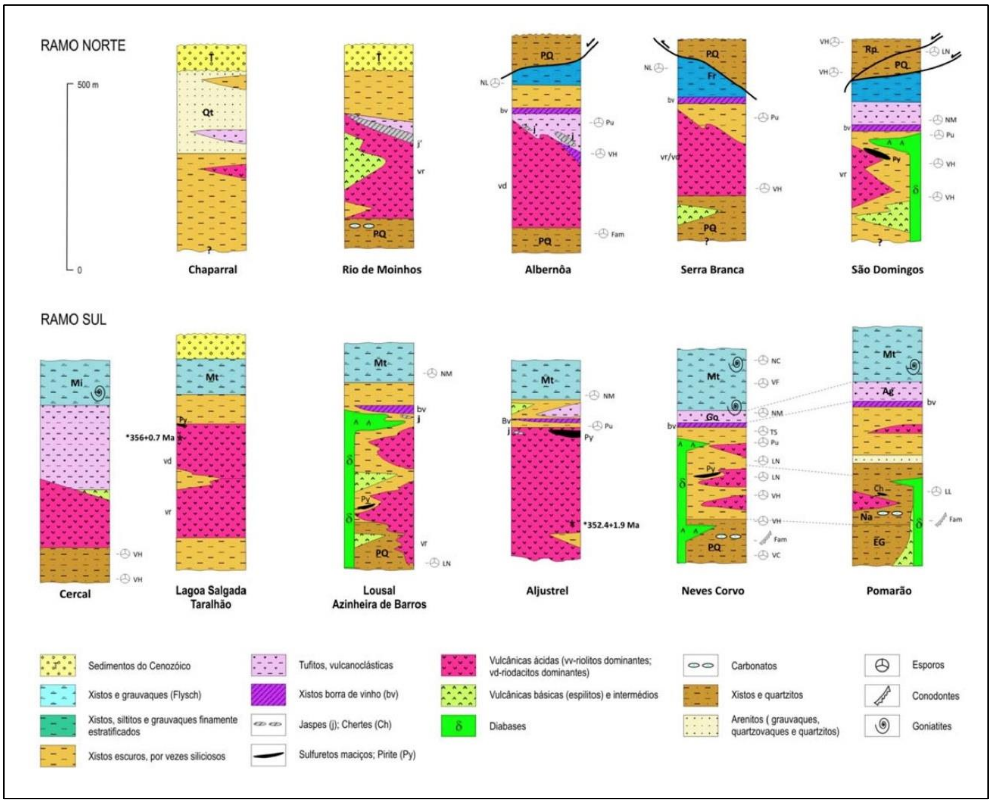
Deformation of the IPB stratigraphic sequence during the Variscan orogeny is characterized by south- to southwest-verging folds, corresponding to a thin-skinned foreland fold and thrust belt (Silva et al. 1990). Low-grade regional metamorphism displays a northward increase from zeolite facies in the south to greenschist facies in the north (Munhá 1990).

7.3 Property Geology

The entire project area is covered by a paleo-fluvial fan that ranges in thickness up to 200 m within the Tertiary Sado Basin and averages 135 m over the LS deposit (Figure 7-2). 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 are represented on the property by a northwest-southeast lineament which is approximately 8.0 km long and over 1 km wide.

The deposit is 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).

Figure 7-2- LS Property – Stratigraphic Column of the LS and other IPB Properties



Source: Ascendant 2019

The mineralization comprises MS and semi-massive sulphide lenses and sulphide veins and veinlets and is mainly hosted by a thick (up to 100 m) and strongly chloritized quartz phyric rhyodacite unit that overlies and is laterally equivalent to the quartz phyric rhyodacitic unit in the south-east.

These two rhyodacites are clearly distinguished by their phenocryst content, and, geochemically, the former corresponds to a more evolved series than the latter. These rhyodacites plot in the andesite field of the diagram after Winchester and Floyd (1977), in contrast with their phenocryst content. This anomalous geochemical classification is interpreted to be caused by low- temperature crustal fusion, which affects the melting of refractory phases (such as zircon) where high field strength elements reside and was previously identified in volcanic rocks from other areas of the IPB by Rosa et al. 2004, 2006. Chloritization by the addition of Mg and Fe has affected most samples, causing the results to plot along a trend toward the chlorite and / or pyrite corner of the alteration box plot. This is interpreted as being typical of chlorite-dominated footwall alteration either in felsic or mafic volcanic rocks (Large et al. 2001).

The architecture of the volcanic and sedimentary units that host the MS mineralization was defined by detailed logging and inspection of slabs and thin sections from drill core of the LS area. Original volcanic and sedimentary textures are typically destroyed or modified in proximity to the zones of more intense hydrothermal alteration and deformation (near the thrust zones). However, primary rock textures are preserved in the less deformed and altered zones.

The quartz-phyric rhyodacite is dominated by coherent facies that is intercalated and grades to overlying monomictic rhyodacitic breccia facies. Intervals of the coherent rhyodacite facies are up to 150 m thick. These facies are evenly quartz-phyric, with ~7 modal percent of embayed euhedral to subhedral, 5-mm-long quartz phenocrysts. The rhyodacite groundmass is flow banded, characterized by 1-mm to 1-cm thick alternating dark and pale bands that may contain abundant chlorite wisps. Pale bands are mainly composed of microcrystalline sericite, with accessory quartz and feldspar, whereas dark bands are composed of microcrystalline quartz, feldspar, and chlorite with accessory sericite. These bands also show abundant relics of recrystallized spherulites (e.g., drillhole LS-1). The coherent facies may show dark (chlorite-rich) and pale (sericite-rich) irregular domains that are probably the result of hydrothermal alteration. The monomictic rhyodacitic breccia facies consists of massive, clast-supported intervals of irregular and polyhedral rhyodacite clasts. These clasts have similar textures to the coherent rhyodacite facies, and their shapes and groundmass textures suggest that fragmentation of the rhyodacite is probably a consequence of autobrecciation. The upper part of the quartz-phyric rhyodacitic unit consists of an interval (up to 50 m thick) of monomictic rhyodacitic breccia facies that encloses the most well-developed sulphide stockwork of the central stockwork zone. This breccia interval has a fault contact with the overlying shale that shows moderate sericitic alteration. The great thickness of coherent facies suggests that the central stockwork zone of LS deposit corresponds to the proximal setting of a felsic volcanic centre (McPhie et al. 1993).

The sequence hosting the MS lens in the north-west comprises a thick (up to 100 m) feldspar- and quartz-phyric rhyodacitic unit that overlies and is laterally equivalent to the quartz phyric rhyodacitic unit in the south-east. The feldspar- and quartz-phyric rhyodacite is typically sericite and chlorite altered and comprises thin intervals of coherent rhyodacite that grade to much thicker intervals (up to 50 m) of monomictic feldspar- and quartz-phyric rhyodacitic breccia. The coherent facies are evenly feldspar-phyric, with ~20 modal percent of feldspar phenocrysts and ~5 modal percent of quartz phenocrysts. The monomictic feldspar- and quartzphyric rhyodacitic breccia is dominated by thick clast-supported intervals, characterized by jigsaw-fit and clast-rotated arrangement of the clasts. The clasts have planar to curviplanar or ragged margins and some are dominantly sericite altered, whereas others are chlorite altered. The monomictic breccia typically hosts a well-developed sulphide stockwork, with the veins occurring preferentially in the matrix of the breccia. Overlying this stockwork occur a MS lens (e.g., drillhole LS5).

Remobilization of clastic components from the feldspar and quartz-phyric rhyodacitic unit defines relatively small (up to 30 m thick × 200 m long) volcaniclastic units. The shapes of the clasts in the monomictic feldspar- and quartz-phyric rhyodacitic breccia and the thick intervals of jigsaw-fit textures suggest that they have formed by quenching of the rhyodacite in contact with water, and that the breccia corresponds to hyaloclastite (Pichler 1965). The great thickness of monomictic feldspar- and quartz-phyric rhyodacitic breccia and the abundant intervals of remobilized rhyodacitic clasts suggest that the rhyodacitic unit probably corresponds to a massive lava (McPhie et al. 1993).

The volcanic units and MS lens are overlain by an irregular and discontinuous layer up to 50 cm thick of hydrothermal chert (e.g., drillholes LS-14 and LS-22; Matos et al. 2000), or a thick interval of shale, locally displaying strong chlorite-sericite alteration. Away from the deposit this shale may host millimetre- to centimetre-sized intercalations of siltstone and graywackes while the cherts probably give way to jaspers, which were recognized to the north of the east-west Alpine age fault shown in **Error! Reference source not found.** (Matos et al. 2000).

LS is further offset by the east-west−trending Alpine-age fault in the north (Figure 7.3), with a 50 m downthrow of the northern block, but whose horizontal amount and sense of displacement is unknown (Matos et al. 2000).

The volcanic sequence has been separated into two units: The Upper Volcanic Unit (UVU) and the Lower Volcanic Unit (LVU) which are described below.

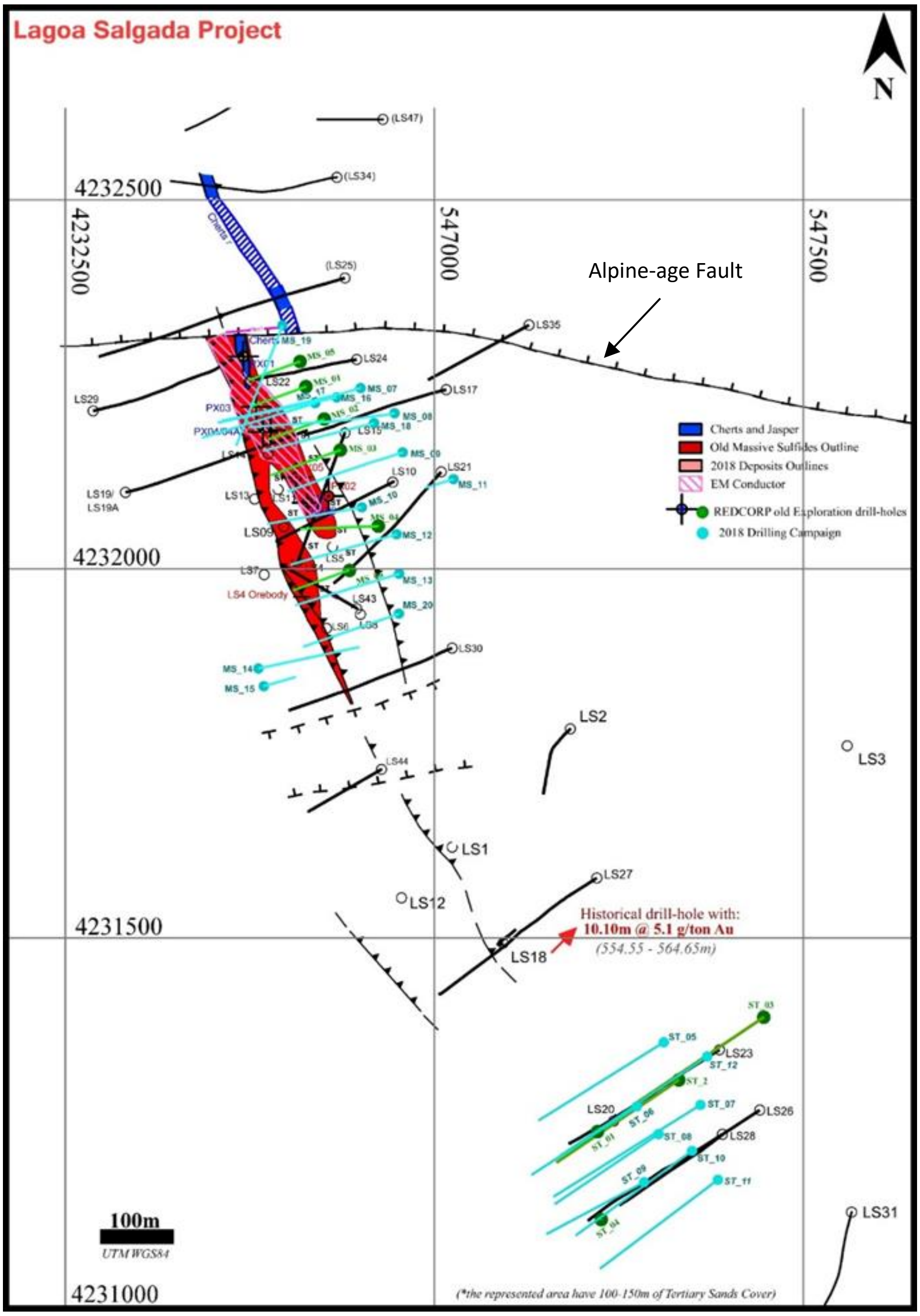
7.3.1 Upper Volcanic Unit

The UVU consists of intermediate to felsic porphyritic tuffs with coarse feldspar phenocrysts, locally including lava facies with porphyritic and auto-breccia textures and fine-grained chlorite-sericite tuffs. Lithogeochemical assays carried out in 2005 classified this rock type as andesite. Hydrothermal alteration of the rock, to chlorite-quartz with disseminated sulphide, is intense close to the MS body where replacement textures are common in the footwall of the sulphide body. Alteration minerals transition gradually to less altered zones composed of chlorite + sericite + carbonates + quartz + sulphides and quartz + carbonates away from the sulphide body.

7.3.2 Lower Volcanic Unit

The LVU is comprised of felsic porphyritic tuffs with abundant quartz phenocrysts (quartz-eye meta volcanic rock) with metre-scaled intercalations of volcano-sedimentary breccias. Whole rock geochemical assays carried out in 2005 classified this unit as dacite. The predominant hydrothermal alteration minerals are sericite + quartz + carbonates + sulphides. Near the sulphide body footwall, the intensity of the alteration increases and is defined by chlorite ± pyrophyllite (Matos et al. 2000).

Figure 7-3 - LS Property – Alpine-Age Fault in the North



Source: Ascendant 2019.

7.3.3 <u>MINERALISATION</u>

There are four types of mineralization at the LS Property: primary MS mineralization, GO mineralization resulting from weathering of the primary mineralization, copper-rich stringer / fissure / stockwork mineralization, and gold-rich silicified zones which appear to be structurally controlled. To date, the mineralized system of the North deposit has been drill tested over a strike extent of approximately 500 m and appears to be open to the south and east. Recent geophysical surveys have found three anomalies, similar in signature to that of the North deposit (former LS-1), continuing to the south-east along strike, over 900 m. The furthest of these anomalies has been drill tested and it is the South deposit (former LS-1 Central deposit).

The MS mineralization occurs in steeply dipping to vertical isoclinal folded volcanic rocks. Primary MS mineralization has been intersected in several diamond drillholes. This mineralization has variable, but significant, base and precious metal values. The best example of this style of mineralization was intersected in drillhole PX-04 and, most recently, in drillholes LS_MS_01 and LS_MS_02. The MS body appears to be cut by post-mineral faults and its relationship to the surrounding stratigraphy is not well understood. The faulting has likely caused a displacement of the continuation of the deposit. The thick overburden cover and the depth of the mineralized body precludes drilling the deposit with a shallow dipping drillhole. For this reason, most of the drilling on the deposit has been either with vertical or steeply dipping drillholes. This has resulted in drillhole intersections that are less than ideal and almost parallel to the primary stratigraphy of the sulphide body. The true thickness of the deposit therefore cannot be determined from single vertical drillhole intersections. The 2017 drillholes were all angled, which helped in the interpretation of the deposit. The thickness of the deposit is inferred as being somewhere between holes that intersected the MS and those that have intersected the footwall or hanging-wall rocks. Additional drilling is required to determine the size of the known MS deposit.

GO mineralization results from the weathering of primary MS mineralization. It is preserved at LS because of the Tertiary sedimentary rocks covering the palaeosurface, in a situation analogous to the Las Cruces copper deposit in Spain. GO mineralization at LS seems to be comprised of a lead rich leached cap, underlain by a precious metal-rich supergene enrichment zone. This is well displayed in hole LS-09.

Copper-rich stringer / fissure stockwork mineralization consists of sulphide veins and stringers in chloritic volcanic rocks, and represents alteration associated with the feeder system to the MS mineralization. This type of mineralization is well-developed in other IPB deposits such as Feitais (Aljustrel) and Neves Corvo and is best typified by the intersections in drillhole LS-20.

8 DEPOSIT TYPES

The LS deposit is a polymetallic, VMS deposit. VMS ore deposits are a type of metal sulphide deposits which are associated with and created by volcanic-associated hydrothermal events in submarine environments. They occur within environments dominated by volcanic or volcanic derived volcano sedimentary rocks, and the deposits 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.

These types of deposits consist of lenses of MS mineralization that were deposited at or near the sea floor because of precipitation from the venting of metalliferous hydrothermal fluids. These fluids typically exploit fault planes as fluid pathways and create a large zone of hydrothermal alteration in the rocks below the deposits. Commonly these form in second and third order basins and are rapidly covered so they can be preserved.

VMS deposits are characterized by clusters of lenses occurring within a distinct stratigraphic layer. The extensive alteration zone on the property suggests that hydrothermal activity was prolonged and that additional lenses associated with separate alteration zones may exist.

The LS deposits were intersected in drillholes and occur within a thick (>700 m) Volcano Sedimentary Complex sequence made up of feldspar- and quartz-phyric rhyodacite, and quartz phyric rhyodacite with intercalations of siltstone, the base of which has not been intersected (Matos et al. 2000). LS is not associated with sedimentary rocks near the MS, contrasting with some of the other MS deposits within the IPB, for example, Neves Corvo and Lousal. True thickness of the stratigraphic sequence is difficult to determine, due to disruption and repetition of the volcaniclastic units by several thrust faults.

9 EXPLORATION

From the discovery period to the present, exploration on the LS Property has been conducted using geophysical techniques (gravity and Induced Polarization (IP)). Much of the earlier exploration work up to 2015 is described in Section 6. This section focuses on the more recent work.

9.1 2016 Petrographic Study from Porto University

In early 2016, Redcorp submitted 20 samples from four of the 2010 to 2012 drillholes to the Porto University Science Faculty, DGAOT – FCUP laboratory for petrographic analysis study. The petrographic study consisted of microscope studies on polished sections using stereo-binocular microscopy and conventional reflected polarized microscopy. Several polished sections were selected for examination using scanning electron microscopy and x-ray microanalysis (MEV-EDS) to confirm the identities of constituent minerals, while others were selected to perform quantitative microanalysis using the electron microprobe technique.

The 20 samples were from representative sections of the stratigraphy that included the following: GO, supergene, chert / jasper, MS, and stockwork. The study report details the mineral suite, textural relationships, primary microstructures, recrystallization textures and chemistries (mineral and whole rock) for the samples. Textural information and association notes are useful here as the samples were noted to be extremely fine-grained and it was concluded as being highly probable that other valuable minerals would also be present in minor to trace amounts, but their characterization was beyond the scope of the study.

The study showed that the individual fragments consisted predominantly of sulphide minerals with non-sulphide gangue minerals being present only in relatively small amounts. Of the sulphides, pyrite was noted to be the most common phase. Both sphalerite and galena were observed in subordinate amounts together with subordinate amounts of arsenopyrite and minor chalcopyrite. Other minerals including tetrahedrite-tennantite and related sulphosalt minerals were noted to be present in very small amounts.

Pyrite and arsenopyrite were both noted to be in the form of granular masses with a wide grain size distribution (<1 micron (µ) to >150 µ for pyrite and 5 µ to >150 µ for arsenopyrite). An intimate association of pyrite with arsenopyrite intergrowth was noted, with pyrite patches within larger arsenopyrite grains, and arsenopyrite intergrowth as inclusions within larger pyrite grains.

Larger sphalerite grains (>20 µ in size) commonly show the presence of fine chalcopyrite, inclusions, but also, less commonly, those of galena and pyrite. Small amounts of sphalerite were noted to occur as very fine (<1 µ) inclusions in pyrite.

Galena was also observed to have a wide distribution of grain sizes, from <1 µ to >150 µ, although most particles were observed to be in the <25 µ range. The associations with pyrite, arsenopyrite, sphalerite and other sulphides are similar to those observed for sphalerite. Galena is commonly intergrown with sulphosalt minerals and chalcopyrite.

Chalcopyrite grains were noted to rarely exceed 25 µ in size, tending to occur either along grain boundaries of larger pyrite grains, or as components of fracture filling assemblages. Most of the chalcopyrite appeared to be fresh or unaltered, although some occurrences of secondary minerals were noted.

Fine cassiterite (<5 µ) was noted as inclusions or intergrowths with sphalerite.

9.2 2017 Mise à la Masse Downhole Survey

In September and October 2017, IGT was contracted to complete a downhole geophysical survey in three drillholes: LS_MS_01, LS_MS_03 AND LS_MS_06.

The following is taken from IGT (2017]:

"The main conclusion to be drawn from the results of this study is that the anomalies produced by the semi-massive sulphide deposit are as sharp as we could expect looking at the theoretical model..."

"The Tertiary cover may relax the potential values at the surface, but this effect does not mask the influence of the conductive orebodies where electrode A was earthed in the surveyed drillholes."

"From the potential maps we interpret that the conductors intersected by LS_MS_01, LS_MS_03 and LS_MS_06 drillholes look the same one. Drillhole LS_MS_01 has intersected it close to its SE end, LS_MS_03 has hit it at its central section, where the orebody shows its maximum thickness and LS_MS_06 shows it at its NW end. This conductive body (semi-massive sulphide deposit) trends following a N160 E for approximately 500 m and is centred in the study area along stations 400 m from line L-2 to line L-7."

9.3 2018-2019 Exploration Geophysics

This Section 9.3 is an extract from a detailed report by Christopher J. Hale, Ph.D., P.Geo. of Intelligent Exploration (I.E.), who has been Redcorp's geophysics consultant for the past three to four years.

Since its discovery by the Portuguese Geological Survey as a gravity anomaly in 1992 (Oliveira et al. 1998) the LS Property has been explored in a succession of geophysical campaigns. Exploration rights were assigned to a consortium including RTZ and government agencies in 1994. Historically both gravity and IP surveys have been used at LS. The exploration history of the property has been summarized in Section 6 of this Technical Report. All previous Technical Reports on the LS Property recommended additional geophysical exploration, particularly in the area separating the North and South deposits.

In 2018, IE selected a suite of samples from the LS drill core to measure physical properties including Specific Gravity, electrical and magnetic characteristics.

The physical properties data were summarized by Hale (2018). The conclusions of that work led to a re-examination of historical Gravity data and the choice of Induced Polarization / Resistivity surveys (IP / Resistivity) to continue exploration in 2018 – 2019. Significant properties for exploration are summarized below.

9.3.1 Physical Characteristics of the LS Core Samples

The MS mineralization is dense (specific gravity up to 4.6), highly conductive (~ 1 Ohm-m resistivity), and Chargeable (~100 millivolts per volt (mV/V)).

The altered volcanic host is moderately dense (SG ~2.8), less conductive (~1,000 Ohm-m), and not chargeable but becomes much less resistive as it is altered to clay.

Stringer or stockwork mineralization is intermediate between these two types. The Specific gravity increases above 3.0 as sulphide mineralization increases. Conductivity presents a variable picture depending on the "connectedness" of the sulphide grains but all samples with sulphide mineralization are chargeable.

The Tertiary cover is much less dense (S.G. ~2.2). The basal conglomerate appears to be conductive over the known deposit, grading to higher resistivity away from the MS. Chargeability is associated with the Tertiary cover rocks, particularly over the deposit.

Some weak magnetic susceptibility was noted in the case of a few samples, but this was generally not far above the detection limit for the probe. Magnetic surveys will not be able to detect this target.

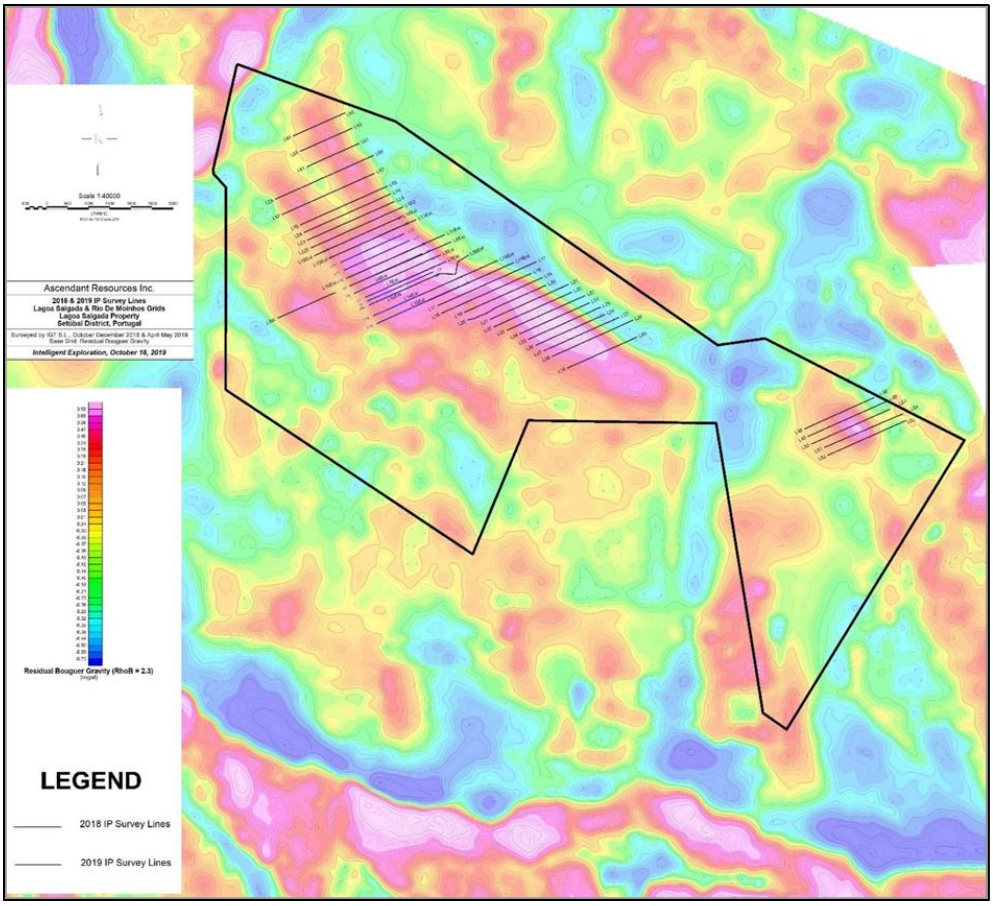
9.3.2 Gravity at LS

The deposit was originally discovered during the drilling of a gravity anomaly, detected at a regional scale by the government geological survey. This was followed up by more detailed gravity work (down to 50 m spacing of stations over the deposit) and additional drilling by RTZ. In 2018, IE replotted a residual Bouguer Gravity map using the residual gravity tabulated by Wright (2007) for a Bouguer density of 2.3.

Figure 9-1 shows the detailed map of the Bouguer Gravity for the Property and the position of the 2019 and earlier IP / Resistivity lines.

The gravity data highlight an anomaly over LS that is not limited to the ~400 m length of the known MS deposit. It extends to the south, reaching a maximum in the Central Zone. The azimuth of the gravity anomaly appears to trend farther east than the projection of the LS-North mineralization, more parallel to the regional gravity trend.

Figure 9-1 - 2019 IP / Resistivity Survey Coverage



Source: I.E. Intelligent Exploration 2019

There is a good agreement between the location of the known massive and stringer / stockwork mineralization and elevated residual Bouguer Gravity. These gravity anomalies also correspond to enhanced chargeability measured in 2018 (Hale and Gilliatt 2018).

Reconnaissance IP / Resistivity surveys were recommended (Hale 2019) to provide the necessary penetration through the ~100 m of Tertiary cover, imaging chargeable targets to depths over 300 m. Lines were also planned to extend coverage to the east of the 2018 lines and to survey the gravity maxima between LS North and Rio de Moinhos.

9.3.3 2019 IP / Resistivity Surveys

9.3.3.1 Survey Methods and Procedures

IP / Resistivity surveys were carried out by IGT using an IRIS Instruments (IRIS) ELREC-PRO Receiver and an IRIS VIP 10,000 W transmitter. The surveys were completed during April and May 2019.

The surveys employed a pole-dipole array. This configuration was chosen because it provides a good balance between depth of penetration and lateral resolution. At each station, 10 dipoles of 75 m were recorded (“a” = 75 m and n = 1 to 10, ~200 m line separation) to achieve a penetration depth over 350 m. Data before 2018 that had been collected with shorter dipoles resulted in very low point-to-point primary voltages and correspondingly noisy profiles. This problem was addressed by increasing the dipole size and using fewer dipoles and a higher power (10 kilowatts (kW) vs. 4 kW) transmitter. The reduced resolution due to the larger dipole size was not significant given that the volume of interest lay under about 120 m of Tertiary sedimentary cover. This cover required emphasis to be placed more on depth penetration and signal strength (Vp) than high resolution.

A transmitting pulse width of 2 seconds was used with alternating polarity, separated by a 2 second “off time” during which the chargeability data were collected.

The receiver recorded in 20 channel Semi-Logarithmic domain mode. This mode provided enough samples early in each decay cycle for calculation of an initial chargeability MIP in addition to the Mx bulk chargeability. Multiple readings were averaged at each station until the standard deviation of the average was less than a specified tolerance. The entire reading and averaging process was repeated for a station if the data failed to reach the quality specified.

Stainless steel rods and ~20 litres (L) of brine were used for current electrodes at each station and potentials were measured using copper sulphate ($CuSO_4$) porous pot electrodes. Lines were surveyed from the southwest to northeast with the local current electrode(s) trailing the receiver electrodes. The “infinity” current electrodes were placed in dug and salted pits lined with aluminum foil and irrigated with several hundred litres of water supplied by a tractor and tank-trailer.

The total survey coverage was 74.4 km. The IP / Resistivity survey coverage is shown in Figure 9-1 above with survey details listed in Table 9-1. Figure 9-1 above shows the extent of 2019 IP / Resistivity coverage (black lines). The 2018 and earlier lines are shown in blue. The black outline is the LS Property boundary. The colour grid displayed as a base map is the Bouguer Gravity map after Wright’s 2007 re-calculation using a density of 2,300 kg/m3 for the Bouguer correction. Table 9-1 is compiled from the data provided to IE by International Geophysical Technologies for daily QA/QC and processing.

	Coordinates (UTM – WGS 84)				**Length (m)**
	Start		**End**		
	X	**Y**	**X**	**Y**	
LS West & North					
L5bExt	547366	4231036	548999	4231898	2,025
L7Ext	547376	4231260	548460	4231260	1,200
L8Ext	547112	4231362	548336	4231907	1,350
L9Ext	546186	4231167	548591	4232217	2,625
L10Ext	546105	4231350	548434	4232387	2,550

	Coordinates (UTM – WGS 84)				Length (m)
	Start		End		
	X	Y	X	Y	
L13Ext	545701	4231828	547756	4232743	2,250
L14Ext	547844	4230948	549694	4231772	2,025
L15Ext	548016	4230806	550072	4231721	2,250
L16Ext	545345	4231888	547401	4232803	2,250
L522	545264	4232071	547453	4233046	2,400
L53	545182	4232254	547238	4233169	2,250
L54	545102	4232437	547157	4233352	2,250
L55	545020	4232619	547075	4233534	2,250
L57	544583	4232863	546771	4233842	2,400
L59	544420	4233228	546749	4234265	2,550
L61	545171	4234000	546404	4234549	1,350
L63	545008	4234365	546240	4234912	1,350
L65	544846	4234730	546080	4235279	1,350
L9W	544473	4230405	545843	4231015	1,500
				Total	**38,175**
Rio De Moinhos					
L48	556968	4227780	558616	4228522	1,800
L49	557050	4227604	558900	4228426	2,025
L50	557131	4227420	558981	4228244	2,025
L51	557440	4227340	559298	4228137	2,025
L52	557522	4227156	559372	4227980	2,025
				Total	**9,900**
LS East					
L17	548280	4230705	550610	4231742	2,550
L18	548362	4230522	550486	4231467	2,325
L19	548900	4230543	550750	4231366	2,025
L20	548981	4230360	550831	4231184	2,025
L21	549519	4230380	551369	4231204	2,025
L22	549600	4230198	551450	4231021	2,025
L23	549911	4230117	551761	4230940	2,025
L24	550220	4230036	551864	4230768	1,800
L25	550416	4229904	552129	4230667	1,875
L26	550588	4229762	552233	4230494	1,800
L27	550784	4229630	552563	4230421	1,950
L28	550979	4229498	552895	4230353	2,100
L30	551348	4229224	552992	429956	1,800
				Total	**26,325**

	Coordinates (UTM – WGS 84)				Length (m)
	Start		End		
	X	Y	X	Y	
Rio De Moinhos					
L48	556968	4227780	558616	4228522	1,800
L49	557050	4227604	558900	4228426	2,025
L50	557131	4227420	558981	4228244	2,025
L51	557440	4227340	559298	4228137	2,025
L52	557522	4227156	559372	4227980	2,025
				Total	**9,900**
				TOTAL	**74,400**

Table 9-1 - IP / Resistivity Survey Coverage

9.3.4 Data Processing and Presentation

The IP / Resistivity data were downloaded daily from the Elrec Pro receiver to a portable computer using PROSYS II software from IRIS. The resulting instrument dump file (*.bin) was edited (spurious readings removed) by IGT field personnel. The clean .bin files were sent to IE for QA/QC review as each line was completed and Cole-Cole parameters were calculated by IE. The edited *.bin data were then exported to a Geosoft format (.dat) file.

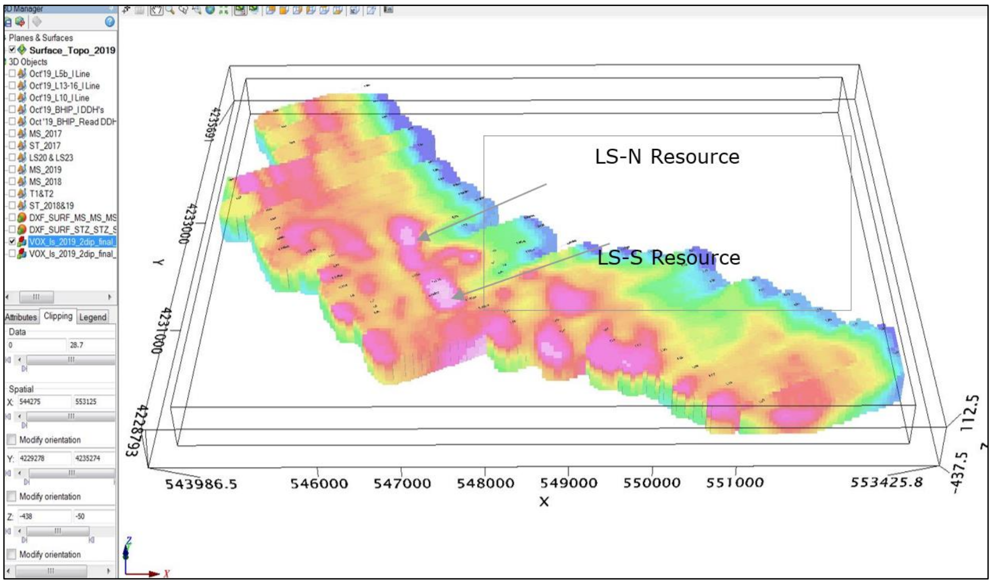
Data files were imported into Geosoft Oasis Montaj® databases (.gdb). Separate data channels were created to store the apparent Resistivity and average IP value (Mx Chargeability) of the middle time slices (~500 to 1,000 msec). Four panel pseudosections with Apparent Resistivity, Chargeability (Mx), Initial Chargeability (Mip) and Decay Time Constant (Tau) were calculated for each line for quality assessment and correlation from line to line.

DCIP2D software developed by the Geophysical Inversion Facility at the University of British Columbia was used to calculate an inverse model for each line. These two-dimensional (2D) inverse models were corelated and re-gridded in 3D blocks to produce the final 2.5D models of resistivity and chargeability.

9.3.5 Surface IP / Resistivity Results

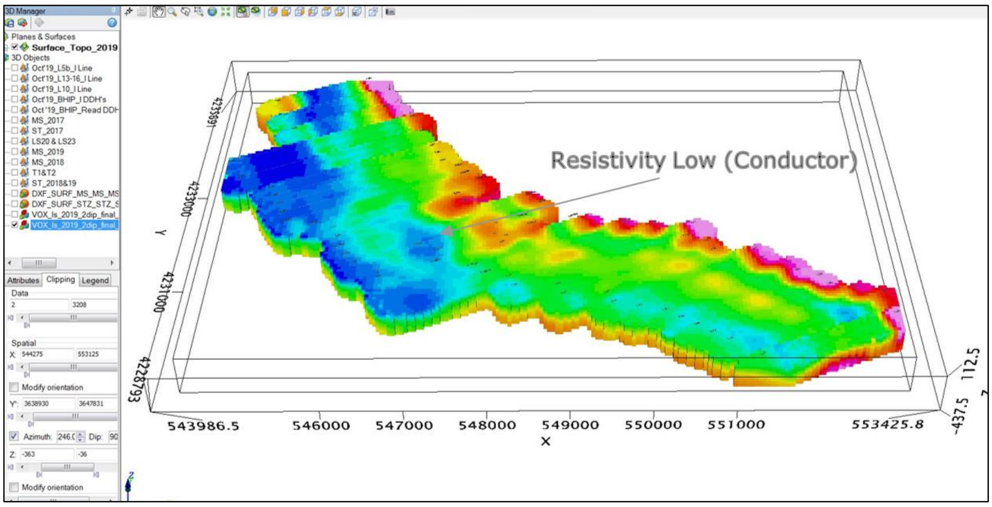
Figure 9-2 and Figure 9-3 show the chargeability and resistivity models respectively, composed from the 2D inverse models calculated for each surface line. The block model has been sectioned at an altitude of -50 m (relative to sea level), approximately 140 m below the surface. This level plan indicates chargeability just below the unconformity that separates the volcanic-sedimentary complex from the overlying Tertiary sedimentary rocks. A clear chargeability maximum corresponds to the known position of the LS North deposit. At this shallow depth a clear Mx peak is also associated with the LS South zone.

Figure 9-2 Mx Model Chargeability at -50 m (relative to sea level, about 140 m below the surface)



Source: I.E. Intelligent Exploration 2019.

Figure 9-3 - Model Resistivity at -50 m (relative to sea level, about 140 m below the surface)



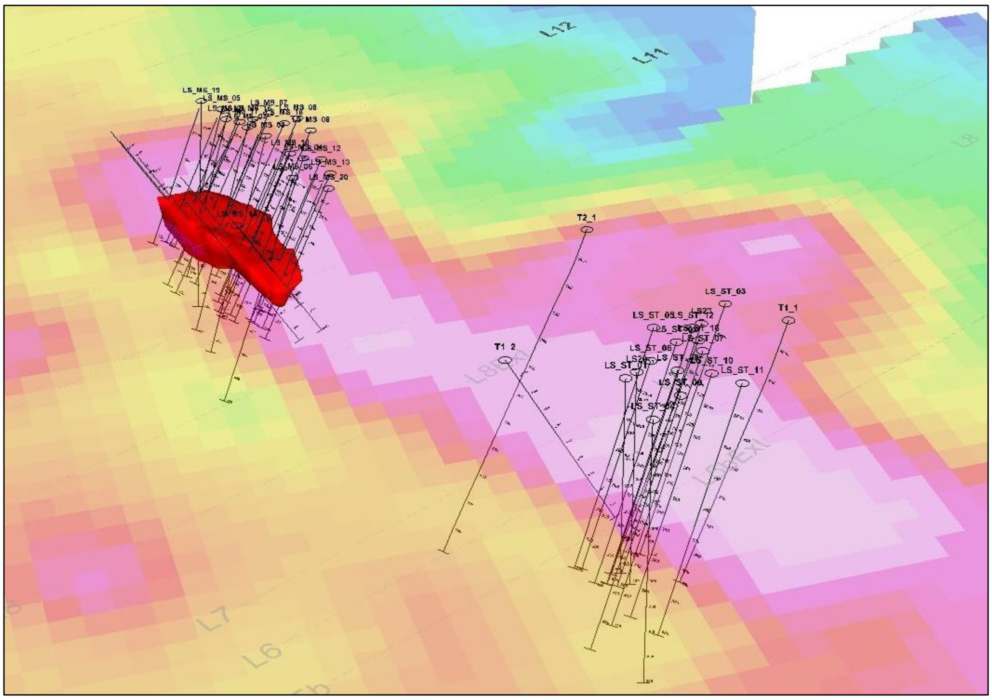
Source: I.E. Intelligent Exploration 2019.

Figure 9-4 shows a closer view of the central part of the IP model, looking toward the north-east, with the depth of the model top adjusted to -108 m, about 200 m below the surface. The IP model can be compared to the drilling in 2019 and earlier years as well as the position of the LS North resource calculated early in 2019, also displayed.

A clear anomaly extends southward from the North Resource, but it appears to be displaced approximately 100 m to the east and 100 m deeper than the known mineralization of the North Resource.

In the LS South (stockwork) zone, the maximum anomaly lies east of most of the drilling that has taken place to date. East to west drill trajectories would have passed above the volume where the maximum chargeability occurs, deeper on the east side of the LS-South deposit. It is not clear that the maximum chargeability corresponds to the greatest concentration of economic mineralization (particularly because pyrite is much more potent than sphalerite in causing chargeability anomalies) but this eastward and deeper extension of the chargeability anomaly should be drill tested.

Figure 9-4 - Model Chargeability at -108 m (about 200 m below the surface)



Source: I.E. Intelligent Exploration 2019

A chargeability anomaly on the apparent east limb of the LS-Rio do Moinhos folded structure, was recommended for drill follow-up early in 2019. This recommendation is still valid in view of the clearer anomaly presented by the 2019 data.

Data from the Rio do Moinhos anomaly were not of the same quality as the remainder of the survey and failed to provide a convincing target for drill follow-up.

9.3.6 <u>BOREHOLE INDUCED POLARIZATION RESULTS</u>

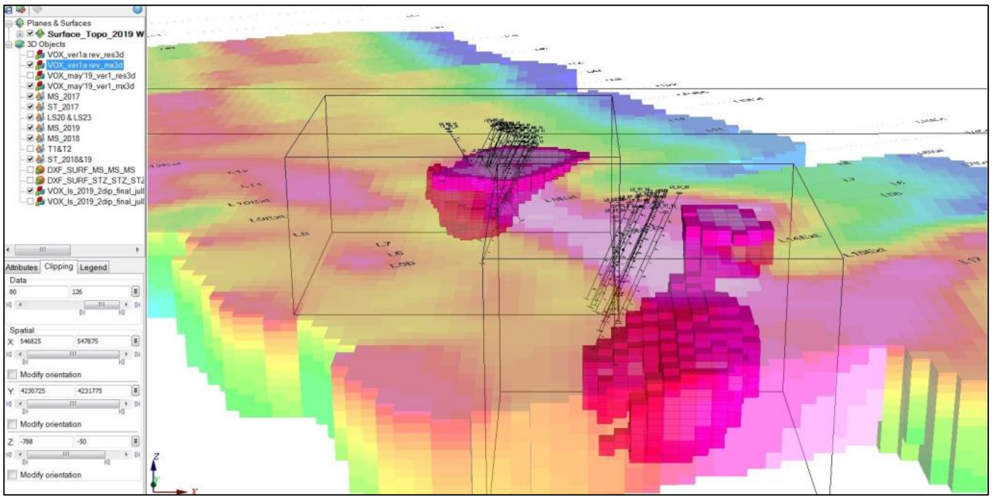
Borehole Induced Polarization (BHIP) was used to increase the resolution of the chargeability models at depth, particularly in the vicinity of the LS North and LS South

Resources when drill step outs were planned in the spring of 2019. Figure 9.5 shows a view (looking north) of two BHIP 3D chargeability models superimposed on the broader 2.5D surface chargeability model. The BHIP models used a fully 3D array of data combining cross-hole measurements from several drillholes in each area of interest following the methodology of Hale and Webster, 2006. Current was injected at depth in the holes, eliminating the de-focusing effect of the Tertiary cover and readings were generally taken with 25 and 50 m dipoles, read every 10 m. An expanding dipole was also measured from the base of the Tertiary (at the end of the steel casing) to the end of each hole to provide a wider search radius around the hole. The BHIP models are shown with a finer (X = 25 m x Y = 25 m x Z = 12.5 m) block size than the surface data. DCIP3D software from the Geophysical Inversion Facility at the University of British Columbia was used to calculate the 3D BHIP models.

Figure 9-5 shows that the spatial correlation between the chargeability model and the mineralization known from drilling is very good for the North Resource but that the maximum chargeability associated with the South Resource occurs to the east and deeper than the drilling to date. Agreement is good between the surface IP models and the BHIP, suggesting that both the North and South Resources are part of a single anomalous zone. The Central Zone remains largely untested by drilling, so the source of this anomaly remains to be identified.

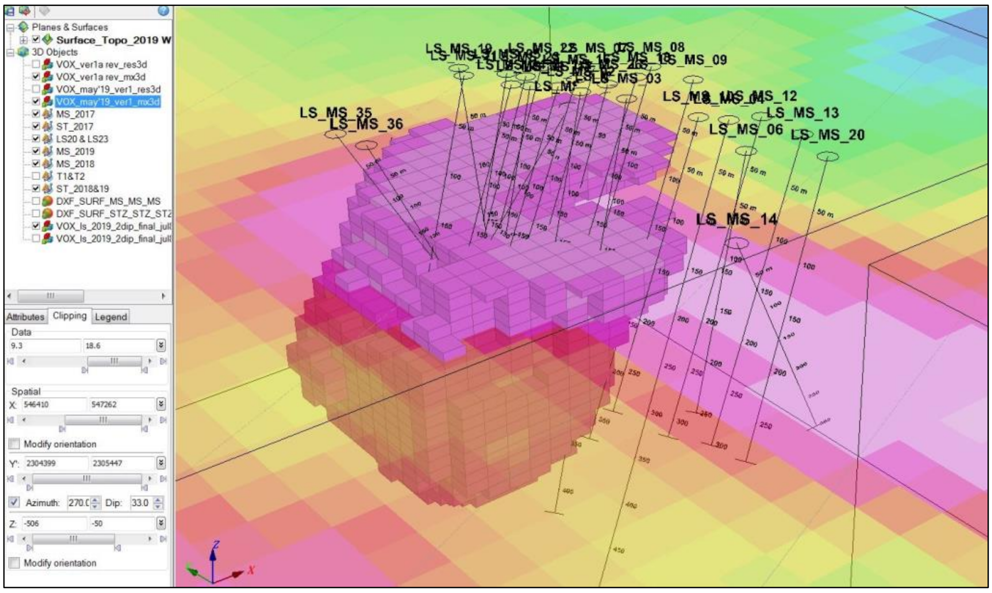
Figure 9-6 shows a closer view of the BHIP model for the North Resource, looking toward the northeast. The top of both the surface IP model and the BHIP has been set to roughly the top of the volcano-sedimentary rocks, about 140 m below the surface. The chargeability determined from BHIP measurements in LS-MS-21 through LS-MS-24 shows that the anomalous chargeability recognized from the surface work extends farther north than the present limit of drilling. The BHIP demonstrated that a continuous volume of MS mineralization extends to the northwest linking LS MS-21 to the mineralization in the LS-MS-22 to 24 section.

Figure 9-5 - BHIP Chargeability Models for N and S Resource



Source: I.E. Intelligent Exploration 2019

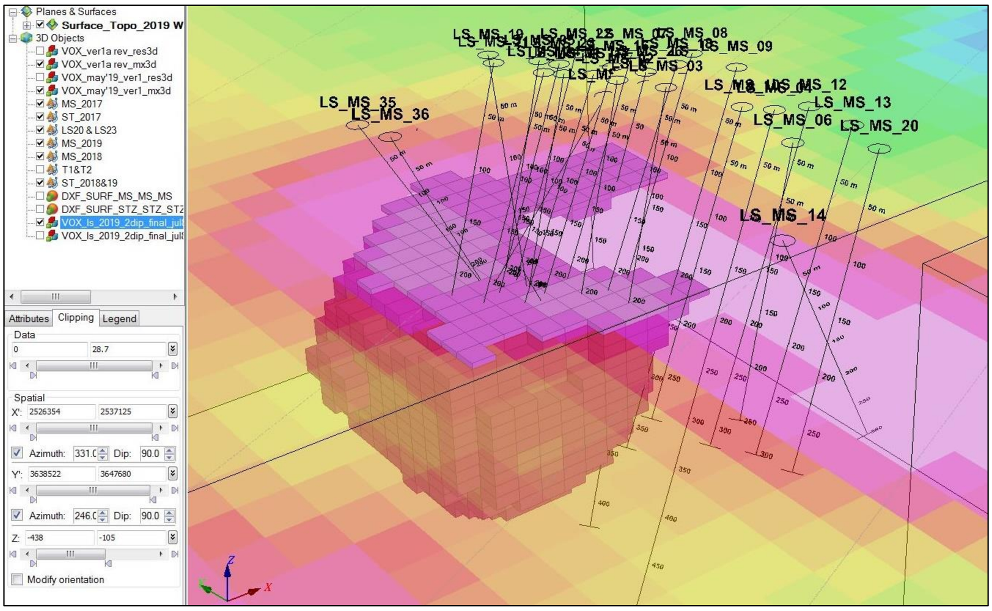
Figure 9-6 - BHIP Chargeability Model for North Resource (looking NE)



Source: I.E. Intelligent Exploration 2019

Figure 9-7 shows the same model as Figure 9-6, but this time the top has been adjusted to present a level plan at -105 m, about 200 m below the surface. At this level, the extension of the mineralization to the northwest of the most recent drilling here (LS-MS-35 and -36) is clear. The extent of this mineralization cannot be known because of the non-uniqueness of the BHIP models but the possibility is suggested that additional mineralization may be drilled with a step-out to the northwest. In the short term it would be useful to refine the BHIP model with measurements from the recent holes in this part of the property.

Figure 9-7 - BHIP Chargeability Model for North Resource (looking NE, 200 m below surface)



Source: I.E. Intelligent Exploration 2019

9.3.7 RESULTS AND CONCLUSIONS

The 2019 geophysical exploration program has clarified the picture offered by the 2018 data with cleaner, more reliable data from both surface and borehole surveys. The principal conclusions to be drawn from this work are the following:

- Both the LS North Resource and LS South Resource appear to be parts of a single band of anomalous chargeability that links them in the surface results. Additional support for the continuity of mineralization is provided by the conductive zone seen in the surface resistivity model and its elevated gravity.
- The chargeability anomaly extends southward from the LS-North Resource, but it appears to be located about 100 m east and 100 m deeper than the North Resource as it is now located. This chargeability may result from either an offset in the LS-North mineralization or the presence of a parallel mineralized structure at this depth, or a non-economic formational source like an underlying sulphidized black shale. There is no way to resolve this uncertainty without drilling a step-out to test this anomaly at depth.
- The main anomaly at the LS South Resource appears to be located to the east and deeper than much of the present drilling. Again, a step-out to the east is necessary to test this anomaly.
- BHIP has been helpful in improving the local definition of the chargeability anomalies, showing that mineralization is linked between the north-western holes in the North Resource. Surface and BHIP models agree well and provide good targets for drill testing.
- Surface IP data indicate several chargeability maxima that are peripheral to the main resource volumes. Where these coincide with low resistivity and elevated gravity, they present good targets for follow-up drilling.

9.3.8 RECOMMENDATIONS FROM THE 2018-2019 GEOPHYSICAL RESULTS

Recommendations pertaining to the North and Central zones, LS West:

- Chargeability, resistivity and gravity data all suggest the possibility of a larger tonnage in the stockwork zone in the central and southern parts of the deposit. Some more aggressive step outs to test the idea of additional tonnage to the east may be useful.
- Recently drilled holes should be surveyed with BHIP, especially those in the Central Zone and extensions of the LS North Resource to the north-west and south-east. A program of 10-15 BHIP holes should be carried out prior to drilling or extending LS North Resource drilling toward the north-west, to optimize future drilling.

Recommendations for the property as a whole:

- Chargeability anomalies are indicated on both the western (LS) and eastern limbs of an apparent anticline indicated on the regional Bouguer gravity map. The anomaly extends from Line 7-9 at the east end of these lines and possibly south to L14 on the eastern limb of the inferred anticline where it has not been drilled. A target could be tested at 4231900N, 548000E, -300 m. This recommendation from the 2018 program is carried forward and validated by the 2019 results.
- Other surface IP / Res chargeability anomalies from the 2019 survey should be considered for follow-up drilling, especially when they are spatially correlated with conductivity and elevated gravity.

9.4 STRUCTURAL MODELLING

Consulting de Geologia y Mineria, S.L. of Spain was contracted by Redcorp in 2018 to produce a 3D structural model of the LS Property deposits. This work is still in progress.

9.5 QP COMMENTS

The QP has reviewed the exploration programs to date and finds that the work conducted is consistent with the work that should be conducted for the mineralization and deposit type that is indicated to be present on the LS Property.

10 DRILLING

10.1 DRILLING SUMMARY

The LS project has been explored by drilling from 1995 to the present. 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 (formerly LS-1 deposit) and the South deposit (formerly LS-1) areas.

Table 10-1 summarizes the drilling on the property and on the LS deposits. Figure 10-1 illustrates the locations of the drillholes on the property and the deposits/rock types intersected

Table 10-1 - Summary of Exploration Drill Programs at the LS Project Since 1995

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
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
Total	**1995 to 2021**	**131**	**51,974**	**HQ**

Of the total, 61 drillholes intersect the North deposit and 24 drillholes intersect the South deposit.

10.2 ASCENDANT / REDCORP DRILLING PROGRAM

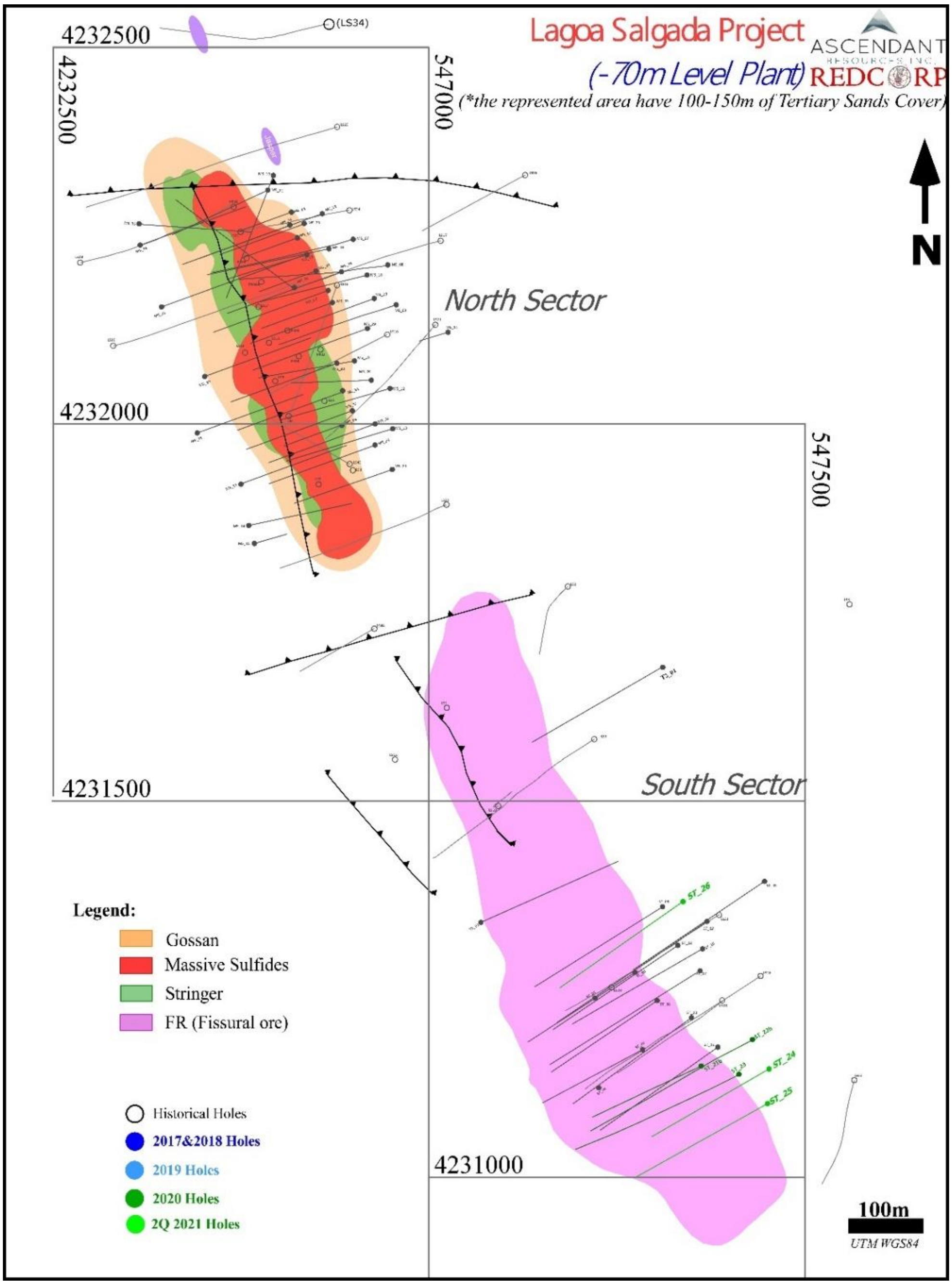
Ascendant / Redcorp have so far conducted two drilling campaigns on the LS Property as shown in Table 10-1 above. In both instances, Drillcon was contracted to undertake the drilling.

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 hole within the Tertiary sedimentary units.

A second drill, comprised of a diamond core drilling rig using HQ size core, was then brought in to continue drilling. Once the drill rods showed signs of stress, the drill core size was reduced to NQ. Most of the core drilling was conducted using HQ.

The drillers used a black marker to label the core boxes and, as the drilling progressed, they also used the black marker to denote the depth of the drillhole on wooden blocks within the core boxes. Once the core boxes were filled, they were transported by Redcorp personnel to the core logging and storage facility. The boxes and metre markers were subsequently retagged by Redcorp with Dymo plastic tags.

Figure 10-1 - Drillhole Location Map



Source: Map provided by Ascendant 2021.

Once the drillhole was completed, an NQ size ABS PVC pipe was inserted down the entire length of the drillhole. This was conducted to prevent the walls of the drillhole from collapsing prior to carrying out downhole geophysical surveys such as the Mise-a-la-Masse surveys.

The drillhole steel collar was retained in-situ and a steel cap was placed on the top of the collars to allow for a hinged cap to cover the drillhole and be locked with a padlock. To keep the drill collars more visible, a 4" blue ABS pipe was used as a collar marker.

10.2.1 Ascendant / Redcorp Core Logging Procedures

The diamond drill core was collected by Redcorp geologists at the drill site and conveyed to the core logging and sampling facility. The drill core was rough logged onto paper logs prior to being transcribed into a Microsoft Excel spreadsheet.

Sampling was conducted in 1.0 m intervals respecting the contacts between different lithologies. The sample tags were inserted into the core box at the beginning of the sample interval.

Lead and zinc standards were inserted roughly every 15 samples within the 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 it as a new sample.

Upon arrival at the core shed, the drill core went through the following steps:

- Core was reassembled in the box and if necessary, cleaned.
- Core was photographed.
- The following information was recorded in a digital spreadsheet:
  - Core recovery.
  - Rock quality designation (RQD).

Geological logging protocols record lithology, structures, alteration, mineralization and oxidation in descriptive columns. Logs are first recorded on paper logging sheets, and later transcribed into a computer database by Redcorp geologists.

10.2.2 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.

Interpreted drill sections of the North and South deposits are shown in Figure 10-2 and Figure 10-3, respectively.

10.2.3 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, as demonstrated in Figure 10-4.

10.2.4 2021 Summary of Drilling Results

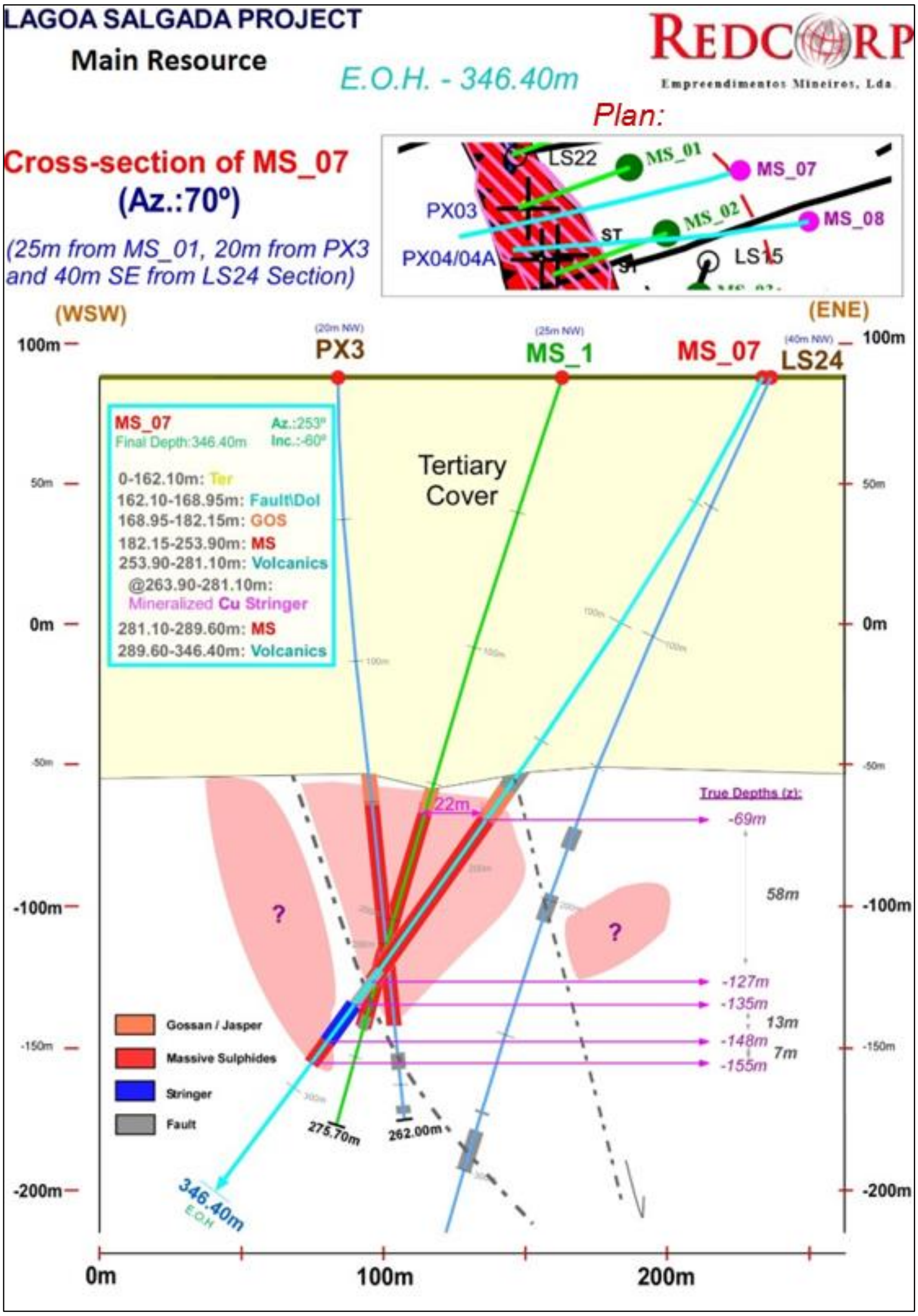
Step-out drilling combined with geological/structural reinterpretation has culminated in merging of the former Central deposit with the southern mineralized envelope to form one continuous deposit, the South deposit, in the South sector in Figure 10-1 above. Critically, drill hole LS 27 confirmed the linkage.

10.3 QP Comments

The drilling results summarized above and, in the sections, (Figure 10-2, Figure 10-3 and Figure 10-4) below demonstrate that the drilling campaigns have progressively yielded encouraging results. However, down dip and lateral extensions remain to be fully tested for each of the two deposits.

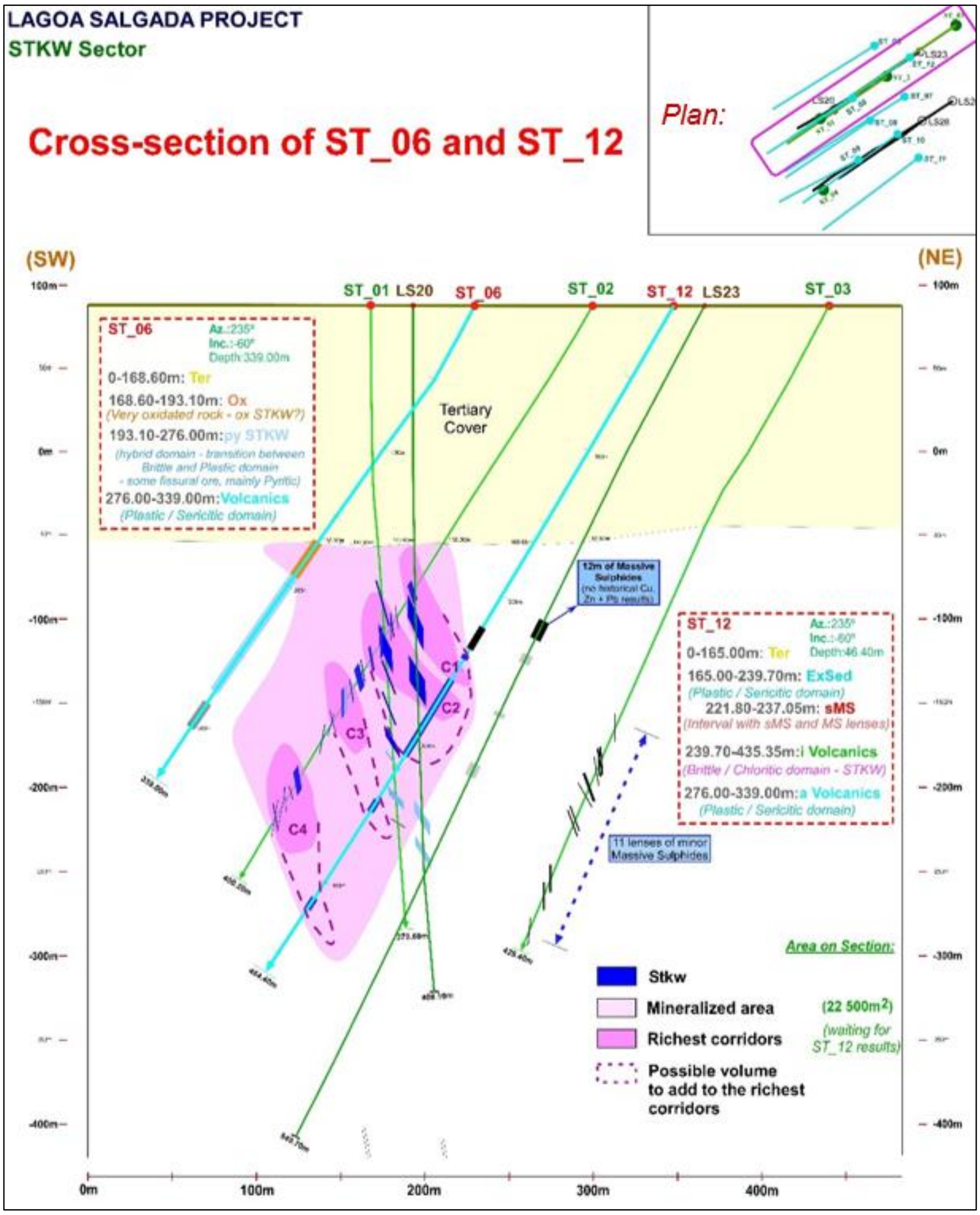
Redcorp's drilling and sampling protocols are in line with the CIM best practice guidelines. Core recoveries beneath the overburden are excellent (+95%) and this ensures good quality samples. The restriction of sample intervals to lithological and mineralization boundaries yields a representativeness of the mineralization types encountered and facilitates geological modelling of the deposits. The QP has not identified any drilling, sampling or recovery factors that could result in sampling bias or otherwise materially impact the accuracy and reliability of the assays and, hence, the resource database.

Figure 10-2 - Drill Section through the Venda Nova North Deposit



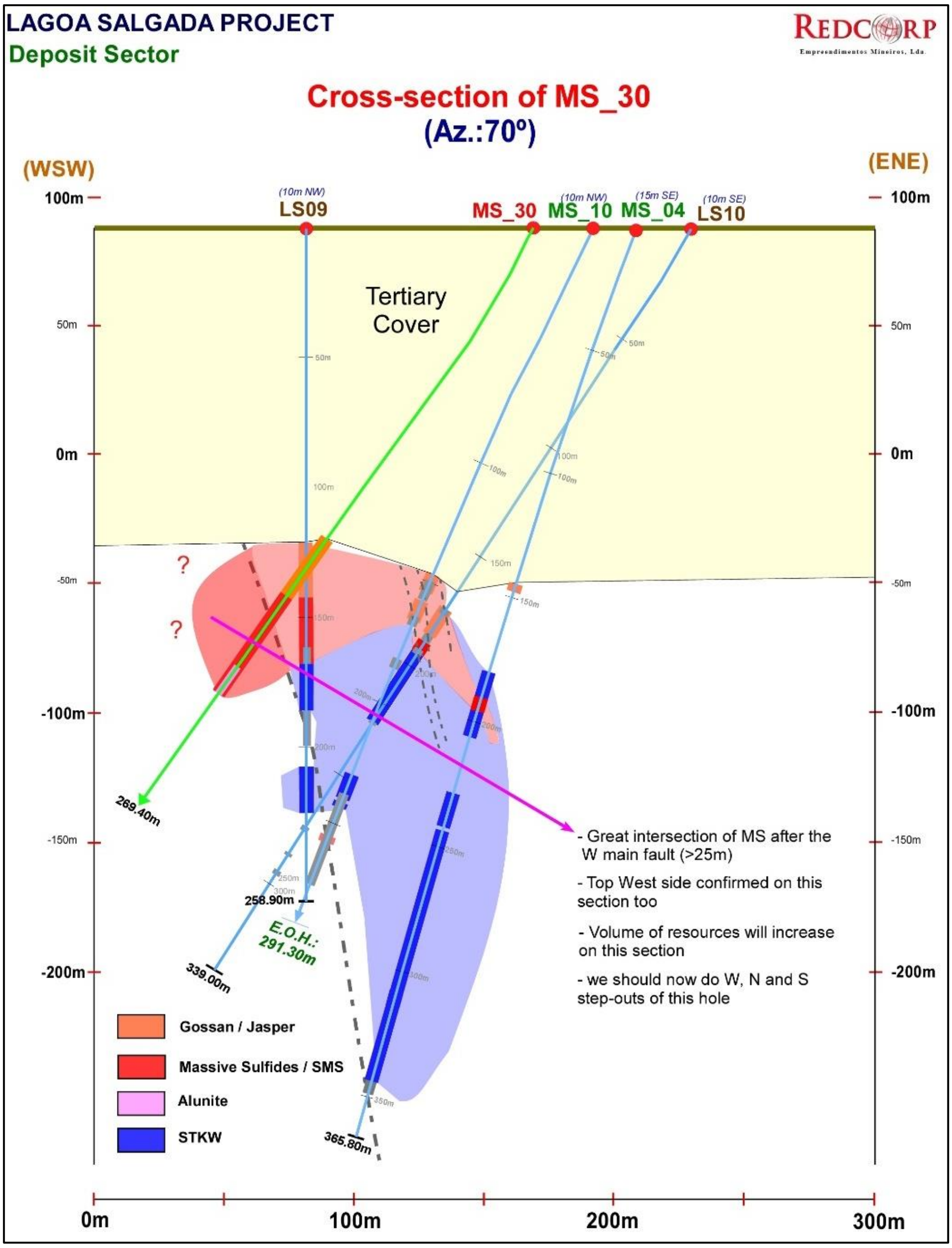
Source: Ascendant 2019/2020.

Figure 10-3 - Drill Section through the Venda Nova South Deposit



Source: Ascendant 2019/2020.

Figure 10-4 - Section Demonstrating the Effects of Infill Drillhole MS_30



Source: Ascendant 2019/2020.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 SAMPLE PREPARATION AND ANALYSES

11.1.1 SAMPLE PREPARATION AT SITE

Drill core in core trays is 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 samples are taken at 1 m intervals terminated 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 the drillhole is sampled. A tag with the sample identification (ID) number is placed in each sample bag before being 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 into 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 15th sample through the series and field duplicates every 40th sample. Two blanks are also placed in every assay batch.

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 rice bags are dispatched to the ALS (Seville) laboratory via 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 analyzing laboratory (ALS Sudbury) 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 analyzed 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).

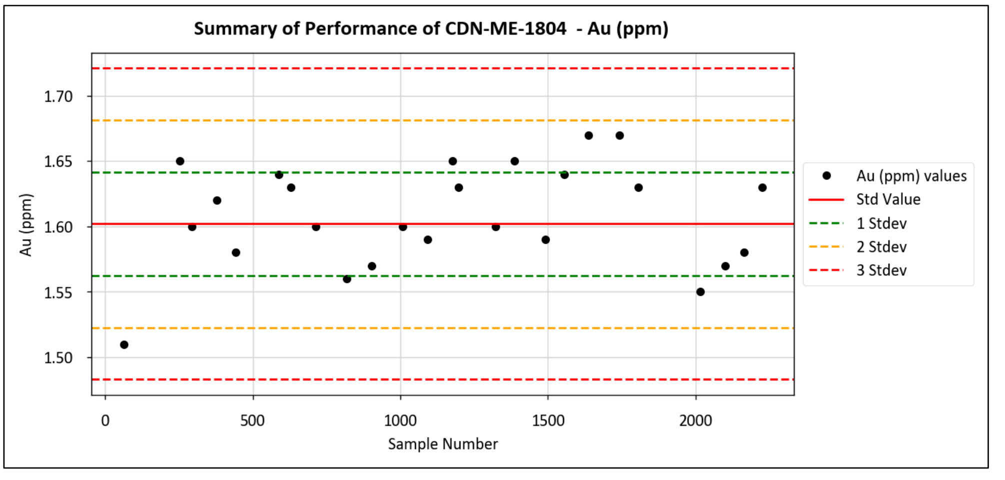
11.2 Bulk Density

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.

11.3 Quality Control Results

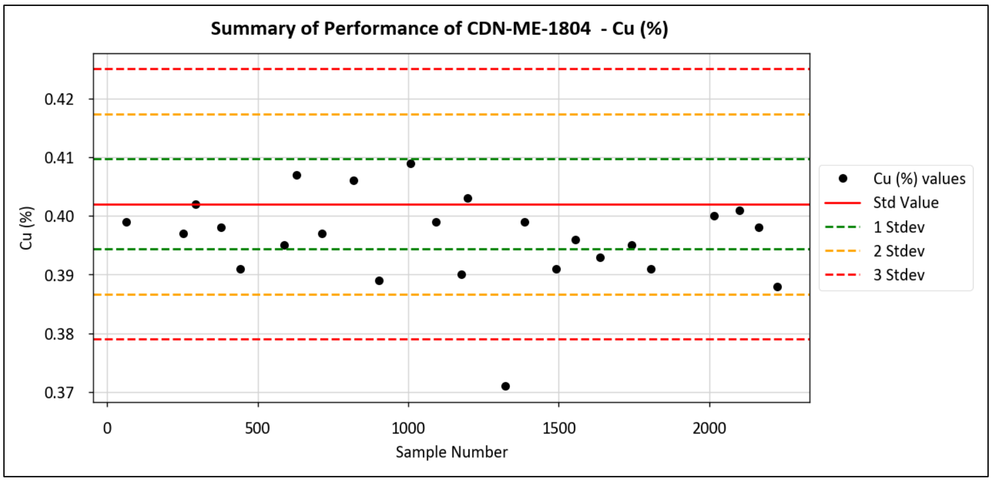
All assays are reported directly to Redcorp via e-mail to designated personnel. Signed assay certificates are sent via courier or post. The monitoring of the performance of the QA/QC samples is conducted immediately after the assay results are received. The assay results for control samples were plotted 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-analyzed. 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. As examples the performance of CRM CDN-ME-1804 is demonstrated in Figure 11-1, Figure 11-2 and Figure 11-3.

Figure 11-1 - Summary of Performance of CRM CDN-ME-1804: Au ppm



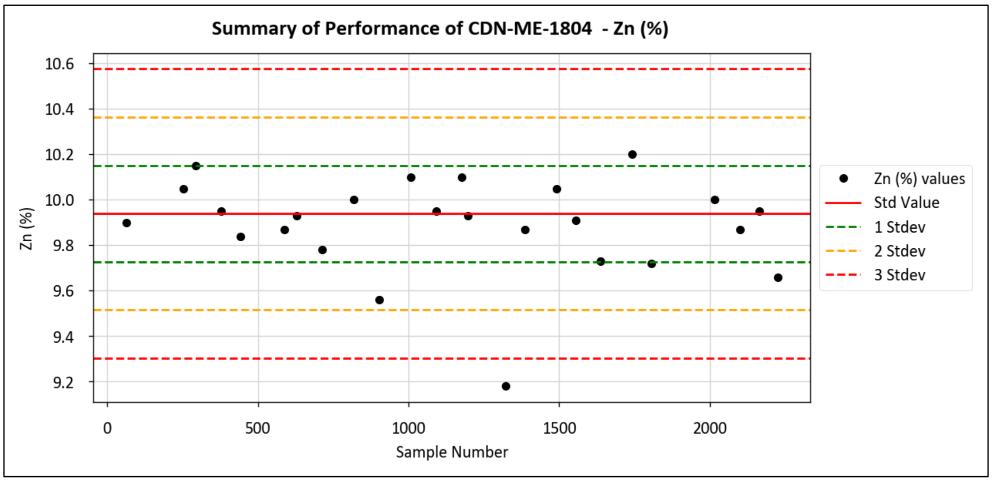
Source: Micon 2019.

Figure 11-2 - Summary of Performance of CRM CDN-ME-1804: Cu (%)



Source: Micon 2019.

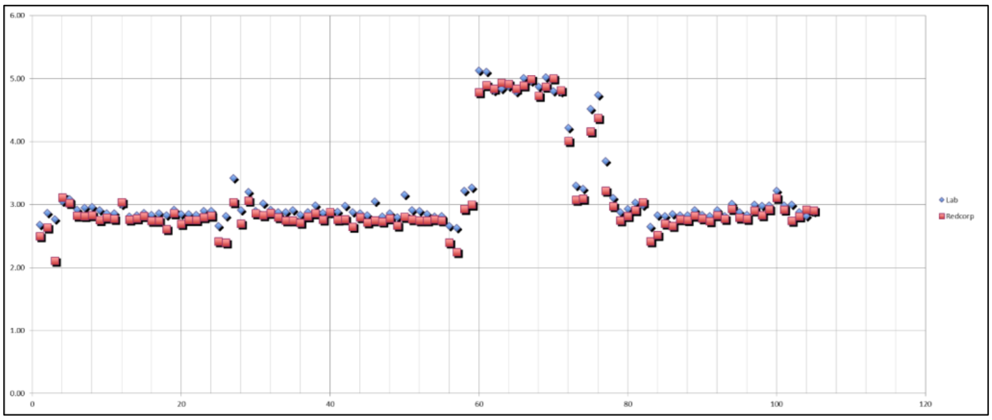
Figure 11-3 - Summary of Performance of CRM CDN-ME-1804: Zn (%)



Source: Micon 2019.

Bulk density checks by the laboratory showed that the in-house determinations were marginally lower, as shown in Figure 11-4 below.

Figure 11-4 - Redcorp Bulk Density vs ALS Laboratory Bulk Density



Notes: Blue = ALS; Red = Redcorp.
Source: Redcorp 2019/2020.

The final bulk density used for each lithology / domain in estimating the Mineral Resource tonnages was taken as the average of the ALS Laboratory determinations.

11.4 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 to verify the data in this Technical Report include three site visits to the Property, analyzing 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 Nunes, MIMMM, visited the LS property on the 23 November 2020. The Redcorp staff in attendance were João Barros (Redcorp Managing Director) and Vítor Arezes (Senior Project Geologist). A second visit was held by Quadrante Project Director, João Nunes, MIMMM and Quadrante Project Director João Horta, MIMMM on the 20 August 2021. The Redcorp attendant was Vitor Arezes (Senior Project Geologist).

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

12.1.1 DISCUSSIONS ON GEOLOGICAL ATTRIBUTES

Discussions held with Redcorp staff centred on the genetic model / attributes of the LS project deposits, including mineralization trends and the role of structures and lithology.

The general consensus is that the subdivision of the LS project into the North and South deposits is arbitrary, being based on the existing drill pattern. All two deposits coalesce into a single zinc-rich VMS system manifesting / displaying its macro-genetic features from secondary GO to primary massive (MS) and primary / secondary stringer / fissure type mineralization in the waning phases of volcanic activity. This interpretation is supported by geophysics which shows that all zones lie on a continuous coincidental IP chargeability anomaly with an estimated geological strike length of 1.7 km in an SSE to NNW direction from the South deposit to beyond the North deposit and terminating against the Alpine fault. The MS intersections observed in drillholes LS 23 and LS-ST 12 on the eastern side of the South deposit suggest the possibility of another volcanic vent.

The overall controlling structure and continuity of the mineralization follow a linear trend in a north westerly direction over about 1.7 km. Both lithological and structural control appears to be significant, with the mineralization exhibiting both global and local trends.

Micon has incorporated these attributes in the modelling of the deposits

12.1.2 FIELD EXAMINATION OF PROJECT AREA AND DRILLING

The LS property was visited to examine the landscape features and diamond drilling techniques, including down-the-hole surveys. Observations on the ground confirm a monotonous flat topography that conforms to the digital terrain model (DTM) provided by Redcorp with the database. Thick sequences of alluvium necessitate 4-wheel drive vehicles in wet conditions.

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 witnessed some of the downhole measurements being conducted and is satisfied that industry standards were upheld. In addition, Micon checked the calibration of the downhole survey instrument and found it to be in good standing as evidenced in Figure 12-1.

12.1.3 EXAMINATION OF DRILL CORES

Most of the drilling on the LS Property 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 DATA COLLECTION TECHNIQUES

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. The protocols are summarized in Figure 12-2.

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

Figure 12-1 - Calibration Details for Reflex Ez-Sot used at the LS Property

REFLEX®
intelligence on demand

Unit 4/5, Upper Stalls
IFORD, Nr Lewes
East Sussex, BN73EJ
UK
+44(0) 1273 483700
www.reflexnow.com

CERTIFICATE OF CALIBRATION

Product Information

Product Type: **Reflex EZ-Shot** Product SN: **0592**

Calibration Information

Calibrated for and on behalf of: **Reflex Instruments Europe Lt**

Issued on: **13 December 2017**

Reference Magnetometer Serial Number: **12EJB121772**

Calibration Test Data

Tool Serial Number	0592
Accelerometer S/N (X,Y,Z)	2018116, 2018115, 2018117
Calibration Temperature	13°C

OFFSET	X	Y	Z
ACCELS	3.85	-117.90	4.20
MAGS	-8.89	-16.51	1.70

Error Levels

ACCELS	RMS DIP ERROR	RMS ROLL ERROR
	0.03° ≤ 0.25°	**0.04°** ≤ 0.25°

MAGS	MAX DIR ERROR	MAX M% ERROR
	0.15° ≤ 0.25°	**0.15%** ≤ 0.25%

Confidence Check

Position 1

Roll	Total [nT]	AZI [°]	DIP [°]
Ref.	49890	168.7	26.6
0°	49860	169.0	26.7
90°	49870	168.8	26.6
180°	49820	169.0	26.7
270°	49870	168.9	26.7

Position 2

Roll	Total [nT]	AZI [°]	DIP [°]
Ref.	48090	99.7	39.4
0°	48120	99.3	39.5
90°	48090	99.3	39.5
180°	48090	99.3	39.5
270°	48120	99.3	39.4

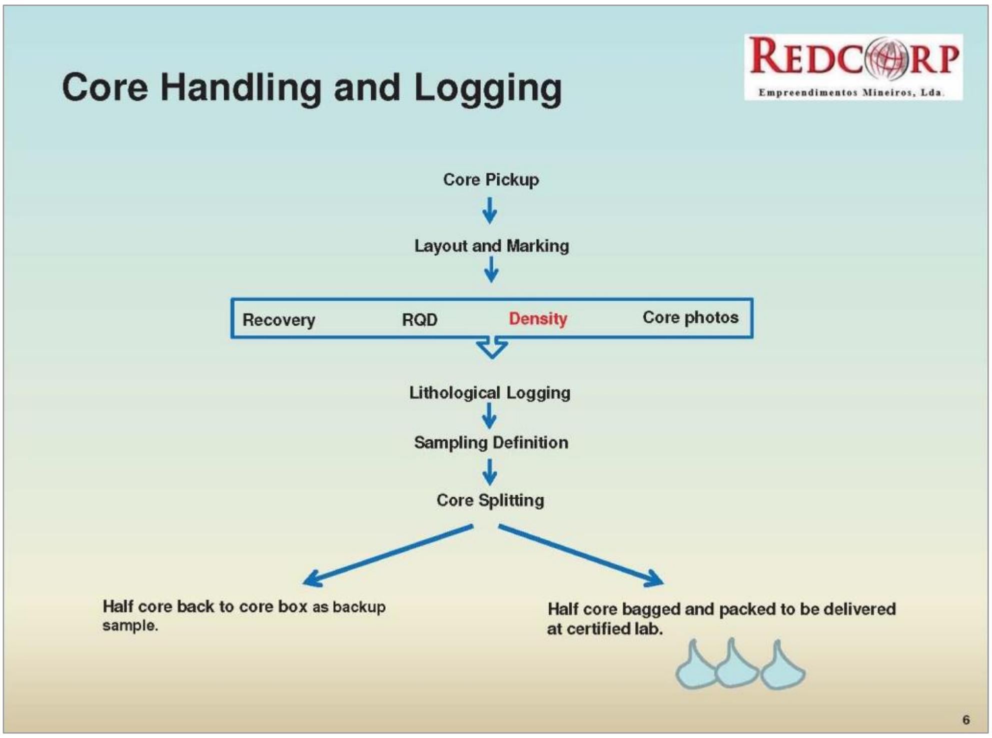
Recalibration recommended no later than: **13 December 2019**

CAL
IMD 21

FQC

Reflex Instruments Europe Ltd certifies that this instrument has been calibrated in accordance with REFLEX Quality System procedures and conforms to product performance specifications. This certificate provides traceability of measurement to recognized national standards or to accepted values of natural physical constants.

Figure 12-2 - Redcorp Protocols for Core Handling and Logging



12.2 Analysis of QA/QC Protocols

Redcorp QA/QC protocols conform to CIM best practice guidelines. The monitoring of the laboratory's performance was conducted on a real time basis and ensured that corrective measures, where needed, were taken at the relevant time, giving confidence in the validity of the assay data.

12.3 Bulk Density

Bulk density measurements were conducted at site by Redcorp technicians using the Archimedes principal technique. Validation of bulk density measurements was conducted by the ALS Laboratory.

Micon reviewed the measurement procedure and found it to be acceptable. However, the in-house site measurements are slightly lower than ALS Laboratory measurements. For the mineral resource tonnages, Micon adopted the average values as determined by the ALS Laboratory, as seen in Figure 12-1.

Table 12-1 - Summary of ALS Laboratory Bulk Density Measurements

Domain	No. of Samples	Average Density
GO_N	100	3.12
MS_N	70	4.76
Str_N	150	2.88
Str_S	165	3

12.4 Database Validation

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 in Micon's Toronto offices, and 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 major errors were found.

12.5 Data Verification Conclusions

The QP has not found any issues with Redcorp's data collection techniques and QA/QC protocols. Based on the verification procedures described above, 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

The metallurgical testwork described in this Section was prepared by Grinding Solutions (GSL). This program of work was managed directly by Ascendant Resources.

Micon has reviewed the testwork report from which the testwork described below was copied and has prepared the conclusions and recommendation of this chapter.

The testwork program described below was completed in 2021 by GSL. Previous phases of testwork are not included although a summary of the historical studies can be found in the NI 43-101 Technical Report by Micon entitled "Updated Mineral Resource Estimate for the South Deposit and PEA for the North Deposit, Lagoa Salgada Property, Setúbal District, Portugal", dated March 2021.

13.1 INTRODUCTION

Several scoping level metallurgical and mineralogical studies have been undertaken for the LS Project by Grinding Solutions (GSL), a metallurgical testing laboratory located in Cornwall, UK.

The aim of the studies was to confirm results previously achieved in Grin Job No 19-1608 on samples with head grades more representative of potential ROM material.

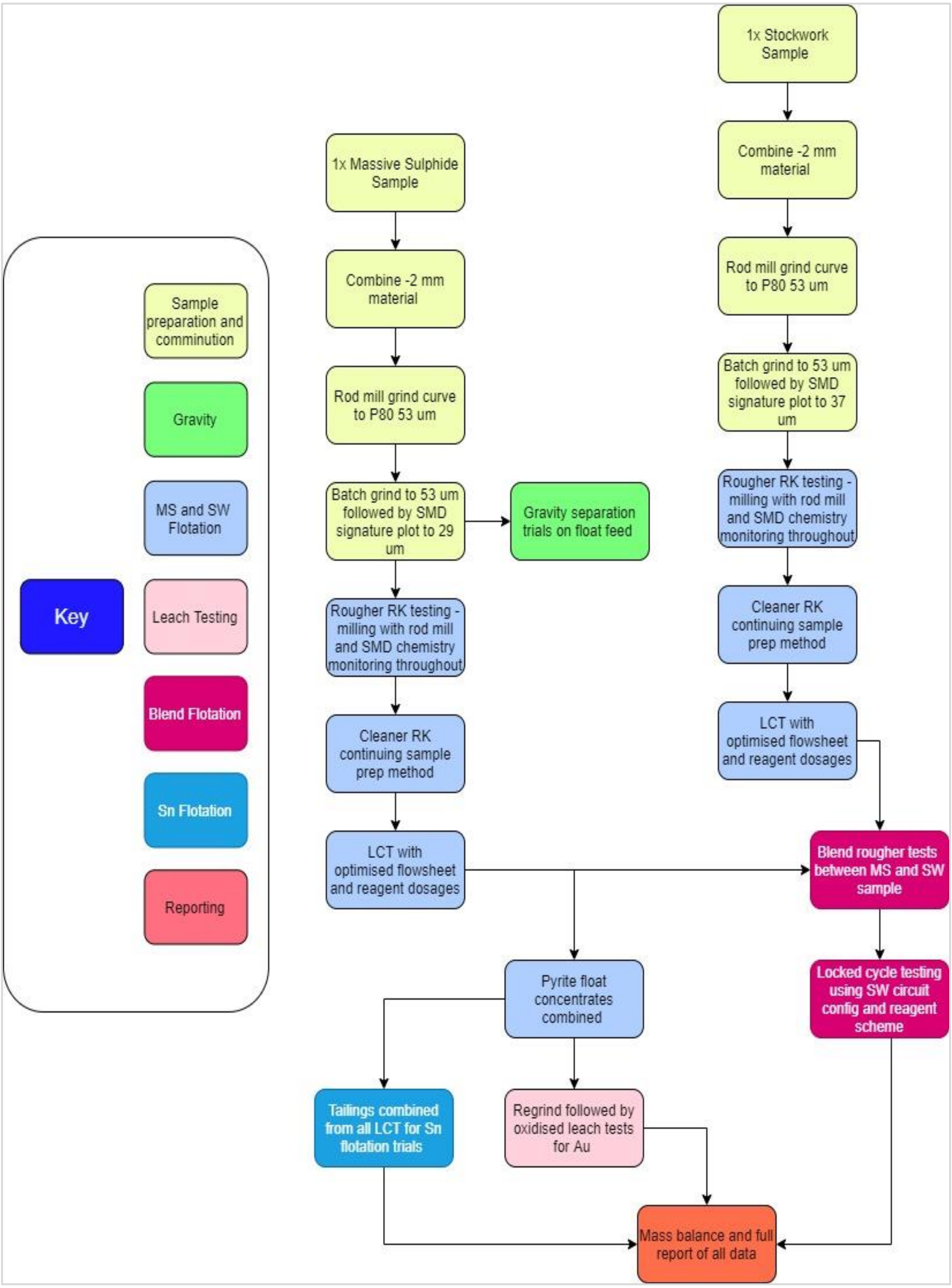
The -2 mm crushed samples selected by Ascendant Resources for testing had been stored without refrigeration for approximately two years and, there was concern that they may have oxidised which would have a significant detrimental effect on the metallurgy. As such, preliminary testing was conducted on samples from the two metallurgical composites, massive sulphide (MS) and stockwork (SW), to assess the extent of the oxidation.

The main testing that followed the preliminary assessment of the samples was split into 6 phases

- Feed characterisation and sample preparation.
  - Head assay.
  - Comminution (rod mill grind curve and SMD Signature plot) to target primary grind sizes.
- Gravity separation trials on Massive Sulphide (MS) sample.
- Flotation evaluation on both MS and Stockwork (SW) samples.
  - Scoping trials.
  - Cleaner float tests.
  - Locked Cycle Testing.
- Leach trials on a bulk pyrite concentrate on the MS.
- Evaluation of potential tin (Sn) extraction post pyrite concentrate.
- Flotation testing to evaluate the potential for processing a blend of MS and SW.

The main work programme is shown in Figure 13-1.

Figure 13-1 - Work programme



13.2 Sample Characterization

A total of 289 kg of crushed (-2 mm) massive sulphide (MS) mineralization consisting of 21 samples and 282 kg of crushed stockwork (SW) mineralization comprising over 38 samples was received by GSL for the test work program. The MS samples were combined into a single composite, cone and quartered and riffled into 1 kg charges for subsequent testing, as were the SW samples. The samples for the test work program were selected by Ascendant Resources with the objective of preparing samples that were representative of the mineral resources metal grades. The crushed assay reject samples selected by Ascendant Resources originated from the 2019 /2020 drilling of the South Zone.

A comparison of the average composite sample grades received from Ascendent Resources and the mineral resources is presented in the table below.

Table 13-1 – Average metallurgical composite grades

Sample	Mass kg	Average Grade					
		Cu%	Pb%	Zn%	Sn%	Au g/t	Ag g/t
MS Composite	285	0.26	4.54	4.29	0.20	0.84	104.1
SW Composite	281	0.44	1.14	1.84	0.02	0.07	17.4
North M&I	-	0.37	2.39	2.12	0.16	0.64	64
South M&I	-	0.42	0.87	1.55	-	0.06	18

13.2.1 Deleterious Elements

Detailed multi-element chemical analyses of the two composite samples were not available at the time of issuing this report. However, it is likely that minerals containing undesirable elements such as arsenic, antimony, and mercury would be present within the sulphide mineralization.

13.2.2 Mineralogy

No mineralogical testwork was undertaken in the 2021 metallurgical study. However, historical work described in the March 2021 NI 43-101 Technical Report noted the following:

"The dominant phase in the MS sample was pyrite with minor amounts of target minerals sphalerite, galena, and traces of chalcopyrite, cassiterite, tetrahedrite, and secondary copper sulphides. Besides pyrite, the main gangue minerals were minor arsenopyrite and quartz with trace gangue phases comprising carbonates, micas, and feldspars.

The dominant phase in the Stockwork sample was mica group minerals. The main ore economic minerals identified were sphalerite, galena, chalcopyrite, and trace secondary copper sulphides. There was only an ultra-trace of cassiterite. Besides micas, the main gangue minerals were major to mirror iron oxides, quartz, pyrite, and carbonates with trace accessory phases."

13.2.3 Preliminary Testing - Sample Oxidation Evaluation

Due to the time the samples had been kept at -2mm without refrigeration, testing was carried out prior to the main testwork programme to determine the extent of the oxidation.

The sample was selected at random by Ascendant Resources from the total bulk and the results should of the testing should be considered indicative rather than fully representative of the total MS and SW sample. The sample head grade was different from both the bulk samples received for the main test programme and from previous test campaigns and as such, this should be considered when reviewing the results.

The process for oxidation evaluation was:

- Sample milling and evaluation of the pH of the product.
- EDTA extraction carried out on milled products.
- Up front comparative floats comparing to previous testing.

13.2.3.1 Milling Tests

The samples were milled in a lab rod mill until a grind size of nominally P_{80} 25 µm for the MS and 35 µm for the SW were achieved. Immediately following milling the pH of the product was measured, with the SW recording pH 7.94 and the MS pH 6.6. Compared to previous testing by GSL on the same deposit, the slurry was more acidic than would be expected.

As the SW sample contains carbonate material, a pH of nominally 10-11 would be expected naturally. The decrease to pH 7.94 would suggest the sulphides have broken down, resulting in free H^+ ions. The larger decrease in pH of the MS sample is likely due to increased levels of sulphide material in the sample compared to the SW.

13.2.3.2 EDTA Extraction

A sub sample of the milled product pulp was taken for EDTA extraction with the residue and solution assayed. Table 13-2 shows the results of Cu, Pb, Zn and Fe for the MS and indicates that the galena has oxidised. An increase in the number of ions in solution could potentially activate other depressed sulphide minerals resulting in poorer selectivity.

Table 13-2 - MS EDTA extraction data

Sample	Mass (%)	Grade (g/t)				Deportment			
		Cu	Pb	Zn	Fe	Cu	Pb	Zn	Fe
EDTA Solids MS	4%	1900	15200	73000	389500	97%	34%	97%	99%
EDTA Solution MS	96%	2.16	1234	104.3	106.1	3%	66%	3%	1%

A similar trend is observed with the SW sample in Table 13-3 albeit with a lower deportment (33% compared to 66%). The difference is likely due to the lower Pb grade and naturally higher pH of the mill pulp from the carbonate material.

Table 13-3 - SW EDTA extraction data

Sample	Mass (%)	Grade (g/t)				Deportment			
		Cu	**Pb**	**Zn**	**Fe**	**Cu**	**Pb**	**Zn**	**Fe**
EDTA Solids SW	4%	7600	20300	36700	130600	97%	67%	99%	99%
EDTA Solution SW	96%	11.42	410.4	12.08	33.35	3%	33%	1%	1%

13.2.3.3 Comparative Flotation

The flowsheet and reagent scheme established in previous testing (GSL job 19-1604) was used as the basis for comparative flotation tests.

From the results of the MS sample (Table 13-4) it can be seen that there were significant differences in flotation response. Previous testing achieved a 27% total mass pull compared to a 94% mass pull in the repeat, indicating that little to no selectivity was achieved and that the pyrite (representing 90% of the mineralogy) was collected throughout. This is a strong indication that achieving successful and equivalent selectivity would be very difficult due to the activation of minerals. As the sample had a higher head grade than previous work a longer residence time is normally required to achieve equivalent recovery.

Table 13-4 - MS comparative float data with previous work

19-1608 Original Data							
Stream	**Mass %**	**Assay (%)**			**Recovery**		
		Cu	**Pb**	**Zn**	**Cu**	**Pb**	**Zn**
Pb Ro Conc	12%	0.52	13.31	4.46	17%	50%	18%
Zn Ro Conc	16%	1.04	4.41	14.31	46%	22%	76%
Zn Ro Tail	73%	0.18	1.23	0.26	37%	28%	6%
Back Calc Head		0.36	3.15	2.95			
22-1936 Confirmation Float							
Stream	**Mass %**	**Assay (%)**			**Recovery**		
		Cu	**Pb**	**Zn**	**Cu**	**Pb**	**Zn**
Pb Ro Conc	48%	0.32	7.30	5.28	66%	74%	46%
Zn Ro Conc	46%	0.16	1.64	3.98	30%	21%	53%
Zn Ro Tail	6%	0.12	3.71	0.92	3%	5%	1%
Back Calc Head		0.23	4.70	5.51			

Table 13-5 shows the SW comparative float data. Although the differences were less than the MS, selectivity was reduced with large losses of Pb to the Cu and losses of Zn to the Pb. This suggests that, as with the MS, the minerals were highly activated and not responding to the depressants. The data however did suggest seen that improvements could be made with kinetic and dosage changes to control the loses to the different concentrate streams.

Table 13-5 - SW comparative float data with previous work

19-1608 Original Data							
Stream	**Mass %**	**Assay (%)**			**Recovery**		
		Cu	**Pb**	**Zn**	**Cu**	**Pb**	**Zn**
Cu Ro Con	13%	10.15	13.88	12.12	84%	38%	24%
Pb Ro Con	11%	1.51	23.46	13.28	10%	53%	21%
Zn Ro Conc	12%	0.44	1.54	25.34	3%	4%	46%
Zn Ro Tails	64%	0.05	0.38	0.99	2%	5%	9%
Back Calc Head		1.59	4.80	6.76			
22-1936 Confirmation Float							
Stream	**Mass %**	**Assay (%)**			**Recovery**		
		Cu	**Pb**	**Zn**	**Cu**	**Pb**	**Zn**
Cu Ro Con	10%	0.55	3.81	9.94	86%	88%	39%
Pb Ro Con	29%	0.09	0.34	2.80	6%	8%	39%
Zn Ro Conc	8%	0.17	0.14	2.75	3%	1%	11%
Zn Ro Tails	52%	0.04	0.07	0.43	5%	3%	11%
Back Calc Head		0.44	1.19	2.10			

13.2.3.4 Conclusions

Based on testing outlined above, GSL advised that the samples were oxidised and likely not suitable for the full float programme as optimal grades and recoveries would be difficult to achieve. Fresh feed was requested. Ascendant advised that this was not possible and requested that testing proceed with the samples designated.

13.3 Main Programme of Testing

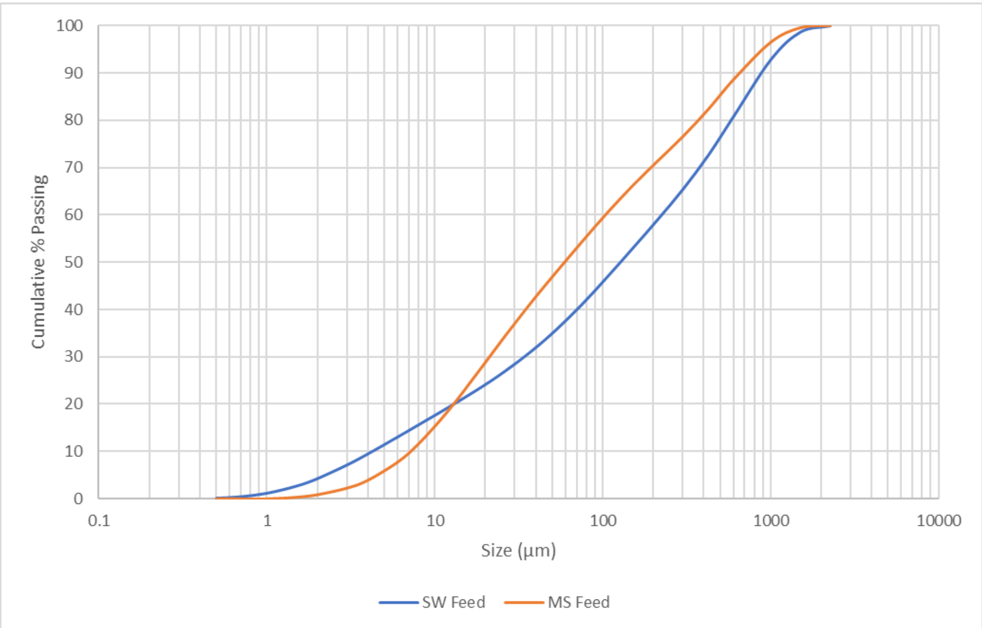
13.3.1 Testing Sample

A total of 289 kg of MS was received over 21 samples and 282 kg of SW over 38 samples. The 21 MS samples were combined, cone and quartered and riffled into 1 kg charges for subsequent testing, as were the SW samples. Table 13-6 and Figure 13-2 - Cumulative % passing for feed PSDs show the size distribution and F-percentiles for the two as received feeds.

Table 13-6- F-percentiles for the two feed PSDs

Sample Name	F-percentiles (µm)								
	F_{10}	F_{20}	F_{50}	F_{60}	F_{75}	F_{80}	F_{90}	F_{95}	F_{97}
SW Feed	4.25	13.0	127	223	470	579	872	1120	1280
MS Feed	7.12	12.9	58.9	103	269	369	646	883	1040

Figure 13-2 - Cumulative % passing for feed PSDs



13.3.2 Experimental Methodology

13.3.2.1 Assays

Two methods were used for assay through the project. A portable X-Ray Fluorescence (pXRF) analyser was used for all products with select products submitted for wet chemical assay with the main suite being:

Cu, Pb, Zn, Fe, As, S, Sn, Au, Ag, Hg, Sb

All pXRF measurements were calibrated against wet chemical data.

13.3.2.2 Sample Preparation

Primary milling was conducted using a laboratory rod. Grind curves was carried out at 60% w/w solids to determine grind times to achieve a target initially of 53 µm for both samples and subsequently 29 µm for MS and 37 µm for SW samples. All sizing was carried out by laser diffraction using a Malvern Mastersizer 3000.

13.3.2.3 SMD Signature Plot

SMD (Stirred Media Detritor) signature plot testing was conducted to establish energy requirements to mill from 53 µm to the finer grind sizes. The method is as follows:

- Tests conducted at 55% w/w solids with 50:50 ratio of media and slurry.
- Kings 3mm 3.8 S.G. grinding media was used, with the size of media selected to mill the sample most efficiently at the feed particle size.
- Multiple samples were taken at different specific energies ranging for 2.5 kWh/t to 30 kWh/t and particle size measured by laser diffraction (Malvern Mastersizer 3000).
- Energy vs size reduction plotted to interpolate a specific energy requirement.

13.3.2.4 Gravity Testing

Two different gravity testing methods were used to evaluate the MS material amenability to gravity separation. A rougher stage separation was simulated using a Falcon L40 separator (Figure 13-3). 1 kg charges at a feed size of 53 µm were used, with 3 feed flowrates assessed and the Falcon speed constant at 60 Hz. Products were dried and assayed.

Figure 13-3 - Falcon L40 separator

Rougher concentrates were cleaned using a Mozley Superpanner (Figure 13-4), with multiple concentrate samples and a single tailings sample collected, dried and assayed.

Figure 13-4 - Mozley Superpanner

13.3.2.5 Flotation

All flotation tests were conducted using a Denver D12 flotation cell (Figure 13-5).

Figure 13-5 - Denver D12 laboratory flotation machine

Rougher Scoping

For each rougher flotation test the following procedure was followed:

- 1 kg feed charge.
- Milled to target grind size in rod mill with required reagents at 60% solids w/w.
- pH, DO and mV recorded throughout test.
- Split concentrates taken to determine kinetic change through test.
- Tailings, and concentrates assayed using pXRF.
- Select products sent for wet chemistry assay for Cu, Pb, Zn, Fe, As, S, Sn, Au, Ag, Hg, Sb.

Cleaner Tests

For each cleaner floatation test the following procedure was followed:

- 1 kg feed charge.
- Milled to target grind size in rod mill with required reagents at 60% solids w/w.

- pH, DO and mV recorded throughout test.
- Bulk rougher floats based on rougher scoping tests.
- Regrinds carried out in benchtop SMD with particle measured using a Malvern Mastersizer 3000.
- Tailings, and concentrates assayed using pXRF.
- Select products sent for wet chemistry assay for Cu, Pb, Zn, Fe, As, S, Sn, Au, Ag, Hg, Sb.

LOCKED CYCLE TESTING

For each locked cycle floatation test the following procedure was followed:

- Circuit determined from scoping rougher and cleaner testing.
- 1 kg feed charge for each cycle.
- Milled to target grind size in rod mill with required reagents at 60% solids w/w.
- pH, DO and mV recorded throughout all cycles for both rougher and cleaner stages.
- All exit streams filtered, dried and assayed using pXRF throughout test.
- All re-circulating product streams filtered with reagent water retained to be used for dilution where necessary.
- On final cycle all re-circulation streams filtered dried and assayed using pXRF.
- On completion, all collected samples sent for wet chemistry assay for Cu, Pb, Zn, Fe, As, S, Sn, Au, Ag, Hg, Sb.
- On samples with sufficient mass, extended full analysis was also carried out final products to evaluate all potential penalty elements.

13.3.2.6 OXYGENATED LEACH TESTING

To generate feed for the oxygenated leach tests a 2kg bulk float will be carried out on a MS sample. Once generated the sample was split into 300g sub-samples with 2 reground in a SMD. The target grind sizes being as produced, 15µm and 5µm.

Three tests were carried out with O_2 pumped through at 1L/min with the other reagents used being Lime for pH control (maintained between 10.5 and 11 as to not produce HCN) and NaCN at 10g/L to leach the remaining Au and Ag. The bottle rolls were left for 48 hours with solution sub-samples at 2,4,6,24 and 48 hours and assayed for Au and Ag. The residue from the test was also assayed for Au and Ag.

13.3.3 Results

13.3.3.1 Comminution

Grinding Curves

Figure 13-6 and Table 13-7 show the grind time plot and the grind time model respectively for the two rods mill grind curves on the MS and SW. The SW required a longer grind time / higher energy input than the MS sample and it is likely that the carbonate clays present in the SW are responsible.

Figure 13-6 - Rod mill grind curve model for MS and SW

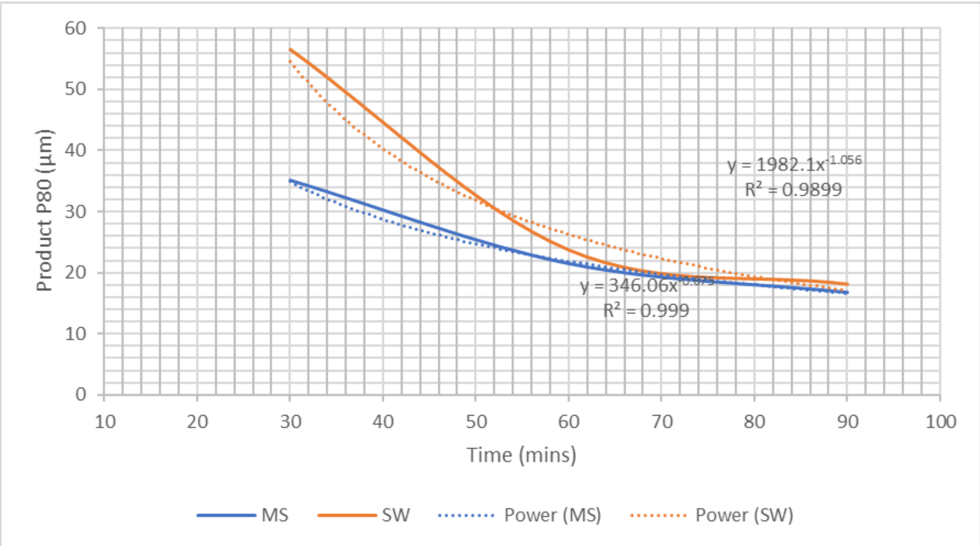


Table 13-7 - Grind time model for rod mill grind curves

Grind Time Model				
	a	**b**	**29µm**	**37µm**
MS	345.06	-0.675	39.2 mins	24.4 mins
SW	1982.1	-1.056	54.6 mins	40.3 mins

SMD Signature Plots

Figure 13-7 and Table 13-8 show the signature plot and power model for the SMD grind on the MS. The given energy requirement for the P_{80} 29 µm target from a F_{80} 53 µm feed was 6.84 kWh/t. This value is equivalent to the Ecs for similar deposits in the region. Generally, the high bearing pyrite ore bodies in the region require an increased energy requirement for ball milling and rod milling, relying on the impact to break down the particle size. The attrition mechanism for milling in the SMD requires less energy.

Figure 13-7 - Signature plot for MS

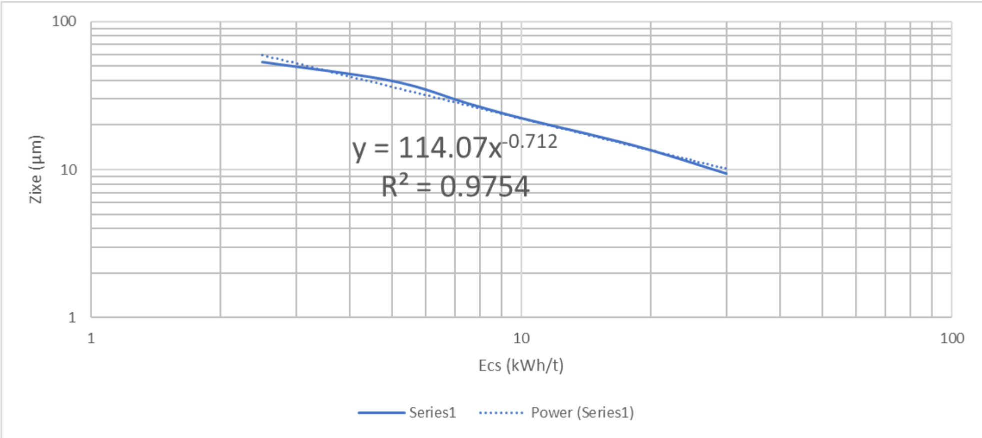


Table 13-8 - Power model for MS SMD signature plot

Equation Variables MS			Calculated Specific Energy (Power Model)	
Equation	**a**	**b**	**29µm**	**37µm**
y = a*x^b	114.07	-0.712	6.8 kWh/t	4.9 kWh/t

The SW signature plot and power model show an energy requirement of 2.06 kWh/t to achieve the target grind size of P_{80} 37 µm. The results were lower than expected which is potentially due to the finer feed size of F_{80} 46 µm from the rod mill. It was noted during the testing that the rod mill grind time for the SW sample was less reliable probably due to the rheological issues associated with the clays. If further work is undertaken it would be advised to generate 30-40 kg of rod mill product at P_{80} 53 µm, undertake a cyclone cut at the target grind sizes and carry out a signature plot on the cyclone underflow. This would give a better understanding of the required energy.

Figure 13-8 - SW signature plot for SW

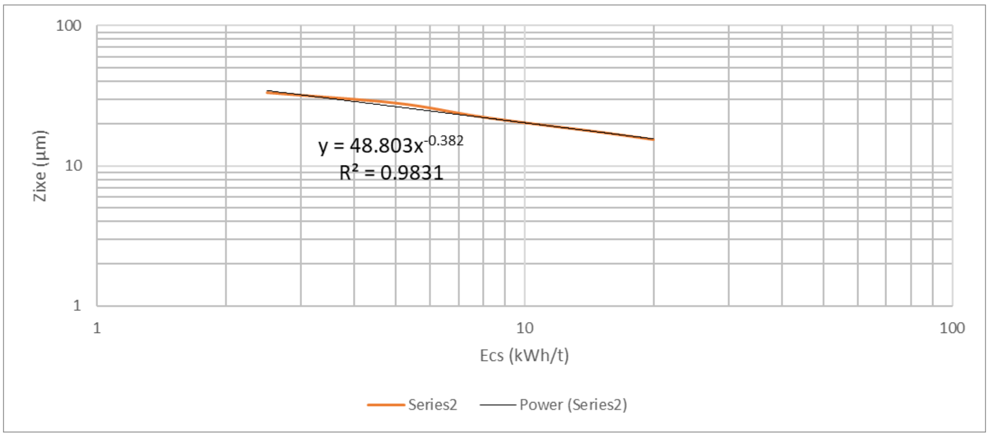


Table 13-9 - Power model for SW SMD signature plot

Equation Variables			Calculated Specific Energy (Power Model)	
Equation	**a**	**b**	**37µm**	**29µm**
y = a*x^b	48.803	-0.382	2.1 kWh/t	3.9 kWh/t

13.3.3.2 Gravity Testing

Table 13-10, Table 13-11 and Table 13-12 show the mass balances for the initial falcon rougher stage and the subsequent Mozley superpanner cleaning stage of the Falcon tests conducted at different feed flowrates. The higher flowrate should result in a lower mass pull to concentrate and therefore a higher grade to the concentrate.

At 1800 ml/min 31% mass pull was achieved in the rougher resulting in 37% Sn recovery, 29% and 30% Au and Ag recovery respectively. There was no upgrading of the target heavy minerals with the recovery being equivalent to mass pull. There is noticeable upgrading of Au and Ag on the cleaner table, but the relative recoveries are poor. The Sn recovery on the table, as with the Falcon matches the mass pull showing recovery and upgrading unlikely through gravity.

The trends seen with the 1800ml/min test were also seen at 3500 ml/min and 6000 ml/min. The recovery of Sn, Au and Ag equated to the mass pull for Sn. However, Au recovery was better on the cleaner table, but the majority of the Au (83% and 89%) was not recovered in the Falcon.

Overall, it can be seen that gravity would not be a viable option for generating a pre-concentrate of either Sn, Au or Ag on the massive sulphide. It is likely due to the density of the pyrite and other sulphide minerals are not allowing for a clean cut. There may be potential for gravity carried out on the flotation tailings once the sulphide has been removed, however this work was not included in the scope for this project.

Table 13-10 - Gravity test 1 at 1800 ml/min

Sample	Mass (%)	Grade			Recovery		
		% Sn	Au (mg/kg)	Ag (mg/kg)	Sn	Au	Ag
Con 1	0%	0.27	23.74	256.1	0%	3%	1%
Con 2	15%	0.27	1.31	92.6	15%	13%	14%
Con 3	8%	0.26	6.42	91.2	8%	34%	7%
Con 4	1%	0.10	2.35	102.8	0%	1%	1%
Tails	77%	0.26	0.93	98.6	77%	48%	77%
	Head Grade	0.3	1.5	97.5			

Sample	Mass (%)	Grade			Recovery		
		% Sn	Au (mg/kg)	Ag (mg/kg)	Sn	Au	Ag
Falcon Con	31%	0.26	0.86	92.4	37%	29%	30%
Falcon Tails	69%	0.20	0.97	100.3	63%	71%	70%
	Head Grade	0.2	0.9	97.8			

Table 13-11 - Gravity test 2 at 3500 ml/min

Sample	Mass (%)	Grade			Recovery		
		% Sn	Au (mg/kg)	Ag (mg/kg)	Sn	Au	Ag
Con 1	0%	0.24	22.94	110.1	0%	15%	1%
Con 2	5%	0.31	1.15	92.8	7%	9%	6%
Con 3	8%	0.26	1.10	90.6	8%	13%	8%
Con 4	7%	0.22	0.84	96.3	6%	8%	8%
Tails	78%	0.26	0.51	91.5	78%	55%	78%
	Head Grade	0.3	0.7	91.9			

Sample	Mass (%)	Grade			Recovery		
		% Sn	Au (mg/kg)	Ag (mg/kg)	Sn	Au	Ag
Falcon Con	24%	0.25	0.57	92.6	28%	17%	22%
Falcon Tails	76%	0.21	0.87	103.6	72%	83%	78%
	Head Grade	0.2	0.8	100.9			

Table 13-12 - Gravity test 3 at 6000 ml/min

Sample	Mass (%)	Grade			Recovery		
		% Sn	Au (mg/kg)	Ag (mg/kg)	Sn	Au	Ag
Con 1	0%	0.27	31.88	95.2	0%	15%	0%
Con 2	28%	0.27	0.96	92.9	30%	28%	29%
Con 3	16%	0.22	0.88	82.2	14%	15%	15%
Con 4	6%	0.18	1.00	96.3	4%	6%	7%
Tails	50%	0.26	0.70	89.3	51%	36%	50%
	Head Grade	0.3	1.0	89.6			

Sample	Mass (%)	Grade			Recovery		
		% Sn	Au (mg/kg)	Ag (mg/kg)	Sn	Au	Ag
Falcon Con	16%	0.25	0.57	92.6	18%	11%	14%
Falcon Tails	84%	0.21	0.87	103.6	82%	89%	86%
	Head Grade	0.2	0.8	101.9			

13.3.3.3 Massive Sulphide Flotation Testing

The original scope for the MS flotation programme was to confirm previous test results using the reagent scheme and circuit configuration determined in previous test programmes and making small adjustments in reagent and residence times to account for the different head grade. Following initial testing and discussions with Ascendant, it was agreed that a full scoping programme was required to achieve results in line with the original testing.

Throughout the testing discussions were carried out with both Ascendant and Setas Pires (consultant acting on behalf of Ascendant) as to the progress of testing and next steps.

Rougher Scoping

Table 13-13 shows the testing parameters for all 23 float tests undertaken on the MS. It should be noted that MS FT1 and MS FT23 were cut short at the end of the Pb stage as the conditions trialled as it was apparent the test conditions were unsuitable. A number of different reagents were trialled outside of the original reagent scheme used in 19-1608, most notably:

- Nascol 2016 - collector reagent with increased selectivity for both Cu and Pb.
- $Na6(PO3)6$ – depressant similar to SMBS that depresses pyrite and sphalerite.
- SIBX – trialled instead of SIPX as reagent has shown better selectivity for sphalerite at operations of similar deposits such as Almina.
- Higher addition rates of lime were used, up to 10 kg/t, to control the pH to the previous targets of pH 9.5 for the Pb and pH 11-12 for the Zn stages.

In addition to reagent changes differing lengths of aeration times conditioning times and float times were trialled to control the mass pull and target a maximum of 20% Zn loss to the Pb concentrate.

Table 13-14 gives a high-level overview of the results for the 22 rougher tests carried out on the MS sample. Rationale behind the test are summarised below:

- FT1 – FT8 were conducted to determine a baseline for lime addition in the mill where it was determined that the mass pull of the floats and subsequently the selectivity was mainly driven by the pH control in the mill.
- FT9-FT11 used the reagent scheme used at the Almina operation with FT12 acting as hybrid between previous tests looking at pH control with lime and the reagent addition and conditioning times of Almina.
- FT14-FT18 evaluated methods of depressing pyrite other than pH control
    - The addition of Na6(PO3)6 and at higher dosages of NaS the mass pull reduced but at the cost of a lower recovery of galena and sphalerite.
    - Concentrate grades dropped showing that all sulphides were being depressed.
- FT19- FT22 increased ZnSO4 dosage, which led to a higher dose (650 g/t) of CuSO4 to reactivate.
    - This controlled the sphalerite losses to Pb concentrate to under 20% in FT21.
    - Overall mass pull reduced to nominally 40%.
    - Increased concentrate grades for both Pb and Zn.
    - Pb recovery reduced however Zn recovery increased.

Figure 13-9 shows the kinetics for FT21 and FT22 (best performing MS rougher floats). The Pb recovery is 64% for both at a cumulative grade of 14%, with a required residence time of 8 minutes. The Zn recovery is 68% with a cumulative grade of nominally 15%. Zn Ro 1 grades were 27% with a concentrate recovery of 60%. The residence time for the Zn concentrate was 16 minutes.

Table 13-13 - MS Rougher float test conditions

Test No.	Aeration (mins)	Grind Size	ZnSO4 g/t	SMBS g/t	NASFROTH 250 g/t	CuSO4 g/t	Lime kg/t	3418A g/t	SIPX g/t	SIBX g/t	Nascol 2016 g/t	Na6((PO3)6) g/t	Na2S g/t	Condition Time (mins)	Float Time (mins)
MS FT1	15	P_{80} 29µm			39		6.4	45						14	14
MS FT2	15	P_{80} 29µm			43	300	10.95	25	105					33	30
MS FT3	15	P_{80} 29µm			43	300	8.6	25	105					33	30
MS FT4	10	P_{80} 29µm	833.3	500	38	300	8.95	22.5	105					33	30
MS FT5	25	P_{80} 29µm			38	300	9		105		22.5			33	30
MS FT6	38	P_{80} 29µm			38	300	8.2							33	30
MS FT7	12	P_{80} 29µm			38	300	7.45	20	105					33	30
MS FT8	12	P_{80} 29µm		7	38	300	8.55	20	105					33	30
MS FT9	14	P_{80} 29µm	388.9		38	300	8.9	20	105					33	30
MS FT10	12.5	P_{80} 29µm	555.6		36	300	9.2	20	105					30	26
MS FT11	50	P_{80} 29µm	388.89		36	300	8.25	20	105					30	26
MS FT12	16.5	P_{80} 29µm	388.89		36	300	8.35	20	105					30	26
MS FT13	43	P_{80} 20µm	388.89		36	300	8.35	20	105					30	26
MS FT14	50	P_{80} 29µm	388.89		36	300	8.55	20	105			500		30	26
MS FT15	39	P_{80} 29µm	388.89		36	300	8.35	20	105				1000	30	26
MS FT16	13	P_{80} 29µm	388.89		36	300	9.5	20	105			2000		30	26
MS FT17	30	P_{80} 29µm	388.89		36	300	7.35	20	105				3000	30	26
MS FT18	13	P_{80} 29µm	388.89		36	300	7.35	20	105				3000	30	26
MS FT19	5	P_{80} 29µm	650		25	450	8.2	20		45				35	23
MS FT20	13	P_{80} 29µm	650		20	450	10.45	20		45				35	23
MS FT21	15	P_{80} 29µm	650		20	450	12.2	25		45				35	23
MS FT22	15	P_{80} 29µm	650		20	450	12.2	25		45				35	23
MS FT23	15	P_{80} 29µm	650		25		10	35						19	15

Table 13-14 - MS rougher rate kinetic summary

Float	Zn Losses to Pb	Zn Ro 1 Grade	Pb Recovery to Pb Conc	Pb Ro 1 Grade	Total Mass Recovery
FT1	87%	n/a	86%	13.2	43%
FT2	37%	28.8	71%	13.2	54%
FT3	43%	28.4	77%	13.9	56%
FT4	52%	25.1	83%	13.5	59%
FT5	38%	24.1	74%	14.9	54%
FT6	61%	20.8	81%	13.8	65%
FT7	41%	27.1	79%	11.3	57%
FT8	54%	22.6	88%	12.2	66%
FT9	45%	27.7	83%	17.9	58%
FT10	37%	24.0	76%	11.7	60%
FT11	25%	26.8	68%	12.2	60%
FT12	29%	28.1	74%	15.0	49%
FT13	29%	27.9	72%	14.6	53%
FT14	37%	26.6	76%	13.8	51%
FT15	28%	28.5	74%	17.2	60%
FT16	31%	26.3	75%	20.2	50%
FT17	20%	13.0	55%	6.5	41%
FT18	22%	3.8	43%	6.5	39%
FT19	28%	14.9	67%	14.9	55%
FT20	28%	21.3	72%	13.3	55%
FT21	19%	27.8	66%	14.6	39%
FT22	20%	24.2	64%	0.2	42%

Figure 13-9 - MS optimal conditions kinetics

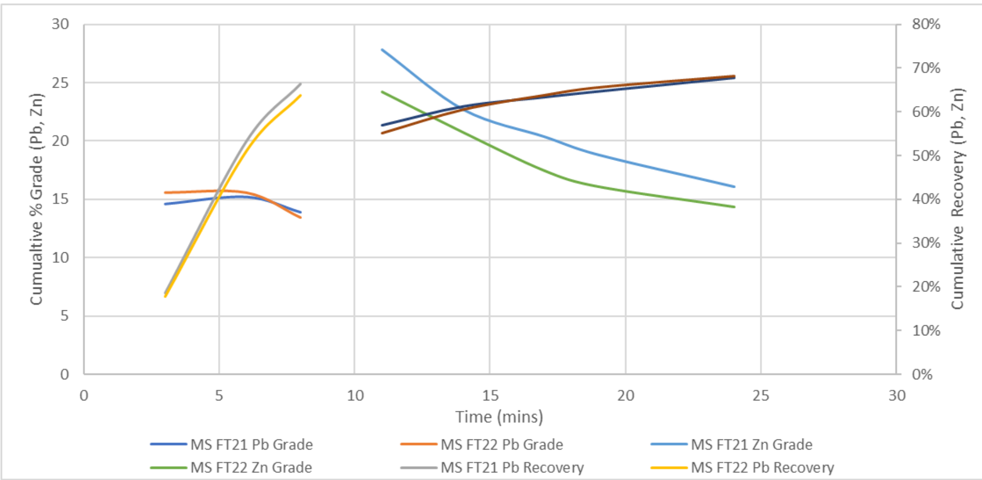


Figure 13-10 and Figure 13-11 show the grade vs recovery for a selection of the best performing MS rougher floats. It can be seen that small differences to the Pb rougher stage made for a large difference to the performance of the Zn rougher.

Overall, it was seen, with this sample, achieving more than 70% Pb recovery and 68% Zn recovery was unlikely with extended Pb floats (i.e FT12) resulting in reduced Zn performance.

The float parameters determined in FT21 and repeated in FT22 showed the best performance and were used for the cleaner and locked cycle tests (LCT).

Figure 13-10 - MS optimal conditions grade recovery curves

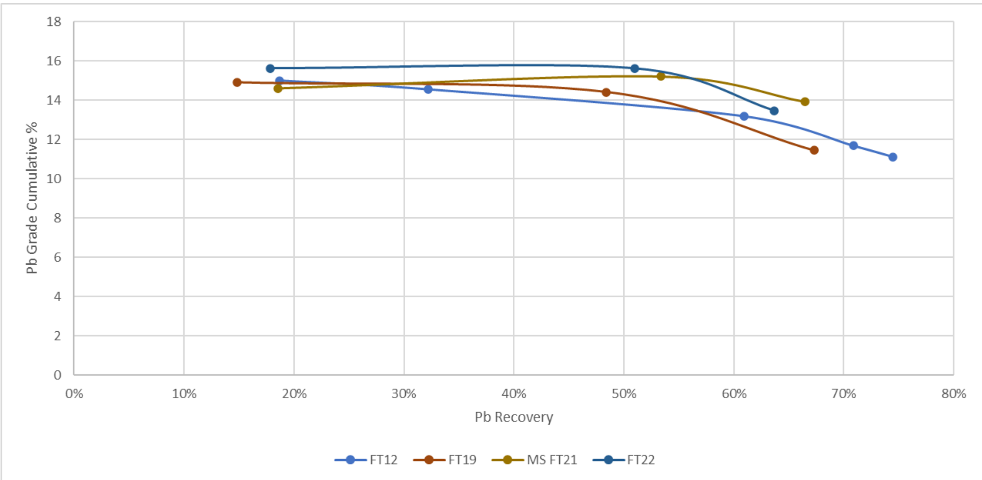
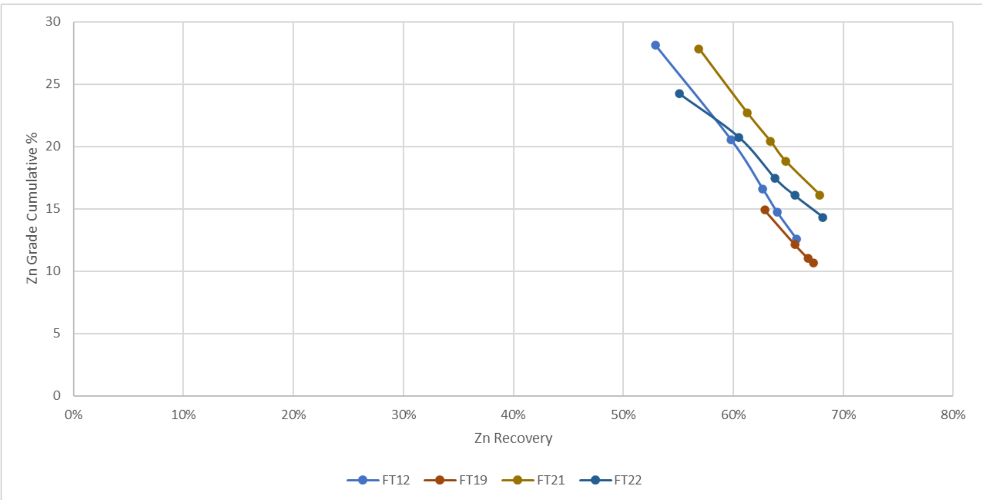


Figure 13-11 - MS optimal conditions grade recovery curves



The selectivity curves (Figure 13-12, Figure 13-13 and Figure 13-14) demonstrate the fine margins between the different float tests. As can be seen there was little variation in the selectivity between Pb and Zn with FT21 showing the lowest loses at 20%. However overall, this was beneficial as near 30% losses seen in FT12 and FT19 meant 10% less recovery to the Zn concentrate.

The main improvements with regards the concentrate grade increases were the selectivity with the Fe and therefore the pyrite. FT21 gave better selectivity between Pb and Fe and a big reduction between Zn and Fe. This reduced the mass pull considerably lowering entrainment and increasing concentrate grade.

Figure 13-12 - Selectivity for Pb vs Zn – FT12, FT19 & FT21

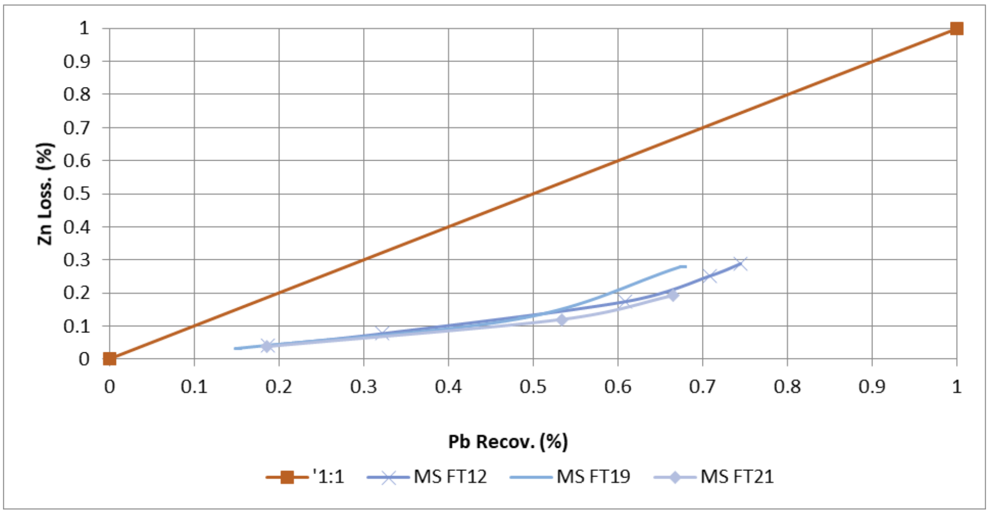


Figure 13-13 - Selectivity for Pb vs Fe – FT12, FT19 & FT21

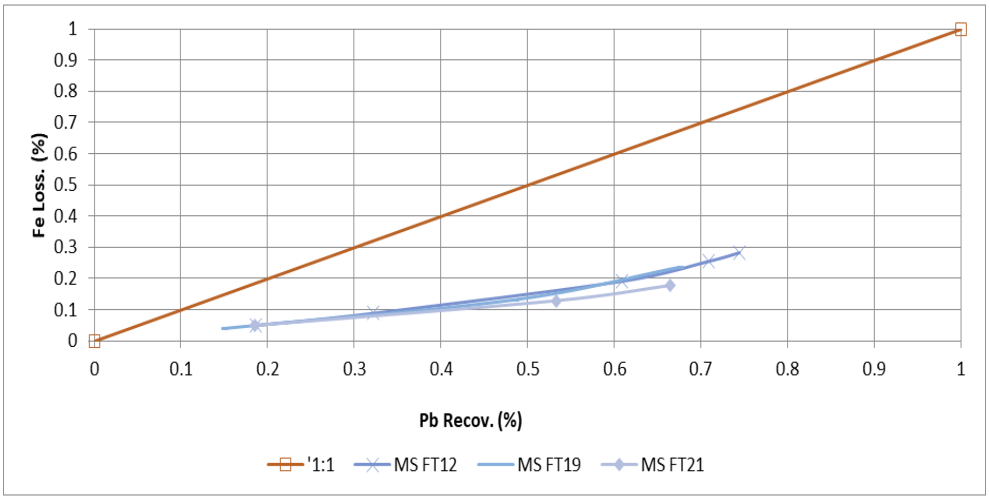
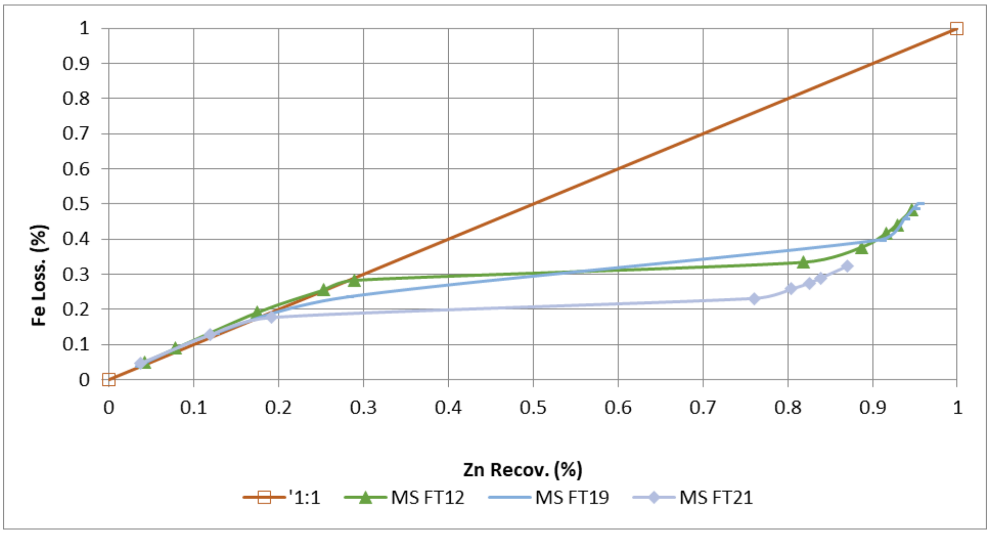


Figure 13-14 - Selectivity for Zn vs Fe – FT12, FT19 & FT21



Cleaner Tests

The reagent scheme from rougher tests FT21 and FT22 were used for all cleaner MS testing. Table 13-15 shows a summary of the 10 cleaner (Cl) float tests carried on the MS. Initially finer regrind sizes were trialled to match the liberation data previously applied to the Lagoa MS deposit in 19-1608. Regrinds were carried out in a small scale SMD to best match pulp chemistry conditions expected on plant.

In addition to the regrinds, lime and ZnSO4 were added to the Pb regrind for pyrite and sphalerite depression and lime added to the Zn regrind to maintain at pH 12 for pyrite depression. Initially 3 cleaning stages were trialled with 5 cleaning stages trialled for FT10.

From the results of the cleaner tests (Table 13-16) the following can be seen:

- There was little improvement in the grade and recovery of both Pb and Zn with modification of the grind size
- With three stages of cleaning 43% Pb recovery could be achieved with a grade of 21.5%.
- 5 stages of cleaning on the Zn gave the highest grade of 46.6% at a recovery 45%
    - It should be noted with the 5 stages of cleaning, collector (SIBX) addition was required to all cleaning stages to maintain the recovery

Overall, it was determined that there was a large amount of variability between the grade and recovery of the Pb concentrates. The cleaner feed grade and mass pull was very variable with the most likely cause the degree of oxidation in the feed material. It was determined that the 5 cleaning stages gave the best consistency.

Unlike the Pb rougher, the Zn rougher was more reliable in terms of mass pull. It was noted that high amounts of collector were required to ensure recovery and grades of up to >45% were achievable.

Table 13-15 - MS cleaner test parameter summary

Test	Regrind Pb Size	Regrind Zn Size	Reagents in Mill	Stages Pb	Stages Zn
MS CL Ft1	5.5µm	8µm	250 g/t $ZnSO_4$ and 250 g/t Lime for Pb 350 g/t Lime in Zn	3	3
MS CL Ft2	5.5µm	8µm	No reagents in mill for either regrind	1	1
MS CL Ft3	5.5µm	8µm	250 g/t $ZnSO_4$ and 250 g/t Lime for Pb 600 g/t Lime in Zn	1	1
MS CL Ft4	15µm	15µm	500 g/t Lime in Pb 500 g/t Lime in Zn	3	3
MS CL Ft5	15µm	15µm	150 g/t $ZnSO_4$ and 450 g/t Lime for Pb 600 g/t Lime in Zn	3	3
MS CL Ft6	15µm	15µm	150 g/t $ZnSO_4$ and 450 g/t Lime for Pb 600 g/t Lime in Zn	3	3
MS CL Ft7	10µm	15µm	150 g/t $ZnSO_4$ and 350 g/t Lime for Pb 1000 g/t Lime in Zn	3	3
MS CL Ft8	6µm	8µm	150 g/t $ZnSO_4$ and 350 g/t Lime for Pb 1500 g/t Lime in Zn	3	3
MS CL Ft9	15µm	15µm	350 g/t $ZnSO_4$ and 450 g/t Lime for Pb 1350 g/t Lime in Zn	3	3
MS CL Ft10	15µm	15µm	350 g/t $ZnSO_4$ and 450 g/t Lime for Pb 1350 g/t Lime in Zn	5	5

Table 13-16 - MS cleaner test result summary

Cleaner Test	Pb Conc		Zn Conc	
	Pb Grade	Pb Rec	Zn Grade	Zn Rec
MS Cl FT1	21.5%	43.0%	23.0%	29.0%
MS Cl FT2	11.3%	49.0%	15.3%	50.6%
MS Cl FT3	11.1%	66.9%	17.6%	49.8%
MS Cl FT4	27.3%	31.0%	35.5%	53.0%
MS Cl FT5	34.8%	23.6%	43.9%	49.4%
MS Cl FT6	30.7%	30.6%	36.8%	22.4%
MS Cl FT7	29.0%	42.0%	32.5%	55.0%
MS Cl FT8	24.6%	40.6%	42.4%	27.1%
MS Cl FT9	25.5%	16.0%	33.6%	60.0%
MS Cl FT10	29.8%	21.0%	46.6%	45.0%

Locked Cycle Testing

Two locked cycle tests were conducted on the MS sample.

Locked Cycle Test 1

The first locked cycle test (LCT1) was carried out with 3 cleaning stages on both the Pb and Zn with all Pb rougher material taken forward to the regrind. The regrind size was maintained at P_{80} 15 µm with the following exit streams taken:

- Pb Cl3 concentrate.
- Zn Cl3 concentrate.
- Zn Cl1 tailings.
- Zn Ro tailings.

Figure 13-15 shows the LCT1 flowsheet.

Figure 13-15 - MS LCT1 Flowsheet

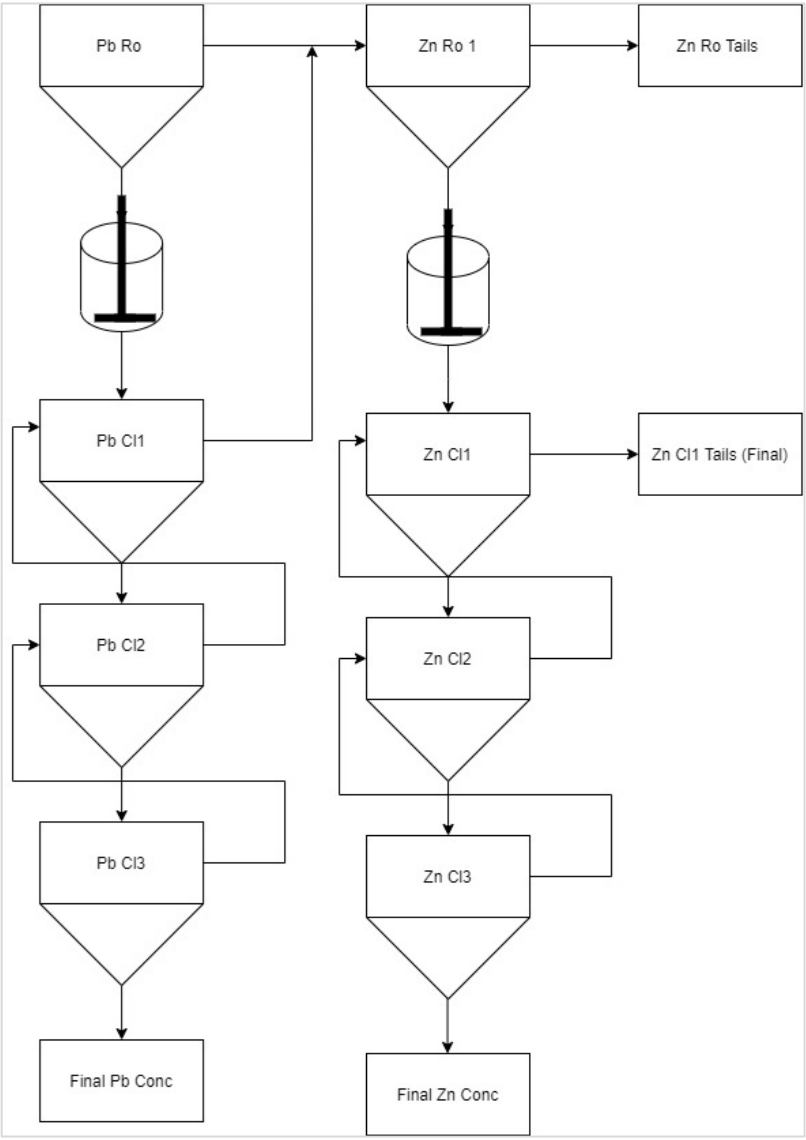


Table 13-17 shows the results of LCT1. The final concentrate grades for both Pb and Zn have decreased considerably from the open circuit testing, however, the recovery has increased. The mass pull to the two concentrates is very high, 26% combined, which when looking at the grade and recovery shows that the concentrates have been heavily diluted by pyrite.

The Sn content, as with the gravity circuit, matches closely with the mass pull and 33% is lost to the concentrates and Zn Cl1 tails. For recovery of the Sn in the Zn rougher tailings, a bulk pyrite sulphide float will be required, and testing conducted to ascertain whether reasonable Sn recovery can be achieved.

The majority of the Au reports to the Zn rougher tailings suggesting that it is mainly associated with the pyrite. 33% of the Ag was recovered the Pb concentrate as it would be associated with the galena. Leach testing carried out on the bulk pyrite concentrate will determine the recovery of Au and Ag from the Zn rougher tailings.

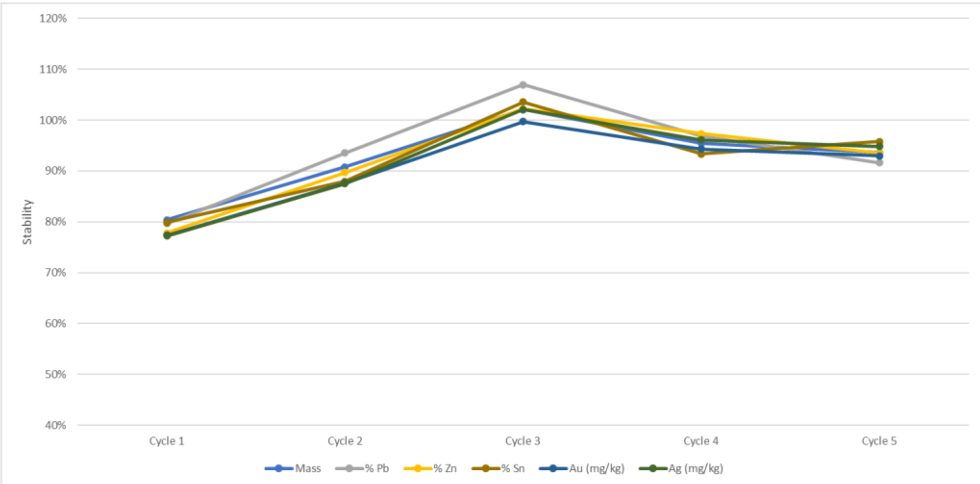
Table 13-17 - MS LCT1 test results

Product	Wt (%)	Assay WJL (%, g/t)					Recovery				
		Pb	Zn	Sn	Au	Ag	Pb	Zn	Sn	Au	Ag
Pb Cl3 Con	15%	17.9	5.1	0.1	1.3	259.5	62%	15%	7%	21%	33%
Zn Cl1 Tails	10%	2.7	1.7	0.3	1.1	105.6	6%	3%	13%	11%	8%
Zn Cl3 Con	11%	4.2	29.0	0.3	1.2	144.6	11%	66%	13%	15%	14%
Zn Ro Tails	64%	1.5	1.2	0.2	0.8	83.6	21%	15%	67%	53%	45%
Total	100%	4.4	5.00	0.2	0.9	119.6	100%	100%	100%	100%	100%

Figure 13-16 shows the stability graph for mass and keys elements. For a MS from the Iberian pyrite belt (IPB) the stability of the LCT was very good. Previous testing on different IPB deposits has shown an inability to stabilise across multiple cycles.

It can be seen that there is an increase of mass in the system over the first three cycles with cycle 4 and 5 showing that the system has achieved steady state.

Figure 13-16 - LCT stability



Overall, the results from LCT1 showed that there were issues with mass recovery to both the Pb and Zn concentrate effecting the concentrate grade achievable. Recoveries had increased considerably from the original testing in 19-1608 however this was a function of the mass.

Selectivity and depression of pyrite have been demonstrated to be the main issues floating this sample.

Locked Cycle Test 2

Following LCT1, a second LCT was conducted with an increased number of cleaning stages and exit streams to try and control the mass recovery issues and achieve a higher grade of concentrate. Figure 13-17 shows the refined MS LCT flowsheet following the completion of LCT1. As can be seen there are a number of changes:

- 5 stages of cleaning carried out on both the Pb and Zn concentrates.
- The Pb rougher was cut short with a scavenger concentrate taken and left open as an exit stream. This allowed for the removal of pyrite at the tail end of the Pb rougher without the mass reporting to the Zn rougher.

Figure 13-17 - MS LCT2 flowsheet

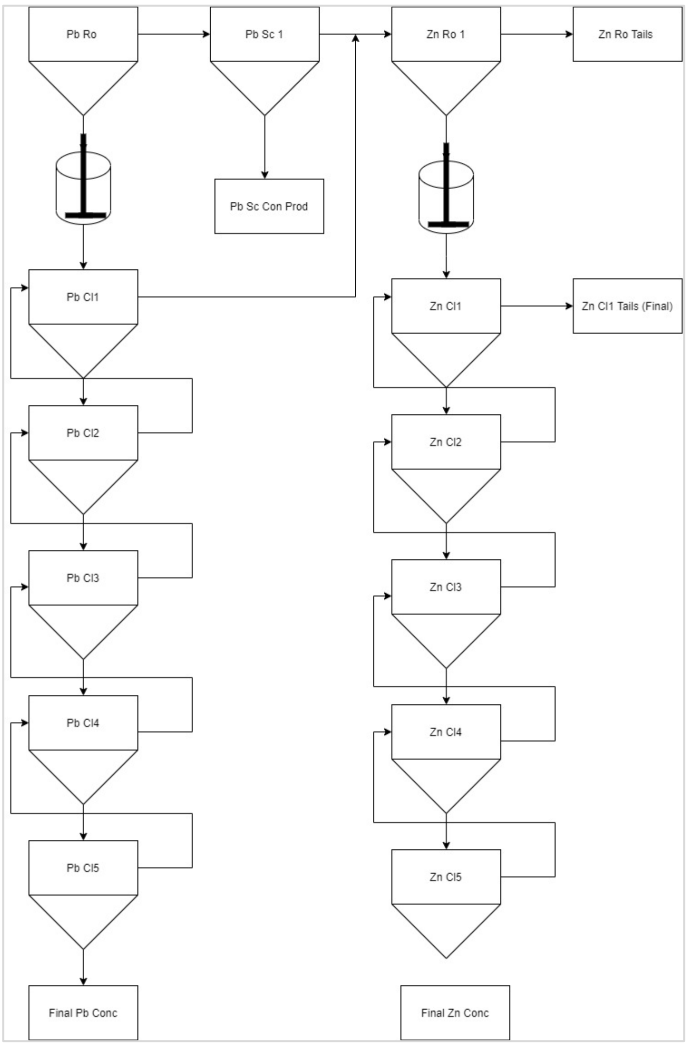


Table 13-18 shows the results for LCT2. As can be seen the main changes seen are an increase in the grade for both Pb and Zn with the Zn recovery increasing and Pb recovery decreasing. The mass pull for the Pb has decreased considerably from 16% to 9% with the Zn concentrate mass pull staying the same. The drop in the Pb recovery can be accounted for with the 16% recovery to the Pb Sc con. By removing this material, the grade only increased to 22% but the recovery dropped by 19%.

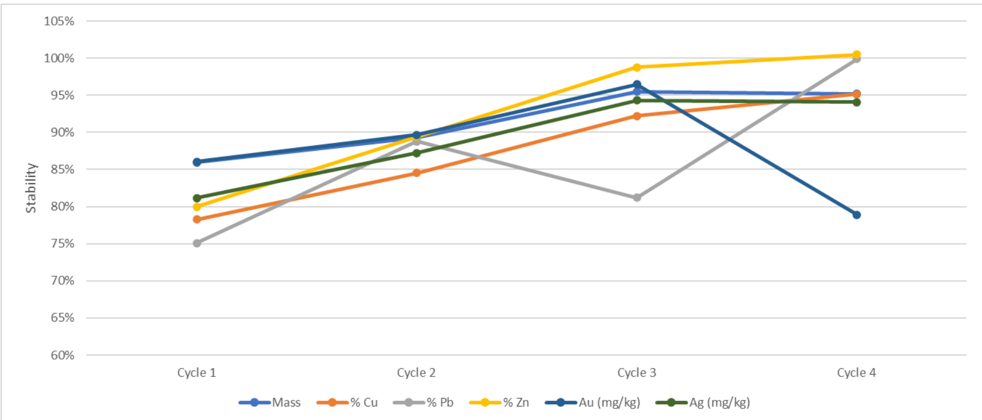
The Au and Ag losses to the Zn rougher tailings have increased with the revised circuit configuration which is likely be due to the mass pull decrease to the rougher and scavenger concentrates. The Sn recovery to the Zn rougher tailings has also increased to 73%.

Table 13-18 - MS LCT2 results

Product	Wt (%)	Assay WJL (%, g/t)						Recovery					
		Cu	Pb	Zn	Sn	Au	Ag	Cu	Pb	Zn	Sn	Au	Ag
Pb Cl5 Con	9%	0.5	21.9	4.6	0.1	1.1	250.9	20%	43%	8%	3%	11%	19%
Zn Cl1 Tails	6%	0.2	2.8	2.1	0.3	1.2	97.2	6%	4%	2%	9%	9%	5%
Zn Cl5 Con	10%	0.3	5.2	34.6	0.2	1.0	129.0	15%	12%	66%	10%	11%	11%
Pb Sc Con	8%	0.5	8.7	4.7	0.1	1.3	170.9	17%	16%	8%	5%	13%	13%
Zn Ro Tails	68%	0.1	1.6	1.2	0.2	0.7	86.7	42%	26%	16%	73%	56%	52%
Total	100%	0.2	4.4	5.1	0.2	0.8	112.4	100%	100%	100%	100%	100%	100%

The stability for LCT2 (Figure 13-18) showed similar trends to LCT1 with the increase in mass. It should be noted however that only 4 cycles were carried out. There were issues encountered at cycle 5 and cycle 6 with pH control out of the mill and as the mass had steadied after cycle 3 and 4 it was decided to keep the recirculating loads back rather than risk the test data being ruined.

Figure 13-18 - MS LCT2 stability



Based off the two LCTs carried out it can be seen that there is comparisons to be made with the previous testwork on the MS. Although highly oxidised, reducing the selectivity achievable, compared with the results achieved in 19-1608 it can be seen that reduction in grade is balanced with the increase in recovery. Further work should be undertaken on fresh sample utilising the findings from both scoping studies to result in a more robust circuit configuration and regent scheme as well as potentially improved performance.

13.3.3.4 Stockwork

As with the MS sample the initial tests showed that the SW is likely oxidised although the performance differences between 19-1608 and the current sample seemed to be closer in the initial float tests.

Rougher Scoping

Figure 13-18 shows the testing parameters for the 10 rougher rate kinetic tests carried out. Testing was carried out to match the sequential floatation method determined in the original work with a separate rougher concentrate for Cu, Pb and Zn. FT1-FT7 explored the sequential method, with FT8-FT10 determining conditions for a bulk Cu/Pb concentrate followed by a Zn concentrate.

Key aspects that were trialled through the testing were:

- Levels of SMBS required to gauge a separation between Cu and Pb in the rougher stage for sequential stage.
- Increased dosages of $ZnSO_4$ to control losses of Zn to the Cu and Pb concentrates.
- Different collectors trialled including Danaflot 247E, SIBX, SIPX, 31418A and Nascol 2016.
- Different frother trialled including MIBC and Nasfroth 250.
- Aeration times and primary grind size were also altered to control Pb and Zn loss to Cu concentrates.

Table 13-19 - SW rougher parameter summary

Test No.	Aeration (mins)	Grind Size	ZnSO 4 g/t	SMBS g/t	NASFROTH 250 g/t	CuSO4 g/t	Lime kg/t	3418A g/t	SIPX g/t	MIBC g/t	Nascol 2016 g/t	NaSil g/t	Na2S g/t	Danaflot 247E g/t2	Condition Time (mins)	Float Time (mins)
SW FT1	10	P_{80} 37µm	3000	2100		450	2.4	13	19	20		1000	3000		39	43.5
SW FT2	10	P_{80} 37µm	2500	2100		450	6.2		18	20		1000	2500	9	39	43.5
SW FT3	10	P_{80} 37µm	2500	2300		450	2.25		18	20	9	1000	2500		39	43.5
SW FT4	10	P_{80} 37µm	2500	2250		450	2.15					2500	2500		39	43.5
SW FT5	10	P_{80} 37µm	5000	2350		450	1.65	13	19	22		2500	5000		39	43.5
SW FT6	10	P_{80} 20µm	5000	2250		450	1.95	13	19	24		3500	5000			
SW FT7	15	P_{80} 37µm	2500	2100	21	650	2.7		19		17	1250	2500		34	29.5
SW FT8	15	P_{80} 37µm	2500		14.5	650	1.9		19		20	1250	2500		23	17
SW FT9	15	P_{80} 37µm	2500		9	650	1.85		21		20	1250	2500		23	17
SW FT10	15	P_{80} 37µm	2500	1000	15	650	1.6		21		20	750	2500		23	17

Table 13-20 shows the summary of SW rougher tests for FT1-FT7 representing the sequential float tests. At all stages there was little upgrading for all three rougher concentrates, with large losses of Pb and Zn to the Cu concentrate and Zn to the Pb concentrate. FT3 using Nascol 2016 gave the best performance with regards to the sequential float for each of the concentrates. Further work with this collector and altered residence time with a unoxidized feed could offer good selectivity for a sequential flotation approach.

Table 13-20 - SW rougher results summary FT1- FT7

Float	Cu Grade	Cu Recovery	Pb Grade	Pb Recovery	Zn Grade	Zn Recovery	Mass Recovery
SW FT1	3.21	86%	1.03	14%	3.46	21%	39%
SW FT2	2.69	60%	2.61	49%	1.08	4%	41%
SW FT3	4.32	82%	1.17	20%	6.29	32%	41%
SW FT4	4.25	85%	0.95	12%	4.15	22%	33%
SW FT5	4.38	87%	0.50	6%	1.46	6%	33%
SW FT6	3.46	90%	0.26	5%	2.24	13%	46%
SW FT7	5.42	88%	0.96	12%	4.25	40%	37%

Table 13-21 shows the result summary for FT8-FT10 which had a Cu/Pb bulk rougher and Zn rougher separately. As can be seen the recovery and performance is improved considerably with a large reduction in mass recovery showing reduction in dilution.

Table 13-21 - SW rougher results summary FT8 - FT10

Float	Cu/Pb Cu Grade	Cu/Pb Cu Recovery	Cu/Pb Pb Grade	Cu/Pb Pb Recovery	Zn Grade	Zn Recovery	Mass Recovery
SW FT8	1.27	92%	3.27	94%	4.54	24%	42%
SW FT9	3.47	93%	7.18	87%	5.88	40%	25%
SW FT10	5.19	88%	10.18	74%	5.43	54%	25%

Figure 13-19 and Figure 13-20 show the grade vs recovery curves for Cu and Pb comparing the performance difference between the sequential float and bulk Cu/Pb concentrate. As can be seen the bulk approach gives performance increases to the Cu with higher grades achieved for equivalent recovery. With the majority of the Pb being lost into the Cu sequentially the recovered Pb to a bulk concentrate can be seen to be far better at 70-80% and a grade of 10%.

Figure 13-19 - Cu grade and recovery curves

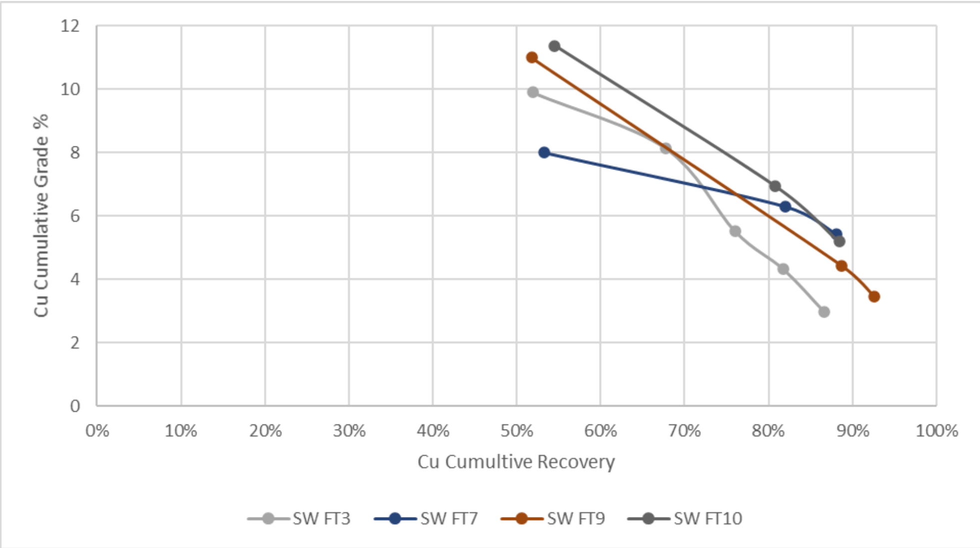


Figure 13-20 - Pb grade and recovery curves

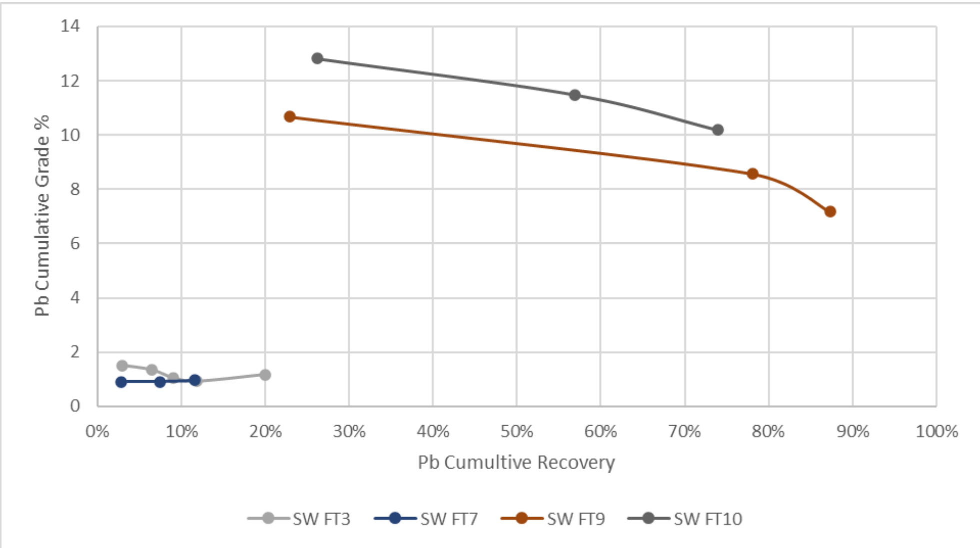
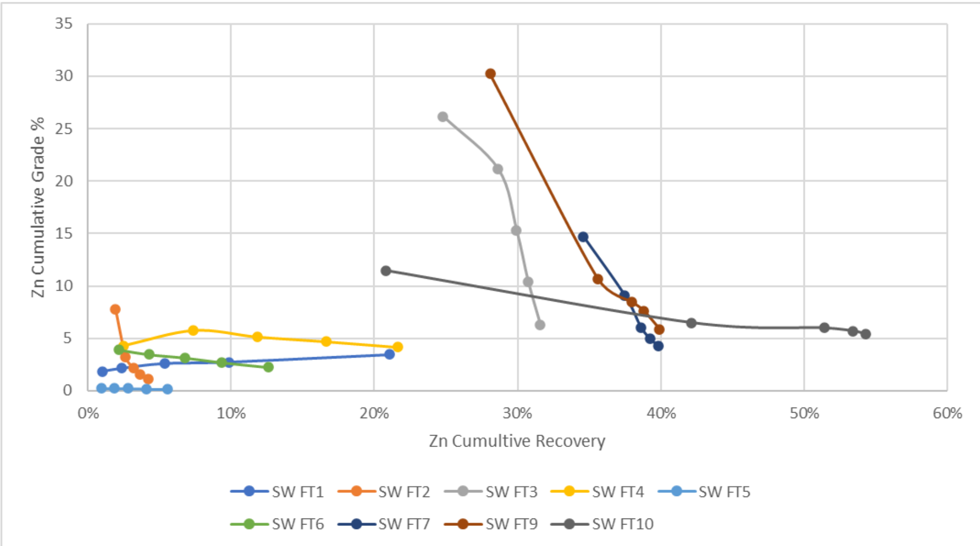


Figure 13-21 shows the Zn grade vs recovery for sequential tests and with a Cu/Pb bulk float. The use of Nascol 2016 in FT3 has led to a good Zn grade but poor recovery. With the bulk float, losses to the Cu/Pb concentrate are reduced considerably allowing for much improved recovery to >50%.

Based off the results the conditions from FT10 were used for subsequent cleaners and locked cycle tests.

Figure 13-21 - Zn grade and recovery curves



CLEANER TESTS

As stated, the conditions from FT10 rougher were used for the cleaning stages. The aim for the Cu/Pb bulk regrind was to depress the galena taken forward with SMBS and allow for a clean Cu concentrate to be pulled. Following a Cu Cl1 the Cu Cl1 tails would be dosed with NaS to reactivate the galena and then generate a concentrate. Table 13-22 outlines the 5 different cleaner tests undertaken.

Undertaking the cleaner tests, the following was determined:

- Pb Cl1 con 1 was predominately Zn, due to the Zn being activated before the Pb.
    - As a result, the first 30 second concentrate from Pb Cl1 was to be added to the Zn Cl1 feed.
- FT3 showed that once the galena had been depressed 3 stages of cleaning gave good Cu concentrate grade at 27.5% and recovery at 57.4%.
- Zn performance in Cl FT5 showed good grade at 51.6% and recovery of 46.9% .
- The Pb recovery in FT3 was poor but it was assumed that an improvement would be seen in the LCT with addition of recirculating loads and reagent water helping depression in the Cu cleaner and activation in the Pb Cl1.

Table 13-22 - Cleaner test parameter summary

Test	Regrind Cu/Pb Size	Regrind Zn Size	Reagents in Mill	Stages Cu	Stages Pb	Stages Zn
SW CL Ft1	15µm	15µm	2000 g/t SMBS in Cu/Pb 200 g/t Lime in Zn	1	0	1
SW CL Ft2	15µm	15µm	4500 g/t SMBS in Cu/Pb 200 g/t Lime in Zn	3	1	3
SW CL Ft3	15µm	15µm	4500 g/t SMBS in Cu/Pb 200 g/t Lime in Zn	3	1	3
SW CL Ft4	15µm	15µm	4500 g/t SMBS in Cu/Pb	5	0	0
SW CL Ft5	15µm	15µm	4500 g/t SMBS, 1000 g/t ZnSO4, 1000g/t NaS in Cu/Pb 200 g/t Lime in Zn	3	1	3

Table 13-23 - Cleaner test result summary

Cleaner Test	Cu Conc		Pb Conc		Zn Conc	
	Cu Grade	Cu Rec	Pb Grade	Pb Rec	Zn Grade	Zn Rec
SW Cl FT1	10.9%	86.6	n/a	n/a	31.1%	55.4%
SW Cl FT2	22.0%	70.0%	17.1%	17.0%	44.4%	51.0%
SW Cl FT3	27.5%	57.4%	25.8%	4.4%	45.0%	48.0%
SW Cl FT4	12.8%	72.0%	-	-	-	-
SW Cl FT5	14.6%	84.1%	-	-	51.6%	46.9%

Locked Cycle Testing

Figure 13-22 shows the circuit configuration for the SW LCT based off the rougher scoping tests and cleaner tests. As can be seen with the separation of the Cu and Pb concentrates post regrind a fairly complex cleaning circuit has resulted. The key aspects are:

- Pb Cl1 Con1 feed combines:
  - Cu Cl2 tailings (assays showed that the main component of the tailings was Pb).

  - Pb Cl2 tailings.
  - Cu Cl1 tailings.
  - For each recirculating stream the reagent water was also used to ensure chemistry would match plant conditions as close as possible.
- Pb Cl1 tailings was kept as an exit stream as there was little Zn present in the open circuit tests.
- Zn Cl1 tailings as an exit stream.
- Regrind sizes at nominally P_{80} 15-20µm.
- 3x concentrates Cu, Pb and Zn.

Figure 13-22 - SW LCT circuit configuration

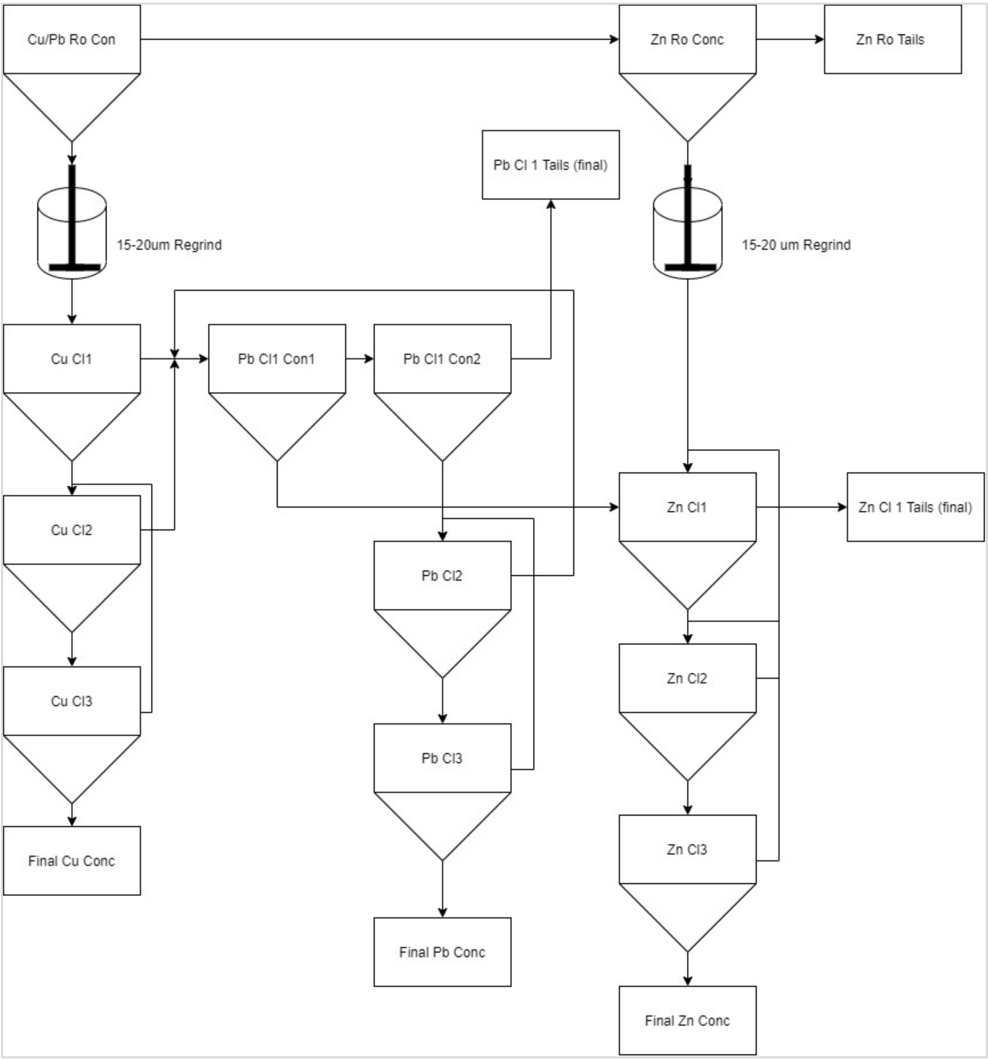


Table 13-24 shows the results from the SW LCT:

- Cu Cl3 concentrate resulted in 25.1% grade at 69% recovery.
  - With the addition of SMBS into the Cu cleaner circuit a very clean Cu concentrate was achieved at good recovery.

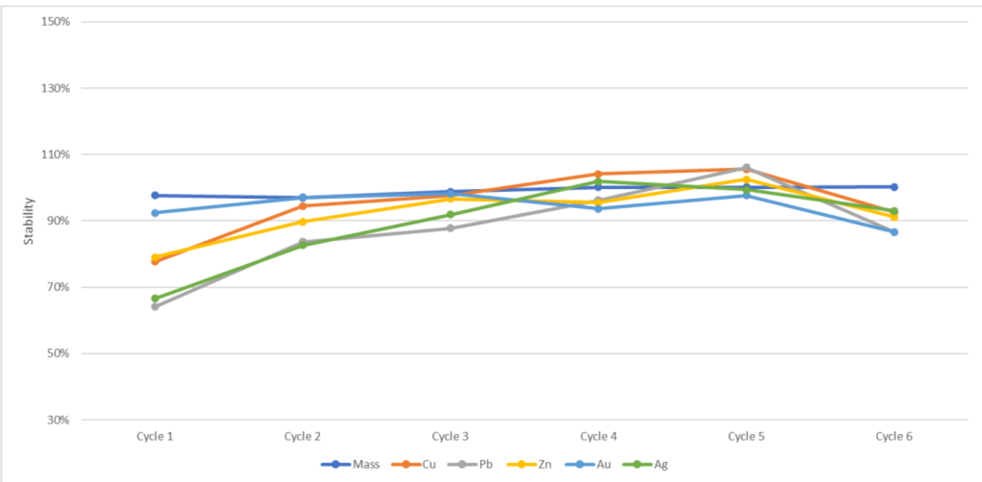
- Pb Cl3 concentrate resulted in 27.7% Pb grade and 16% recovery.
    - Grade and recovery increased throughout the LCT.
    - Initially the concentrate from Pb Cl1 con1 was high in Zn and was sent to the Zn cleaning circuit. However, $ZnSO_4$ added into the Pb cleaners increased depression of Zn in Pb Cl1 con1 during the LCT (ultimately leading it to reporting to Pb Cl1 con1 tails) and led to Pb recovered in this concentrate being added into Zn circuit rather than Pb.
- Zn Cl3 concentrate resulted in 43.7% grade at 54% recovery.
    - Initial grades showed that >50% grade was possible.
    - Pb from Pb Cl1 con1 diluted the Zn concentrate.
    - Au recovery to concentrate was 16% and Ag recovery to contrate was 65%.

Table 13-24 - SW LCT results

By Cycle		Assay Calibrated %, g/t					Recovery Calibrated				
Product	Wt (%)	Cu	Pb	Zn	Au	Ag	Cu	Pb	Zn	Au	Ag
Cu Cl3 Con	1%	25.1	15.1	13.0	0.5	355.5	69%	21%	10%	8%	28%
Pb Cl3 Con	1%	4.4	27.7	14.3	0.6	335.9	5%	16%	4%	5%	11%
Pb Cl1 Tails	5%	0.1	1.7	4.1	0.1	21.9	1%	9%	11%	9%	6%
Zn Cl3 Con	2%	3.1	16.2	43.7	0.1	199.8	14%	37%	54%	3%	26%
Zn Cl1 Tails	6%	0.34	1.04	0.9	0.1	26.4	4%	7%	3%	10%	10%
Zn Ro Tails	85%	0.04	0.13	0.4	0.1	3.9	7%	11%	17%	65%	19%
Total	100%	0.50	0.98	1.8	0.1	17.2	100%	100%	100%	100%	100%

Figure 13-23 shows the stability graph for the SW LCT over 6 cycles. The process was very stable showing that steady state was achieved within the first 3 cycles.

Figure 13-23 - SW LCT Stability



Overall, the results of the LCT were promising with the bulk Cu/Pb concentrate. With some circuit configuration changes, and once the effect of recirculating reagent water is taken into account, there is potential to generate 3 good concentrates with good recoveries.

By keeping the Pb cleaner circuit closed (i.e., no concentrate to Zn cleaner feed) the Pb concentrate recovery would increase and hopefully with higher mass in the system the grade would increase also. The Zn cleaner concentrate would also likely achieve a higher grade of closer to 50%.

An additional LCT with revisions suggested would have been recommended however time constraints on the overall study meant that a repeat was not possible.

13.3.3.5 Blend Testing

Following the conclusion of the MS and SW scoping study blend tests between the two feed were trialled. The ratios trialled were:

- 25:75 - MS:SW
- 50:50 – MS:SW
- 75:25 – MS:SW

As the Cu from the SW would be recoverable, the reagent scheme from the SW rougher and cleaner was used with the bulk float to recover the Cu, Pb and Zn.

Rougher Scoping

Figure 13-24 shows the grade vs recovery curves for Cu on the different blend ratios. With the head grade of Cu on the MS being very low the best performing test was FT2 with the 75% SW feed. 77% recovery at a cumulative grade of 4.2% was achieved outperforming the other blends.

It should be noted that due to carbonate present in the SW at the higher SW blend ratios no additional lime was required in the mill to maintain the targeted pH. With the higher MS blends lime was added to maintain the pH to make the tests comparable.

Figure 13-24 - Cu grade vs recovery curves for blend ratios

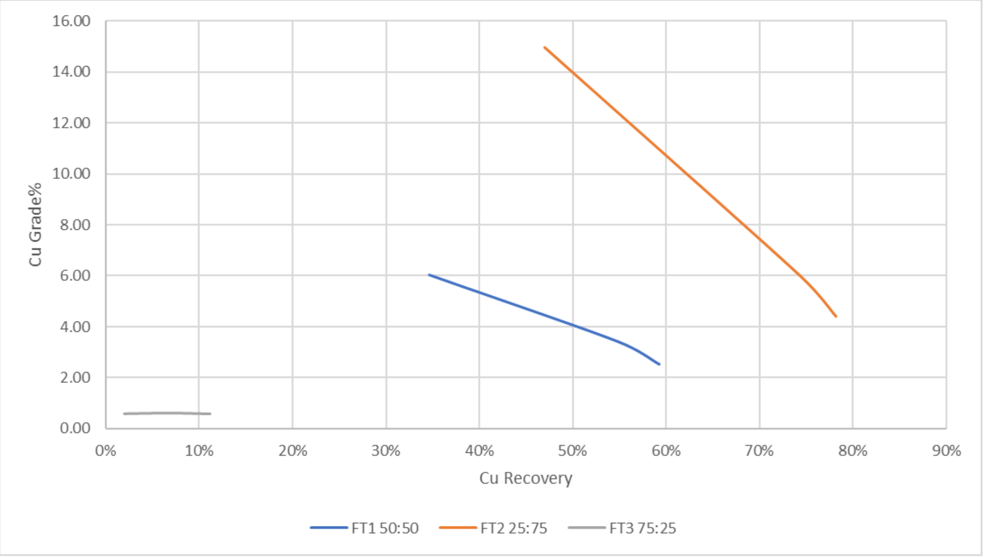


Figure 13-25 and Figure 13-26 shown the grade vs recovery curves for Pb and Zn respectively. As can be seen like the Cu the 25:75 (MS:SW) gave the best performance. This is likely to be because of the higher deportment of pyrite that will be present with the higher MS ratios. The increased pyrite deportment would require variations to the lime dosing and other depressants such as SMBS to control the grade.

Figure 13-25 - Pb grade vs recovery for blend ratios

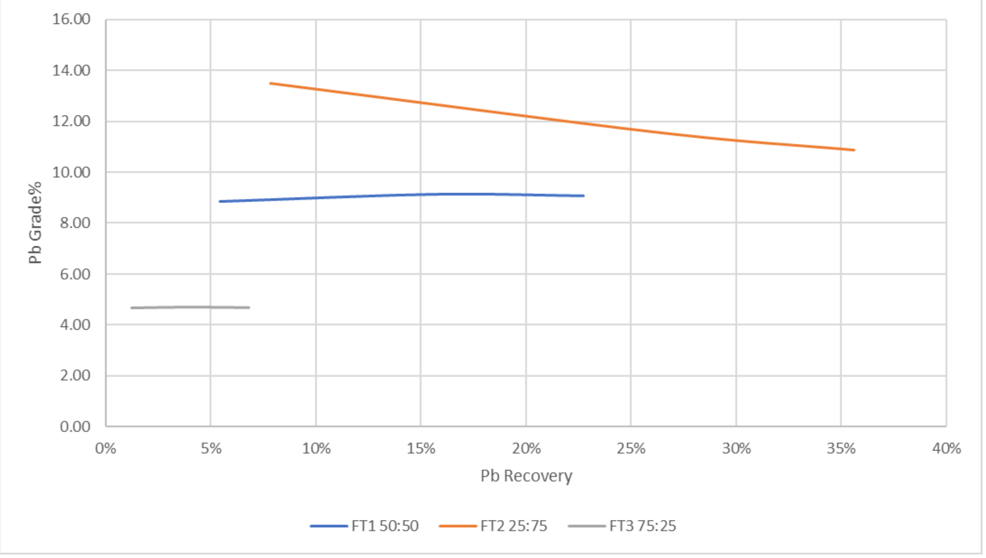
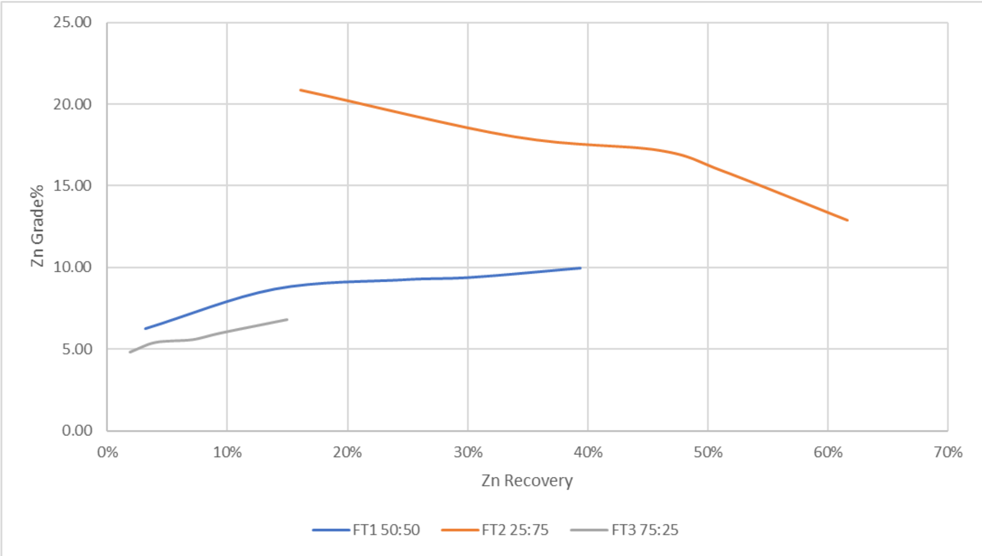


Figure 13-26 - Zn grade vs recovery for blend ratios



Overall, for Cu, Pb and Zn it can be seen that the 25:75 ratio was the best suited going forward to the LCT. Once further evaluation of the resource is made and a ratio of the SW and MS that will be potentially processed together is known, a better scoping study can be determined to fully evaluate and optimise the blended float. Initial floats however show there is scope to float both ore bodies in at the same time.

Locked Cycle Testing

Based off the results from the SW LCT revisions were made to the flowsheet for the MS:SW blend testing (Figure 13-27). The main changes made include:

- Cu Cl2 tailings recirculating to Cu Cl1 feed.
- Combined Pb Cl1 concentrate taken forward to Pb Cl2 feed.

Pb Cl1 tailings added back into Zn Ro feed. Table 13-25 shows the results for the blend LCT. Key points from the test work are:

- Cu grade at 23.5% matches closely with the SW Cu grade, showing good repeatability. Recovery has decreased to 53% which is likely to be associated with the unrecoverable Cu present in the MS.
- Pb grade and recovery is poor at 12% grade and 5% recovery
  - The majority of the Pb was lost to the Zn concentrate suggesting that an increased residence time in the bulk Cu/Pb rougher may potentially recover more Pb to the bulk concentrate.
- Zn grade decreased from >40% to 28% grade, as stated due to the Pb reporting to the Zn Ro.

Figure 13-27- Blend LCT circuit configuration

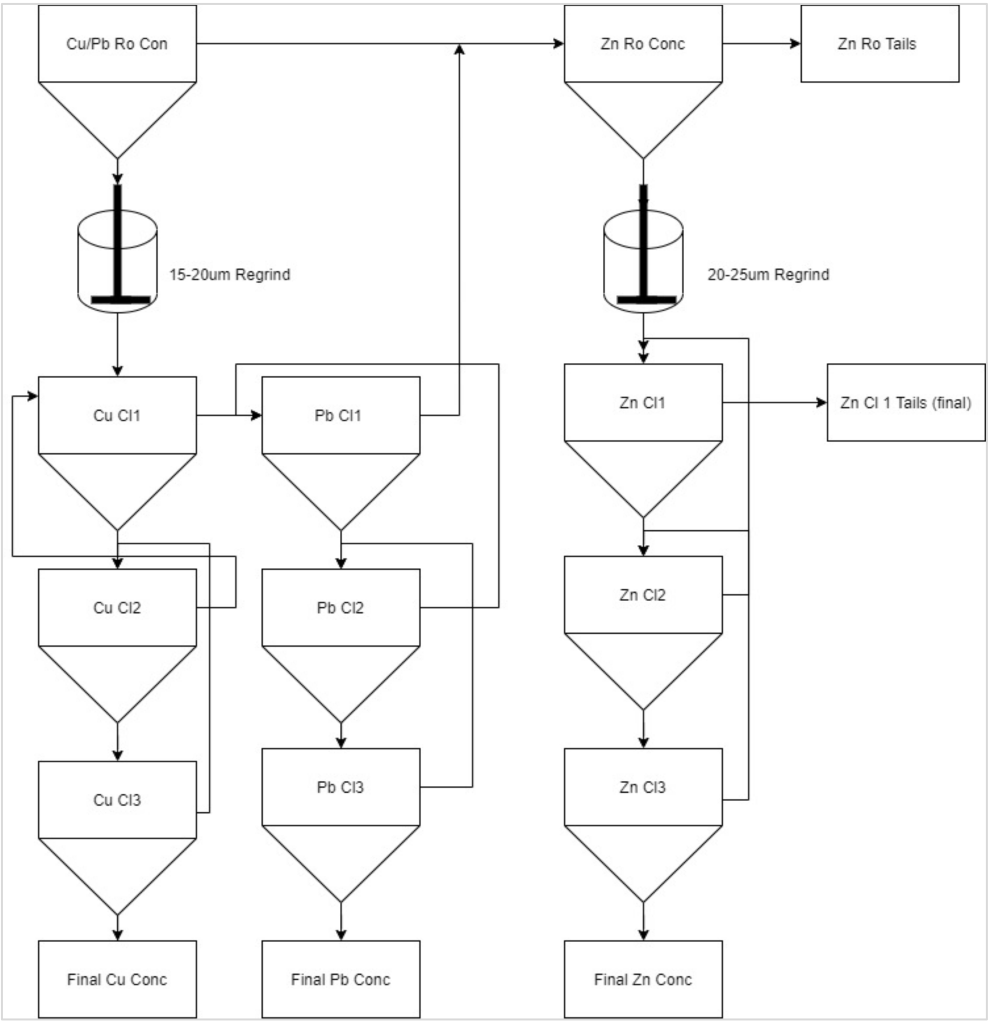
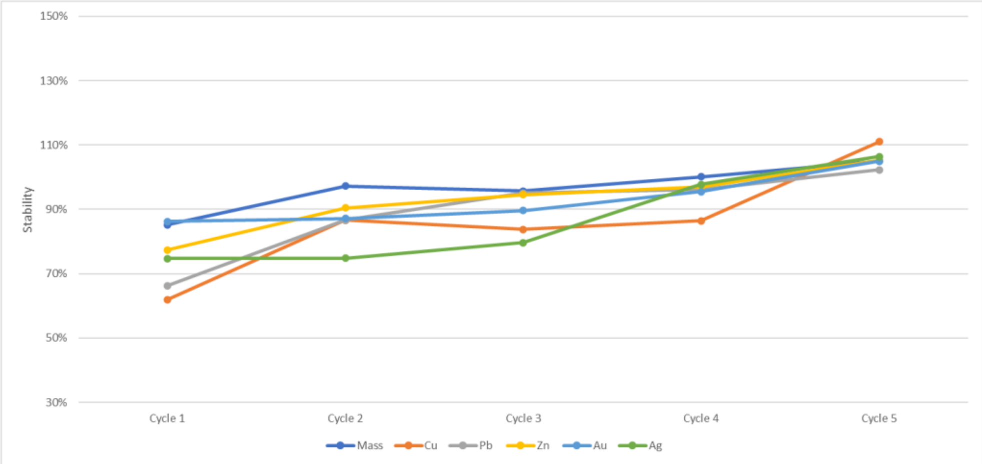


Table 13-25 – Blend LCT results

By Cycle		Assay WJL %, g/t						Recovery Calibrated %					
Product	**Wt (%)**	**Cu**	**Pb**	**Zn**	**Sn**	**Au**	**Ag**	**Cu**	**Pb**	**Zn**	**Sn**	**Au**	**Ag**
Cu Cl3 Con	1%	23.5	6.5	9.6	0.1	0.7	293.1	53%	3%	4%	1%	3%	7%
Pb Cl3 Con	1%	6.2	12.3	13.0	0.1	0.7	293.8	12%	5%	4%	1%	2%	6%
Zn Cl3 Con	5%	0.9	18.2	27.9	0.1	0.5	161.5	11%	53%	64%	6%	10%	23%
Zn Cl1 Tails	7%	0.3	2.0	1.0	0.1	0.3	45.6	4%	8%	3%	9%	11%	9%
Zn Ro Tails	86%	0.1	0.7	0.7	0.1	0.2	23.8	20%	31%	25%	83%	74%	55%
Total	100%	0.4	1.8	2.3	0.1	0.4	37.2	100%	100%	100%	100%	100%	100%

Figure 13-28 shows the stability for the blend LCT over 5 cycles. The first 4 cycles were very stable with little mass and deportment change. Cycle 5 however showed a slight increase in mass. For repeats on the LCT with the blend it would be suggested to run an elongated LCT with nominally 8-10 cycles to fully evaluate the stability and determine whether cycle 5 was anomalous or not.

Figure 13-28 - Blend LCT stability



Overall, the results from the blend testing are promising with the Cu present in the SW still resulting in good grade and recovery. Further rougher optimisation would be required to maximise the Pb and Zn recovery to the relative concentrates.

As an initial test to evaluate the potential for processing a blend the results showed good performance especially considering both feeds were oxidised. With fresh feed and a known ratio based on resource the two ore bodies could be processed together.

13.3.3.6 LEACH TESTING

Flotation pyrite concentrates were submitted to intensive cyanide leach testing. Table 13-26 shows the back calculated head grades from each of the leach tests. The gold grade of the pyrite concentrate is indicated at 0.62 g/t whilst the silver grade is indicated at 59.16 g/t. It should be noted that the feed samples for each of the leach tests were riffled from one bulk feed sample.

Table 13-26 - Back Calculated Head Grades for Intensive Cyanidation Testing of Pyrite Float Concentrate

CN Test	Back Calculated Head mg/kg			
	Au	Ag	Cu	Zn
CN1 33 µm	0.65	61.63	1104	7005
CN2 15 µm	0.64	60.87	1122	6873
CN3 5 µm	0.56	54.98	1095	6194
Average	0.62	59.16	1107	6691

A summary of the leach results is shown in Table 13-27.

Table 13-27 - Summary of Intensive Cyanidation Leach Results

CN Test	48 Hour Extraction %				Reagent Consumption	
	Au	Ag	Cu	Zn	NaCN kg/t	Ca(OH)2 kg/t
CN1 33 µm	9.4	39.5	62.0	12.9	6.84	3.66
CN2 15 µm	15.9	47.4	66.1	12.7	14.06	4.79
CN3 5 µm	25.2	65.2	81.7	24.1	40.46	5.82

The leach results show that gold extraction from the concentrate was low for all tests at between 9.4 % and 25.2 %. A clear trend is apparent that decreasing grind size permits higher gold extraction to the solution. A similar trend is shown for silver where extractions ranged between 39.5 % and 65.2%.

The cyanide content of the leach was maintained at 10 g/l NaCN whilst the dissolved oxygen content of the leach was elevated by the sparging of oxygen during the tests. DO2 content of the tests was recorded at around 20 ppm indicating that the low gold extraction is not due to insufficient reagent. Although it is likely that the low gold extractions are liberation related, it would be worth re-leaching the residue again using an addition of lead nitrate to investigate whether coatings on the gold surface are inhibiting the leach mechanics.

Copper dissolution for the tests reached a maximum during the finest regrind test where 81.7 % of the copper was extracted to solution which provided a final copper in solution concentration of ~250 mg/l. Zinc concentration in solution was also elevated at around 300 mg/l.

Whilst decreasing grind size allowed for high metal extraction it also resulted in higher reagent consumption due to increased surface area of reactive minerals. Cyanide consumption ranged from 6.84 kg/t NaCN when testing at the flotation grind of 33 µm and increased to 40.46 kg/t NaCN when the concentrate was reground to 5 µm. Again, lime consumption was 3.66 kg/t testing at 33 µm and increased to 5.82 kg/t when tested at a regrind of 5 µm.

13.4 CONCLUSIONS

The two composite samples selected by Ascendant Resources originated from the 2019/2020 mineral resources definition drilling of the South Zone that intersected the mineral resources and the average grades of these two composite samples were similar to the average mineral resource grades presented in Section 14 of this Technical Report.

Based on a sample oxidation evaluation by GSL, the samples used for this program of testwork, which is the basis for this PEA, were not truly representative of fresh mineralization that would feed a mineral processing facility.

Micon notes that although the original aim of the GSL testwork program was to undertake a standardised flowsheet development testwork study on a sample of massive sulphide and stockwork from Lagoa Salgada, due to the tainted samples, GSL and Ascendent Resources decided to undertake a more investigatory scoping program to adjust the circuit configuration and reagent scheme to achieve comparative results.

SMD (Stirred Media Detritor) signature plot tests for the fine grinding of samples from a target grind size of approximately 80% passing (P_{80}) 53 µm to float feed target P_{80} sizes of 29 µm for the MS sample and 37 µm for the SW sample gave energy requirements of 6.84 kWh/t and 2.06 kWh/t for Ms and SW, respectively.

The requirement for a very fine primary grind will result in relatively high comminution capital costs for multiple crushing and grinding stages and high operating costs for grinding power and grinding media.

Gravity separation tests on the MS sample gave no selectivity, recoveries were proportional to mass pull.

Intensive cyanide leach tests on flotation pyrite concentrates gave metal extractions after 48 hours of between 9% and 25% for gold, between 40% and 65% for silver, between 62% to 82% for copper and between 13% and 24% for zinc.

The process flowsheet used for the locked cycle tests (see Figure 13-27- Blend LCT circuit configuration) comprised the following:

- Primary grind to target P_{80} sizes of29 µm for the MS sample and 37 µm for the SW sample.
- Rougher copper/lead flotation followed by zinc flotation.
- Regrinding of Cu/Pb and zinc rougher concentrates to target P_{80} size of 15 to 20 µm.
- Three stages of copper, lead and zinc flotation cleaning.
- Dewatering of separate Cu, Pb and Zn concentrates.
- Tailings dewatering and disposal.

The LCT completed for the MS samples resulted in Pb concentrate grade of 22% with 43% recovery and a Zn concentrate grade of 35% at 66% recovery. Separate Cu and Pb concentrates were not recovered for the MS tests.

The LCT completed for the SW sample resulted in Cu concentrate grade of 25% at 69% recovery, Pb concentrate grade of 28% at 16% recovery and Zn grade of 44% at 54% recovery.

A LCT using a blend of MS and SW resulted in Cu concentrate grade of 24% at 53% recovery, Pb concentrate grade 12% at 5% recovery Zn concentrate grade 28% at 64% recovery.

13.5 RECOMMENDATIONS

A review of the available metallurgical testwork results to date suggests that there is insufficient metallurgical data available to allow an accurate forecast of metallurgical performance for the deposit. The process design criteria for this PEA should include the comminution and flotation requirements developed in this testwork program although any additional recovery of gold and tin using gravity separation or leaching technologies cannot be justified at this time due to lack of any supporting data.

Metallurgical recoveries and concentrate grades assumed for the PEA should take into account the results presented above. Although the samples used were oxidized, these flotation test results, together with other historical work, suggest that the poly-metallic mineralization Lagoa Salgada is complex, and it will be challenging to produce high quality concentrate products with high recoveries.

Although detailed final concentrate characterization was not included in the most recent program of metallurgical testwork, historical results suggest that the concentrates produced may include deleterious elements (to be confirmed with additional test work). Depending on the additional test work results, penalties may need to be applied in future techno-economic studies.

A new program of testwork is recommended using fresh drill core samples that represent the different lithologies found within the mineral resources. This program should use the recent and historical testwork as a basis and include the following

- Detailed mineralogical characterization including particle liberation analyses and valuable metal deportment.
- Detailed multi-element analyses of the samples.
- Comminution tests to develop design criteria to support the crushing, primary coarse and fine grinding and concentrate regrinding circuits.
- Flotation tests to provide detailed design information and to quantify final concentrate recoveries and grades that can be used to support a preliminary technical study.
- Detailed characterization of flotation concentrates and preliminary concentrate marketing studies.
- Preliminary thickening and filtration tests to support the design of dewatering equipment included in the process flowsheet.
- Preliminary geochemical studies of tailings, waste rock and economic mineralization.

14 MINERAL RESOURCE ESTIMATES

Micon estimated the LS project mineral resources in September 2019. In 2020/2021, Ascendant/Redcorp drilled 6 step-out drill holes on the South deposit. The 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. The merging has necessitated an update of the South deposit resource. However, the North deposit resource remains current as no additional drilling and/or reinterpretation has been conducted on the deposit. The estimation protocols/methodologies used in 2019 for the North deposit and in the 2021 updating of the South deposit resource are essentially the same and are described below.

14.1 EXPLORATORY DATA ANALYSIS

14.1.1 DATABASE DESCRIPTION

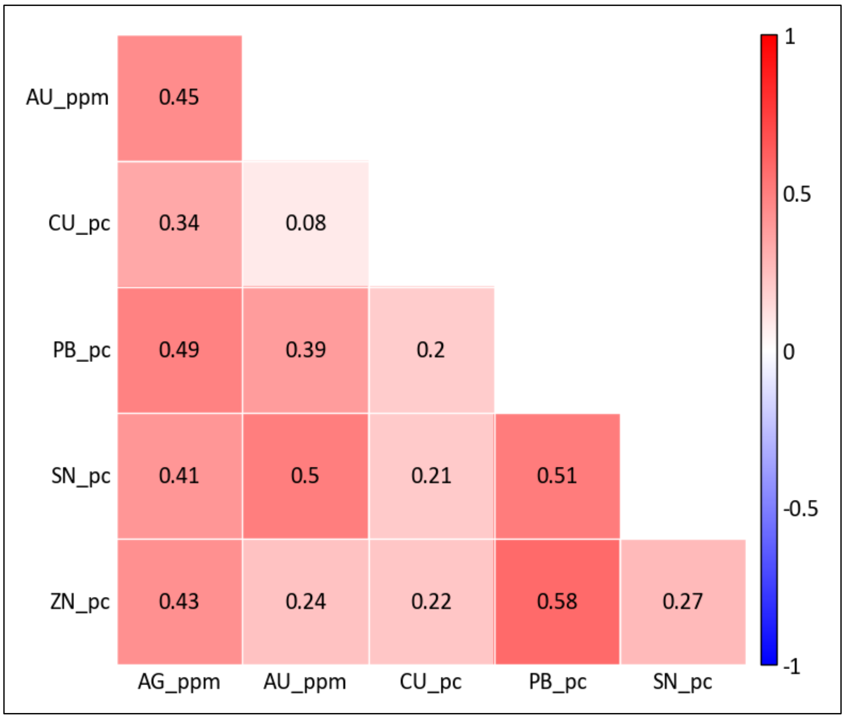
The LS property deposits have been tested by diamond drilling over a cumulative strike length of approximately 1.7 km and down to a vertical depth of about 600 m. The resource database is derived from 82 surface diamond drillholes, all of which were utilized in the resource estimation. Original assay certificates from the laboratory were provided as csv documents. A detailed DTM and overburden depth model were provided as dxf surfaces.

The average drillhole spacing in the best drilled areas of the project is about 20 m; the spacing in the more poorly drilled areas is between 40 and 150 m.

14.1.2 DEPOSIT COMPONENTS

The LS property is comprised of multi-metal deposits whose chief components are zinc, lead, copper, gold, silver, and tin. The global correlation matrix (Figure 14-1) shows that, save for zinc and lead, the coefficients of correlation between the deposit components are generally poor despite these elements occurring together within the deposits. This poor correlation is partly attributed to post mineralization processes such as metamorphism and remobilization which affected the metals differently.

Figure 14-1 - Global Correlation Matrix for the LS Deposit



Source: Micon 2019/2021.

14.2 OVERVIEW OF ESTIMATION METHODOLOGY

Following the completion of the database validation as outlined in Section 12.0 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.

The estimation was conducted using both the ED50 co-ordinate reference system (CRS) and projection to UTM Zone 37N, and WGS 84.

14.3 GEOLOGICAL INTERPRETATION

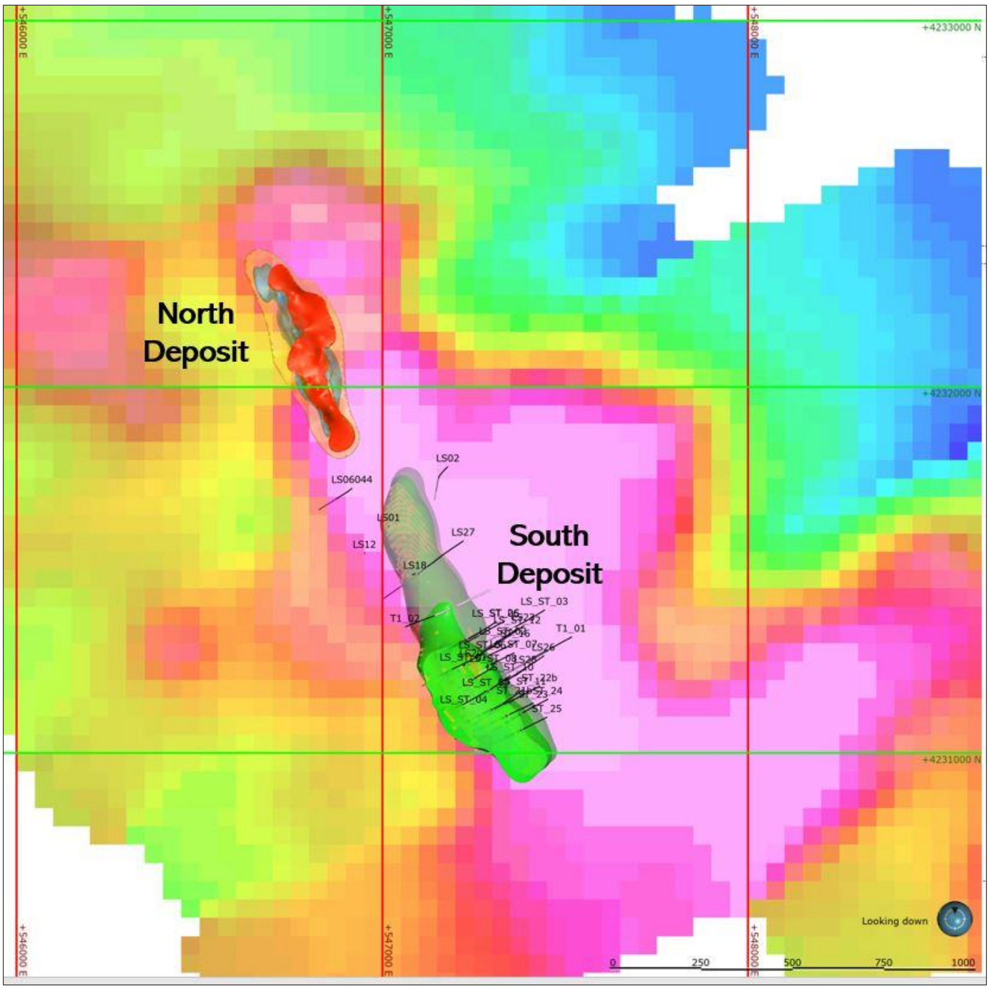
The mineralization extends continuously beneath Tertiary cover rocks over the entire drilled strike length of about 1.7 km. However, two deposits are recognized; these are the North deposit (formerly LS-1 deposit), and the South deposit (formerly LS-1 now merged/combined with the former Central sector). The respective locations are shown in Figure 14-2. There is currently an undrilled gap of 250 m separating the two deposits, but Geophysics shows that mineralization is continuous between the two and beyond in either direction along the NW – SW trend.

The North deposit is complex in that it hosts three types of mineralization: GO mineralization, primary sulphide mineralization, and stringer mineralization beneath, and on the periphery of, the primary sulphide zone. The GO mineralization resulted from the weathering of the underlying primary sulphide mineralization.

The South deposit comprises copper-rich stringer / fissure / stockwork mineralization and isolated gold-rich silicified zones which appear to be structurally controlled. It is hosted in a unit locally known as the iVfr. The iVfr is heterogeneous, being 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 iVfr 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 primary sulphide zone of the North deposit whereas zinc, lead, copper, gold, and silver are common to all the deposits. However, copper is apparently dominant over zinc, lead, gold, and silver in the South deposit.

Figure 14-2 - Geophysical Map Showing the Location of the Currently Known Deposits on the LS Property



Source: Redcorp/Ascendant 2019/2021.

14.4 Selection and Modelling of Estimation Domains

14.4.1 North Deposit

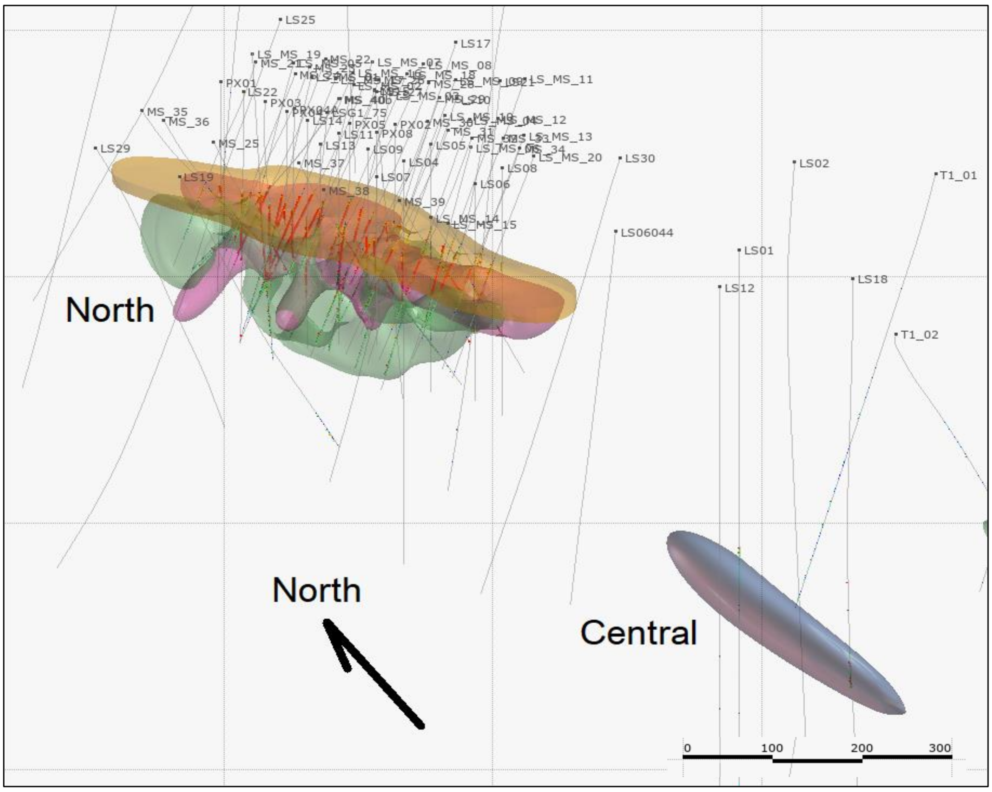
Micon's estimation domain selection criterion is based purely on geology for the GO and MS domains and on the zinc equivalent (ZnEq) threshold value for the stringer / fissure / stockwork domains. To obtain the ZnEq% threshold value for modelling, the threshold grade for each metal in the stringer zones was obtained from probability / log-probability plots; thereafter, the threshold grades for each metal were combined into a ZnEq% threshold value using the following formula:

*ZnEq% = ((Zn Grade*25.35)+(Pb Grade*23.15)+(Cu Grade * 67.24)+(Au Grade*40.19)+(Ag Grade*0.62)+(Sn Grade * 191.75))/25.35*

The ZnEq threshold value for the stringer zone North deposit was established as 1.53% while that for the Central deposit was established as 0.95% ZnEq. The South deposit modelling is detailed in Section 14.4.2**Error! Reference source not found.**.

Drillhole intercepts were coded using the geological and ZnEq criteria described above. Following coding, domain wireframes were created by implicit modelling using the Leapfrog mining software. The modelled domain wireframes are shown in Figure 14-3.

Figure 14-3 - 3D Perspective of the LS Project North Deposit Domain



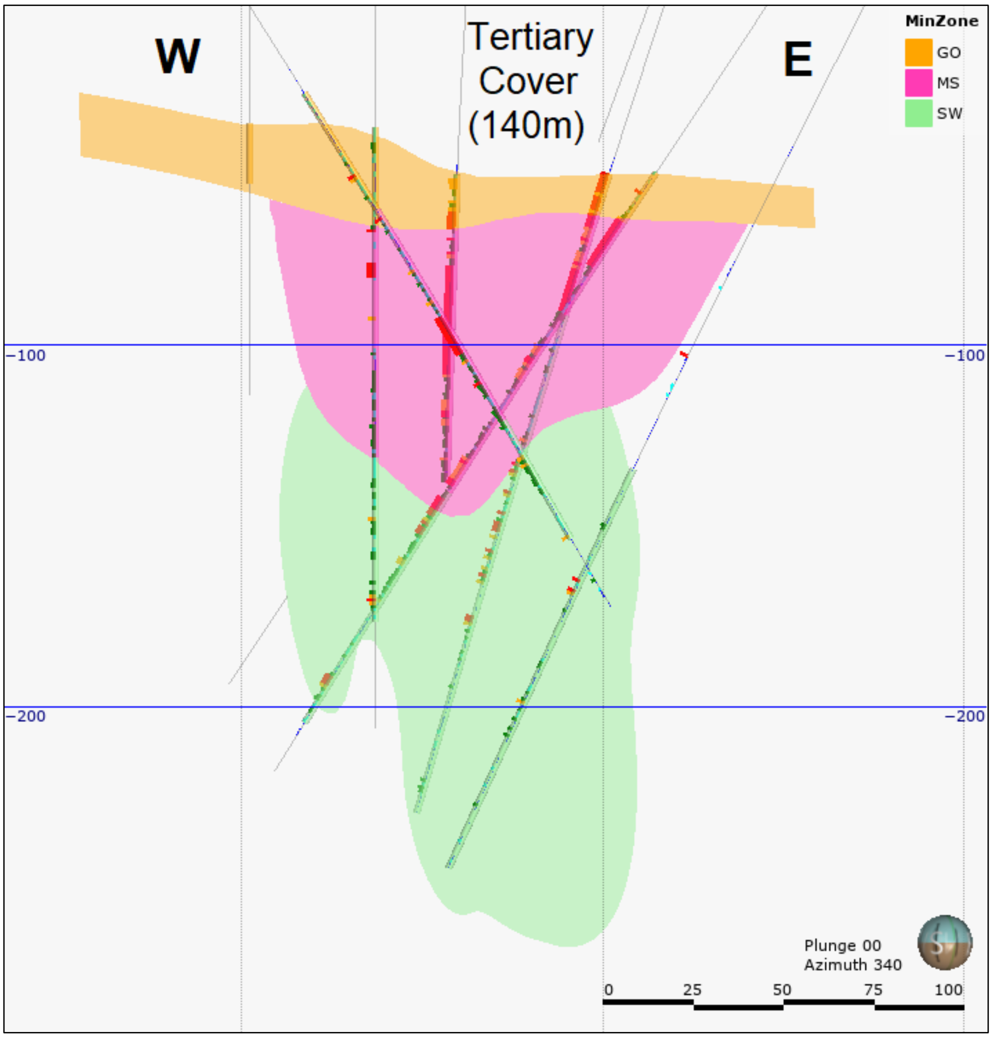
Source: Micon 2019.

In summary, the selected domains are as follows:

- GO (brown / yellow)
- MS (purple)
- Stringer zone (green)

Boundaries between the domains are naturally hard as dictated by the geology. A section through the North deposit which has three domains, namely GO, MS, and stringer zone (SW) is shown in Figure 14-4.

Figure 14-4 - East-west Section through the North deposit estimation domains



Source: Micon 2019.

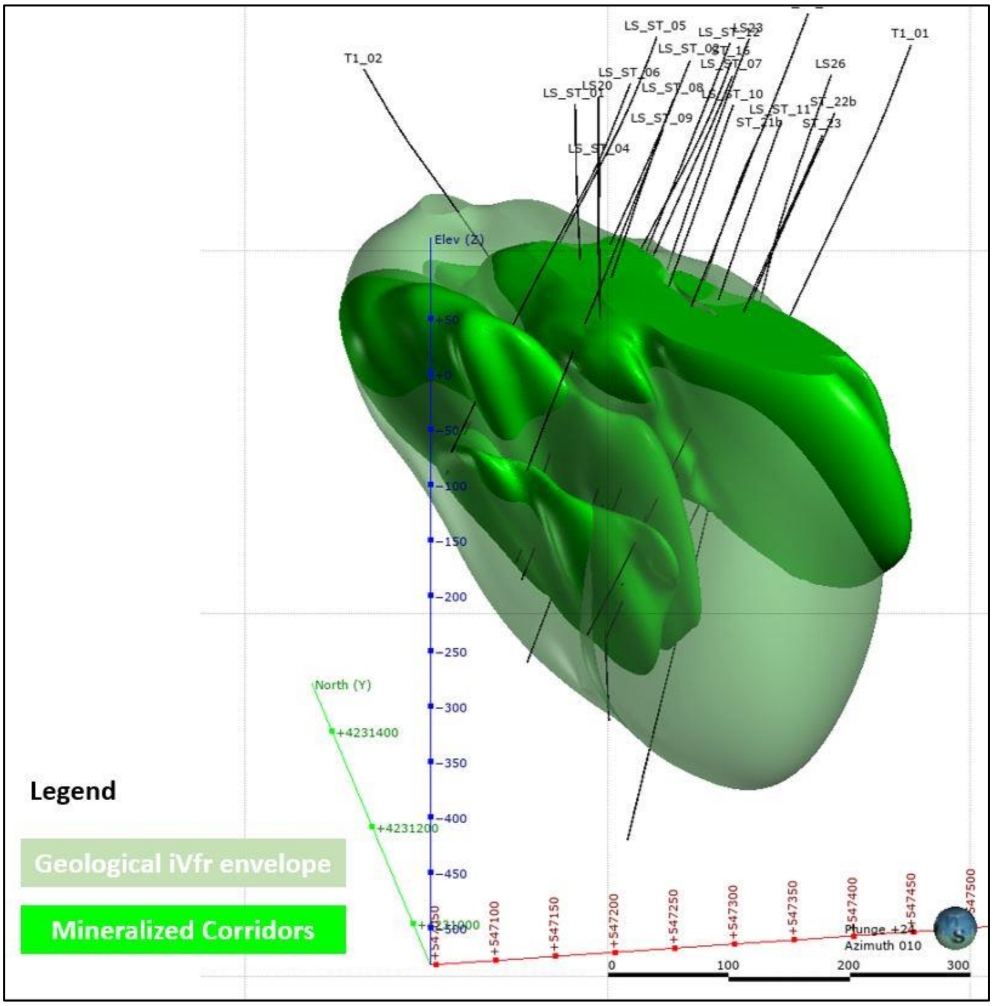
14.4.2 South Deposit

The modelling involved the following steps:

- Defining the boundary/outer limit of the iVfr unit using geological logs
- Assigning each sample within the iVfr, a copper equivalent value (CuEq) using the formula: CuEq% = ((Cu Grade * 67.24) +(Zn Grade*25.35) + (Pb Grade*23.15) + (Au Grade*40.19) + (Ag Grade*0.62))/67.24 and calculating the mean CuEq grade which was found to equal 60%CuEq.
- Wireframing/modelling the main corridors of mineralization using the mean 60%CuEq as the threshold value of the main mineralization corridors within the iVfr unit.

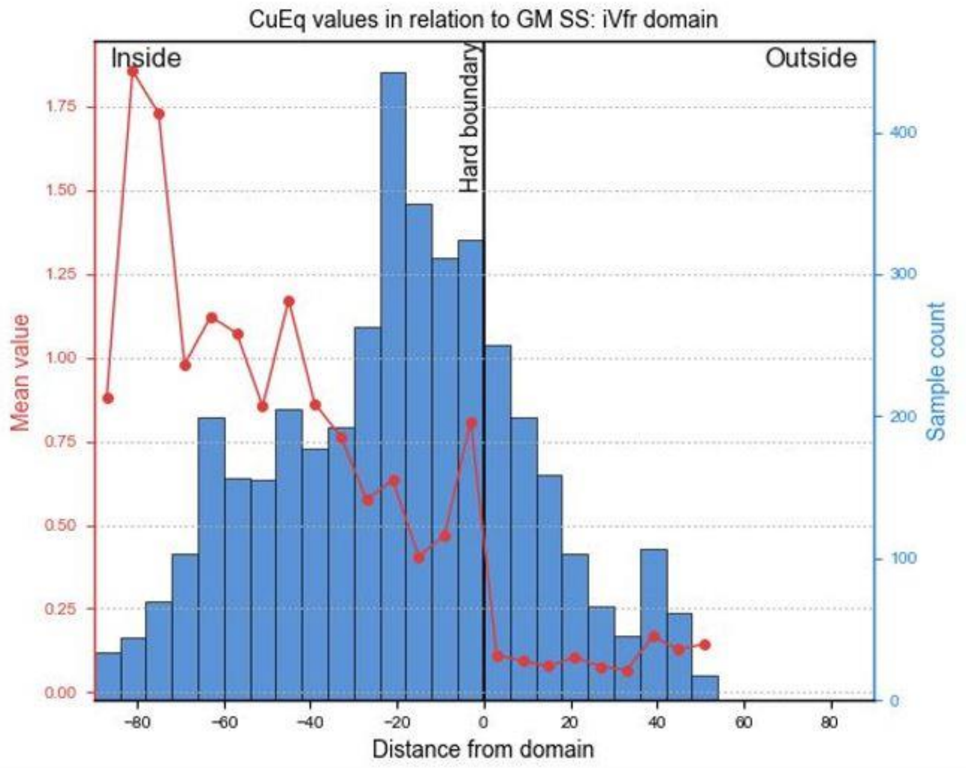
The relationship between the iVfr unit and wireframe of the mineralized corridors is shown in Figure 14-5.

Figure 14-5 - 3D Perspective of the South Deposit



Statistical analysis confirms that the boundary between the geological envelope and the mineralized corridors is hard as shown in Figure 14-6.

Figure 14-6 - South Deposit Boundary Analysis Between Mineralized Corridor and Geological Model

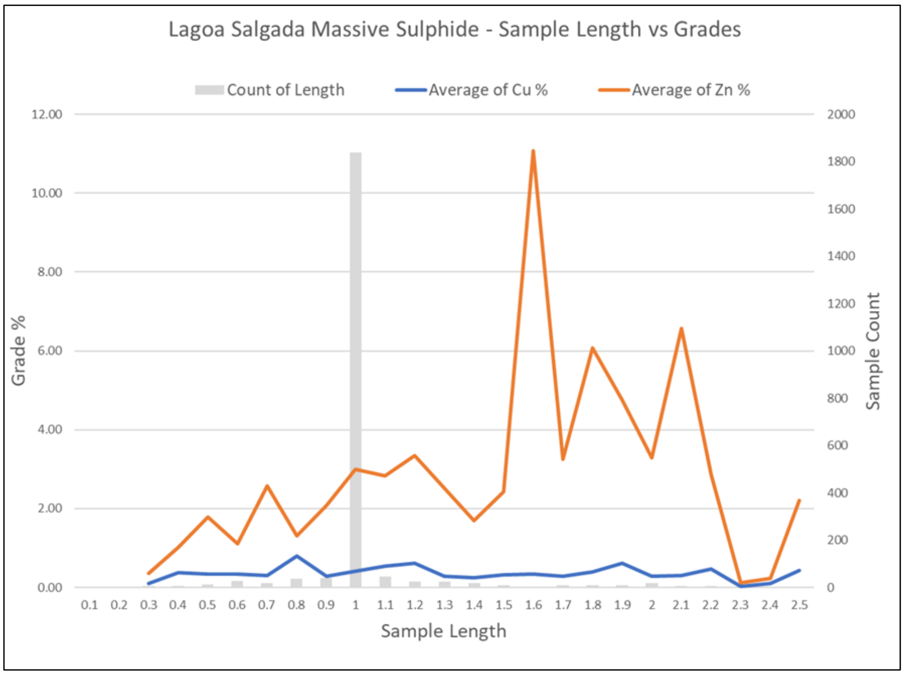


14.5 GRADE CAPPING, COMPOSITING, STATISTICS AND VARIOGRAPHY

14.5.1 GRADE CAPPING, COMPOSITING AND STATISTICS

Micon investigated 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 as illustrated in Figure 14-7. Thus, the determination of grade capping threshold values was conducted on raw samples using population histograms and probability / log probability plots. The summary statistics and log-probability plots for the MS domain (i.e., the best/highest grade mineralized domain) are shown in Figure 14-8 to Figure 14-13. Grade capping values are indicated in red on the plots for each element. The same procedure was followed for the other domains, including the South deposit.

Figure 14-7 - Grade versus Sample Length in the MS Domain



Source: Micon 2019/2021.

Figure 14-8 - Domain MS Summary Statistics and Probability / Log-probability Plot for Zn

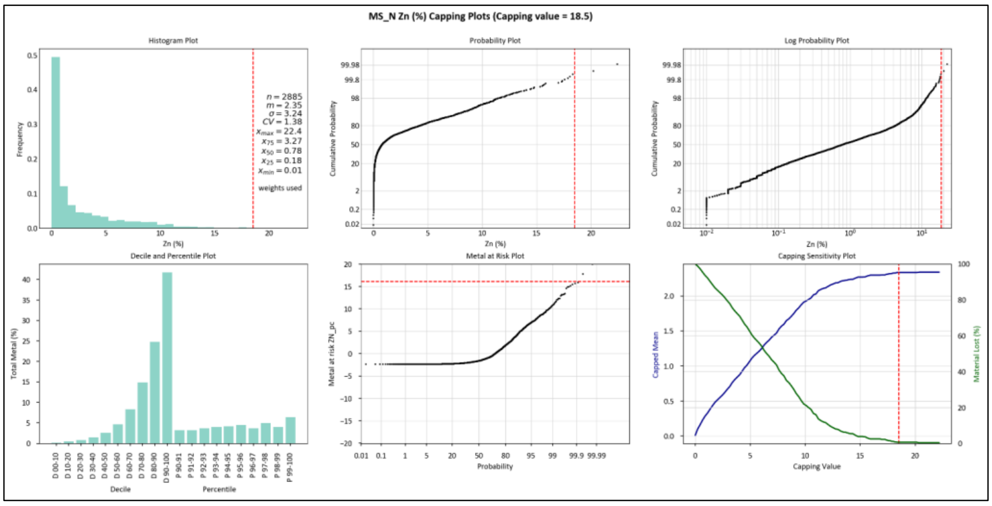


Figure 14-9 - Domain MS Summary Statistics and Log-probability Plot for Pb

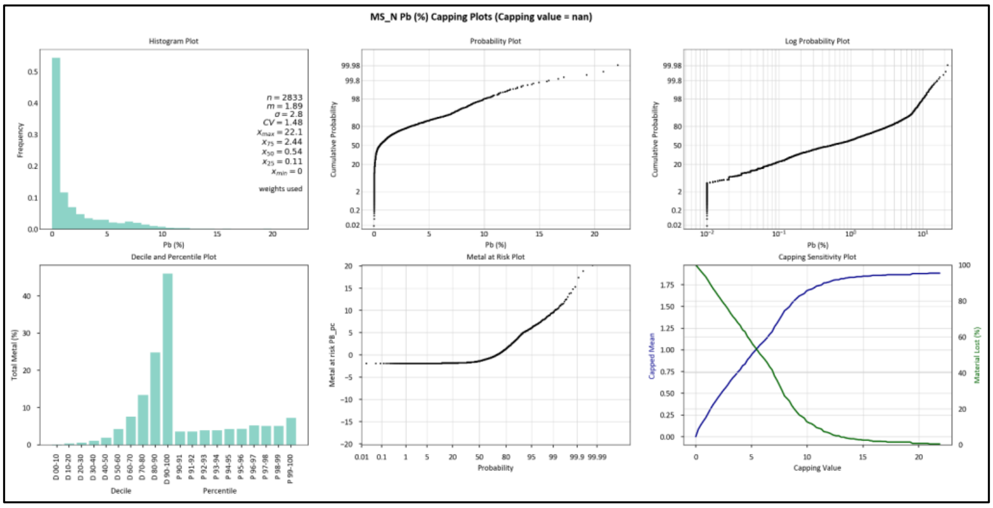


Figure 14-10 - Domain MS Summary Statistics and Log-probability Plot for Cu

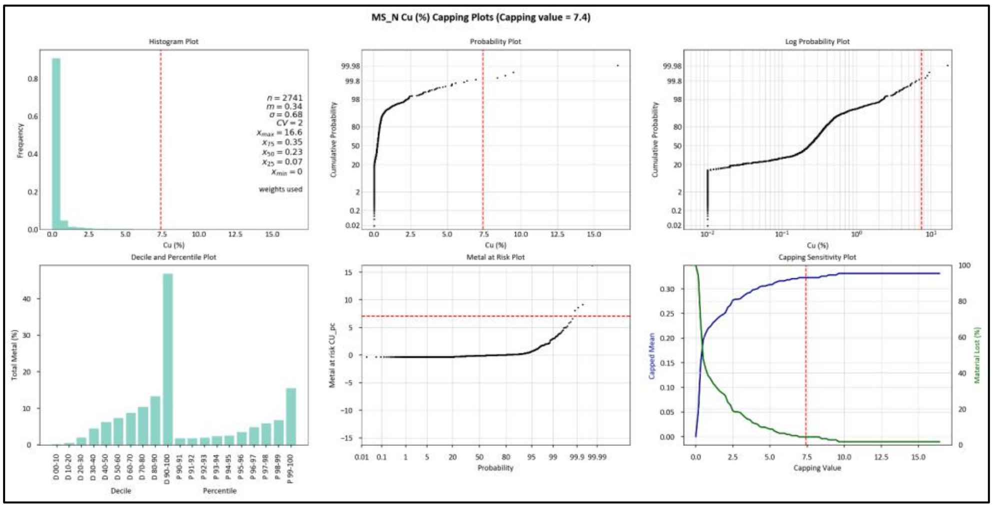


Figure 14-11 - Domain MS Summary Statistics and Log-probability Plot for Au

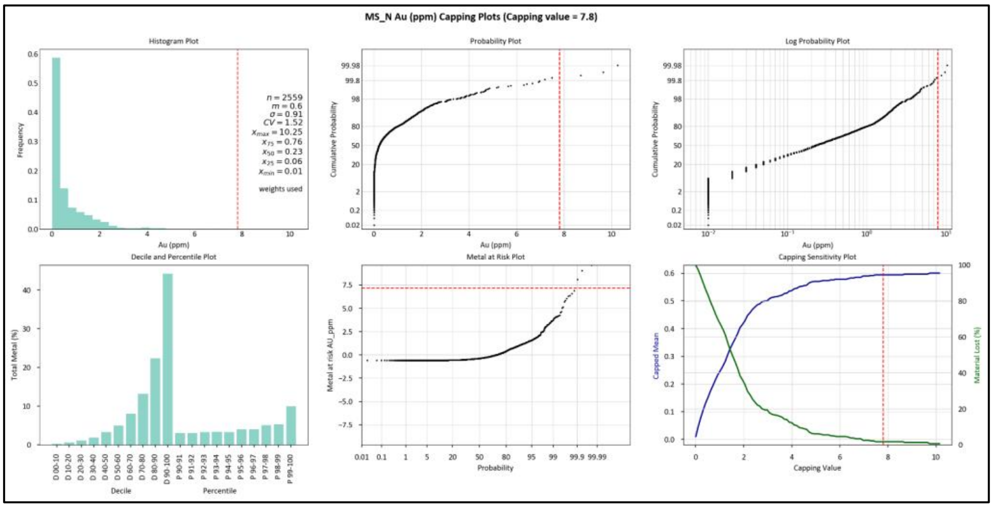


Figure 14-12 - Domain MS Summary Statistics and Log-probability Plot for Ag

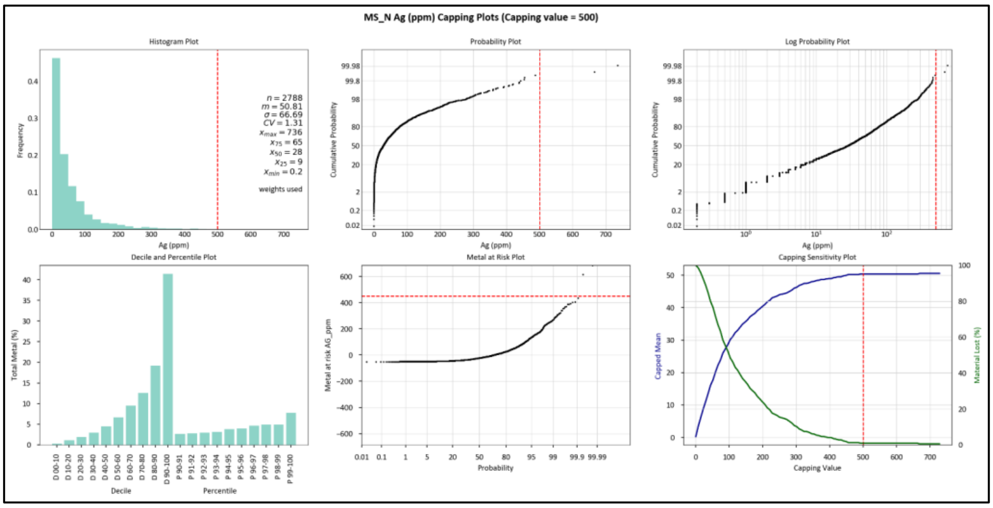
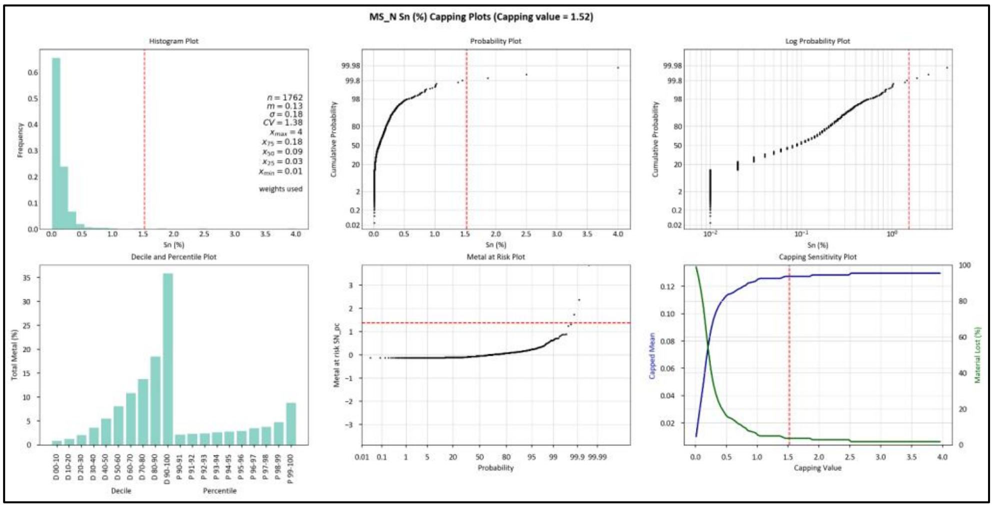


Figure 14-13 - Domain MS Summary Statistics and Log-probability Plot for Sn

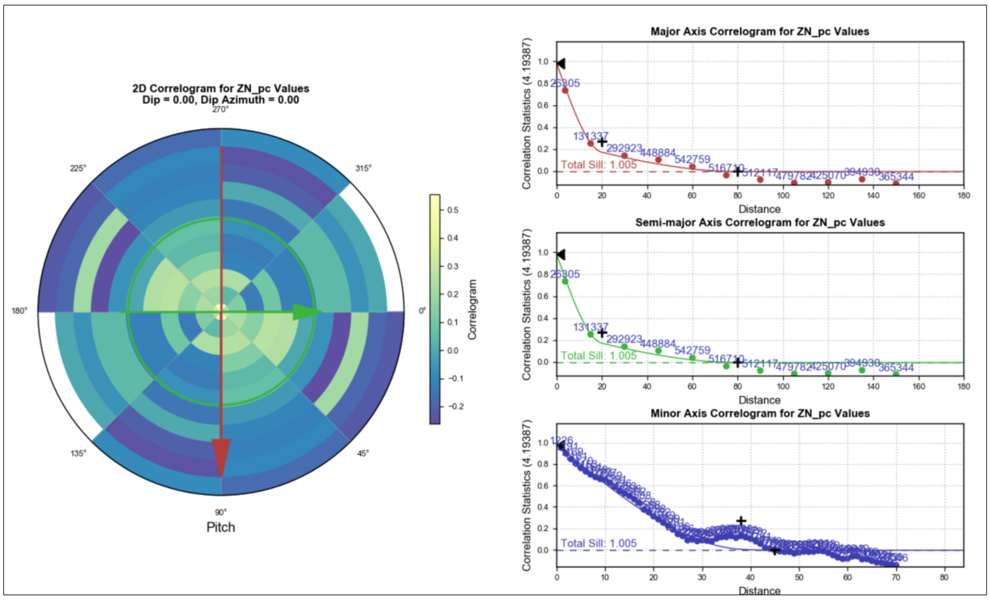


The capping was applied after compositing to 2 m to give equal weighting to the values prior to variography. The mode (average) of the sample lengths within the modelled estimation domains is 1 m and the standard practice would be to use this as the composite length. However, given the significant number of samples greater than 1.0 m in the LS-series drillholes (Figure 14-5) Micon's view is that 2 m is the best option. By taking this option, Micon does not believe the choice of 1 m versus 2 m would make a material difference to the estimation process, providing that the estimation searches are optimized. The summary statistics of the capped and un-capped composites are shown in Table 14-1.

14.5.2 VARIOGRAPHY

Micon completed a geostatistical analysis of all domains to determine the optimum grade interpolation parameters. Conventional variograms were difficult to discern and correlograms were computed instead. Examples of the correlograms are shown in Figure 14-14 and Figure 14-15, which clearly demonstrate how the search ellipse ranges where determined. Note that Zn and Cu have been selected because the resources are based on Zn Eq (for North Deposit) and CuEq (for South Deposit).

Figure 14-14 - North Deposit Domain MS Correlogram for Zn



N.B. The similar long ranges (80 m) of influence along the major and semi-major directions confirm the isotropic nature of the MS domain.

Figure 14-15 - South Deposit Correlogram for Cu

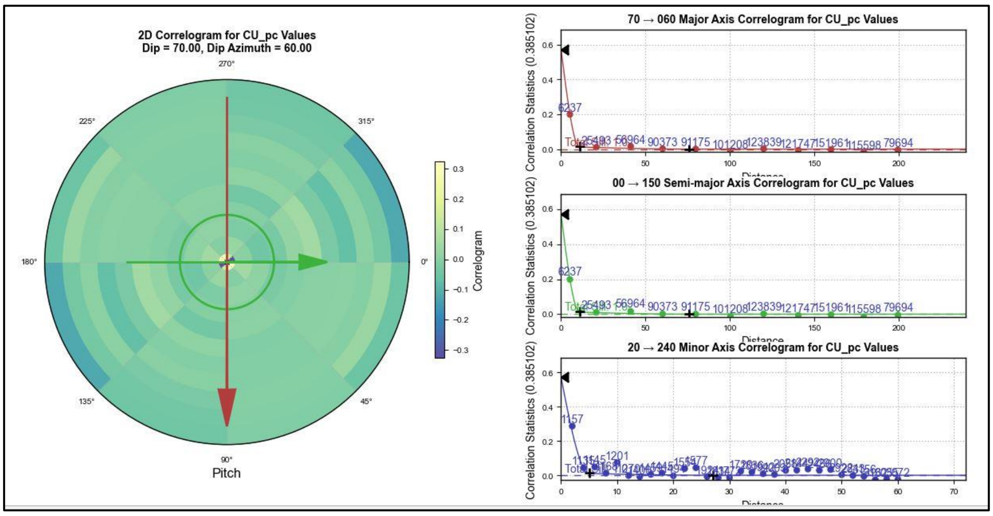


Table 14-1 - Summary Statistics of the Capped and Un-capped Composites

		AG_CAP	AG_ppm	AU_CAP	AU_ppm	CU_CAP	CU_pc	PB_CAP	PB_pc	SN_CAP	SN_pc	ZN_CAP	ZN_pc
Gossan	Count	373	373	373	373	373	373	373	373	250	250	373	373
	Length	733.62	733.62	733.62	733.62	733.62	733.62	733.62	733.62	490.03	490.03	733.62	733.62
	Capped Comps	2	0	1	0	2	0	1	0	0	0	2	0
	Mean	30.65	31.45	0.58	0.60	0.10	0.10	2.14	2.15	0.16	0.16	0.48	0.48
	Min	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.01
	Median	6.12	6.12	0.10	0.10	0.04	0.04	0.54	0.54	0.02	0.02	0.36	0.36
	Max	520.00	740.50	20.00	27.58	1.50	3.55	30.50	35.67	1.92	1.92	2.40	4.56
	CoV	2.33	2.47	2.78	3.11	1.88	2.53	1.72	1.77	2.02	2.02	0.83	0.90
Massive	Count	1146	1146	1146	1146	1146	1146	1146	1146	795	795	1146	1146
	Length	2283.45	2283.45	2283.45	2283.45	2283.45	2283.45	2283.45	2283.45	1583.41	1583.41	2283.45	2283.45
	Capped Comps	1	0	1	0	0.00	0	0	0	2	0	0	1
	Mean	62.71	62.72	0.68	0.68	0.40	0.40	2.42	2.42	0.14	0.15	2.90	2.90
	Min	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.01
	Median	40.59	40.59	0.34	0.34	0.28	0.28	1.23	1.23	0.11	0.11	1.52	1.52
	Max	500.00	510.21	7.80	8.68	6.85	6.85	20.98	20.98	1.52	2.64	18.50	19.71
	CoV	1.07	1.07	1.31	1.32	1.53	1.53	1.17	1.17	1.04	1.17	1.15	1.15
Stringer (North)	Count	1215	1215	1215	1215	1215	1215	1215	1215	925	925	1215	1215
	Length	2408.12	2408.12	2408.12	2408.12	2408.12	2408.12	2408.12	2408.12	1831.52	1831.52	2408.12	2408.12
	Capped Comps	1	0	0	0	0	0	2	0	2	0	3	0
	Mean	9.54	9.57	0.07	0.07	0.14	0.14	0.17	0.17	0.02	0.02	0.59	0.60
	Min	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.01
	Median	6.33	6.33	0.03	0.03	0.04	0.04	0.08	0.08	0.01	0.01	0.44	0.44
	Max	175.00	202.91	2.17	2.17	3.02	3.02	3.30	10.71	1.25	1.48	6.00	14.25
	CoV	1.36	1.39	2.18	2.18	1.82	1.82	1.75	2.46	2.91	3.12	1.00	1.13
Stringer (South)	Count	792	792	792	792	792	792	792	792	96	96	792	792
	Length	1569.95	1569.95	1569.95	1569.95	1569.95	1569.95	1569.95	1569.95	191.00	191.00	1569.95	1569.95
	Capped Comps	1	0	2	0	3	0	2	0	0	0	0	0
	Mean	12.56	12.62	0.05	0.05	0.34	0.35	0.72	0.72	0.01	0.01	1.26	1.26
	Min	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.01
	Median	5.00	5.00	0.02	0.02	0.10	0.10	0.33	0.33	0.01	0.01	0.65	0.65
	Max	255.00	300.71	0.58	0.63	6.40	10.07	12.00	12.85	0.04	0.04	11.83	11.83
	CoV	1.80	1.84	1.56	1.57	2.12	2.32	1.61	1.63	0.78	0.78	1.34	1.34

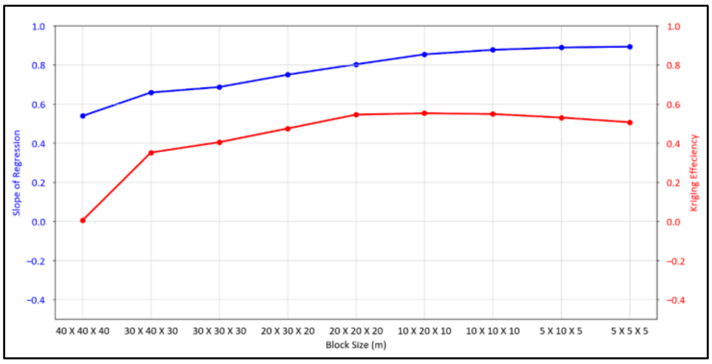
14.6 ESTIMATION

14.6.1 BLOCK SIZE ANALYSIS

14.6.1.1 NORTH DEPOSIT

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 MS 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. The results indicate an optimum block size of 5 x 10 x 5 m, as demonstrated in Figure 14-16 below.

Figure 14-16 - North Deposit MS Domain Block Model Sensitivity Analysis

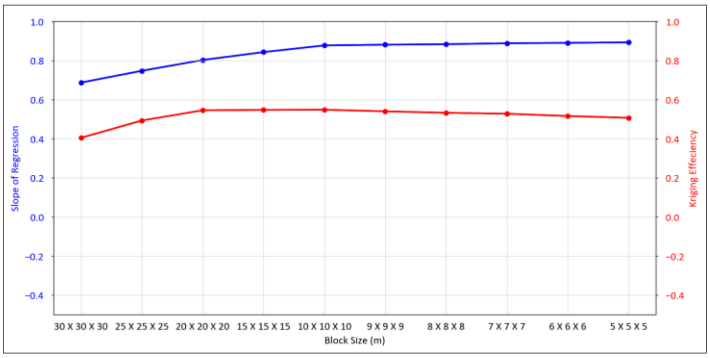


Source: Micon 2019/2020.

14.6.1.2 SOUTH DEPOSIT

A similar exercise on the South deposit indicates an optimum block size of 10 x 10 x 10 m as shown in Figure 14-17.

Figure 14-17 - South Deposit Block Model Sensitivity Analysis



14.6.2 Resource Block Model Definition

The North deposit block model attributes are presented in Table 14-2. The upper limit (Z) is approximately 40 m above the GO domain contact with the overlying Tertiary cover rocks. The block size is based on the sensitivity analysis (Figure 14-16), drillhole spacing, the envisaged selective mining unit (SMU) and the geometry of the deposit.

Table 14-2 - LS North Deposit Block Model Definition

Item	X	Y	Z
Origin Coordinates	547200	4231080	3
Extents	445	1600	655
Block Size m (parent)	5	10	5
Azimuth (Degrees)	140		

The South deposit block model attributes are presented in Table 14-3 - LS South Deposit Block Model Definition. The block size is based on the sensitivity analysis shown in Figure 14-17 above.

Table 14-3 - LS South Deposit Block Model Definition

Item	X	Y	Z
Origin Coordinates	547200	4231550	150
Extents	340	630	640
Block Size m (parent)	10	10	10
Azimuth (Degrees)	140		

14.6.3 Bulk Density

Bulk density measurements were conducted as described in Section 12. The average calculated density values used to estimate the tonnage in each domain are as follows:

- GO = 3.12

- MS = 4.76
- SW (North deposit stringer zone) = 2.88
- South deposit = 3.0

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 two-pass estimation procedure for all domains was used for the interpolation. For both passes, the maximum 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 14-4 and in all cases respect the variogram ranges for each element.

Table 14-4 - Summary of Search Parameters for the North Deposit

Domain	Element	Pass	Interpol. method	Y (m)	X (m)	Z (m)	Dip (º)	Dip Az. (º)	Pitch (º)	Min S	Max S	Max S/DH
GO	Au	1	OK	60	40	15	0	0	59	9	18	3
	Ag	1	OK	50	40	15	0	0	57	9	18	3
	Cu	1	OK	60	40	15	0	0	58	9	18	3
	Zn	1	OK	50	45	20	0	0	56	9	18	3
	Pb	1	OK	80	40	20	0	0	56	9	18	3
	Sn	1	OK	60	40	15	0	0	32	9	18	3
Massive Main (MS)	Au	1	OK	80	40	30	63	70	12	9	18	3
	Ag	1	OK	80	50	40	63	70	168	9	18	3
	Cu	1	OK	100	50	30	63	70	168	9	18	3
	Zn	1	OK	100	50	40	63	70	146	9	18	3
	Pb	1	OK	90	40	30	63	70	12	9	18	3
	Sn	1	OK	100	40	40	63	70	168	9	18	3
Stringer (SW)	Au	1	OK	60	40	40	0	0	78	9	18	3
	Ag	1	OK	80	40	40	0	0	56	9	18	3
	Cu	1	OK	50	40	40	0	0	57	9	18	3
	Zn	1	OK	60	40	40	0	0	57	9	18	3
	Pb	1	OK	60	40	40	0	0	56	9	18	3
	Sn	1	OK	40	40	40	0	0	32	9	18	3
All	All	2	OK	P1x2	P1x2	P1x2	As P1	As P1	As P1	1	12	3

14.6.4.2 SOUTH DEPOSIT

The search parameters are summarized in Table 14-5. Pass 1 equals the variogram range; Pass 2 equals variogram range x 1.5; Pass 3 equals variogram range x 2; and Pass 4 equals variogram range x 3.

Table 14-5 - Summary of Search parameters for the South Deposit

Pass	Interpol. method	Y	X	Z	Dip	Dip Az	Pitch	Min. S	Max. S	Max. S/DH
1	OK	75	30	75	70	60	90	15	20	5
2	OK	110	45	110	70	60	90	15	20	5
3	OK	150	60	150	70	60	90	15	20	5
4	OK	225	90	225	70	60	90	15	20	5

The variogram ranges chosen for Pass1 were obtained from Au and Cu correlograms which displayed the shortest range and therefore, are conservative for the other elements (Zn, Pb, and Ag) with slightly longer ranges.

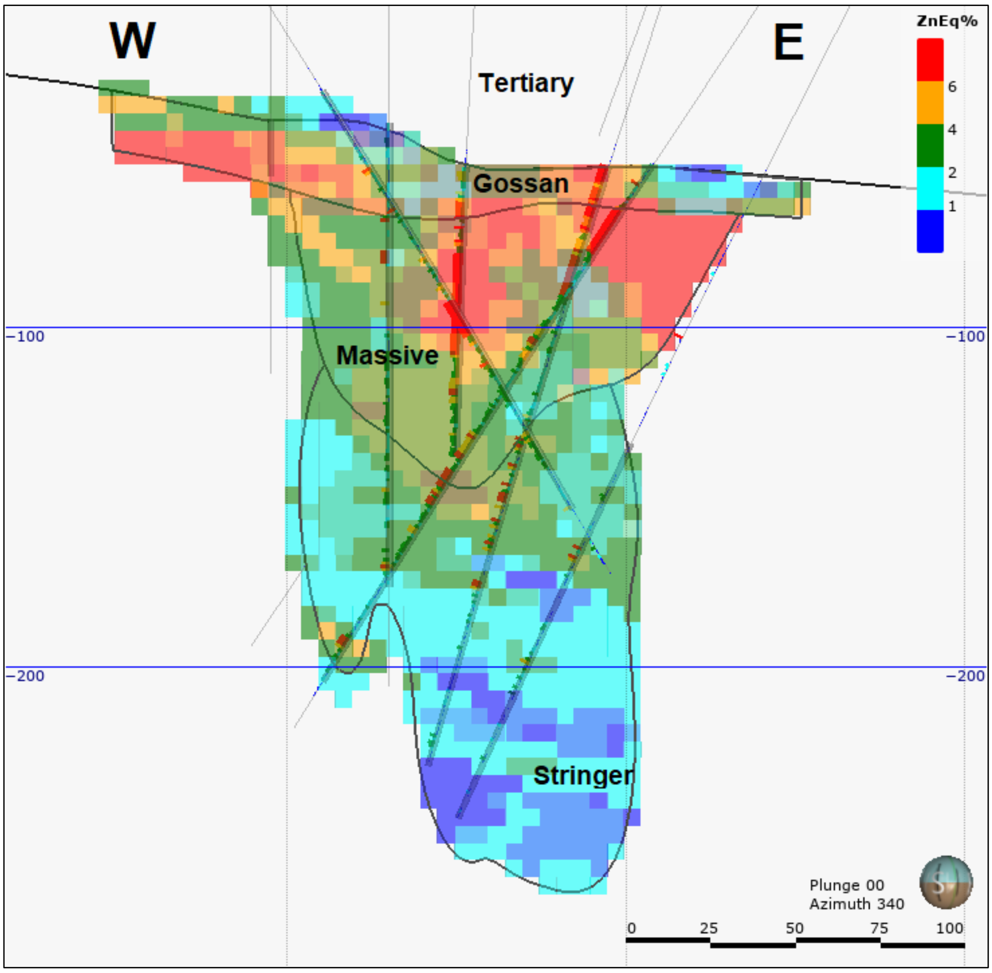
14.6.5 GRADE INTERPOLATION AND VALIDATION

Ordinary kriging (OK) was used for grade interpolation for both the North deposit domains. The block grades were validated as described below.

14.6.5.1 VISUAL VALIDATION

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 is given in Figure 14-18.

Figure 14-18 - Section through the MS Domain Showing the Match Between Block and Composite Grade

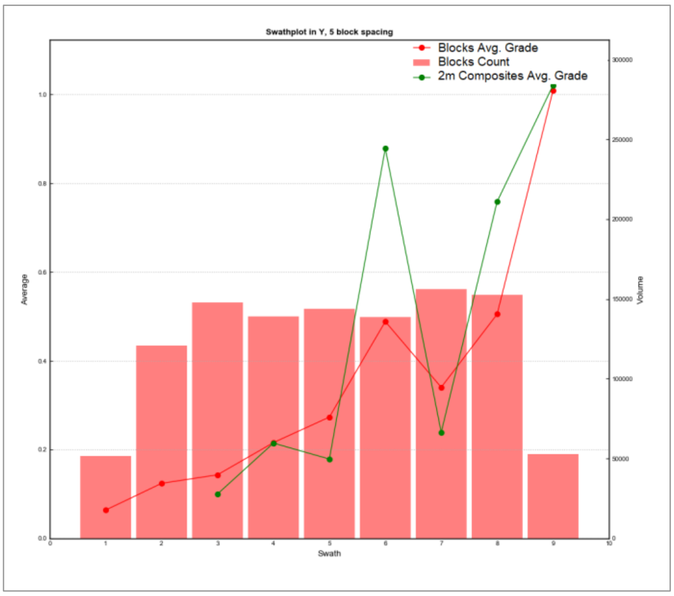


Source: Micon 2019/2020.

14.6.5.2 Validation by Swath Plots

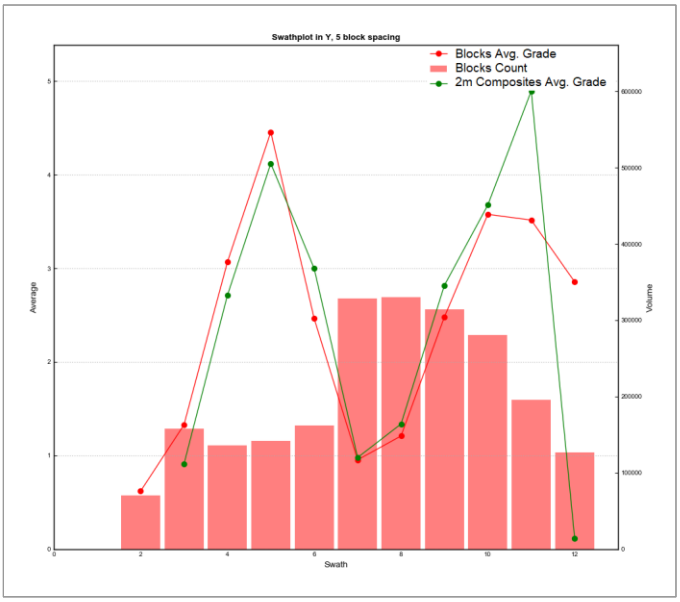
Validation using swath plots produced satisfactory results. Examples are given in Figure 14-19 to Figure 14-22. In all cases, a satisfactory overall match is reflected between block grades and composites.

Figure 14-19 - North Deposit GO Domain Au Swath Plot



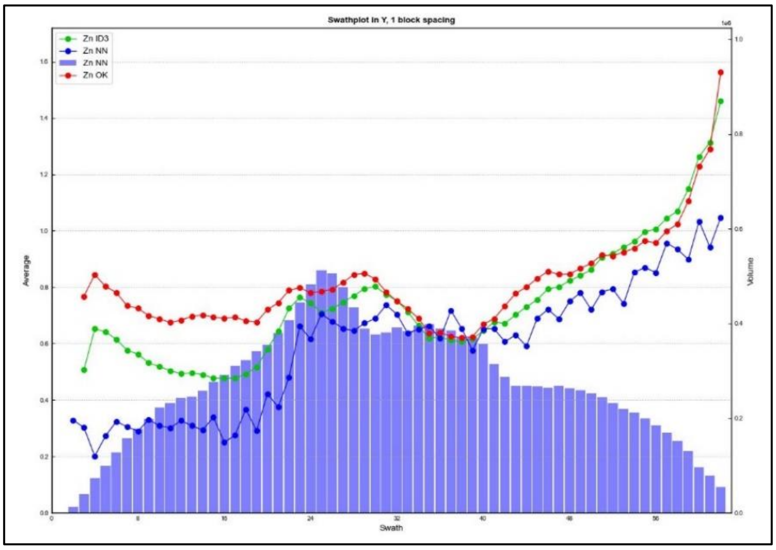
Source: Micon 2019/2020.

Figure 14-20 - North Deposit MS Domain Zn Swath Plot



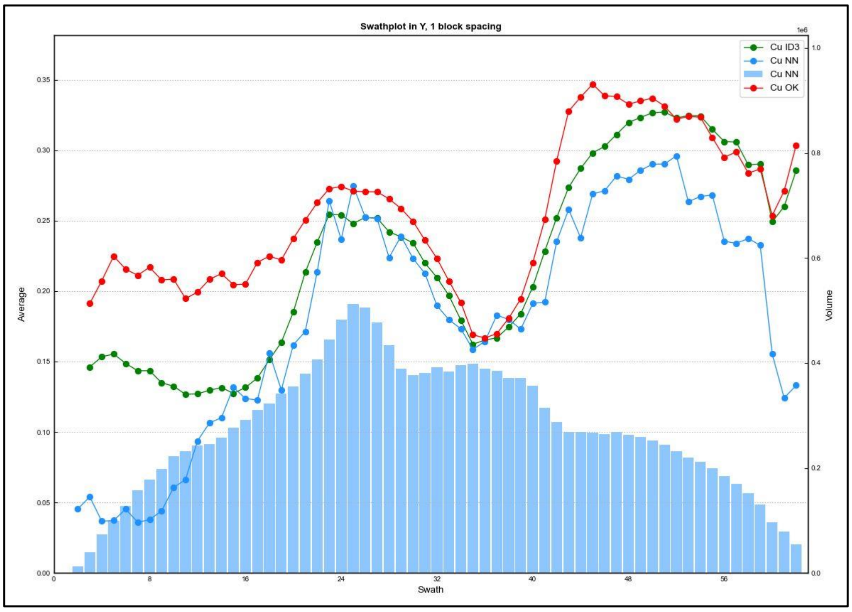
Source: Micon 2019/2020.

Figure 14-21 - South Deposit Zn Swath Plot Along the Y Direction



Source: Micon 2021.

Figure 14-22 - South Deposit Cu Swath Plot Along the Y Direction



Source: Micon 2021.

14.6.5.3 VALIDATION BY GLOBAL STATISTICS

The final validation was conducted by comparing global statistics of the block grades utilizing the inverse distance cubed (ID^3) and nearest neighbour (NN) estimation techniques on the MS domain, which contains most the resource. The results for zinc shown in Table 14-6 below indicate a favourable match with the original OK method used for the estimate.

Table 14-6 - Comparison Between ID^3, NN, and OK Estimation Results for Zn in the MS Domain

Technique	Block Count	Mean	Std Dev	Coeff. V	Median
OK	9027	2.27	2.01	0.89	1.63
NN	9052	2.39	2.11	0.89	1.59
ID^3	9052	2.32	2.25	0.97	1.46

14.6.5.4 Overall Comments and Sensitivity Tables

All the three methods used to validate block grade estimation supported the estimation results. Table 14-7 presents the North deposit mineral inventory's sensitivity to cut-off grade for ZnEq and Table 14-8 presents the South deposit mineral inventory's sensitivity to cut-off grade for copper equivalent (CuEq).

Table 14-7 - LS Project – North Deposit Global Mineralized Tonnes at Various ZnEq Cut-off Grades

Category	ZnEq cut-off	Tonnes (kt)	Average grade							
			ZnEq (%)	Cu (%)	Zn (%)	Pb (%)	Sn (%)	Ag (g/t)	Au (g/t)	AuEq (g/t)
GO	6.0%	1,074	10.61	0.11	0.55	4.54	0.41	40.64	0.95	6.70
	5.5%	1,158	10.26	0.10	0.54	4.39	0.40	40.22	0.90	6.48
	5.0%	1,294	9.73	0.10	0.53	4.14	0.38	39.63	0.84	6.14
	4.5%	1,472	9.13	0.10	0.52	3.88	0.35	39.01	0.78	5.76
	4.0%	1,713	8.44	0.09	0.51	3.55	0.31	38.14	0.71	5.32
	3.5%	1,959	7.85	0.09	0.49	3.26	0.29	37.52	0.66	4.95
	3.0%	2,219	7.31	0.09	0.48	3.00	0.26	36.56	0.61	4.61
	2.5%	2,527	6.75	0.09	0.47	2.74	0.24	35.14	0.55	4.26
	2.0%	2,907	6.16	0.08	0.46	2.46	0.21	33.24	0.50	3.89
	1.5%	3,417	5.50	0.08	0.46	2.15	0.19	30.22	0.45	3.47
	1.0%	4,085	4.81	0.07	0.45	1.84	0.16	26.95	0.39	3.03
	0.0%	4,448	4.48	0.07	0.45	1.70	0.15	25.28	0.36	2.83
	Total	**4,448**	**4.48**	**0.07**	**0.45**	**1.70**	**0.15**	**25.28**	**0.36**	**2.83**
MS	6.0%	6,419	11.72	0.45	3.10	3.05	0.15	88.44	0.83	7.40
	5.5%	6,878	11.33	0.44	3.00	2.92	0.15	85.22	0.81	7.15
	5.0%	7,313	10.96	0.43	2.91	2.80	0.15	82.26	0.78	6.92
	4.5%	7,793	10.58	0.42	2.81	2.68	0.14	79.12	0.75	6.68
	4.0%	8,340	10.17	0.41	2.70	2.55	0.14	75.76	0.72	6.42
	3.5%	8,864	9.79	0.41	2.59	2.43	0.14	72.67	0.69	6.18
	3.0%	9,431	9.39	0.40	2.49	2.31	0.13	69.47	0.66	5.93
	2.5%	9,941	9.05	0.39	2.40	2.21	0.13	66.70	0.63	5.71
	2.0%	10,302	8.82	0.39	2.34	2.15	0.12	64.78	0.61	5.56
	1.5%	10,489	8.69	0.38	2.31	2.11	0.12	63.78	0.60	5.49
	1.0%	10,626	8.60	0.38	2.29	2.09	0.12	63.05	0.60	5.43
	0.0%	10,640	8.59	0.38	2.28	2.08	0.12	62.97	0.60	5.42
	Total	**10,640**	**8.59**	**0.38**	**2.28**	**2.08**	**0.12**	**62.97**	**0.60**	**5.42**

Category	ZnEq cut-off	Tonnes (kt)	Average grade							
			ZnEq (%)	Cu (%)	Zn (%)	Pb (%)	Sn (%)	Ag (g/t)	Au (g/t)	AuEq (g/t)
SW	6.0%	12	7.68	0.61	0.61	0.12	0.56	42.57	0.03	4.85
	5.5%	17	7.16	0.59	0.61	0.12	0.51	39.86	0.03	4.52
	5.0%	24	6.56	0.66	0.67	0.12	0.41	36.26	0.03	4.14
	4.5%	40	5.86	0.63	0.81	0.19	0.31	33.90	0.03	3.70
	4.0%	83	5.01	0.54	0.98	0.25	0.21	27.66	0.05	3.16
	3.5%	174	4.33	0.47	1.05	0.29	0.14	23.16	0.07	2.73
	3.0%	414	3.68	0.38	1.03	0.29	0.10	19.31	0.08	2.32
	2.5%	878	3.17	0.33	0.93	0.26	0.08	17.02	0.08	2.00
	2.0%	1,864	2.67	0.27	0.84	0.23	0.06	14.46	0.07	1.68
	1.5%	3,691	2.20	0.21	0.73	0.21	0.04	12.11	0.06	1.39
	1.0%	6,389	1.79	0.16	0.64	0.17	0.03	9.82	0.06	1.13
	0.0%	8,222	1.57	0.14	0.57	0.15	0.03	8.57	0.06	0.99
	Total	**8,222**	**1.57**	**0.14**	**0.57**	**0.15**	**0.03**	**8.57**	**0.06**	**0.99**

Table 14-8 - LS Project - South Deposit Global Mineralized Tonnes at various CuEq Cut-off Grades

		Average Grade						Contained Metal					
Cut-off	Tonnes	CuEq	Cu	Zn	Pb	Ag	Au	CuEq	Cu	Zn	Pb	Ag	Au
	kt	%	%	%	%	g/t	g/t	kt	kt	kt	kt	k oz	k oz
0.9	29,310	1.20	0.31	0.81	0.44	14.30	0.51	351.87	90.97	236.54	129.15	13,477	477
1.0	24,231	1.25	0.32	0.82	0.44	14.56	0.57	303.55	76.76	198.78	107.34	11,344	443
1.1	14,871	1.39	0.34	1.00	0.55	15.40	0.57	206.18	50.06	148.57	81.80	7,362	273
1.2	10,218	1.50	0.36	1.09	0.60	15.80	0.62	153.10	37.27	111.74	61.10	5,192	203
1.3	5,868	1.69	0.40	1.36	0.77	18.80	0.57	99.19	23.35	80.06	44.95	3,548	108
1.4	4,284	1.82	0.42	1.41	0.79	19.74	0.69	77.87	17.99	60.42	33.81	2,719	95
1.5	3,198	1.94	0.43	1.43	0.81	20.46	0.85	62.17	13.67	45.88	25.86	2,104	86

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.

Based on three-year trailing averages, the forecasted metal commodity prices are zinc = \$2,535/t, lead = \$2,315/t, copper = \$6,724/t, gold = \$1,250/oz, silver = \$19.40/oz, and tin = \$19,175/t. The ZnEq and CuEq values are calculated as follows:

*ZnEq% = ((Zn Grade*25.35) + (Pb Grade*23.15) + (Cu Grade * 67.24) + (Au Grade*40.19) + (Ag Grade*0.62) + (Sn Grade * 191.75)) / 25.35*

*CuEq% = ((Zn Grade*25.35) + (Pb Grade*23.15) + (Cu Grade * 67.24) + (Au Grade*40.19) + (Ag Grade*0.62) + (Sn Grade * 191.75)) / 67.24*

Metal recoveries are expected to average about 60 to 70% based on the preliminary testwork completed by Grinding Solutions Mineral Processing Services. The preliminary testwork results also suggest that recoveries will be higher for the stringer / stockwork type mineralization (South deposit) than for the MS (North deposit). The South resources are reported at a copper equivalent grade of 1.1% CuEq since they are relatively more enriched in copper than zinc / lead. The North deposit resource is reported at 3% ZnEq (MS) and 2.5% ZnEq (GO and stringer) in line with the expected lower recoveries in the MS mineralization. Table 14-9 summarizes the underground economic assumptions upon which the resource estimate for the LS deposits is based.

Table 14-9 - Summary of Economic Assumptions for the Conceptual Underground Mine at the LS Project

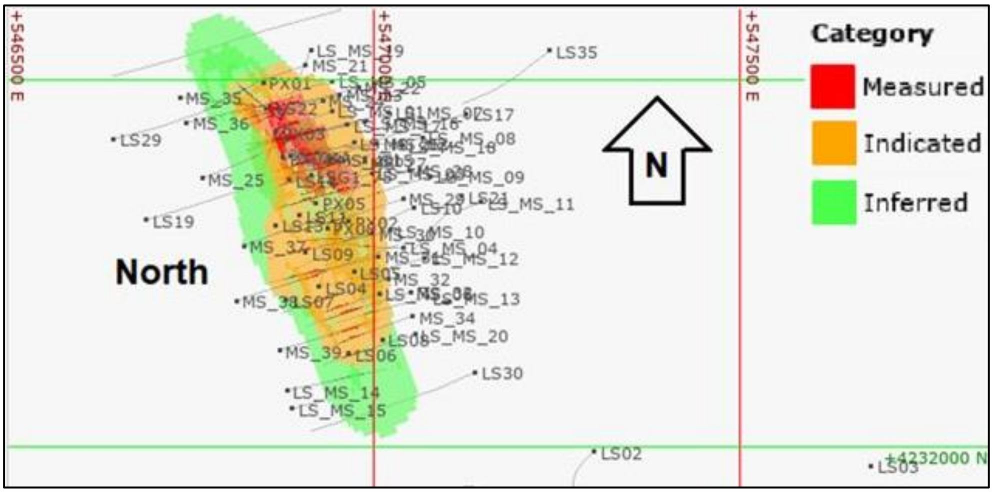
Description	Value Used
Mining cost ($/t)	$65
Processing cost ($/t)	$20
General & administration ($/t)	$5
Average metallurgical recovery – North deposit	65%
Average metallurgical recovery – South deposit	78%

14.7.2 Classification of the Mineral Resource

14.7.2.1 North deposit

Micon has classified the North deposit mineral resource estimate in the Measured, Indicated, and Inferred categories. A plan view of the resource categorization is shown in Figure 14-23.

Figure 14-23 - Plan View of the North Deposit Showing Mineral Resource Classification

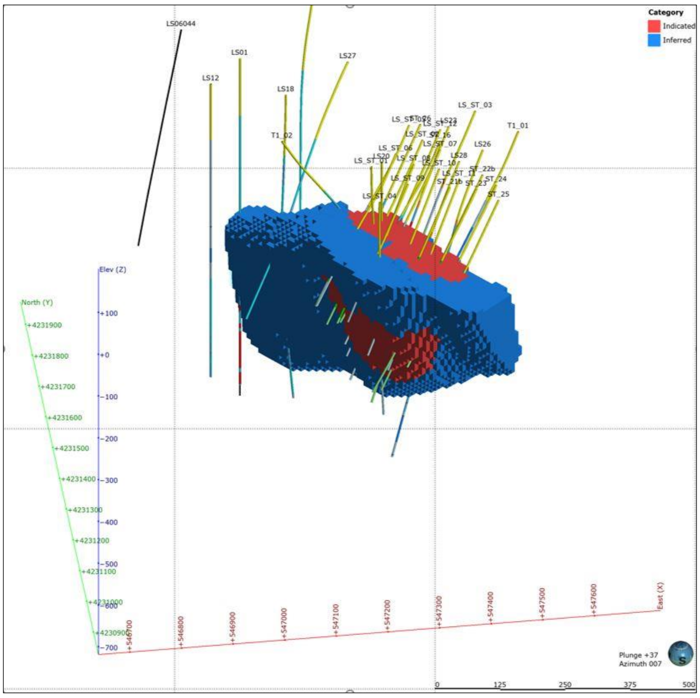


The approach used to categorize the Measured resource was to select those blocks informed by more than 4 drillholes and within a 20-30 m distance from the closest composite. The approach used to categorize the Indicated resource was to select those blocks informed by more than 3 drillholes and within a 30-60 m distance from closest composite. 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 were classified in the Inferred category.

14.7.2.2 SOUTH DEPOSIT

Micon has classified the South deposit mineral resource estimate in the Indicated and Inferred categories. A 3D perspective view of the resource categorization is shown in Figure 14-24.

Figure 14-24 - 3D Perspective of the South Deposit Resource Classification



The approach used to categorize the Indicated Resource was to select those blocks in Pass 1 interpolation (i.e., blocks informed by more than 4 drillholes and within a search ellipse of 75 x 75 x 30). 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 2, 3 and 4) were classified in the Inferred category. The contribution from Pass 4 is negligible (<2%).

14.7.3 MINERAL RESOURCE STATEMENT

The mineral resources for the LS project North and South deposits are summarized in Table 14-10 and Table 14-11. 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 14-10 - LS Property Mineral Resource estimate of the North Deposit as of 5 September 2019 at Cut-off grades Shown in Table

Deposit	Category	Min Zones	Cut-off ZnEq (%)	Tonnes (kt)	Cu (%)	Zn (%)	Pb (%)	Sn (%)	Ag (g/t)	Au (g/t)	ZnEq (%)	AuEq (g/t)	Cu (kt)	Zn (kt)	Pb (kt)	Sn (kt)	Ag (koz)	Au (koz)
North	Measured (M)	GO	2.5	234	0.13	0.70	4.32	0.36	51	1.50	11.38	7.18	0.3	1.6	10.1	0.9	385.2	11.3
	Indicated	GO	2.5	1,462	0.08	0.43	2.55	0.26	37	0.51	6.63	4.18	1.2	6.2	37.3	3.8	1,742.1	23.8
	M & I	**GO**	**2.5**	**1,696**	**0.09**	**0.47**	**2.79**	**0.27**	**39**	**0.64**	**7.28**	**4.60**	**1.5**	**7.9**	**47.4**	**4.6**	**2,127.2**	**35.1**
	Inferred	GO	2.5	831	0.08	0.48	2.62	0.17	27	0.37	5.66	3.57	0.7	4.0	21.8	1.4	727.6	9.9
	Measured	MS	3.0	2,444	0.40	3.12	2.97	0.15	72	0.74	10.95	6.91	9.7	76.3	72.5	3.7	5,623.9	58.4
	Indicated	MS	3.0	5,457	0.45	2.35	2.30	0.13	75	0.67	9.55	6.03	24.5	128.1	125.6	7.3	13,221.5	116.9
	M & I	**MS**	**3.0**	**7,902**	**0.43**	**2.59**	**2.51**	**0.14**	**74**	**0.69**	**9.98**	**6.30**	**34.2**	**204.4**	**198.1**	**10.9**	**18,845.5**	**175.2**
	Inferred	MS	3.0	1,529	0.23	1.96	1.32	0.09	45	0.49	6.36	4.01	3.6	30.0	20.2	1.4	2,219.7	24.0
	Measured	Str	2.5	94	0.37	0.88	0.28	0.05	17	0.12	3.08	1.94	0.3	0.8	0.3	0.0	51.0	0.4
	Indicated	Str	2.5	643	0.34	0.90	0.23	0.09	17	0.06	3.23	2.04	2.2	5.8	1.5	0.6	354.0	1.3
	M & I	**Str**	**2.5**	**737**	**0.34**	**0.90**	**0.24**	**0.09**	**17**	**0.07**	**3.21**	**2.03**	**2.5**	**6.6**	**1.7**	**0.6**	**405.0**	**1.7**
	Inferred	Str	2.5	142	0.24	1.12	0.39	0.04	17	0.09	2.95	1.86	0.3	1.6	0.6	0.1	75.6	0.4
North	**M & I**	**All zones**	**2.9**	**10,334**	**0.37**	**2.12**	**2.39**	**0.16**	**64**	**0.64**	**9.06**	**5.72**	**38.2**	**219.0**	**247.2**	**16.2**	**21,377.7**	**212.0**
North	Inferred	All zones	2.8	2,502	0.18	1.42	1.70	0.12	38	0.43	5.93	3.74	4.6	35.6	42.6	2.9	3,022.8	34.3

Notes:

9. 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.
10. The mineral resources have been estimated in accordance with the CIM Best Practice Guidelines (2019) and the CIM Definition Standards (2014).
11. Mineralized Zones: GO=Gossan, MS=Massive, Str=Stringer, Str/Fr=Stockwork.
12. ZnEq% = ((Zn Grade*25.35) + (Pb Grade*23.15) + (Cu Grade * 67.24) + (Au Grade*40.19) + (Ag Grade*0.62) + (Sn Grade*191.75)) / 25.35.
13. AuEq g/t = ((Zn Grade*25.35) + (Pb Grade*23.15) + (Cu Grade * 67.24) + (Au Grade*40.19) + (Ag Grade*0.62) + (Sn Grade*191.75)) / 40.19.
14. CuEq% = ((Cu Grade * 67.24) + (Zn Grade*25.35) + (Pb Grade*23.15) + (Au Grade*40.19) + (Ag Grade*0.62)) / 67.24
15. Metal Prices: Cu $6,724/t, Zn $2,535/t, Pb $2,315/t, Au $1,250/oz, Ag $19.40/oz, Sn $19,175/t.
16. Densities: GO=3.12, MS=4.76, Str=2.88.

Table 14-11 - LS Property South Deposit Resources as of June 14, 2021 at 1.10% CuEq Cut-off Grade

		Average Grade						Contained Metal					
Category	**Ton**	**CuEq**	**Cu**	**Zn**	**Pb**	**Ag**	**Au**	**CuEq**	**Cu**	**Zn**	**Pb**	**Ag**	**Au**
	kt	%	%	%	%	g/t	g/t	kt	kt	kt	kt	k oz	k oz
Indicated	4,044	1.50	0.42	1.55	0.87	17.64	0.06	60.54	16.94	62.53	35.13	2,294	7
Inferred	10,827	1.35	0.31	0.79	0.43	14.56	0.76	145.6	33.12	86.03	46.67	5,068	266

Notes:

6. 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.
7. The mineral resources have been estimated in accordance with the CIM Best Practice Guidelines (2019) and the CIM Definition Standards (2014).
8. CuEq % = ((Cu Grade * 67.24) +(Zn Grade*25.35) + (Pb Grade*23.15) + (Au Grade*40.19) + (Ag Grade*0.62))/67.24
9. Metal Prices: Cu $6,724/t, Zn $2,535/t, Pb $2,315/t, Au $1,250/oz, Ag $19.40/oz, Sn $19,175/t.
10. Density = 3.

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

This section is not relevant to this report, as no Mineral Reserves have been estimated for this report.

16 MINING METHODS

The mine plan is partly based on inferred mineral resources that are considered too speculative geologically for the application of the economic considerations that would enable them to be categorised as mineral reserves, and there is no certainty that the PEA based on these mineral resources will be realised. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

16.1 INTRODUCTION

The Lagoa Salgada Project is planned to be mined using underground mining methods. The option of open-pit mining was discarded due to the characteristics and depth of the orebody.

16.2 ANALYSIS OF MINING METHOD ALTERNATIVES

The mining method was selected using the UBC Mining Method Selection, a modification of the Nicholas methodology (Nicholas, 1981).

The Nicholas classification determines the feasibility of the mining method considering the orebody geometry and distribution, and the hangingwall and footwall geotechnical properties. This method provides a numerical ranking based on the geometry of the deposit and the material properties (including hangingwall, ore and footwall strength).

The Nicholas method is applied to the three mineralised materials found in the deposit:

- Gossan.
- Massive sulphides.
- Stockwork.

Both methods (i.e., Nicholas and modified Nicholas) have been applied to obtain two different ranking values for each material.

The modified Nicholas method was chosen due to the fact that it gives greater importance to the mineralised hangingwall. The Nicholas method assigns the same value to both the mineralised hangingwall and the footwall.

The geometrical and geotechnical parameters used to choose the mining method are based on the reports listed in.

Table 16-1, provided by Ascendant, and the author’s own experience in similar projects in the Iberian Pyrite Belt.

Table 16-1 – Documents provided by ASCN used in the mining method selection

Document	Author
719037 Ascendant Resources Lagoa Salgada PEA Report 25 February 2020	AMC Mining Consultants (Canada) Ltd (AMC)
43-101 TR 2021 Project 2112 South Deposit MRE & North Deposit PEA (March 2021)	Micon International Limited (Micon)

16.2.1 GOSSAN

This part of the Report is based on the following assumptions:

- Moderate geomechanical quality in the hangingwall.
- Tabular geometry.
- Uniform grade distribution.
- Narrow thickness.
- Flat inclination.

Considering the description of the parameters included in the Nicholas method (Nicholas, 1981), the assumptions presented in Table 16-2 were made for this material.

Table 16-2 – Parameters for Gossan mineralisation

Geometry	
General shape	Tabular
Thickness	Very thick: >100m
Dip	Flat: <20°
Grade distribution	Uniform
Strength (Uniaxial Strength / Overburden Pressure)	
Hangingwall	Strong: >15 MPa
Ore body	Weak: <8 MPa
Footwall	Weak: < 8 MPa
Facture frequency	
Hangingwall	Wide: RQD 40–70
Orebody	Very Close: RQD 0–20
Footwall	Very Close: RQD 0–20
Fracture strength	
Hangingwall	Moderate
Ore body	Weak
Footwall	Weak

The results for each methodology (i.e., Nicholas and Modified Nicholas) and the mining method are presented in Table 16-3.

Table 16-3 – Gossan: Results for Nicholas and Modified Nicholas

Method	Nicholas	Modified Nicholas
Open pit mining	37	35
Cut&fill stoping	35	31
Square-set stoping	39	35
Block caving	35	34
Longwall mining	-16	-18
Room and pillar mining	-30	-30
Top slicing	29	28
Shrinkage stoping	23	20
Sublevel caving	19	18
Sublevel stoping	-32	-48

16.2.2 MASSIVE SULPHIDES

The following assumptions have been made regarding massive sulphides:

- Massive geometry.
- Gradual grade distribution.
- Intermediate thickness (30 – 100 m).
- 20°-55° inclination.

Table 16-4 – Parameters for Massive Sulphide mineralization

Geometry	
General shape	Equidimensional
Thickness	Thick: 30–100m
Dip	Intermediate: 20°–55°
Grade distribution	Gradational
Strength (Uniaxial Strength / Overburden Pressure)	
Hangingwall	Moderate: 8–15
Ore body	Strong: >15
Footwall	Moderate: 8–15
Facture frequency	
Hangingwall	Wide: RQD 40–70
Ore body	Very Wide: RQD 70–100
Footwall	Wide: RQD 40–70
Fracture strength	
Hangingwall	Moderate
Ore body	Strong

Footwall	Moderate

The results for each methodology (i.e., Nicholas or Modified Nicholas) and mining method are presented in Table 16-5.

Table 16-5 – Massive Sulphides: Results for Nicholas and Modified Nicholas

Method	Nicholas	Modified Nicholas
Open pit mining	47	44
Cut&fill stoping	27	24
Square-set stoping	26	23
Block caving	29	26
Longwall mining	-77	-81
Room and pillar mining	-18	-21
Top slicing	37	34
Shrinkage stoping	37	35
Sublevel caving	35	32
Sublevel stoping	33	32

16.2.3 STOCKWORK

This scenario is based on the following assumptions:

- Tabular geometry.
- Gradual grade distribution.
- Thick 30–100m.
- >55° inclination.

Table 16-6 – Parameters for Stockwork mineralization

Geometry	
General form	Tabular
Thickness	Thick: 30–100m
Dip	Steep: >55°
Grade distribution	Gradational

Strength (Uniaxial Strength / Overburden Pressure)	
Hangingwall	Strong: >15
Ore body	Moderate: 8–15
Footwall	Moderate: 8–15
Facture 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 rankings of the various mining methods for this ore type are shown in Table 16-7.

Table 16-7 – Stockwork: Results for Nicholas and Modified Nicholas

Method	Nicholas	Modified Nicholas
Open pit mining	45	42
Cut&fill stoping	33	30
Square-set stoping	30	27
Block caving	27	25
Longwall mining	-78	-80
Room and pillar mining	-20	-21
Top slicing	27	26
Shrinkage stoping	31	29
Sublevel caving	32	30
Sublevel stoping	33	32

16.3 Geotechnical Stope Design

This section presents a preliminary stability study for stopes using the Mathews empirical method. This method has only been applied to the massive sulphides and the stockwork, which will be mined by sublevel stoping.

16.3.1 Description of the Mathews Method

The Mathews method for mine design was first proposed in 1980 by Mathews (Mathews et al., 1980). Over the years, several authors (Potvin et al., 1998; Stewart and Forsyth, 1995; Trueman et al., 2000) have improved the method, which has by now become an industry standard for stoping stability assessment and dimensioning.

The design 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.

If these factors are entered into a graph, the stability number on the y-axis and the hydraulic radius on the x-axis, the stability of the stope can be established. The graph is divided into zones representing the stability or instability of the mined void.

16.3.2 STABILITY NUMBER

The stability number, N’, is defined as:

$$N' = Q' \times A \times B \times C$$

Q parameter

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 the joint water reduction parameter (Jw) and the stress reduction factor (SRF) are one = Q’.

The Q’ parameter has been estimated by reviewing the lithological boreholes and the geotechnical borehole (Geotechnical_LOG_MS_32) provided by ASND. Two values have been extrapolated from this data, considering the borehole sections more representative for the materials: 9.15 for the massive sulphides and 6.41 for the stockwork.

Stress Factor – A

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 as shown in Figure 16-1. The uniaxial compressive strength is usually obtained from laboratory tests.

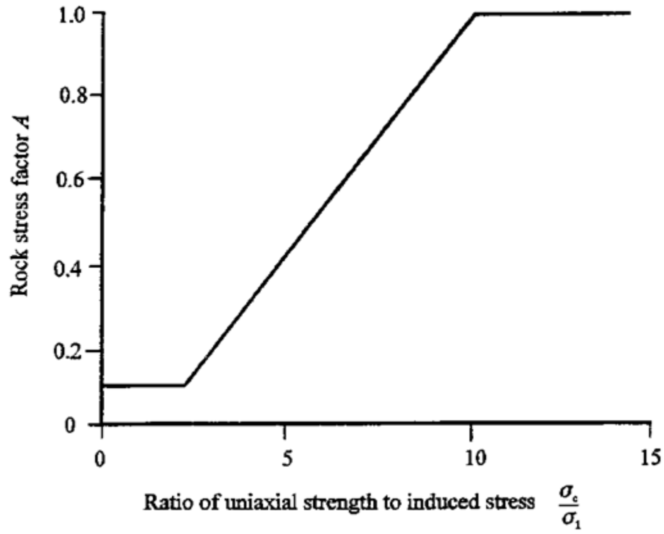


Figure 16-1 – Diagram for the estimation of Stress Factor

The author 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).

Joint Orientation Factor – B

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 16-2.

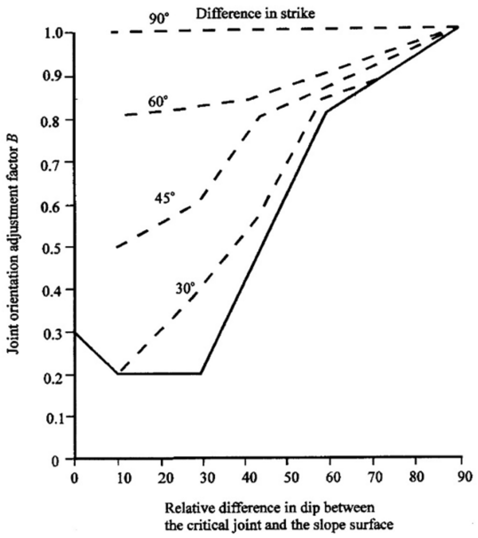


Figure 16-2 – Diagram for the estimation of the Joint Orientation Factor

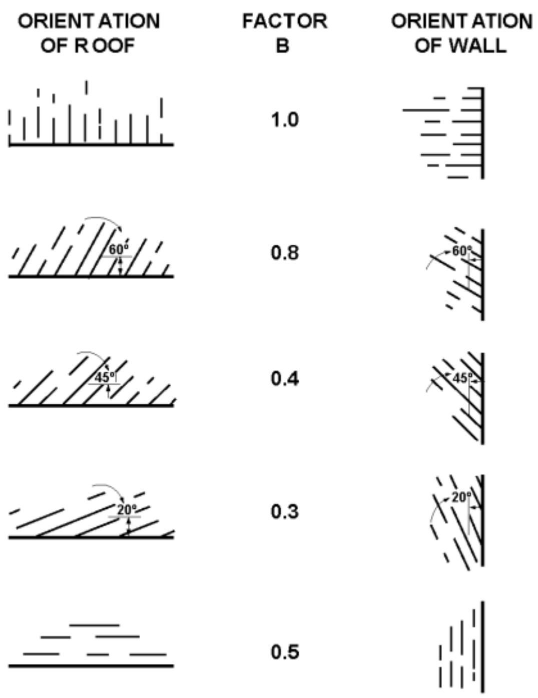


Figure 16-3 – Diagram for the estimation of the Joint Orientation Factor when the difference in direction is zero (according to Stewart and Forsyth, 1995)

Considering both the stope surface to critical joint relative dip difference and the strike difference between discontinuity and plane trace in Figure 16-3, all of the possible B factors specified in the table have been evaluated.

The table highlights the values, resulting from the combinations between the different stopes surfaces and the three possible representative joint types that provide the lowest discontinuity reduction value.

The values for the B factor are 0.6 for the roof and 0.3 for the floor and sidewalls. Due to the lack of data about joints and critical structures, this factor has been estimated based on the author's own experience and conservative principles. The first value, 0.6, has been estimated based on the fact that the critical structure is vertical and the stope roof has a maximum dip of 40°. The latter value, 0.3, indicates that the critical structure is parallel to the front of the stope.

Gravity Adjustment Factor – C

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 16-4 or the following equation:

$$C = 8 - 6 \cdot \cos \alpha$$

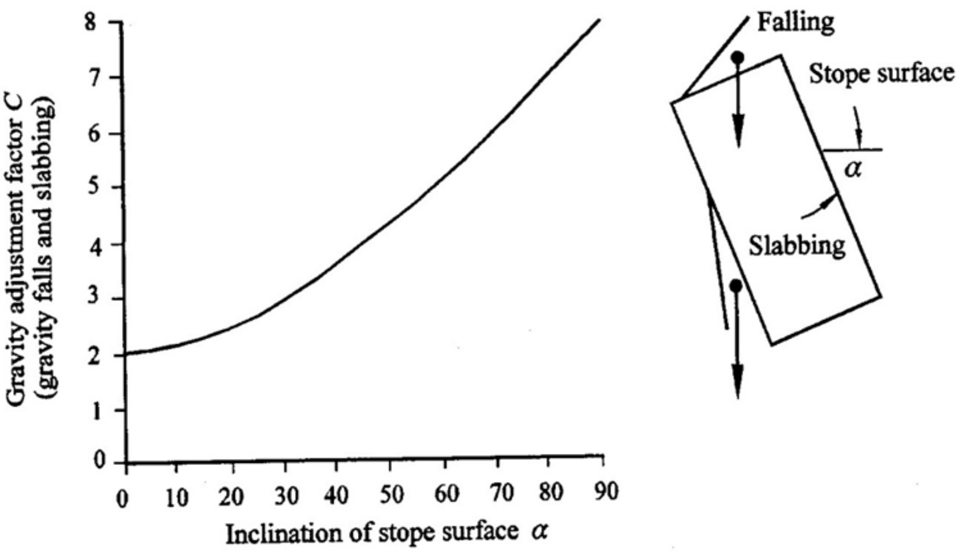


Figure 16-4 – Graph for the determination of the "C" factor by gravity or slabbing mechanisms (according to Potvin, 1988)

Sliding failures depend on the inclination (β) of the critical joints, and the value of C factor can be obtained from Figure 16-5.

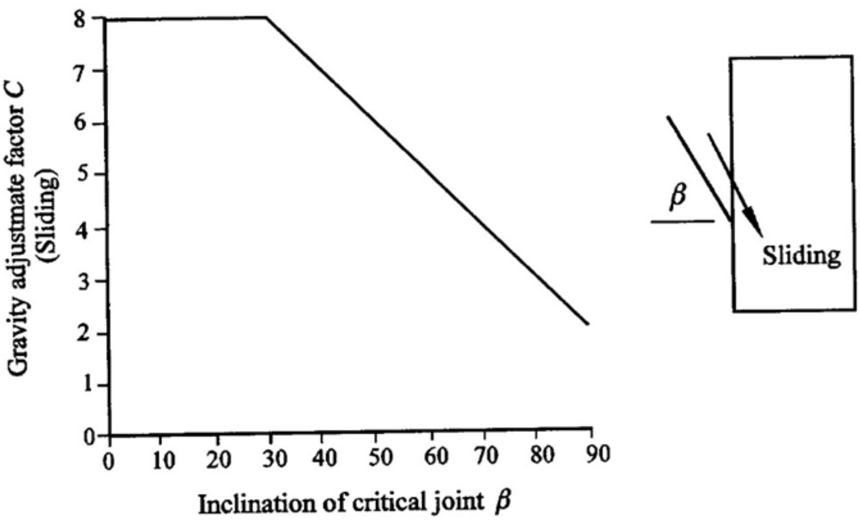


Figure 16-5 – Graph for the determination of the "C" factor by gravity or slabbing mechanisms (according to Potvin, 1988)

The C factor considers the orientation of the analysed surface. The vertical wall design has a value of 8 whereas the value for a horizontal roof is 2. 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 2 and 6 have been considered for the roof and sidewalls, respectively.

16.3.3 Hydraulic Radius

The hydraulic radius, or shape factor, for the surface under consideration is calculated as follows:

$$S = \frac{Area}{Perimeter}$$

16.3.4 Stope Dimension Stability Limit Determinations

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 make preliminary estimates of the stopes dimensions.

$$HR_{limit} = 10^{[0.573+0.338 \cdot \log N]}$$

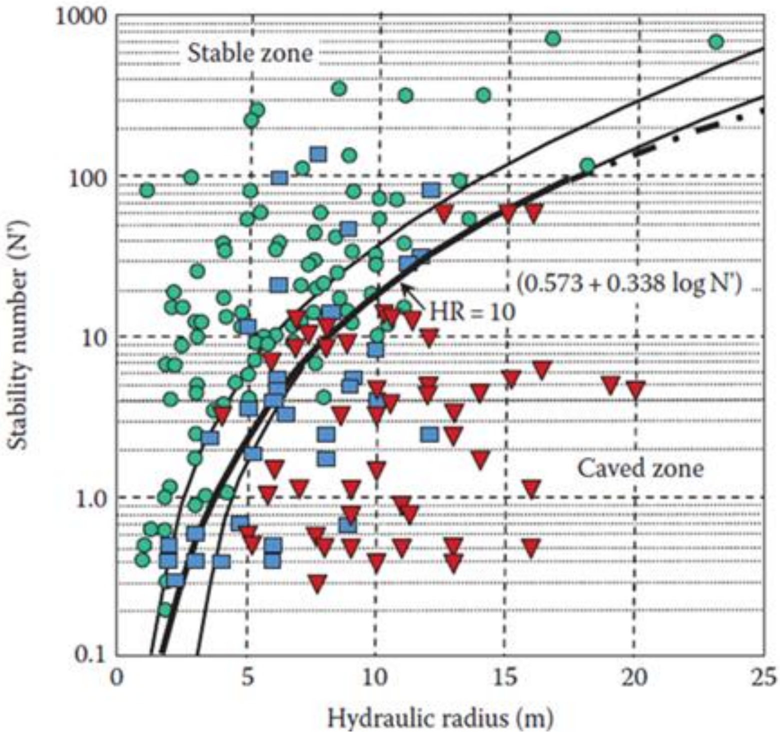


Figure 16-6 – Stability graph and determination of the stability limit (Nickson, 1992)

As the transition between stockwork and massive sulphide materials is expected to be similar throughout the mine, considering single values of N and Q for all stope geometries is regarded as adequate for a PEA-level study.

Based on the most restrictive value of Q' (6.4) and the values of A, B and C determined in the previous sections, stability numbers of 7.7 and 11.5 have been calculated for the roof and sidewalls, respectively. These values are shown in Table 16-8.

Table 16-8 – Stability graph parameters for the stope roofs and sidewalls

Surface	N'	A	B	C
Roof	7.7	1	0.6	2
Sidewalls	11.5	1	0.3	6

A stope width of 15 m and height of 25 m have been selected for this Project. These dimensions are industry standard values in other operations in the IPB and facilitate operations using the type of mobile equipment fleet that is usually employed on site.

As shown in Figure 16-7, the value of 5.8 corresponds to a stope with a length of 50 m and a width of 15 m. This value is below the calculated maximum limit for the transition zone (i.e., the zone of semi-stability without any support).

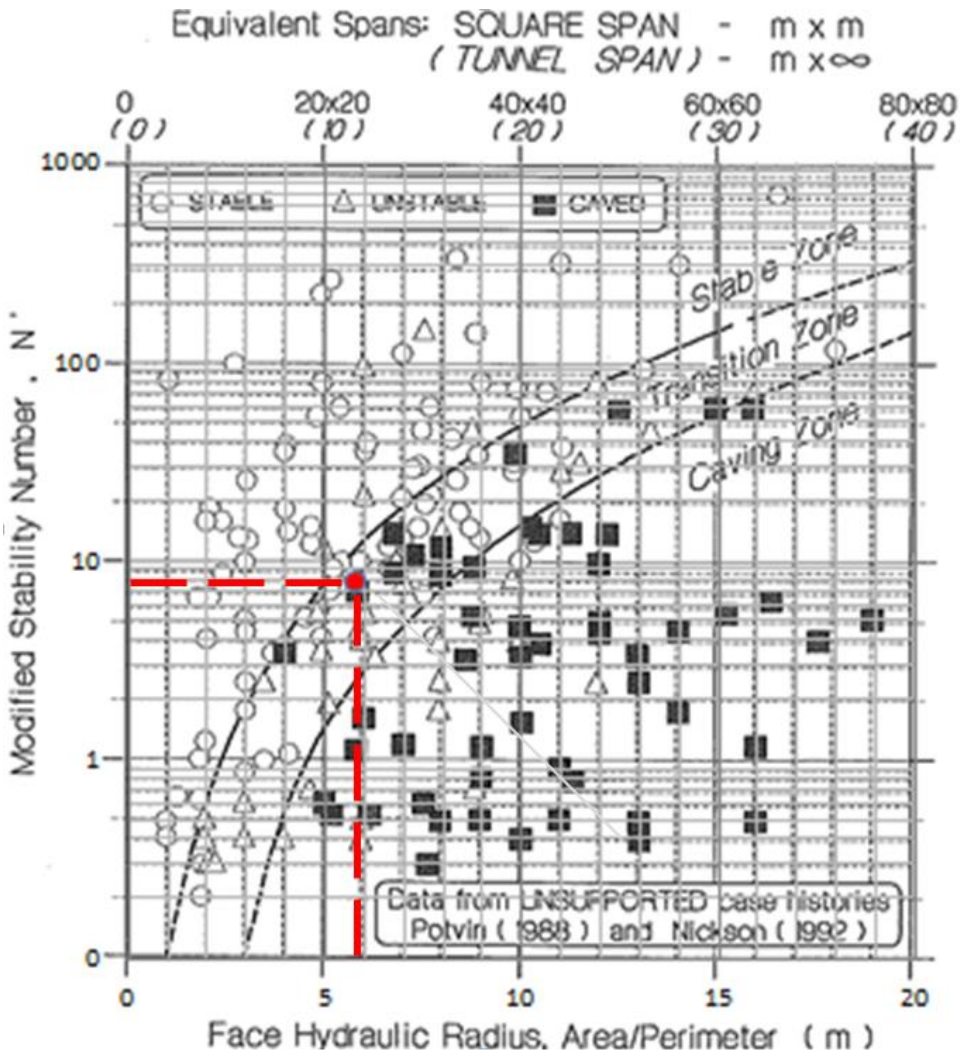


Figure 16-7 – Graph for the stope roof without support

The length of the stopes may reach 60 m, and they will remain within the transition zone.

Table 16-9 – Factors A, B, and C for N' calculation (roof)

Parameter	Value
Factor A	1.0
Factor B	0.6
Factor C	2.0
N'	**7.7**

Table 16-10 – Hydraulic Radius calculation (roof)

Parameter	Value
L	50
W	15
S	**5.8**

When considering a typical support pattern for sublevel stopping support (Table 16-11), the stope roof is within the stability values, as shown in Figure 16-8.

Table 16-11 – Spacing and length recommended for cablebolt support

Parameter	Value
Support element	double strand cable bolt
Spacing	2m
Length	6-9m

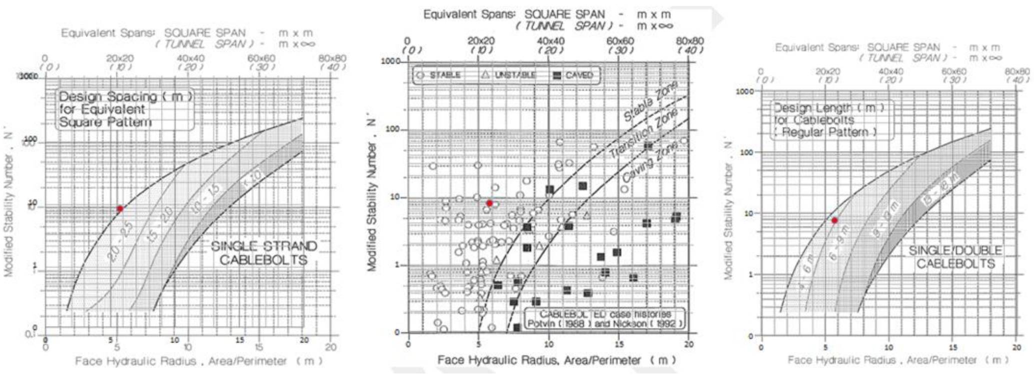


Figure 16-8 – Graph for stope roofs with support

In the case of sidewalls and considering a stope length of 50 m and a height of 25 m, the stope would be below the theoretical calculated limit value and thus, within the stable or transition zone.

Table 16-12 – Factors A, B, and C for N' calculation (sidewall)

Parameter	Value
Factor A	1.0
Factor B	0.3
Factor C	6.0
N'	**11.5**

Table 16-13 – Hydraulic Radius calculation (sidewall)

Parameter	Value
L	50
W	25
S	**8.3**

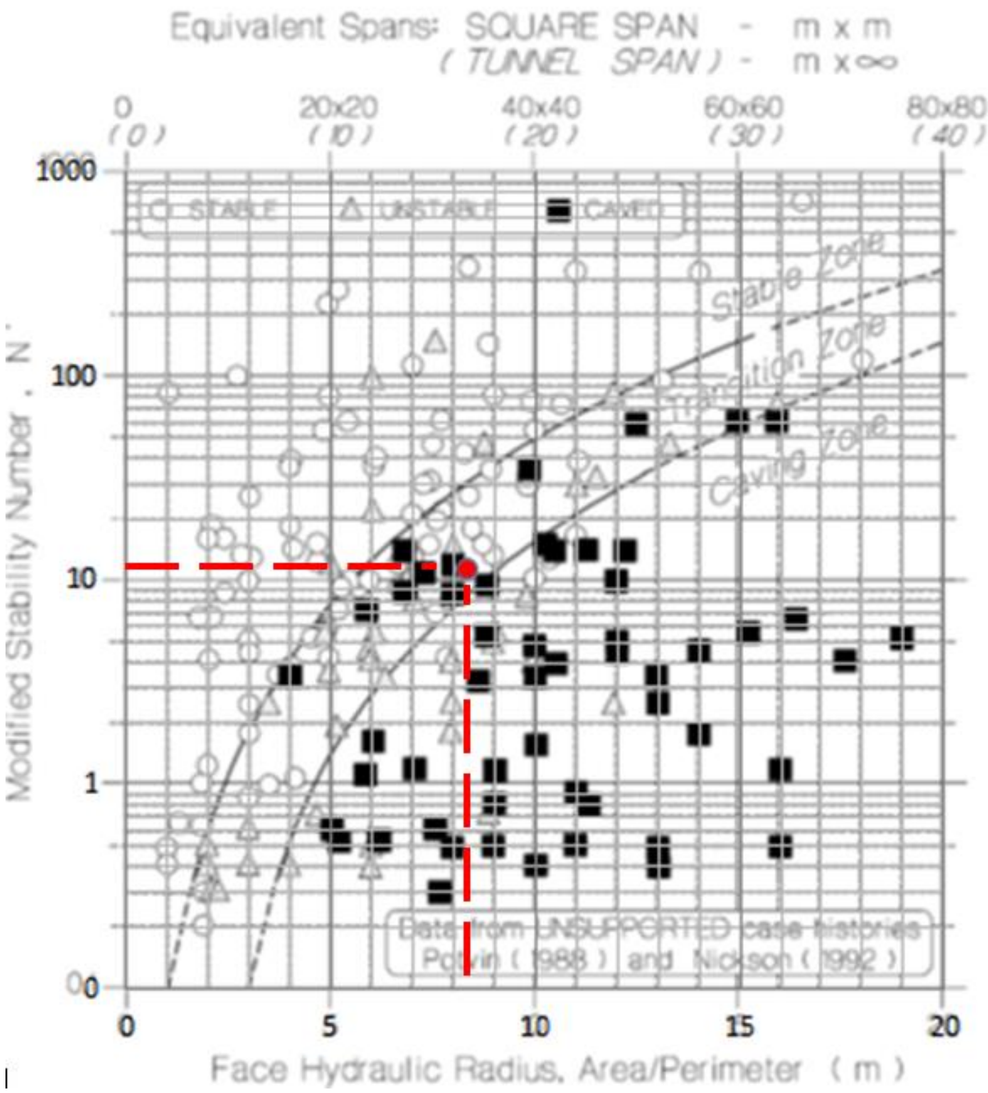


Figure 16-9 – Graph for the stope sidewall without support

16.3.5 SUPPORT PATTERN

A basic design of a typical support pattern is presented in Figure 16-10 and Table 16-14, with a cable length that meets the criteria obtained from the stability analysis.

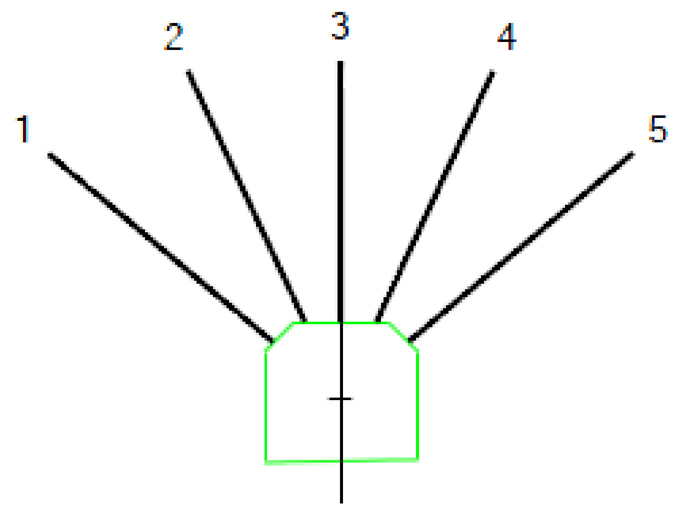


Figure 16-10 – Typical Cablebolt pattern

Table 16-14 – Cablebolt support design parameters

Cablebolt Number	Length	Type
1	9m	Double cablebolt
2	7m	Double cablebolt
3	6m	Double cablebolt
4	7m	Double cablebolt
5	9m	Double cablebolt

16.4 GEOTECHNICAL INFRASTRUCTURE SUPPORT DESIGN

This section describes the ground support required for the underground development of the Project. It defines the minimum safety conditions that must be met by the support of the underground excavations.

16.4.1 RAMP ACCESS AND TRANSPORT DRIFT SECTIONS

Considering the geotechnical information available, the Abacus of Barton (Abacus of Grinstand & Barton, 1993) was used to design a preliminary support. The Barton's Q index, which is related to Bieniawski's RMR, has been 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" (De) of the excavation:

$$D_e = \frac{B}{ESR}$$

Where:

- B – Width of the excavation (m);
- ESR – Stress level parameter (obtained from Table 16-15)

Table 16-15 – Evaluation of the ESR (Grinstad and Barton, 1993)

Type or use of underground opening	ESR
Temporary mine openings	3-5
Vertical shafts, rectangular and circular respectively	2.0-2.5
Water tunnels, permanent mine openings, adits, drifts	1.6
Storage caverns, road tunnels with little traffic, access tunnels, etc.	1.3
Power stations, road and railway tunnels with heavy traffic, civil defence shelters, etc.	1.0
Nuclear power plants, railroad stations, sport arenas, etc.	0.8

Knowing the "Equivalent dimension" and the Q value, the support to be used can be estimated by means of the abacus, presented by NGI (2013) based on the abacus of Grimstad and Barton (2000), and shown in Figure 16-11.

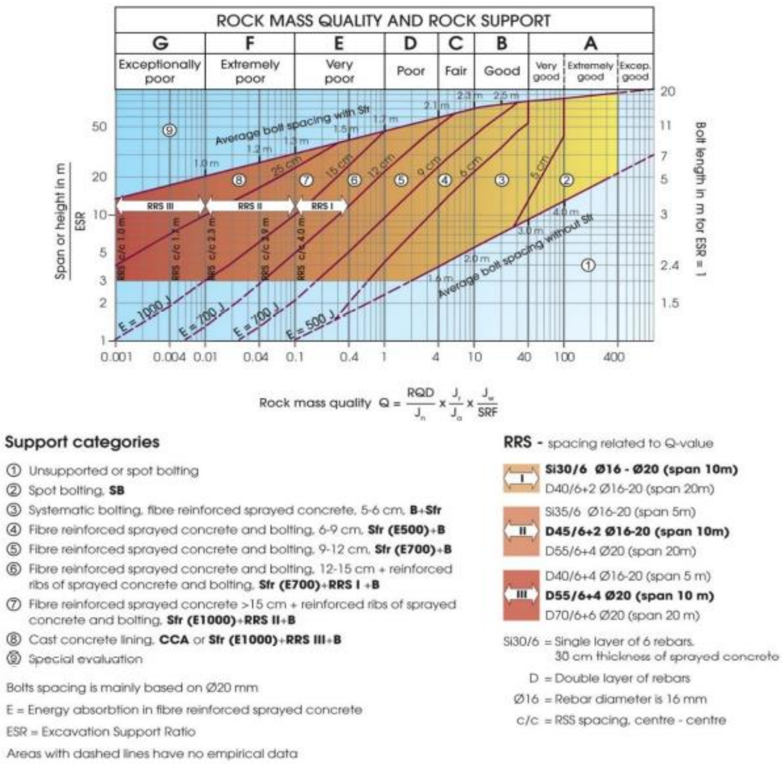


Figure 16-11 – Approximate determination of the support of a tunnel. Carried out by NGI from the abacus of Grinstad and Barton, 1993

16.4.2 MAIN DECLINE AND CAPITAL LATERAL DEVELOPMENT SUPPORT DESIGN

This section includes an estimation of the support that needs to be applied to the different ground types that are expected to be crossed during the execution of the main decline and capital lateral development.

The following is an estimate of the support that needs to be applied in the different materials that are expected to be crossed by the access ramp and the development of the mine.

The equivalent dimension of the excavation is show in Table 16-16.

Table 16-16 – Evaluation of the equivalent dimension (main decline and capital lateral development)

Parameter	Value	Comments
Section Height	5.5m	
Section Width	5.5m	
ESR	1.6	Permanently opened 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 16-12.

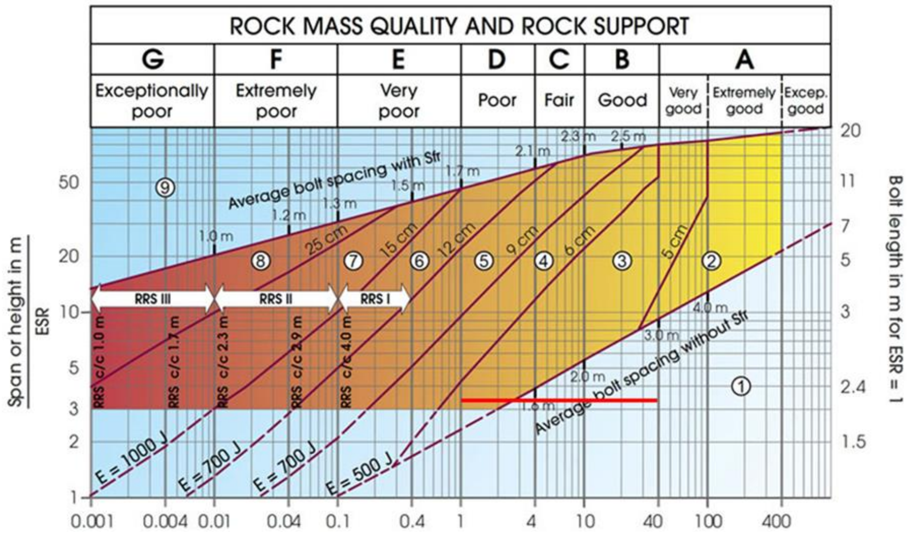


Figure 16-12 – Approximate determination of support for poor, fair, and good rock quality

Based on the empirical results obtained from the Abacus and the author's own experience in similar operations, the support recommended for good quality material is formed by sections of 8 bolts with a length of 2.4m. These bolts can be galvanised split-sets or cemented/resin rebar bolts. The spacing between the sections will be between 1.5m and 1.8m. No mesh is required.

In ground of fair rock quality, the type of support will consist of 9 bolts with a length of 2.4 m. These bolts can also be galvanised split sets or cemented/resin rebar bolts. The spacing between the rings will be 1.5 m, and additionally, 5 cm of shotcrete will be sprayed into the main drift.

In poor-quality terrain, it is recommended to install 9 bolts with a length of 2.4 m per section, with a reinforcement of 8 cm of shotcrete in the main drift and 5 cm of shotcrete in the transverse drift, as well as mesh in the main drift.

It is possible that the main drift or the transport drift are intersected by zones of very poor quality or fault zones, in which case it will be necessary to specifically study each zone and to apply exceptional supports, such as trusses, micropiles and self-drilling bolts where needed, or drainage systems if water is present. Table 16-17 shows a support pattern for the main decline.

Table 16-17 – Support design parameters (main decline)

Rock Quality	Q	Bolt Spacing	Section Spacing	Bolts / Section	Mesh	Shotcrete
Good	10 – 40	1.5m	1.5/1.8m	8	No	No
Fair	4 – 10	1.5m	1.5m	9	No	5cm
Poor	1 – 4	1.5m	1.5m	9	Yes	8cm
Very Poor	0.1 – 1	1.0m	1.0m	11	Yes	10cm

The remainder of the capital lateral development (i.e., footwall drifts, ventilation, and orepass accesses) would require the support as indicated in Table 16-18.

Table 16-18 – Support design parameters (capital lateral development)

Rock Quality	Q	Bolt Spacing	Section Spacing	Bolts / Section	Mesh	Shotcrete
Good	10 – 40	1.5m	1.5/1.8m	8	No	No
Fair	4 – 10	1.5m	1.5m	9	No	No
Poor	1 – 4	1.5m	1.5m	9	Yes	5cm
Very poor	0.1 – 1	1.0m	1.0m	11	Yes	8cm

16.4.3 Stope Access and C&F Access Support Design

The equivalent dimension of these temporary openings, wich will be used to access the stopes and C&F areas, is described in Table 16-19.

Table 16-19 – Evaluation of the equivalent dimension (access to stopes and C&F areas)

Parameter	Value	Comments
Section Height	5.0m	
Section Width	5.0m	
ESR	3	Temporary openings
Equivalent dimension	1.6	

Based on the current geotechnical knowledge, it has been 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.

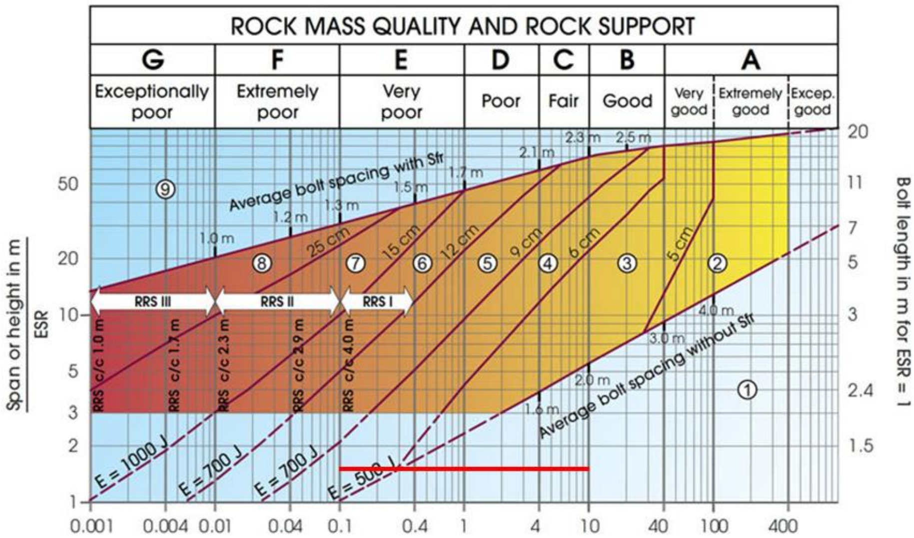


Figure 16-13 – Approximate determination of support for very poor, poor, and fair rock quality

The recommended type of support in fair quality material consists of 8 bolts with a length of 2.4m. These bolts can be galvanised split-sets and the spacing between the sections will be 1.5m.

When intersecting poor quality terrain, it is recommended to install 8 bolts with a length of 2.4m per section, with an additional 5cm shotcrete reinforcement.

In case that the drifts for access to the stopes and cut-and-fill areas intersect very-poor-quality zones or fault zones, it will be necessary to review and study each zone and to apply exceptional supports such as trusses, micropiles or self-drilling bolts where needed. The type of standard support recommended in the case of very-poor-quality ground is shown in Table 16-20.

Table 16-20 – Support design parameters (access to stopes and C&F areas)

Rock Quality	Q	Bolt Spacing	Section Spacing	Bolts / Section	Mesh	Shotcrete
Fair	4 – 10	1.5m	1.5m	8	No	No
Poor	1 – 4	1.5m	1.5m	8	No	5cm
Very poor	0.1 – 1	1.0m	1.0m	9	Yes	8cm

16.4.4 C&F Areas Support Design

In areas, where the C&F method is applied, the excavations will have the equivalent dimension shown in Table 16-21.

Table 16-21 – Evaluation of the equivalent dimension (C&F areas)

Parameter	Value	Comments
Section Height	5.0m	
Section Width	5.0m	
ESR	3	Temporary openings
Equivalent dimension	1.6	

It is 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.

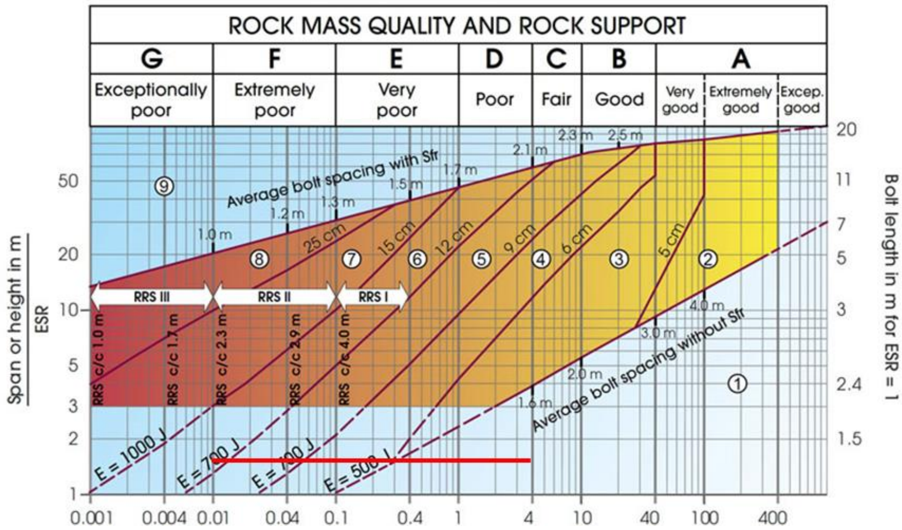


Figure 16-14 – Approximate determination of support for extremely poor, very poor, and poor rock quality

A determining characteristic for the type of support to be placed in the C&F drift, is that the exposure time (the time during which these drifts need to be open) is very small. A typical C&F face will be excavated and filled within a few months.

In these drifts it is recommended to install swellex type bolts, since this type of bolt is cheap and easy to install and its bearing capacity is sufficient and appropriate.

When crossing poor-quality terrain, it is recommended to install 8 swellex bolts with a length of 2.4 m per section, with a 5 cm shotcrete reinforcement.

If the material is very poor, it is necessary to install 9 swellex bolts with a length of 2.4 m per section, with a 1 m spacing between the rings and an 8 cm shotcrete reinforcement. Mesh also needs to be installed in this case.

Where cut-and-fill developments cross extremely poor material, it will also be necessary to reduce the drilled length of each round by means of rib-bar-type shotcrete ribs, in addition to the recommended support.

Table 16-22 – Support design parameters (C&F areas)

Rock Quality	Q	Bolt Spacing	Section Spacing	Bolts / Section	Mesh	Shotcrete
Poor	1 – 4	1.5m	1.5m	8	No	5cm
Very Poor	0.1 – 1	1.0m	1.0m	9	Yes	8cm
Extremely Poor	0.01 – 0.1	1.0m	1.0m	9	Yes	10cm

16.5 BACKFILL

16.5.1 INTRODUCTION

The preliminary design of a mine backfill system is largely dependent on the available materials. Potential sources of fine aggregate locally available to the mine were identified as tailings from the future Lagoa Salgada Processing Plant.

16.5.2 TAILINGS SUITABILITY FOR PASTEFILL SYSTEM

Considering the preliminary metallurgical testwork that has been done, it will be possible to have a good quality pastefill system using the tailings that will be produced by the ore treatment process.

The tailings estimation was done according to the mineral processing and metallurgical testing reports developed by Grinding Solutions.

There are two relevant types of ore, Massive Sulphide and Stockwork, each of them with different characteristics. The oxide ore was not considered as it is not relevant for the tailings production aspect of the operation. The ore from Lagoa Salgada is very similar to the ore from other IPB mining operations. These types of tailings have normally good characteristics for a paste backfill system.

The reports related to Mineral Liberation Analysis (MLA) were studied and used to support the suitability of the tailings to produce a good quality pastefill system. The PSDs from these reports were compared with standard PSD curves for each type of backfill.

Based on the analysis, the PSD suits quite well for pastefill. It is possible to conclude that there is good correlation between the PSD, density, and the filtration capacity also for a Dry Tailings Disposal System. Therefore, it is expected that the tailings can also have a good filtering capacity.

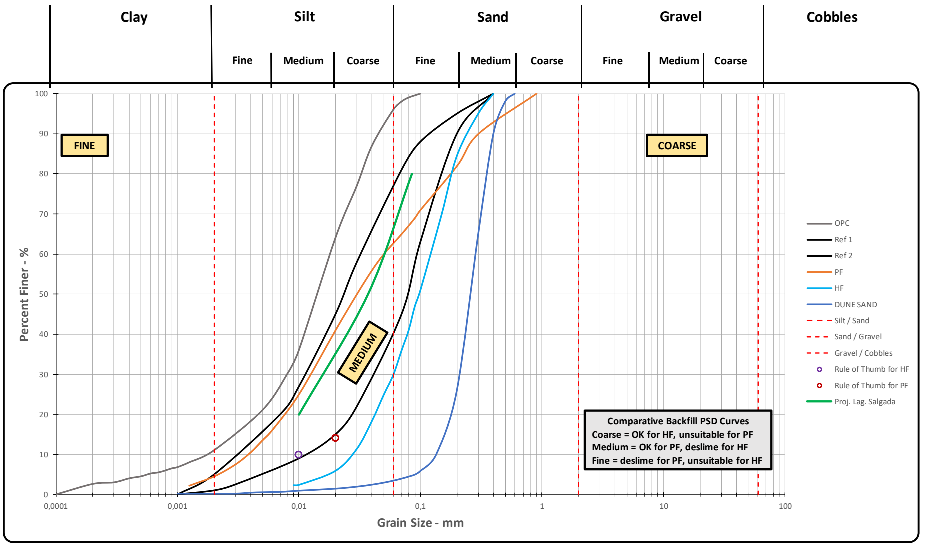


Figure 16-15 – Particle size distribution comparison

Figure 16-15 represents a comparation between different types of backfill. The PSD from Lagoa Salgada (in green) sits in the middle of the two reference PSDs, Ref 1 and Ref 2 (in black). All the PSDs that fit in the middle of these two references are suitable for Paste Backfill. The PF curve (in orange) is a PSD of the tailings for pastefill in one operation on the IPB, with similar geology, mineralogy, and chemical characteristics to Lagoa Salgada.

16.5.3 Pastefill Strength Requirements

With the stope dimensions and characteristics of the future tailings, we can calculate the pastefill strength requirements. These requirements are different for primary and secondary stopes.

Primary Stopes

For the estimation of the backfill strength requirements of the primary stopes it was considered the worst-case scenario. This scenario consists in having a confined block without shear resistance mechanism of frictionless fill (adapted from Micthell et al. 1982). This can happen when a primary stope is backfilled and mining of both contiguous secondary stopes is started. The exposure of the pastefill block of the primary stope on both sides is in a frictionless fill face situation.

- Mitchell Analysis

The Micthell Analysis was used to determine the unconfined compressive strength (UCS) that the primary stopes should have, according to the assumptions stated above. The parameters that were used were:

Table 16-23 – Parameters used in Mitchell analysis

Parameter	Symbol	Unit	Value
Height of stope	H	m	25
Width of stope	W	m	15
Length of stope	L	m	50
Internal friction angle	ϕ	Degrees	34
Bulk unit weight	ϒ	kN/m³	19.62
Factor of safety	FS	-	1.5

The equation for the stability of a free standing backfill face, based on centrifugal modeling tests (Mitchell, 1983) is:

$$UCS_{design} = \frac{\gamma LH}{L+H} FS$$

Where:

- ϒ – fill bulk unit weight
- L – strike length of stope
- H – total height of stope fill
- FS – Factor of safety

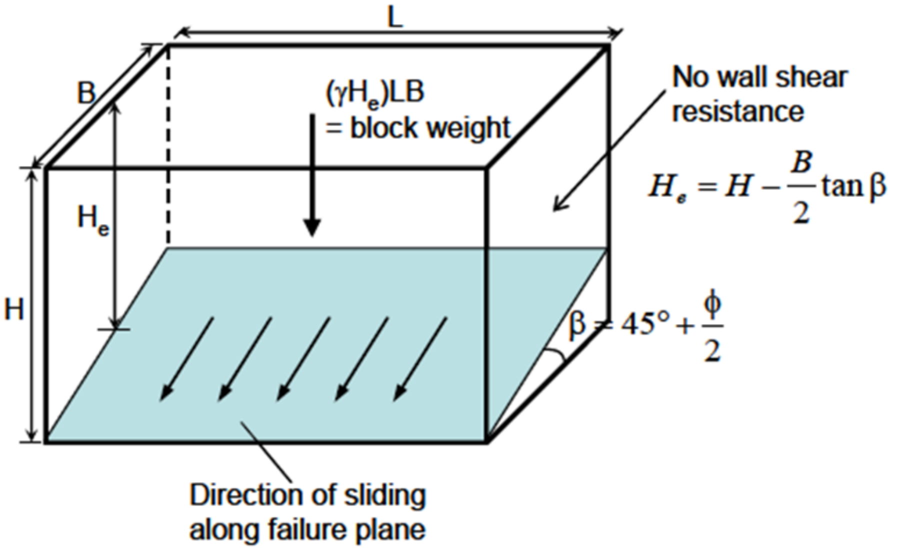


Figure 16-16 – Confined block without shear resistance mechanism of frictionless fill (adapted from Mitchell et al. 1982)

According to the parameters of the Lagoa Salgada project, considering a factor of safety of 1.5 the design compressive strength of the backfill should be 490,5 kPa.

Based on the Paste Backfill of other mines on Iberian Pyrite Belt with similar mineralogy this UCS is easily achieved.

- Terzaghi Analysis

The bearing capacity calculation is based in the Terzaghi's expression of a square foundation to represent the contact area between the vehicle tire and the backfill surface. The vehicle that was considered for this analysis was a loader with full bucket having the maximum weight of 40 tons.

The parameters that were considered are listed below:

Parameter	Symbol	Unit	Value
Backfill bulk unit weight	ϒ	kN/m^3	19.62
Backfill shear strength	C	kPa	100
Internal friction angle	ϕ	Degrees	20
Tyre loading force	F_t	kN	196.20
Tyre pressure	p	kN/m^2	750

- Step 1 – Calculate the Bearing Area (B)

$$B = \sqrt{\frac{Ft}{p}}$$

B = 0.51 m2

- Step 2 – Calculate the Surcharge Bearing Capacity Factor (Nq)

$$Nq = tan^2\left(45 + \frac{\emptyset}{2}\right)e^{(\pi\, tan\emptyset)}$$

Nq = 6.40

- Step 3 – Calculate the Cohesive Bearing Capacity (Nc)

$$Nc = \frac{(Nq - 1)}{\tan\emptyset}$$

Nc = 14.83

- Step 4 – Calculate the Unit Weight Bearing Capacity (Nϒ)

$$N\gamma = 1.8(Nq - 1)\tan\emptyset$$

N_{Υ} = 3.5

- Step 5 – Calculate Ultimate Bearing Capacity (Qf)

$$Qf = 0.4\gamma BN\gamma + 1.2CNc$$

Qf = 1779.6 kPa

- Step 6 – Determine Factor of Safety

$$FS = \frac{Qf}{p}$$

FS = 2.37

According Terzaghi Analysis, for a minimum factor of safety of 2.37, the pastefill should have a minimum strength of 100 kPa, based on an underground loader with a maximum front axle weight of 40 tons. This strength reflects the minimum strength to prevent the loader from sinking. However, it only considers the equipment standing on top of the backfill without movement and it does not consider slippery surface situation during routine mucking and also does not consider over-mucking of the floor if the surface is too soft.

In most of operations that use pastefill, a minimum floor strength of 350 kPa is targeted, with the addition of a layer of waste rock of approximately 300 mm on top to prevent slippage of the equipment.

Considering that the height of the stopes is not higher than 25 meters, it is recommended that the same recipe for the pastefill should be used in all primary stopes maintaining a minimum UCS of 490.5 kPa.

The Figure 16-17 represents USC tests results of Paste Backfill from other mine in Iberian Pyrite Belt. With 4.5% of cement (for a pastefill with a slump of 9”) the USC of 490.5 kPa is achieved in less than 28 days of curing time.

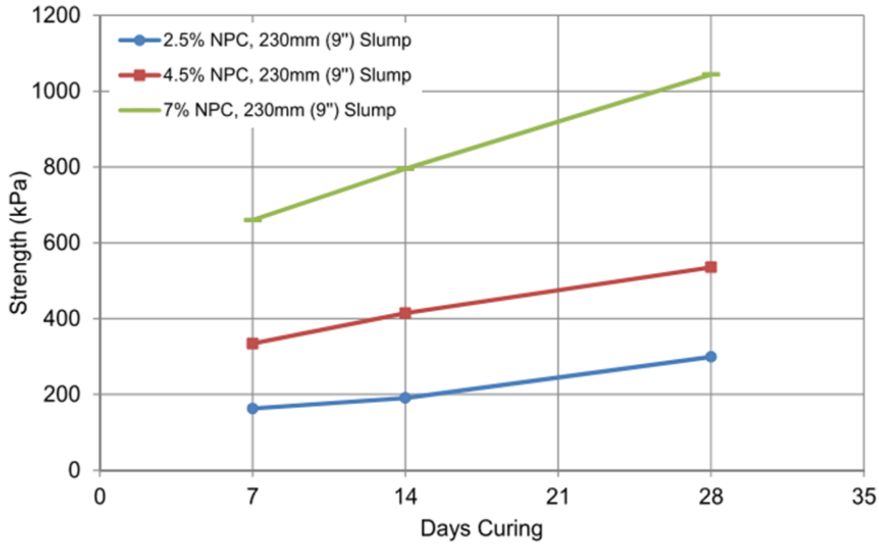


Figure 16-17 – Pastefill UCS test results from a IPB operation

Secondary Stopes

For the secondary stopes, the stability of the free standing backfill wall does not apply. When the secondary stopes are filled with paste, the walls of paste are not going to be exposed. So, in this situation, it is only important to consider the bearing capacity with the same approach that was used for the primary stopes.

Considering the Terzaghi Analysis that was done for the primary stopes, we conclude that the minimum pastefill strength for a 40 ton equipment to operate on top of that pastefill is 100 kPa.

16.5.4 Backfill Reticulation System

Hydraulics is one of the most important aspect to determine if pastefill can flow only by gravity or if it must be pumped. It is one important step to check if the location of the Pastefill Plant is the most adequate. If the pastefill can flow by gravity, the opex and capex needs will be lower due to the lack of necessity of having a positive displacement (PD) pump. The power consumption of such a pump is considerable, and the acquisition cost of a PD pump is high.

The hydraulics were calculated according to hypothetical Pastefill Plant options for location and the simulation was done considering an “Iberian Pyrite Belt Typical Pastefill”. It was made for both North and South sectors of the orebody.

It was considered 8’’ Schedule 80 and Schedule 40 pipes. The simulation was done also with a pumping capacity of 100 m^3/h of paste. According with the LOM, the paste backfill capacity will be of 100 m^3/h.

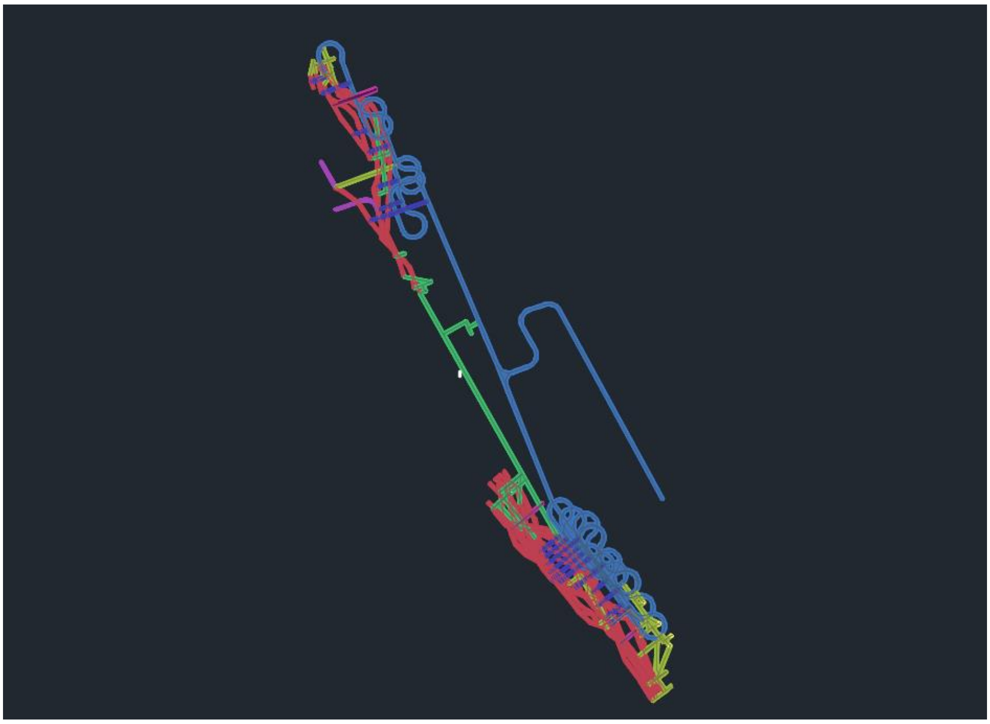


Figure 16-18 – Lagoa Salgada infrastructure with pastefill borehole (in white) between North and South sectors

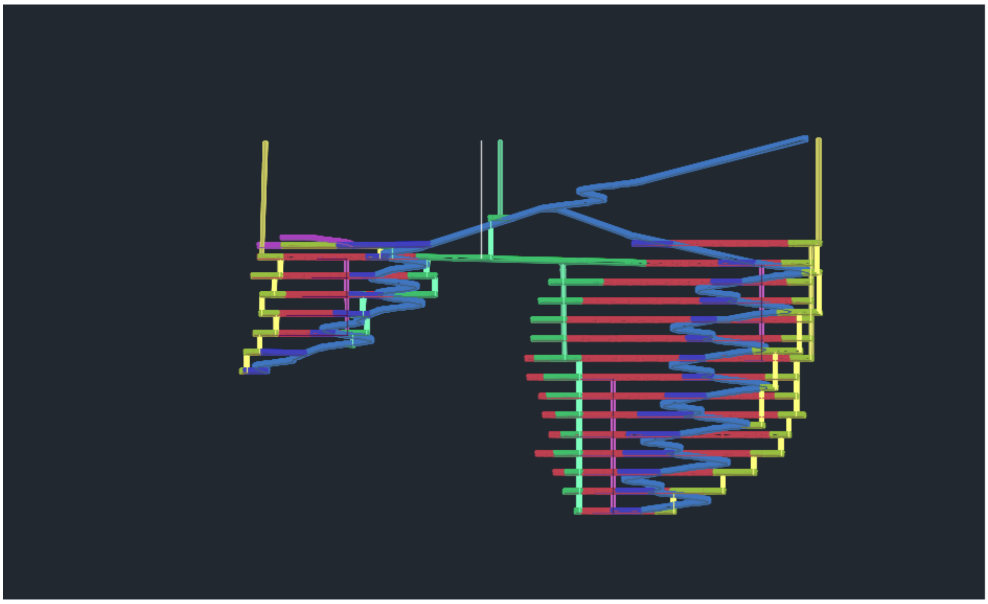


Figure 16-19 – Pastefill borehole representation in section view

Running the Hydraulics with this location:

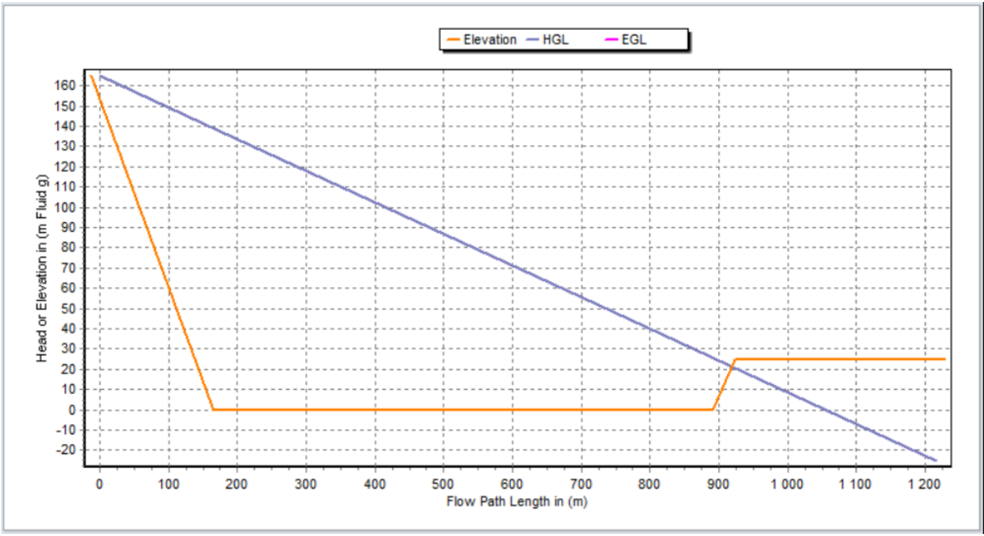


Figure 16-20 – Hydraulic analysis for level -50 by gravity

The Figure 16-20 shows that the Hydraulic Grade Line intercepts the piping profile, and it represents that the paste cannot reach all the entire – 50 Level. For this level the pastefill will need to be pumped.

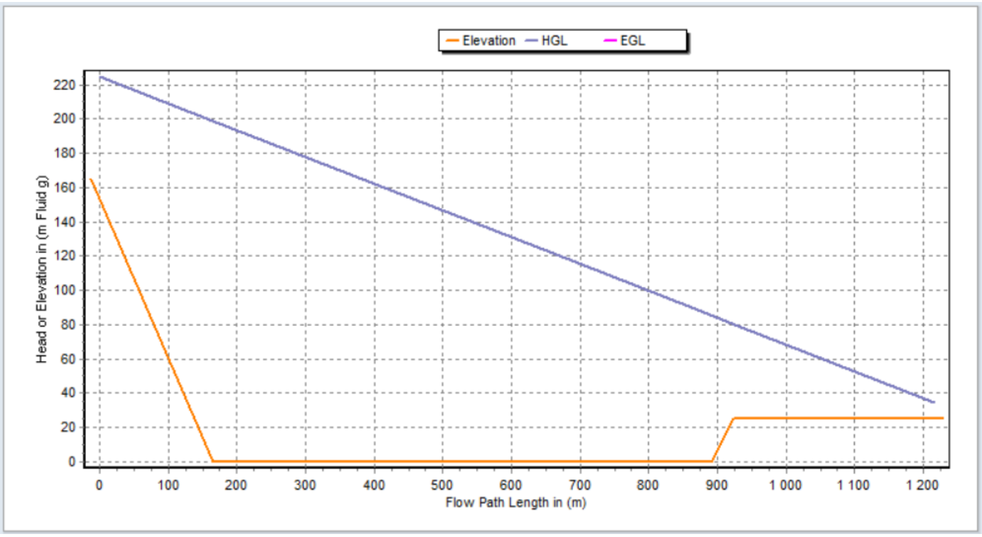


Figure 16-21 – Hydraulic analysis for level -50 with 15 Bar pumping

On Figure 16-21 it can be seen that with a pumping pressure of 15 Bar the pastefill can flow to the entire -50 Level.

The same problems to fill only by gravity occurs also in -75 Level, but in this case with a few adjustments on the reticulation system and pastefill slump, the problems could be solved without the need of pumping.

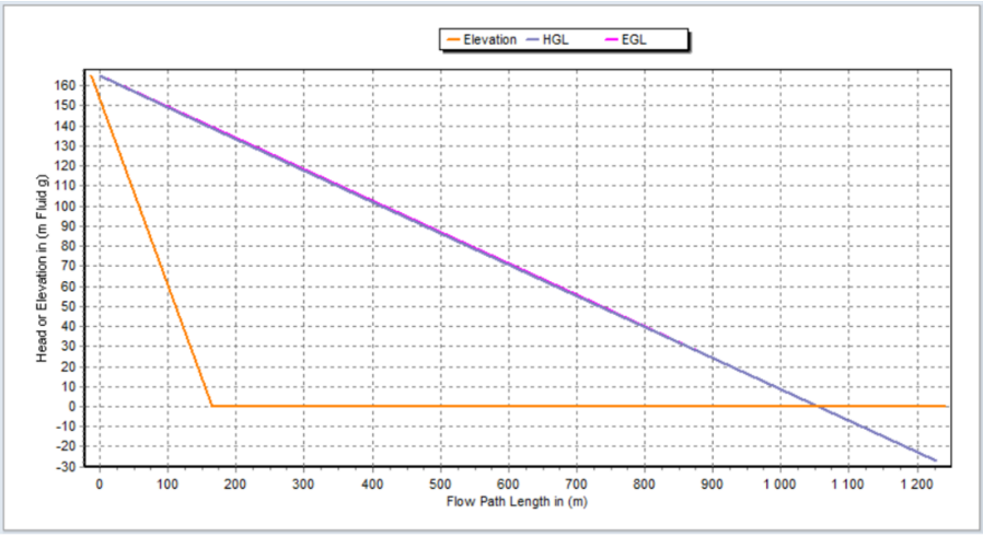


Figure 16-22 – Hydraulic analysis for Level -75 by gravity

For all the levels below -75, for both North and South sections, the Paste Backfill can flow by gravity and without slack flow (that could seriously damage the reticulation system).

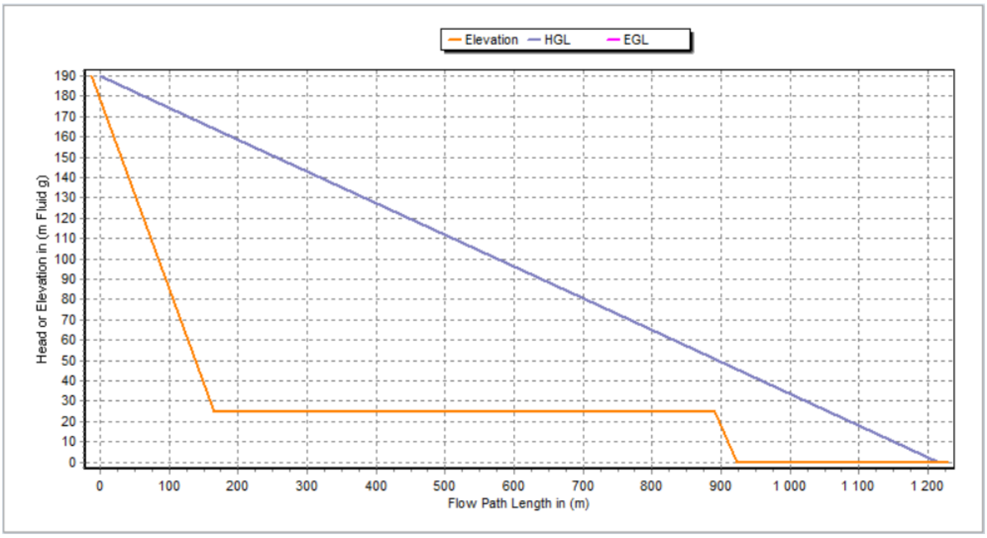


Figure 16-23 – Hydraulic analysis for Level -100 by gravity

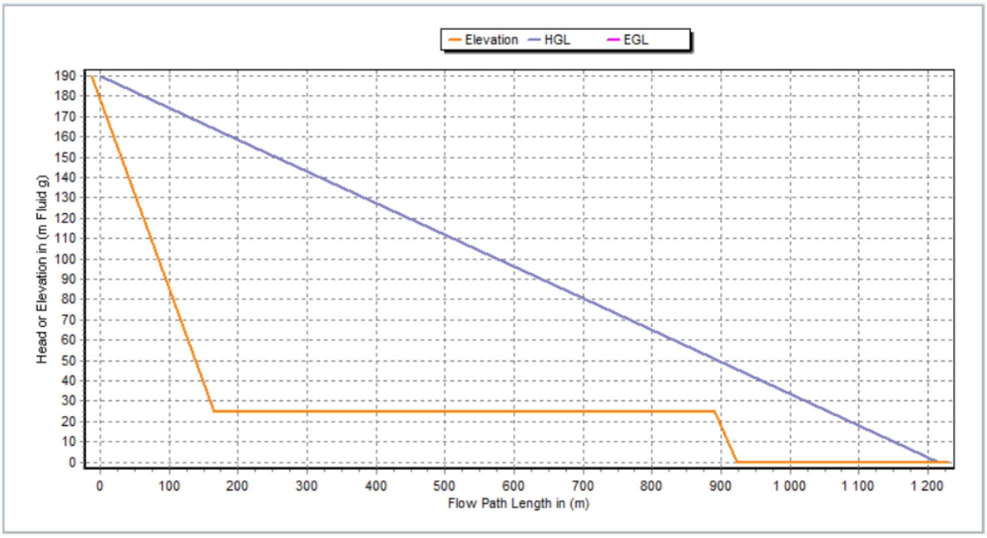


Figure 16-24 – Hydraulic analysis for Level -125 by gravity

16.6 NET SMELTER RETURN MODELLING

The author has generated NSR values for the mineral resource block models. The expected metal recoveries and payables were provided by ASND. The capital and operational costs, as well as other economic assumptions, are detailed in Chapter 21.

The NSR value of each block in the resource model has been calculated as the sum of payable net revenue generated by the recovered zinc, lead, copper, silver, gold, and tin metal masses. The NSR contributions of each metal to the sellable products (zinc/lead/copper concentrates, flotation tailings leached concentrate) has been calculated individually and then, combined, and assigned to each block in the resource model. The generalided formula for the calculation is as follows:

$$NSR\left(\frac{USD}{t}\right) = \sum_{j=1}^{k}\sum_{i=1}^{n} Metal_i \times \%Payable_i \times \%Recovery_i$$

Where:

- j – Sellable product (i.e., Zn/Pb/Cu concentrates, flotation tailings leach);
- i – Payable metal (i.e., Zn, Pb, Cu, Ag, Au, Sn) in j-eth product.

16.7 CUT-OFF CALCULATION

The NSR cut-off grades have been established following Mortimer's approach, 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 tonne.

The former, also known as "boundary cut-off", guarantees that no losses are incurred by mining and treating a volume of rock. The latter, the "volume cut-off" assures that the rock volume yields a certain average level of profit per treated tonne, so that shared costs such as general & administration expenses are covered by the average production grade. Table 16-24 details the parameters and values of both cut-off values for each of the mining methods that have been conseidered.

Table 16-24 – NSR cut-off values per mining method

Parameter	Unit	LHOS	C&F
Total Mining Costs	USD/t	20.0	47.2
Total Processing Costs	USD/t	15.9	15.9
Boundary Cut-Off	**USD/t**	**35.9**	**63.1**
General & Administration	USD/t	7.1	7.1
Volume Cut-Off	**USD/t**	**43.0**	**70.1**

These values, together with modifying factors for mining dilution and recovery, have been considered in the determination of the mining inventory tonnage and grades.

16.8 OPERATING COSTS

Mine operating, processing, and general & administration costs are estimated to a level of accuracy that is adequate for a PEA study. The costs are based on cost database information published by Mining Cost Service and information provided by ASND. Costing information from regional operations has also been used to validate the order of magnitude of the attained costs.

The costs estimates presented in Table 16-24 are considered very conservative, as they have been benchmarked with operations with a considerably lower production rate than that estimated for the Lagoa Salgada Project. For the purpose of calculating the NSR cut-off value, these cost assumptions have been retained without modifications. However, for the economic analysis calculation, the costs have been refined to reflect the economies of scale effect.

16.9 MINE DESIGN

The proposed underground mine design supports the extraction of 2.0 million tonnes of ore per year ("Mtpa") through a combination of transverse sublevel stoping and cut&fill. Paste backfill is used in both mining methods to maximise ore recovery and productivity.

A main decline, starting from the surface portal located close to the processing plant, will be used to access the mine. This main decline then splits into two ramps, one for each mine zone (i.e., "North and South").

A fleet of LHDs (load-haul dumps) and trucks will be used for the loading and hauling of material from production areas to the orepass system. From the orepass collecting points, trucks will haul the ore to the surface. Waste is also transported to the surface by trucks.

A pre-production development programme will be required to provide access to the initial stoping levels in the North Zone during the first two years. Production will start in the second year, reaching the nominal plant feed in the fourth year.

16.9.1 Access and Ramp Infrastructure

The mine portal will be situated close to the processing plant – Figure 16-25 indicates the portal's location as well as the general mine layout.

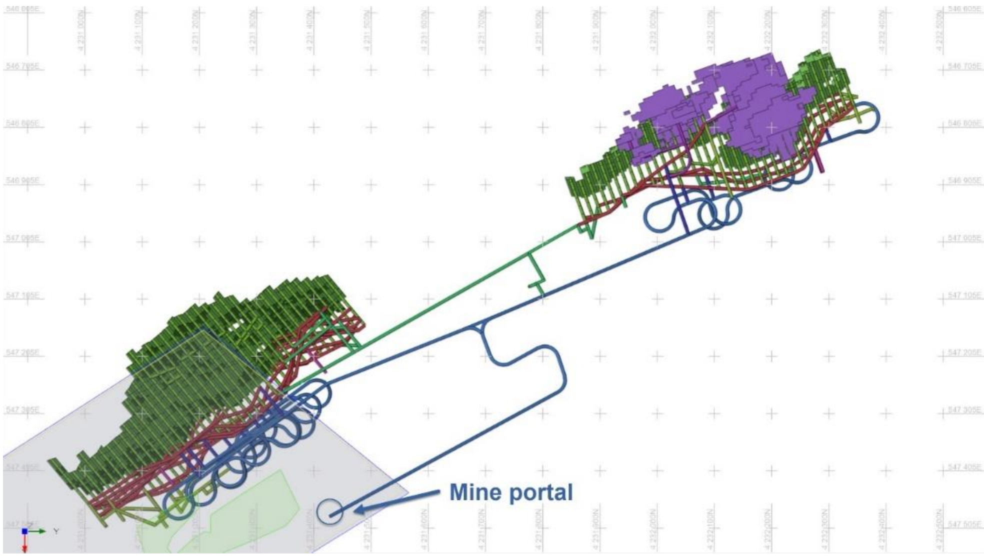


Figure 16-25 – Mine access and general mine layout (plan view)

The main ramp bifurcates into two independent declines, one for each mine area, 15 Level, as shown in Figure 16-26 (declines in red). In each zone, the internal decline connects all working levels.

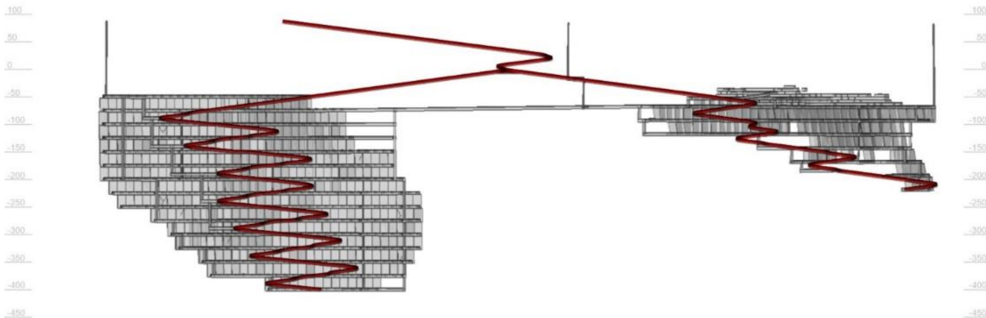


Figure 16-26 – Decline design (long section)

The use of an independent decline for each orebody, instead of one decline serving both zones, was chosen to reduce the initial Capital Cost ("CAPEX") considering that production will start earlier in the North Zone.

The ramps were designed with a 25m turning radius and a maximum gradient of 13%. Although the design does not provide for passing bays, it allows for an additional 25% of decline design meters if future developments require it.

16.9.2 LEVEL DEVELOPMENT

Working levels were placed at vertical intervals of a 25 m, defined by the stope height, and accessed through the internal declines. The footwall drives and declines were set back from the ore contact at a minimum of 40 m and 70 m, respectively. The typical stoping level configuration is shown in Figure 16-27 – fresh and return airways are placed at the ends of the sublevels and the level access is located at the middle of the footwall drive.

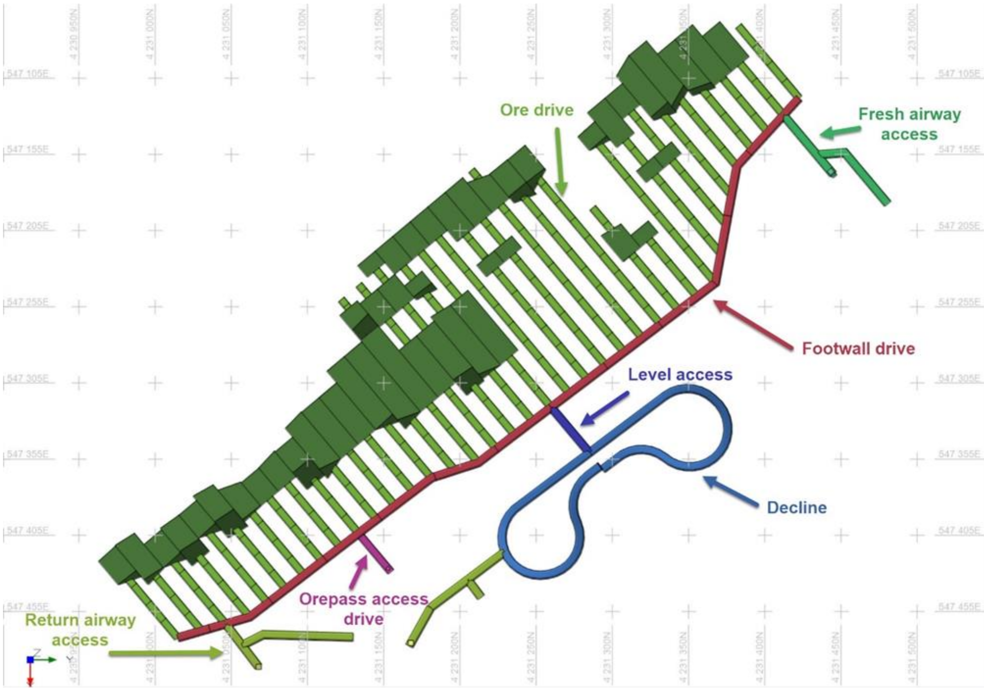


Figure 16-27 – Typical level layout (plan view – 220 South Level)

The cut&fill mining method has been selected for the upper levels in the North orebody (levels 42, 50 and 67). The design for these levels is presented in Figure 16-28 and Figure 16-29.

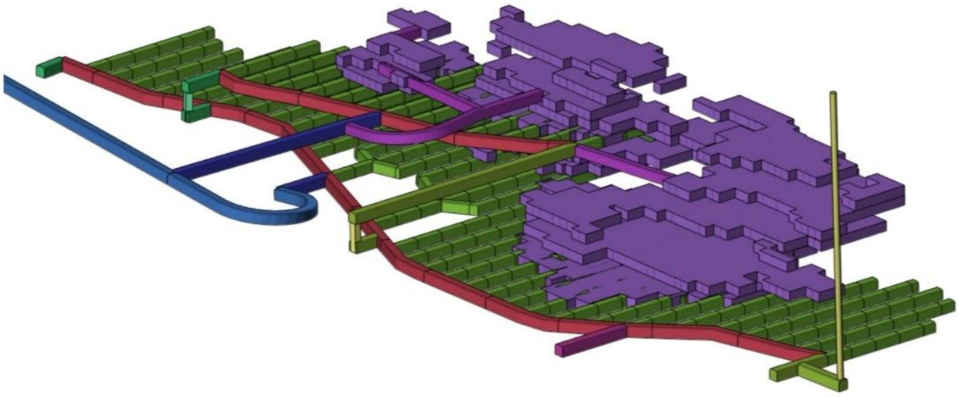


Figure 16-28 – Cut&Fill design (general view)

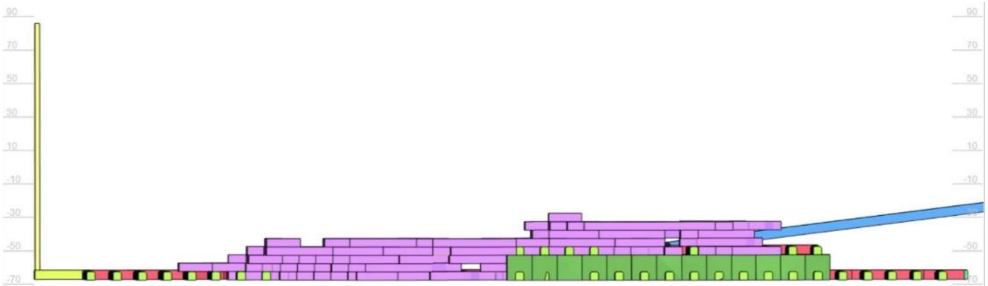


Figure 16-29 – Cut&Fill design (long section)

All other areas will be mined using the sublevel stoping method. The general infrastructure for these areas is shown in Figure 16-30 and Figure 16-31.

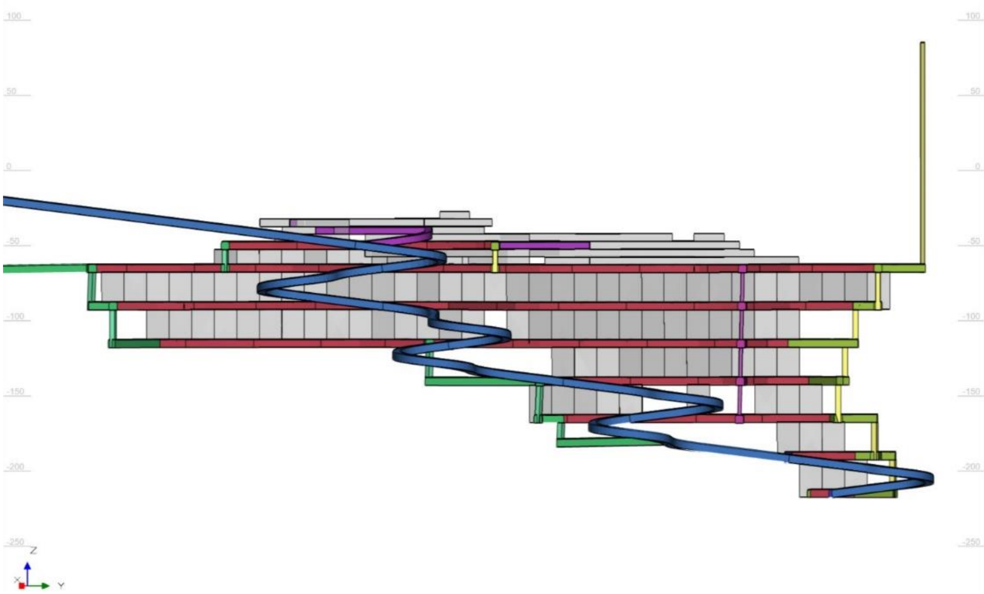


Figure 16-30 – North zone level design (long section)

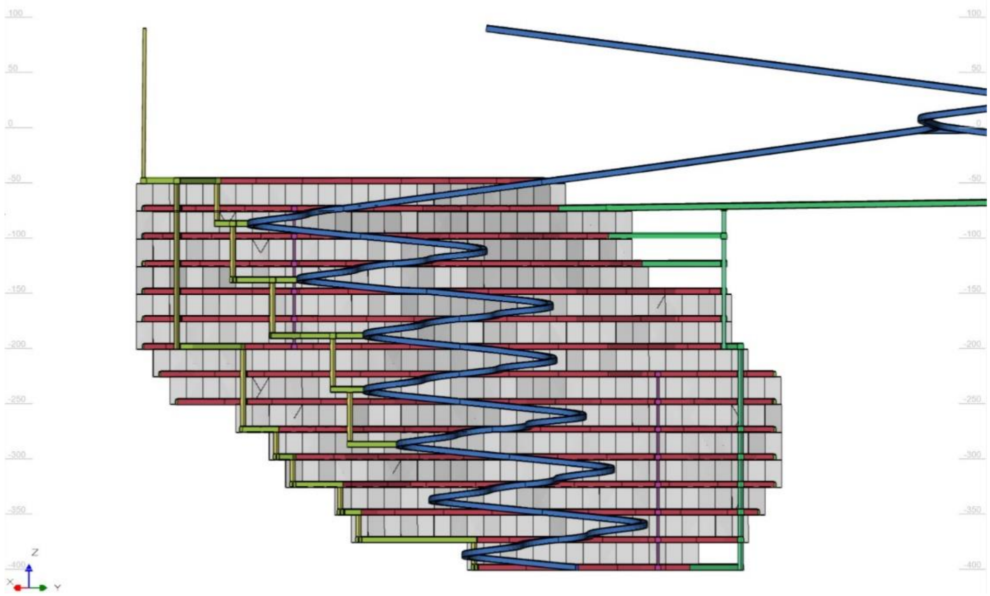


Figure 16-31 – South zone level design (long section)

Given the PEA level of the study, gradients have only been applied to the ramps. The operational and capital development has been designed without any gradients. The design parameters and criteria are listed in Table 16-25.

Table 16-25 – Mining design parameters

Parameter	Value	Comments
Maximum decline gradient	13%	Only used in ramps
Curve radius	25m	
Level access length	30m	Minimum value
C&F access length	15m	Minimum value
Orepass access length	30m	Minimum value
Ventilation raise access length	30m	Minimum value

The development sections have been defined as those sections suitable for accommodating mining equipment (selected according to the mining rates) and mining services (where applicable). The typical sections for each type of development are described in Table 16-26.

Table 16-26 – Sections for mining development

Development type	Section (m x m)
Ore access drive	5.0 x 5.0
C&F access	5.0 x 5.0
Footwall drive	5.5 x 5.5
Level access	5.5 x 5.5
Ramp	5.5 x 5.5
Ventilation raise access	5.5 x 5.5
Orepass access	5.5 x 5.5
Vent raise (Raisebored)	Ø 3.0
Vent raise (Blasted)	3.5 x 3.5
Orepass	Ø 2.4

Considering the mining design criteria mentioned above, it is estimated that 83.7km of horizontal development and 1.8km of vertical development will be needed for the entire LOM period. The details are shown in Table 16-27.

Table 16-27 – Horizontal and vertical development meters

Development type	Distance
Ore access drive	61 209
C&F access	1 329
Footwall drive	9 113
Level access	1 247
Ramp	6 105
Ventilation raise access – Fresh airway	1 937
Ventilation raise access – Return airway	1 937
Orepass access	995
Vent raise (Raisebored) – Fresh airway	206
Vent raise (Raisebored) – Return airway	444
Vent raise (Blasted) – Fresh airway	312
Vent raise (Blasted) – Return airway	501
Orepass	318
Total Horizontal Development	**83 731**
Total Vertical Development	**1 781**

16.9.3 Stope Design

The Deswik.SO optimisation software has been used for the generating the mine production stope and cut&fill shapes for each orebody. Table 16-28 lists the geometrical and angular parameters used for the stopes and the optimisation of the cut&fill mining methods.

Table 16-28 – Geometrical and angular parameters

Parameter	Value
Maximum width	15m
Maximum length	60m
Minimum height	15m
Maximum height	20m
Minimum angle – Front	60m
Minimum angle – Back	60m
Maximum angle – Front	90m
Maximum angle – Back	90m
C&F height	5m
C&F width	5m
Maximum C&F length	300m

For each mine zone and level, the combination of mining methods yielding the largest tonnage has been selected. Table 16-29 details the tonnage and NSR values obtained for the stope optimisation. Note that these values are reported on an in-situ basis, that is, prior to dilution and recovery being applied and that the results have been rounded to two significant figures.

Table 16-29 – Stope Optimizer results

Level	Mining Method	NSR (USD/t)	Tonnage (kt)	No. of# Stopes
North 42	C&F	82	120	-
North 50	C&F	92	310	-
North 67	C&F	140	620	-
North 67	Stoping	93	390	18
North 92	Stoping	110	3 500	53
North 117	Stoping	100	2 400	40
North 142	Stoping	100	950	17
North 167	Stoping	51	390	12
North 192	Stoping	68	200	3

Level	Mining Method	NSR (USD/t)	Tonnage (kt)	No. of# Stopes
North 217	Stoping	64	190	3
South 75	Stoping	51	850	37
South 100	Stoping	49	1 400	56
South 125	Stoping	52	1 400	54
South 150	Stoping	55	1 500	48
South 175	Stoping	55	1 600	49
South 200	Stoping	53	1 400	47
South 225	Stoping	51	1 500	47
South 250	Stoping	50	1 300	43
South 275	Stoping	50	1 300	43
South 300	Stoping	52	1 500	41
South 325	Stoping	53	1 300	38
South 350	Stoping	48	1 200	31
South 375	Stoping	44	810	26
South 400	Stoping	41	400	14
Total		**67**	**26 000**	**720**

Figure 16-32 and Figure 16-33 show the mineable shapes obtained for the optimisation of the North and South mine zones (respectively). These figures also show, the orebody shapes.

Figure 16-32 – Mineable shapes (North Zone)

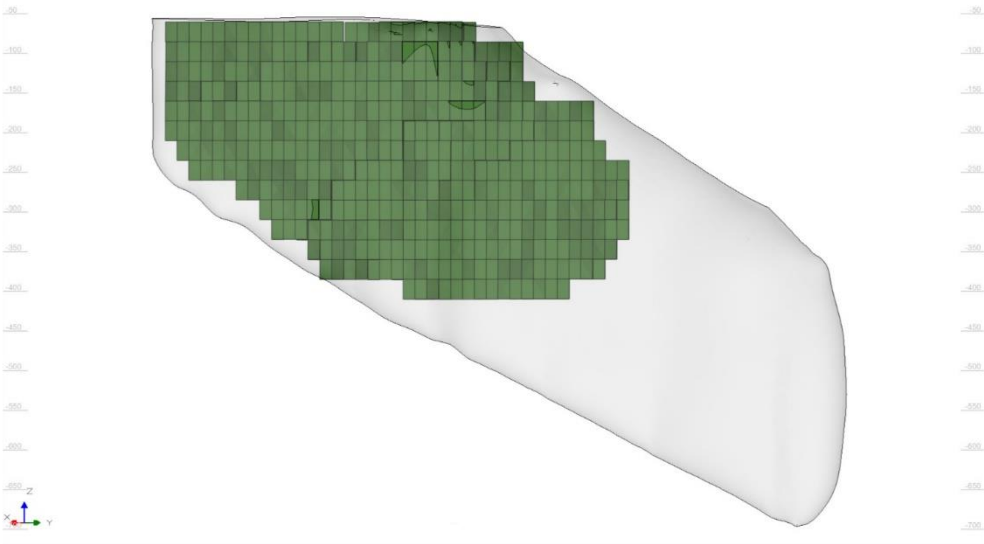


Figure 16-33 – Mineable shapes (South Zone)

16.10 MINE SCHEDULING

16.10.1 MODIFYING FACTORS

The mine plan has been prepared and reported on the basis of the diluted and recovered mass and grade. In doing so, the values listed in Table 16-30 have been used.

Table 16-30 – Dilution and recovery factors

Mining Method	Dilution	Recovery
LHOS – Primary stopes	10%	95%
LHOS – Secondary stopes	10%	95%
C&F	10%	95%
Ore development	10%	95%

16.10.2 MINE PRODUCTION SEQUENCING

Stope Production Sequence

The typical stope production cycle is as follows:

- Cablebolt support starts seven days after the stope access development has been completed. This provides time for stope preparation work (e.g., the cablebolt layout design, geological mapping, surveys, services installation);
- Production drilling in the stopes starts a minimum of two days after the cablebolt support has been installed;
- Once production has been completed, the stopes are paste-filled;

- A three-week curing period is maintained for the pastefill. This ensures that the backfill has adequate strength to ensure top, bottom and sidewall stability.

The cut&fill production cycle considers the following aspects:

- A four-week curing period is assumed for the paste fill;
- The paste fill starts three days after the end of production.

Geotechnical Sequence

In developing the mine plan, the following set of geotechnical sequencing rules have been considered:

- An inverse pyramidal, primary-secondary ascending sublevel retreat sequence is followed in the stoping zones.
- A retreat, ascending sublevel sequence is followed in the cut&fill zones.
- The secondary stope-access drifts on each level can only be started after their neighbouring primary stopes have been backfilled and a curing period of at least three weeks has elapsed.
- The retreat sequence in the stoping drifts is achieved by starting production drilling in the next stope to be mined, after the previous stope has been backfilled and a curing period of at least four weeks has elapsed.
- The primary stopes within a column and their associated development can have a maximum lead of three levels relative to their adjacent secondary stope columns.
- The cut&fill production starts after a four-week curing period for any stopes that have been backfilled on the same working level.
- A four-week curing period is maintained to ensure stability before opening a new level or sublevel.
- The South orebody is divided into two mining blocks by a pillar at Level 225. This pillar allows production to start earlier, as the infrastructure development will only need to reach Level 220 before mining can begin.

16.10.3 Resource Task Rates

The resource task rates considered in the creation of the mine plans are based on the author's own experience and information available from similar mining operations. These rates are described in Table 16-31 and Table 16-32.

Table 16-31 – Stoping resources task rates

Resource	Unit	Rate
Stope production – LHD	t/d	1 200
Pastefill production	m^3/d	2 000
Production drilling – Longhole drill rig	m/d	210
Cablebolting	m/d	220

Table 16-32 – Mine development rates

Development	Unit	Rate
Ore access drive	m/mo	60
C&F access	m/mo	60
Footwall drive	m/mo	90
Level access	m/mo	90
Ramp	m/mo	90
Ventilation raise access	m/mo	60
Orepass access	m/mo	60
Vent raising shaft	m/d	4
Vent blasting shaft	m/d	4
Orepass raising	m/d	4

The ramp development rates have been increased to 120 m/mo in the sections shown in blue in Figure 16-34.

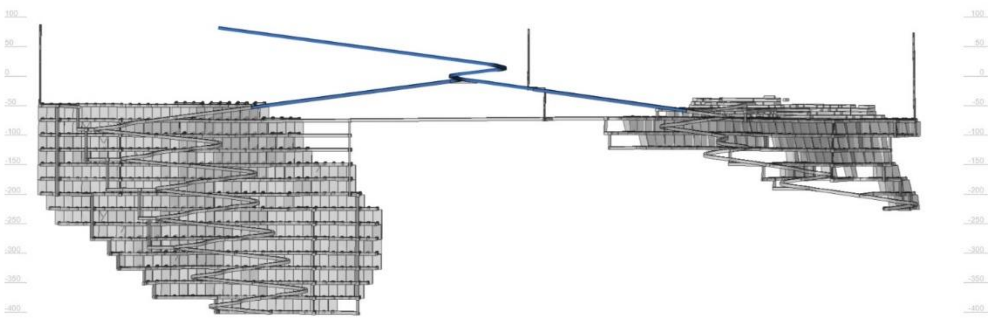


Figure 16-34 – Ramps with increased development rates

16.10.4 Resource and Production Limits

In the mine plan, the ore production has been limited to 2.0 Mtpa. No ramp-up rate has been established, as the mine is assumed to be the bottleneck of the operation. The development capacity is initially assumed to be 180 m per month per jumbo unit. An annual limit for the ore production by mine area has also been set, as shown in Table 16-33 and Table 16-34.

Table 16-33 – Ore production limits (North Zone)

Period	Limit (ktpa)
2022-2026	2 000
2026	1 500
2027	1 250
2028	1 400
2029	400
2030 onwards	200

Table 16-34 – Ore production limits (South Zone)

Period	Limit (ktpa)
2022-2026	-
2026	700
2027	950
2028	650
2029	1 600
2030 onwards	2 200

16.11 Development and Production Schedule mine Plan Results

16.11.1 Pre-Production Development

Pre-production development will take place over a period of approximately 21 months before the first stope is mined. The first priority for the development strategy is the North Zone, followed by the upper levels of the South Zone.

The first stopes will be mined at 92 and 117 Level of the North zone. The critical path pre-production activities include the following:

- Decline development to access the production levels in the North Zone and to reach 50 and 75 South levels.
- A fresh airway drive connecting both mine zones and the establishment of the fresh airway shaft in the decline and the drift.
- Development of the 67, 92 and 117 North levels and the return airway shaft.
- Development of the 50 and 75 South levels to set the return airway shafts.

A total development of 10 672 horizontal metres (capital and operational development) and 503 vertical metres is planned during the first 21 months.

16.11.2 LOM DEVELOPMENT SCHEDULE

Developing the North Zone alone could not sustain the established ore production rate of 2.0 Mtpa. For this reason, the main infrastructure of the South Zone should also be developed to open up working levels in this area. Prioritising the North Zone facilitates the promotion of higher grades during the life of mine as soon as possible. The results of the development strategy are shown inTable 16-35. The horizontal development metres allow for an extra decline of 25%, as well as footwall drifts and level access design metres for any additional development that may be required (e.g., passing bays, ancillary service cuddies).

Table 16-35 – Planned capital and operational development

Year	Capex Development		Opex Development		Total Development	
	Horizontal	Vertical	Horizontal	Vertical	Horizontal	Vertical
Year -2	2 226	102	-	-	2 226	102
Year -1	6 841	510	4 350	-	11 190	510
Year 1	5 676	515	7 812	-	13 489	515
Year 2	3 711	143	4 083	-	7 794	143
Year 3	2 230	135	6 167	-	8 397	135
Year 4	2 950	85	6 077	-	9 027	85
Year 5	1 675	290	4 091	-	5 767	290
Year 6	-	-	5 241	-	5 241	-
Year 7	-	-	5 452	-	5 452	-
Year 8	-	-	4 136	-	4 136	-
Year 9	-	-	7 008	-	7 008	-
Year 10	-	-	2 972	-	2 972	-
Year 11	-	-	6 650	-	6 650	-
Year 12	63	-	5 748	-	5 810	-
Year 13	119	-	3 258	-	3 377	-
Year 14	-	-	2 033	-	2 033	-

The profile of the capital and operational horizontal development requirements is visually analysed in the chart in Figure 16-35.

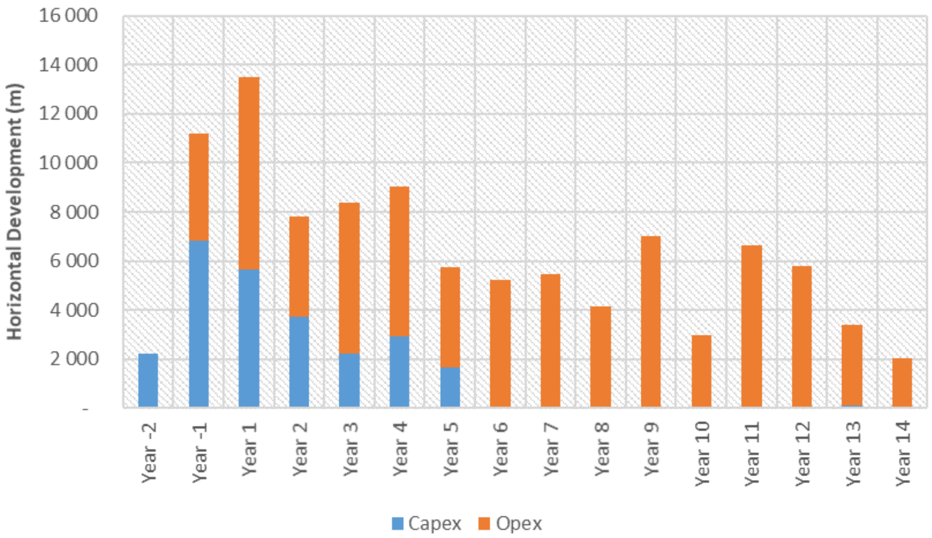


Figure 16-35 – Horizontal development profile for the LOM

16.11.3 LOM PRODUCTION SCHEDULE

Table 16-36 presents the annual plant feed and NSR values that have been obtained.

Table 16-36 – Annual plant feed and average NSR values

Year	Ore Production (kt)	NSR (USD/t)
Year -2	-	-
Year -1	280	81
Year 1	1 800	87
Year 2	2 000	88
Year 3	2 000	75
Year 4	2 000	80
Year 5	2 000	80
Year 6	2 000	61
Year 7	2 000	56
Year 8	2 000	51
Year 9	2 000	53
Year 10	2 000	48
Year 11	2 000	54
Year 12	1 900	51
Year 13	1 500	51
Year 14	590	51

The chart in Figure 16-36 illustrates the production and NSR values during the life of mine. The nominal plant feed is reached in the fourth year and maintained until the Year 12 when the ramp-down starts.

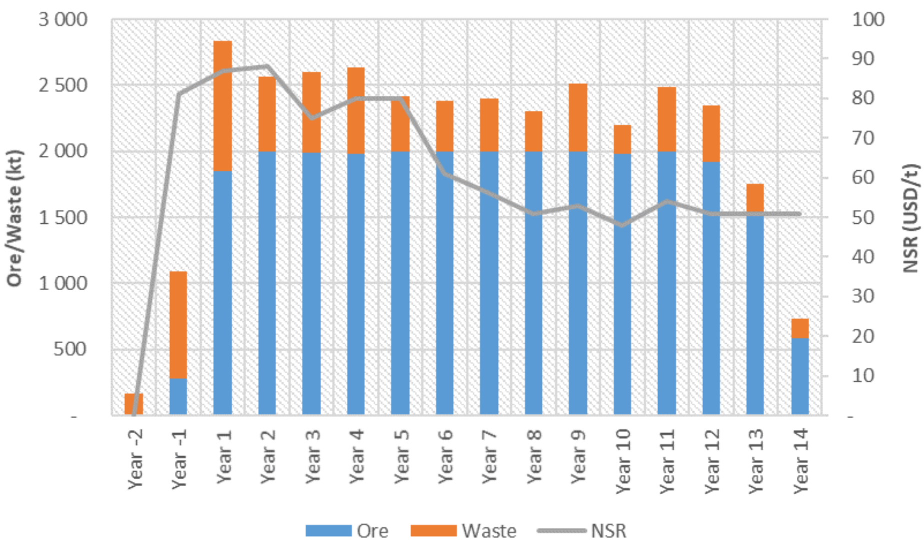


Figure 16-36 – LOM production Schedule and average NSR

16.12 VENTILATION

Ventilation is required to maintain safe and sustainable underground mining and the appropriate environmental conditions. For this purpose, it is necessary to dilute gaseous particulate pollutants and dust to concentrations which are not harmful to the health and safety of workers and maintain thermal comfort through the provision of adequate air velocity.

The general ventilation system is shown in Figure 16-37, it consists of a parallel system, where fresh airflow is provided by the main ramp and one central raise (FAW raise), to North and South zones. The return airflow is exhausted by the ventilation raises located at the extreme of each area. The main fans are located on the surface, at the top of these raises.

Fresh air is introduced to each level through the ramp and internal raises connected to the FAW raise. After passing across the footwall drive, the airflow is exhausted through internal raises connected to the RAW raise. The airflow quantity circulating at each level is controlled by ventilation regulators and fans located in the airway connecting the footwall drive to the internal return raise.

The return raises connecting the decline of the South Zone with the RAW raise have been designed with the objective of improving decline development. Once internal return raises have been developed that establish a connection to the footwall drive, raises connecting to the ramp will no longer be necessary for this purpose.

Fresh airflow is delivered to the production areas via secondary fans located on the footwall drive.

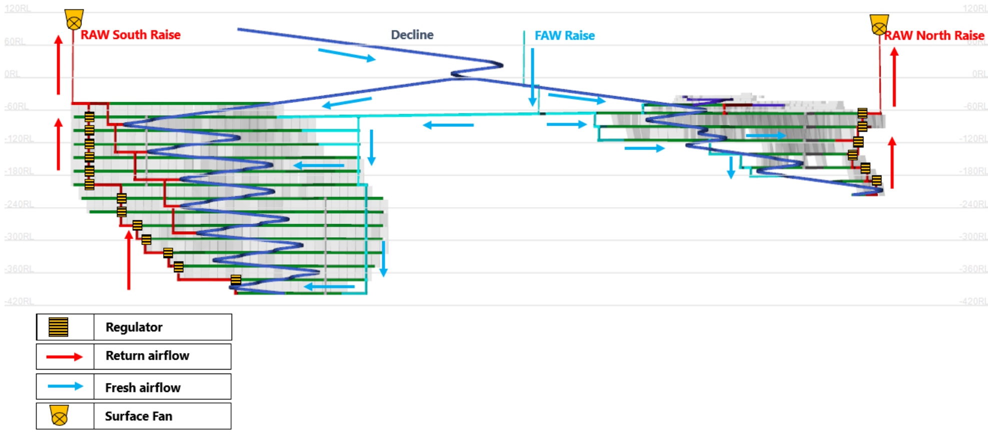


Figure 16-37 – Ventilation circuit (long section)

The required airflow has been calculated using a ratio of 0.0003vm^3/s/t (300 m^3/s/Mt for the sublevel stoping method). This ratio has been benchmarked with mines featuring the same production levels and mining method. This has also been validated with other mining operations in the IPB. This means that an amount of ~600 m^3/s will be necessary for the underground mining activities. This total ventilation requirement will supply the North and South Zones based on the required activity levels at any given time.

For typical development sections, and considering that the maximum air velocity recommended for a transport decline is 6 m/s, the maximum airflow that should circulate through the decline is 150 m^3/s. The FAW raise should provide 450 m^3/s. As the maximum airflow for raises bored in poor rock is 12 m/s, the FAW raise diameter should be 3.5 m. If, after further geotechnical investigation, it is confirmed that the raise will be done in competent rock, the airflow speed can be increased, and the diameter of the raises decreased in turn. The RAW raises should exhaust 600 m^3/s, which is why both raises have been designed with a diameter of 4 m.

17 RECOVERY METHODS

The process design criteria, flowsheet with major process equipment and a description of the process facilities are presented in this section.

The information is based on the existing metallurgical testwork as described in detail in Chapter 13 and in benchmark and author experience previously at Aljustrel and Neves-Corvo operations. Both operations are located in the Iberian Pyrite Belt as Lagoa Salgada Project.

In addition, in this chapter the expected process plant labor requirements for general, plant operations and plant maintenance are detailed.

17.1 Conceptual Process Flowsheet

The Process Design Criteria for the project is developed based on available data including the project schedule and benchmark IPB operation details.

The plant is designed for a total of 2Mtpa or 250tph throughput to produce two concentrates. Based on this, the plant LOM is expected to be 14 years.

The crushing plant is designed at 65% availability, while the main plant will operate at 92% availability.

Based on head grades of 0.31% for Cu, 1.44% for Zn, and 1.22% for Pb the average commodity recoveries are 80% for the Cu SW and 25% for the Cu MS, 80% for the Zn, and 75% for the Pb respectively. It is to be noted that due to fluctuations of the head grades of the different minerals from the initial years of operation through the LOM, the plant has been designed for the range of head grades of each mineral, rather than on the average grades presented in Table 17-1. Same logic has been used as part of the the equipment selection and sizing.

A summary of the conceptual process design criteria is provided in Table 17-1.

Table 17-1 - Process Design Criteria

Criteria	Units	Value	Source
Plant Throughput	tpa	2,000,000	Project Schedule
Plant Throughput	wmth	250	Project Schedule
Plant Operations	years	14	Project Schedule
Plant Availability (Crusher)	%	65	Benchmark IPB Op
Plant Availability (Other)	%	92	Benchmark IPB Op
Head Grade Cu	%	0.31	Project Schedule
Head Grade Zn	%	1.44	Project Schedule
Head Grade Pb	%	1.24	Project Schedule
Total Zn Recovery	%	80	Benchmark IPB Op
Total Pb Recovery	%	65	Benchmark IPB Op
Total Cu Recovery SW	%	80	Benchmark IPB Op
Total Cu Recovery MS	%	25	Benchmark IPB Op

17.2 CONCEPTUAL PROCESS FLOWSHEET

The conceptual methodology process flowsheet envisioned for this deposit consists of:

- Primary, Secondary, and Tertiary crushing.
- Coarse material stockpiling.
- Primary ball mill grinding.
- Secondary ball mill grinding and cyclone classification.
- Lead Flotation.
- Zinc Flotation.
- Concentrates Thickening.
- Concentrates Filtration.
- Final Concentrates Stockpiling.
- Tailings Disposal.

A simplified process flowsheet for Lagoa Salgada Project is presented in Figure 17-1 and Figure 17-2.

Figure 17-1 – Ball mill grinding diagram

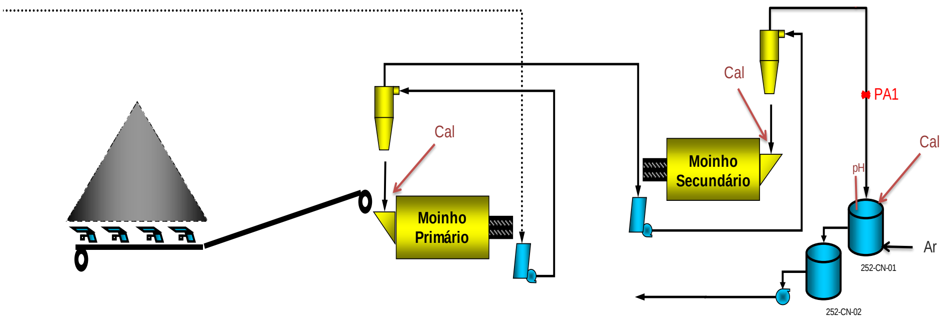
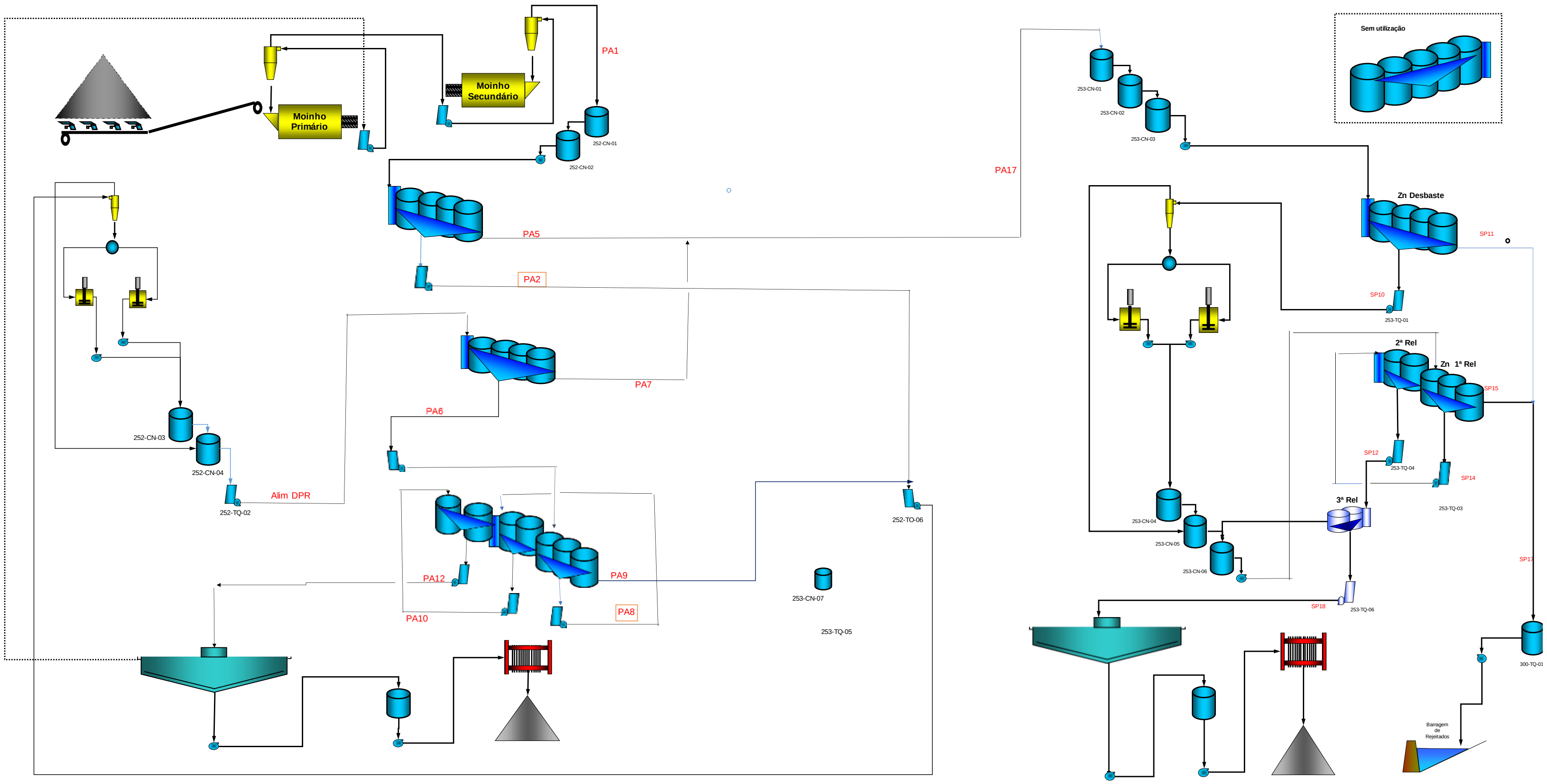


Figure 17-2 – Lagoa Salgada flotation circuit



17.3 Process Description

17.3.1 Crushing Area

Run-of-mine (ROM) material is hauled from the mines by truck and either stockpiled or directly dumped into the primary crusher.

The crushing plant consists of three stages of crushers – primary, secondary and tertiary.

The crushed material is transported by an apron feeder and short conveyor to the overland conveyor system, which transports the material to a crushed material stockpile for further processing.

17.3.2 Grinding Area

The overland conveyor transports the crushed material to a stockpile. Apron feeders withdraw material from the stockpile and transport it via conveyor to the grinding mill. The primary grinding ball mill operates in a closed circuit, with a battery of cyclones. The underflow of these primary cyclones returns to the primary milling joining the new feed. The overflow of the primary cyclones is sent to the sump of the secondary mill, as the final product of the primary milling, where it is combined with the product of the discharge of that mill and water. Sump's product from the secondary mill is pumped by one of the pumps to the secondary cyclone battery. The overflow of these cyclones constitutes the final product of the grinding and it is sent to the Lead/Zinc flotation.

17.3.3 Lead Flotation Area

The product of the griding gravitates to two (2) stages of conditioning:

- Conditioning #1 – Pulp aeration
- Conditioning #2 – Collector addition.

From the conditioner #2 the pulp is pumped by one of the flotation feeds pumps to the coarse rougher flotation consisting of a set of four (4) flotation cells of 100 m^3 each.

The concentrate of the coarse rougher flotation gravitates directly to the regrind feed sump where the tailings from the first cleaner flotation join, becoming the feed to the regrind, pumped to the regrind cyclones battery.

The underflow of the cyclones feed two (2) SMD mills dedicated to lead regrind.

The final product of the regrind it’s constituted by the mixture of cyclones overflow and the outflow of the SMD mills, that come together into the sump that feed rougher regrind product. Finally, it’s pumped to lead rougher regrind flotation.

The tailings from DPR join the tailings from coarse scavenger flotation, constituting the set of these two flows, the final tailings from lead flotation.

The concentrate from DPR, gravitate to the feed sump of the first cleaner flotation, where joins the tailings from the second cleaner flotation, constituting the set of these two flows the feed of the first cleaner flotation, that is pumped to the first cleaner flotation.

The tailings from the first cleaner flotation are resent to the regrinding.

The concentrate from the first cleaner flotation, gravitate to the feed sump of the second cleaner flotation and then it's pumped to the second cleaner flotation, where join the tailings of the third cleaner flotation, constituting the set of these two flows the feed of the second cleaner flotation.

The tailings from the second cleaner flotation, gravitate to the feed sump of the first cleaner flotation.

The concentrate of the second cleaner flotation gravitates to the feed sump of the third cleaner flotation. The third cleaner flotation concentrate it's the final concentrate, and the tailings are recirculated directly to the second cleaner flotation.

The final concentrate it's sent to the concentrate thickener, where it's thickened to 65-70% solids, and sent to two (2) tanks that feed a concentrate pressure filter, where the concentrate is filtered to a moisture content of 9-11%.

The final concentrate is stockpiled for transportation away from the processing plant.

17.3.4 ZINC FLOTATION AND REGRIND AREA

The final tailing from the lead flotation it's the feed of the zinc flotation. This flow it's pumped sequentially for three (3) stages of conditioning:

- Conditioning #1 – pH regulation, adding lime.
- Conditioning #2 – Blend activation by addition of $CuSO_4$.
- Conditioning #3 – Collector is added.

From the Conditioning #3 stage, the pulp is pumped to the Zinc rougher flotation, constituted by a battery of flotation cells.

The rougher flotation concentrate is directly sent to the regrinding, constituted by two (2) SMD mills that operates in reverse closed circuit with battery of cyclones.

The final product of the regrinding is sent to the first cleaner flotation. The first cleaner flotation is carried out in a set of three (3) flotation cells. The concentrate is sent to first the second cleaner flotation and the tailings joins the tailings from the rougher flotation, constituting the whole, the final plant tailings that is sent to the paste fill or tailing storage facility.

The second cleaner flotation is performed in a set of two (2) flotation cells. The concentrate is sent to the third cleaner flotation and the tailings are recirculated to first cleaner flotation.

The third cleaner flotation is performed in a set of two (2) flotation cells. The concentrate is the final zinc concentrate and the tailings are recirculated to the second cleaner flotation.

The final concentrate is stockpiled for transportation away from the processing plant.

The final product of the regrinding, constituted by the mixture of the cyclones overflow and the outflow of SMD mills, feeds the DPR done in a set of three (3) flotation cells.

The DPR concentrate is sent to the first cleaner flotation. The tailings join the rougher tailings, constituting the plant final tailings that is sent to the paste fill or tailings storage facility.

All concentrates will be transported by truck or railway to the nearby seaports.

17.3.5 Dewatering Area

The dewatering area includes Pb and Zn concentrates thickeners and pressure filters.

The thickeners are responsible for reducing the concentrates water content to approximately 65-70% solids and the pressure filters – to 9-11% moisture.

17.3.6 Reagents Area

The reagents area includes the flotation reagents and flocculants to support the processes including flotation frothers, promoters, depressants, activators, pH regulators, and thickeners flocculants to enhance the settling processes.

Each type of reagent depending on the state it is supplied includes feeding and/or mixing tanks and distribution metering pumps.

17.3.7 Plant Tailings Area

The overall processing plant tailings comprise of the first Zn cleaner flotation tailings joined with the tailings from the Zn rougher flotation. The final plant tailings is sent to the paste fill or the Tailing Management Facility (TMF).

17.3.8 Plant Water Area

The processing plant water area includes:

- Fresh water.
- Process water.
- Fire water.
- Gland seal water.

The process plant water is pumped to a process water tank located at the main plant area and from there it is delivered to grinding, Pb and Zn flotation, and flotation reagents mixing.

The excess water from the concentrates thickeners and pressure filters is recirculated back to the plant through the process water tank.

The plant area includes fire water system with fire water tank and pumps.

The plant area is also equipped with gland water system including fire water tank and pumps.

17.3.9 Plant Utilities Area

Plant utilities and support facilities include:

- Grinding media receiving and storage.
- Spares receiving and storage.
- Dust collection system.
- Air blowers and compressors and air supply.
- Assay and metallurgical laboratory.
- Standby power generator.
- Lift trucks.
- On stream analysers and sampling systems.
- Process Control Room and systems.
- Electrical distribution systems.
- Security systems and product storage.

17.4 Process Plant Equipment

The process plant equipment consists of:

- Grinding and regrind equipment including ball and SMD mills and cyclones
- Pb and Zn flotation conditioning tanks and cells
- Concentrates thickeners and pressure filters.

The Major Process Equipment List is presented in Table 17-2.

Table 17-2 – Major Process Equipment List

Equipment	Number	Size
Primary Ball Mill	1	5.1m x 6.3 m
Primary Cyclones	3	Ø= 520 mm
Secondary Cyclones	10	Ø= 250 mm
SMD Mill	4	
Flotation Cells	15	30 m^3
Flotation Cells	6	15 m^3
Flotation Cells	4	100 m^3
Thickener	2	
Conditioner	5	

17.5 PROCESS LABOUR REQUIREMENTS

Process plant labour includes plant management, technical staff, operations, and maintenance personnel, and administrative staff.

The overall personnel peaks at 73 persons in the first two years of operation.

A summary of the required process plant labour is presented in Table 17-3.

Table 17-3 - Mill Production and Maintenance Personnel

Position	Personnel
Mill Operations	
Mill Manager	1
Metallurgist	1
Mill General Foreman	1
Mill Foreman	4
Sampler/Assayer	2
Mill Clerk	1
Crusher Operator	4
Grinding Operator	4
Flotation Operator	8
Dewatering Operator	8
Utilities Operator	4
Control Room Operator	2
Labourer	10
Subtotal	***50***
Mill Maintenance	
Mill Maintenance Manager	1
Mill Maintenance Planner	1
Milwright	4
Electrician/Instruments	4
Welder	5
Labourer	8
Subtotal	***23***
TOTAL	**73**

18 PROJECT INFRASTRUCTURE

18.1 INTRODUCTION

As discussed earlier in this Report, the Lagoa Salgada Project will be implemented at a greenfield site. The main Project infrastructure components include the support infrastructure for the mine and the processing plant, the tailings storage facility (TSF), internal roads, the supply and distribution of power, and a water treatment plant.

A conceptual general layout of the Project infrastructure is shown in Figure 18-1, including:

1. The existing gravel access road to site.
2. The surface infrastructure layout.
3. The TSF area.
4. The temporary fan for initial development stage.
5. The North main fan.

A more detailed view of the proposed infrastructure layout is shown in Figure 18-2.

18.2 ROADS & ACCESSES

The Property can be accessed via good, paved roads from Cilha do Pascoal. There is a highway link from Lisbon to Grândola, the Setúbal port facilities and the Sines industrial area, allowing for fast connections to most important populations centres.

From Cilha do Pascoal, access to the LS site takes place via a 4 km gravel road that may require some improvements to accommodate heavy construction traffic (Figure 18-3).

All traffic will use these roads to access the site, including light vehicles for personnel access and heavy vehicles for the delivery of material and the trucking of the concentrate to be produced. Concentrate trucking, the transport of cement to the site (for paste fill) and the transport of diesel for the fuel tanks will be the main causes of heavy traffic on the roads. In terms of light vehicles, the access of workers, mainly using personal vehicles, will be the main cause of traffic.

Figure 18-1 – Project infrastructure – General layout

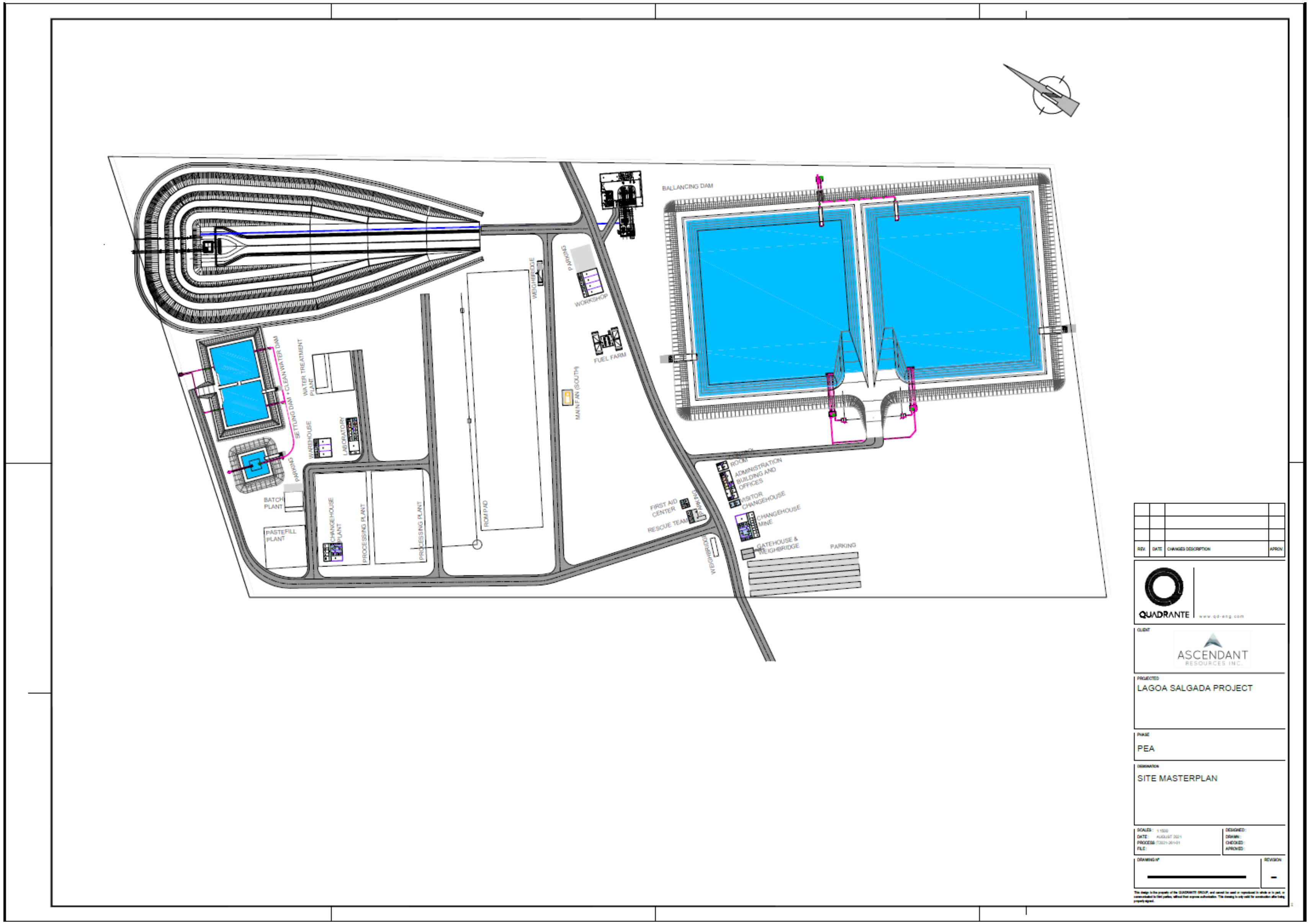


Figure 18-2 – Project infrastructure – Surface infrastructure layout

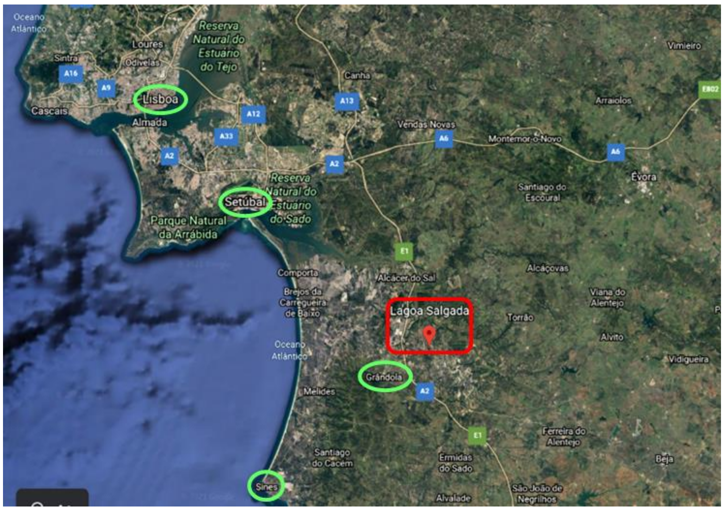


Figure 18-3 – Location of Lagoa Salgada and the main access point (source: Google Earth)

18.3 Stockpiles

The surface stockpiles will temporarily accommodate both waste and ore materials trucked from underground. The area projected for stockpiles has a size 15,000 m^2 (250 m x 60 m) and is located between the mine portal and the processing plant. The ore and waste disposal will be done using mine trucks and organised using front wheel loaders, which will transport the material to a conveyor belt system connecting the stockpile area and the processing plant.

18.4 Waste Storage Facilities

More waste will be generated in the first two years of the Project due to all the capital development that is necessary to start ore production. The amount of waste will gradually reduce in the following years as reflected in the LOM plan.

The waste will be stored temporarily in the stockpile area and moved by truck to its final destination, i.e., the dry stack storage facility used primarily for plant tailings. The waste will be extended with a bulldozer and compacted using a compacting roller.

18.5 Tailings Storage Facility

18.5.1 Introduction

The planned tailings disposal system for the Lagoa Salgada Project takes the form of dry stack tailings. This type of tailings disposal offers an improved solution for tailings management compared to other disposal systems (e.g., subaqueous disposal, paste disposal), as it has a lower operating risk profile and results in a stable landform that greatly simplifies mine closure.

This disposal system also represents a commitment to the safe storage and sound management of mining tailings. As tailings management traditionally represents one of the areas of greatest environmental risk in mining projects, this method is also expected to have a positive impact vis-à-vis the Project's stakeholders, including local communities and mining regulators.

18.5.2 Location and Capacity of the Tailings Storage Facility

Due to its location and topography, the dry lagoon area (Lagoa Salgada) – see the layout in Figure 18-1 – is best suited for conversion into a tailings storage facility. This evaporated salty lagoon has an area of 13.5 ha. As the aerial photos show, the density of vegetation is low compared to the surrounding areas.

According to the LOM plan, the estimated average production will be approximately 2.0 Mtpa. The tailings are estimated to amount to approximately 1,9 Mtpa. At an ore density 3, this corresponds to 633,400 m^3 of tailings per year.

During the LOM, a total of 26,070 kt will be processed, and 1,443 kt of concentrates will be produced. This balance shows that a total of 24,625 kt of tailings will be disposed of, either as paste fill into the stopes (14,776 kt, or 60% of the total) or as dry stack tailings in the TSF (9,850 kt, or 40% of the total).

Considering a compacted density of 2.5 for the dry stack tailings, a total of 3.94 Mm^3 will need to be disposed of in the TSF. Given that according to the LOM plan, around 3.8 Mm^3 of waste will be generated, the TSF area will have to be expanded in the future, unless additional voids are filled with waste (e.g., the development voids of mined-out areas or ore passes).

Considering only the generation of dry stack tailings, the available area of 13.5 ha (135,000 m^2) will accommodate all the tailings foreseen in the LOM plan, at a final height of 29 m. This height can be considered suitable for the type of disposal, given that the capacity of the lagoon will be sufficient for the tailings based on the actual LOM.

18.5.3 Suitability of Dry Stack Tailings Disposal System

Despite its lower impact, dry stacking present a number of risks compared to the other disposal systems. For example, in locations where strong winds, high temperatures and low humidity occur, dust generation is an issue, while high rainfall environments can create day-to-day management problems for the trafficability of haulage and compaction equipment.

An analysis of the location and climatic data for the Lagoa Salgada Project has confirmed the suitability of this location for the establishment of a dry tailings disposal system.

In a later phase of the Project, detailed geotechnical studies for this area must be carried out. At this stage, it has been assumed that this area is geotechnically suitable to be converted into a tailings disposal facility.

18.6 WATER SYSTEMS AND MANAGEMENT

18.6.1 WATER SYSTEMS

Raw water will be required at the site for the following applications:

- Make-up potable water;
- Process water;
- Mine water supply;
- Fire protection.

Raw Water

It has been assumed that raw water will be obtained from local groundwater wells, which are either already existing or will be drilled on or adjacent to the site. Raw water will be directed to the raw water tank (from which it will feed the potable water treatment system) and to the balancing dam that serves as a source of process and mine water.

Potable Water

Raw water will be sourced from local groundwater wells and processed through filtration and disinfection equipment before being stored in the potable water tank. Filtration and disinfection equipment will be available in order to obtain water for human consumption in accordance with the drinking water regulations established by Portuguese law.

There is the option to use the same company that supplies water to the town of Grândola for the supply of drinking water to the site infrastructure. At a later stage, this option will be analysed in order to reduce the cost of producing potable water on site.

Treated water will be used for human consumption, showers, toilets and sinks in the buildings and ancillary facilities. A pump house will be constructed adjacent to the potable water tank. A booster station will convey water from the tank directly to the buried distribution network.

Fire Water

Raw water will be sourced from local groundwater wells and stored in the fire water tank. There will be an electric and a diesel-powered fire water pumping system. The fire water system will consist of a buried fire water ring main and several fire hydrants at the plant site, ancillary buildings and the processing plant. Hose cabinets will be placed at the fire hydrant locations, and the system will be supplemented with portable fire extinguishers placed within the processing facilities.

Process Water

The water used for the processing facilities will be recirculated to limit the amount of raw water required. Process water will be supplied from a combination of treated water, excess mine dewatering water, treated effluent, storm water (rainfall falling onto the terraces on site) and raw water.

- Process water will be treated and returned for re-use in processing.
- Excess mine dewatering water will be conveyed to the balancing dam.
- Oily/water will be directed to an oil/water separator (vendor package) via a buried network for the gross removal of hydrocarbon contaminants. From the oil/water separator, the water will be delivered to the balancing dam. Oil from the oil/water separator will be stored in waste oil totes and sent off site for final disposal.
- Treated effluent will be delivered from the WWTP to the balancing dam via a treated effluent pump and pipeline.
- The dirty water runoff from the site will be conveyed to the balancing dam in a lined stormwater channel.

Raw water from the groundwater wells will be supplied as a contingency to the balancing dam in the event that insufficient water resources are available for re-use on site.

Mine Water

Water for use in mining activities, road dust suppression and the mine truck wash will be supplied by mine dewatering. Mine dewatering water will be discharged from the mine dewatering pumps to the settling dams. From the settling dams, the water will be directed to the clean water dam. Clean overflow water from the clean water dam will be sent down to the UG mine water reticulation system to be re-used in the mining operations.

Settling ponds with two compartments will be constructed on the surface to ensure that one of them can be cleaned without disrupting the water supply into the mine.

Sewage Treatment

Sewage generated at the Project site will be collected via an underground sanitary sewer network to a common location where it will be treated by a sewage treatment plant (vendor package). Treated sewage effluent will be conveyed to the balancing dam.

18.6.2 Site Wide Water Balance

Deterministic water balance modelling has been carried out for the operating phase in order 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 18-4 and Figure 18-5 presents the total water balance of the site in January and July, i.e., the months requiring the smallest and the greatest amount of make-up water respectively.

The modelling results show that in January, the recirculated water can (on average) meet the process and mine water demands of the Project without withdrawing any additional water from aquifers through groundwater wells.

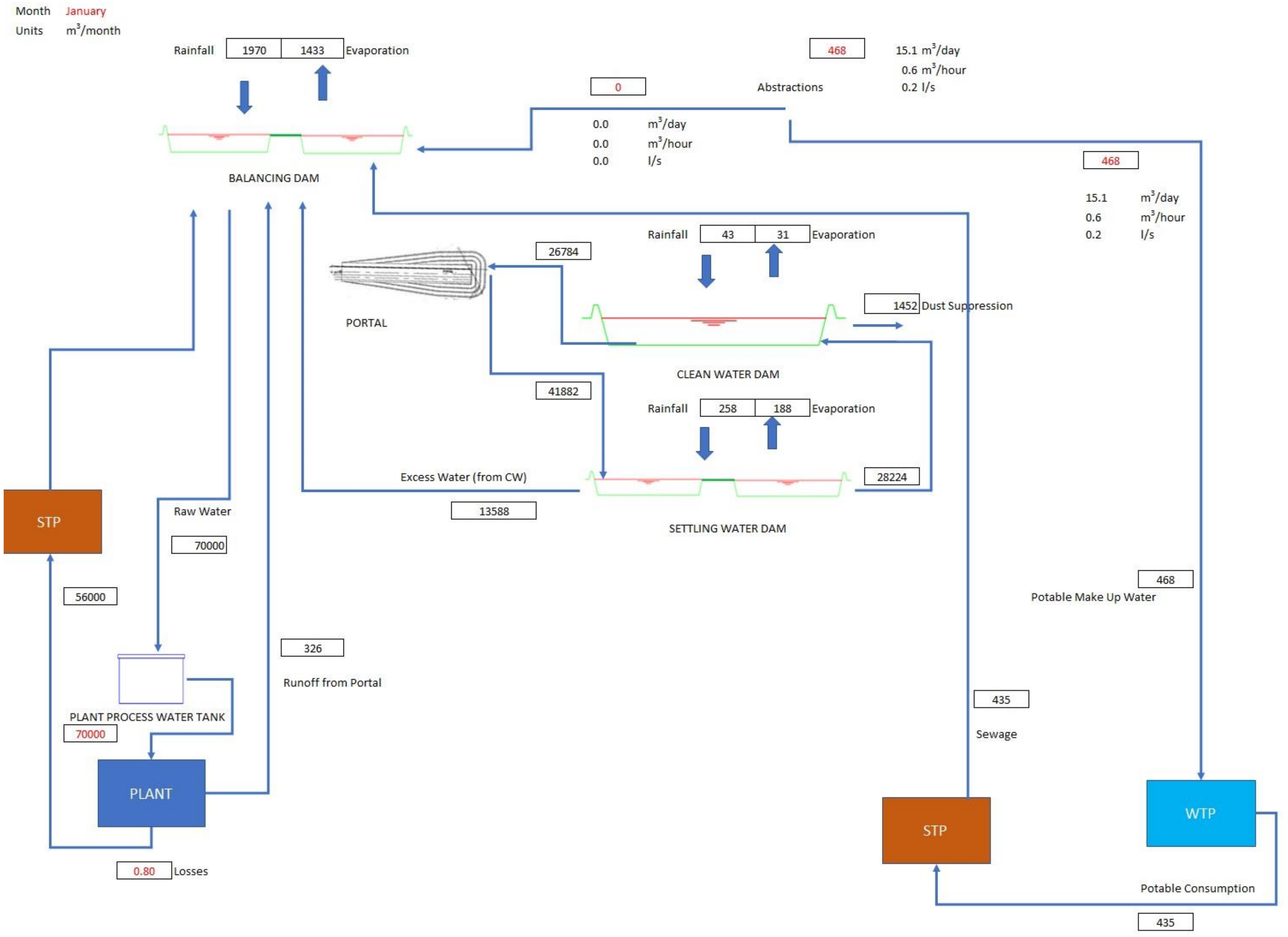


Figure 18-4 – Site water balance – January

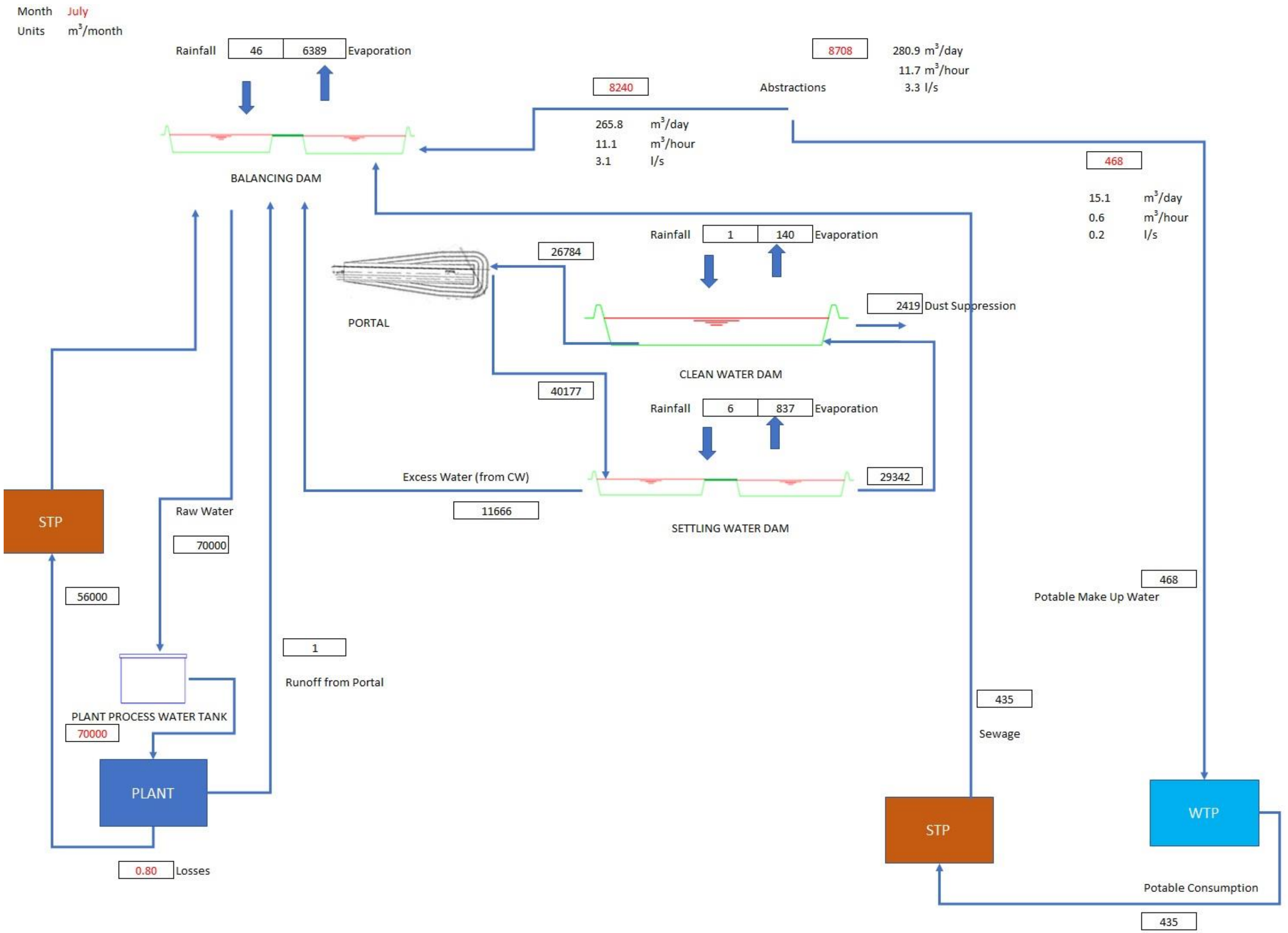


Figure 18-5 – Site water balance – July

18.6.3 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 utilised.

The majority of the water that will need to be managed at the site comes from the dewatering of underground workings, followed by surface run-off around the site, especially during major precipitation events (such as 1-in-100-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 is 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 (balancing dam). 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. Three dams will be constructed on the surface to manage the contact and process water. The details of the ponds and effluent treatment plant are summarised below:

- Balancing dam – A lined containment pond with a capacity of approximately 70,000 m^3 will be constructed to store excess mine dewatering water, treated water from the effluent and the treatment plants as well as runoff water from the site. The dam has two compartments to ensure that one of them can be cleaned without disrupting the water supply into the mine. 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 1,500 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 boxcut, and will have a capacity of approximately 500 m^3. This dam stores water for use in the mining operations.

18.7 CAMPS AND ACCOMMODATION

The Project site is close to communities with the infrastructures needed to provide accommodation to workers coming from outside the area. The majority of the workers will be recruited from these local communities. These workers will therefore not require relocation from their homes.

No on-site camps and accommodation will be needed.

18.8 BUILDINGS AND BUILT INFRASTRUCTURE

The main ancillary buildings proposed for the production phase of the Project are indicated in Table 18-1.

Table 18-1 – List of ancillary buildings

Building	Gross Area (m^2)
Main Building	420 (ground floor) 288 (first floor)
Warehouse	304
Workshop	477
Rescue Team HQ	57
Laboratory	359
First Aid Facility	63
Mine Change House	459
Gatehouse	59

A short description of each of these building is provided below.

18.8.1 MAIN BUILDING

The Main Building is a complex of three buildings: the Visitor Change House (53 sqm), the Administration & Offices building (288 sqm) and the Control & Automation facility (79 sqm).

The Visitor Change House will have two change rooms (female and male) and is estimated to be used by up to 10 visitors at any time.

The Administration & Offices building will function as a main office facility for the management of the site and is estimated to be used by 112 permanent workers. It will have a reception area with waiting room, several office spaces, meeting rooms, shift change rooms, a kitchenette and toilets.

The Control & Automation building is designed to accommodate the people and equipment needed for the control of the mining operations. It has also been conceived as an area for the operation of teleremote/autonomous mining equipment if required.

18.8.2 Warehouse

The warehouse will serve as a storage facility for equipment, parts, materials and consumables. It is estimated to be used by two permanent workers. It will have an open space for storage, office space, other smaller storage areas, an entrance and reception area for receiving products and materials, a kitchenette and toilets.

18.8.3 Rescue Team HQ

This building will be the rescue operation centre. It will also be used for storing firefighting/rescue equipment, emergency vehicles and personal protective equipment for crisis/emergency situations. It is estimated to be used by thee permanent workers. It will have office spaces, toilets and a storage room.

18.8.4 Laboratory

The Laboratory will provide information regarding the processing plant sampling, concentrate shipping, exploration, infill and production diamond drilling, and water sampling. It has been conceived for a total of six users. To fulfil its functional purpose, it contains the following spaces: male and female toilets/change room, laboratories, sample preparation and storage rooms, a meeting room, offices and a kitchenette.

18.8.5 First Aid Facility

This building will function as the mine’s main first aid facility for immediate primary treatment in case of emergencies and other non-urgent medical consultations. It is estimated to be used by four permanent workers. It will have a reception and waiting room, a medical room and infirmary, a storage area, a kitchenette and toilets.

18.8.6 Mine Change house

This building is the change house for the mine operation workers and the general supervisory staff. It has a capacity of 40 miners/shift, 172 lockers and space for two permanent staff. To fulfil its functional purpose, it contains the following spaces: a reception, a laundry, male and female change rooms, locker rooms, a storage area and a lamp room.

18.8.7 Gatehouse

This building is designed to be used by security personnel for controlling the entrance and exit flows of workers, contractors and visitors. It can accommodate three permanent security staff and will have a reception area for visitors with a waiting area, a weighbridge control room, an office, toilets and a kitchenette.

18.9 POWER AND ELECTRICAL

18.9.1 INTRODUCTION

The Lagoa Salgada Project will be equipped with an electrical supply system, sized to the operational loads required during the construction and operation of the mine, including the surface and underground infrastructure.

From a power supply point of view, this will be a new connection to the Rede Elétrica de Serviço Público (Public Service Electric Network – RESP), with a distribution voltage of 60 kV and a power requirement of 21 MVA, as outlined in the chapters below. This connection will consist in the construction of a 60 kV double overhead line (OHL) to an existing 60 kV OHL that connects two substations of the distribution system operator (E-REDES). It is also expected that a new 60 kV switching station with three bays will be required by E-REDES to complete this connection. Both the new OHL and the switching station will belong to E-REDES.

At the Project site, a new step-down substation will be constructed to reduce the grid connection voltage level (60 kV) to the internal distribution voltage levels (15 kV and 6 kV), and to distribute the power from the substation to the loads. The configuration of the proposed electrical power supply system for the Project is shown in the single line diagram in (Figure 18-6).

The connection point to the site will be located in Grândola, Setúbal, approximately at the following coordinates: 38°13'37.02"N |8°27'18.91"W.

The electrical system of the Lagoa Salgada Project will have the general characteristics presented in Table 18-2. All equipment to be installed for this Project must be compatible with these characteristics, to avoid, compromising the proper operation of the entire electrical system.

Table 18-2 – General characteristics of the electrical system

Characteristics	HV Level	MV Level	MV Level	LV Level
Number of poles/phases	3	3	3	3
Nominal voltage (U_n)	60 kV	15 kV	6 kV	400 V
Highest voltage (U_m)	72,5 kV	17,5 kV	7,2 kV	1000 V
Initial symmetrical short-circuit current	25 kA	12,5 kA	TBD	TBD
Peak short-circuit current	63 kA	31,5 kA	TBD	TBD
Nominal frequency	50 Hz	50 Hz	50 Hz	50 Hz

Characteristics	HV Level	MV Level	MV Level	LV Level
System neutral circuit	Solidly Earthed	Resistance Earthed	Resistance Earthed	Solidly Earthed

Given the importance of the sustainability of the Project's power supply, the implementation of a renewable energy auto consumption (Unidade de Produção de Auto Consumo – UPAC) system should be considered from the start of the Project or during the operation stage. While this solution would have an impact on the Project's CAPEX, it would allow the OPEX to be reduced through energy cost savings. The implementation of such a UPAC system would also have a positive impact on the environmental impact study phase.

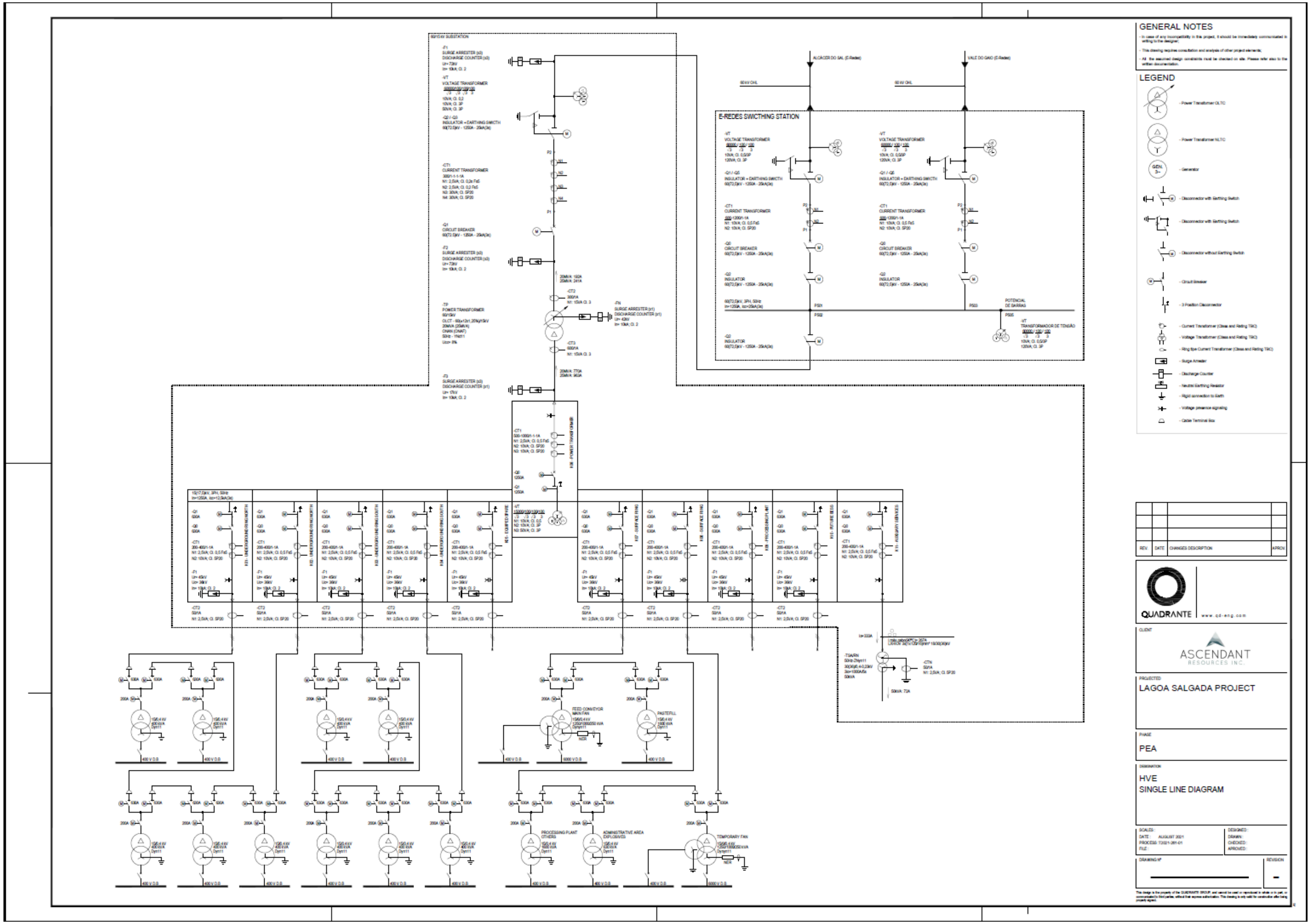


Figure 18-6 – Single line electrical diagram

18.9.2 POWER ESTIMATION

A preliminary estimate has been carried out to define the amount of power required for the Lagoa Salgada Project (Table 18-3).

Table 18-3 – Power estimation

Area		Installed Power (MW)
Pastefill Plant		1.5
Processing Plant		6.9
UG	Mining Equipment	1.6
	Secondary Ventilation	1.1
	Pumping	0.5
Plant Feed Conveyors		0.3
Main Fans		1.5
Batch Plant		0.3
Water Management (Pumps)		0.8
Water Treatment Plant		0.01
Laboratory		0.3
Warehouse		0.005
Workshop		0.1
Fuel Farm		0.005
Explosive Magazine		0.005
Administrative Buildings		0.5
TOTAL (MW)		**15.3**
Power Factor		**0.8**
Diversity Factor		**90%**
Reserve		**20%**
Required Power (MVA)		**20.7**

Based on a future reserve of 20% and the available information, we estimate that the Lagoa Salgada Project will require around 21 MVA of installed power.

18.9.3 GRID CONNECTION

This section compares the different grid connection options for the Lagoa Salgada Project's estimated power requirement of 21 MVA. In general, there are two options: to connect the site to an existing substation, or to connect it to an existing overhead line. Substations owned by the national distribution grid and the national transport grid will be considered for this purpose, alongside the corresponding overhead lines.

All the preliminary studies and options presented in this chapter will have to be analysed and approved by E-REDES.

Option 1 – Connection to Existing Substation

The nearest substations are presented in Table 18-4 below.

Table 18-4 – Details of existing substations

Substation	Owner	Voltage level	Installed Power	Available Power (MV)	Distance
Vale do Gaio	E-REDES	60/30 kV	16 MVA	6,5 MVA	13.6 km
Alcácer do Sal	E-REDES	60/30 kV	20 MVA	0 MVA	18.6 km
Santiago do Cacém	E-REDES	60/30 kV	40 MVA	0 MVA	28.9 km
Comporta	E-REDES	60/30 kV	20 MVA	0 MVA	33.0 km
Ferreira do Alentejo	REN	400/150/60 kV	239 MVA (150/60 kV)	-	34.7 km
Sines	REN	400/150/60 kV	366 MVA (150/60 kV)	-	36.4 km

Connection to the medium-voltage level of the four E-REDES substations is not an option, as they do not have enough remaining capacity. Even though the Santiago do Cacém substation apparently has enough capacity remaining to satisfy the 21 MVA requirement, it is awaiting the allocation of another 12.43 MVA load which will exclude this possibility.

Alternatively, given that there is no available medium-voltage capacity at the existing E-REDES substations, the Project site could be connected to one of these substations at high-voltage level (60 kV).

The Alcácer do Sal substation, despite having enough space on the switchyard, is a less desirable solution due to its location in a city with a high population and housing density. Despite its better location, the Vale do Gaio substation lacks space for further expansion as it is already awaiting new connections from other projects. Connection to the Comporta substation at high-voltage level is a viable option from a technical point of view, but the cost may be high, given the distance of 33 km from the Lagoa Salgada Project.

Two other possible solutions would be connection to the Ferreira do Alentejo or Santiago do Cacém substations, also at the high-voltage level of 60 kV, which are located at 34.7 km and 28.9 km from the Project, respectively.

Lastly, connection to the Sines substation is also a possibility, but given that the Santiago do Cacém substation is closer, it has not been taken into consideration.

Option 2 – Connectoin to Overhead Lines

Given the 21 MVA required for the Lagoa Salgada Project, the grid connection should be done at the high-voltage level of 60 kV.

In the proximity of Lagoa Salgada project there is an existing 60kV overhead line that connects Vale do Gaio to Alcácer do Sal substations. The intersection point is approximately 11.6km away from the project site and a new double overhead line could be executed. This solution would also require the construction of a switching station to E-REDES in order to allow the opening of the existing overhead line, respecting the conditions of the DSO.

Figure 18-7 shows the relative positions of the connection point, the six substations and the existing line.

Figure 18-7 – The existing infrastructure and the connection point

18.9.4 INTERNAL DISTRIBUTION NETWORK

The electrical supply system of the Lagoa Salgada Project will start at the 60/15 kV step-down substation that will connect it to the national distribution grid at 60 kV level. This substation will be equipped with an AIS (air insulated substation) switchyard for the installation of the 60 kV apparatus and the 60/15 kV power transformer, as well as a building for installation of the 15 kV medium-voltage (MV) switchgear that will feed all the Project loads, the auxiliary services, and the protection, control and communication systems.

To reticulate energy to the Project, a 15 kV reticulation network will be required. To step down the 15 kV voltage to the consumer voltage levels, transformer stations and isolated or pole-mounted transformers will be distributed strategically throughout the Project site (on the surface and underground).

Generally, all the mine loads will be supplied at 400/230 V, but some types of major equipment such as mills and the main ventilation fans will be supplied with power at 6 kV, which will be reduced from the 15 kV voltage by means of transformers.

18.9.5 SOLAR PV PLANT – UPAC SYSTEM

A UPAC consists of an energy generation system, usually from renewable sources, that is used for self-consumption purposes. These installations have simple permitting processes compared to the production of energy intended for sale to the grid, which is more complex from a regulatory point of view.

As explained previously, the implementation of a UPAC system is a good strategy for a number of reasons:

- It reduces the mine's CO_2 emissions by producing renewable energy on site, which translates into a lower environmental impact, thereby also supporting the environmental study.
- It reduces the OPEX costs during the life cycle of the operation.
- In future, it will also be feasible to store the energy produced once BESSs (battery energy storage systems) have become more competitive.

The implementation of a UPAC will also bring some constraints:

- It will increase the Project's CAPEX;
- The installation of the energy production system will require additional space.

Given the location of Lagoa Salgada Project, a solar PV plant seems to be the best solution for the implementation of the UPAC system. Table 18-5 presents a preliminary list of the main characteristics of the proposed UPAC system.

Table 18-5 – Characteristics of the proposed Solar PV Plant

Characteristic	Description
Technology	Solar PV Plant
Peak Power	27 MWp
Nominal Power	21 MVA
Connection point	15kV at 60/15kV substation
Estimated required area	Between 15 and 20 ha
Estimated cost (CAPEX)	1.6 M€
Payback time (average)	Between 5 and 7 years

18.10 FUEL

The fuel supply station on the surface will be located in the middle of the industrial area close to the main facilities. It will supply all surface activities (light vehicles and heavy mining equipment) and will be equipped with a high-output pumping system.

The mining equipment that operates exclusively underground will be refuelled by means of underground tanker trucks, in accordance with Portuguese law. The fuel storage capacity will ensure the site's autonomous operation for a period of seven days.

19 MARKET STUDIES AND CONTRACTS

There are no offtake contracts in place for the zinc, lead and copper concentrates that will be produced at LS, which is reasonable given the Project's development stage. It is expected that 100% of the production will be absorbed by the market once production commences.

As Europe is expected to remain an important buyer of zinc, lead and copper concentrates, it is assumed that the sales profile will involve shipping the majority of concentrate production within Europe.

19.1 MARKETS

19.1.1 COPPER

Copper prices reached an all-time high during 2021, marking a 130% growth since March 2020.

The growth in the price of copper over the course of the past year was primarily driven by high demand from China, the top copper consumer, as well as growing optimism about the overall economic recovery in view of the COVID-19 vaccine rollout. The demand for copper is expected to rise further amid rising concerns about low copper inventories.

Copper is the most widely used metal in energy generation, energy transmission infrastructure and energy storage. It is the most frequently used metal after aluminium and steel in the construction, telecommunications, transportation and automobile manufacturing sectors. Factors such as rising demand from the renewable energy sector, the growing use of copper in smart home appliances, the expansion of the construction and electrical equipment manufacturing industries and the growth in the supply of scrap copper may shape the growth of the copper industry.

However, this growth would be hampered by a decline in copper ore grades and the challenges associated with greater amounts of mineral waste. Notable trends include an upsurge in copper used in passenger vehicles, accelerating copper demand in China and new copper mine projects in the pipeline.

19.1.2 ZINC

After a volatile 2019, when the zinc market was hard hit by the coronavirus, the second half of 2020 told a different story, with prices reaching record levels.

In terms of market fundamentals, COVID-19 caused huge disruptions in China's demand for refined zinc in 2020.

As economies started to reopen, a recovery in demand from the steel sector took place, with China seeing the most growth. However, the construction and steel activity in China is likely to slow down in the second half of 2021, which will weigh on global zinc demand.

Zinc demand is expected to continue to recover in 2021, driven by strong steel mill orders and infrastructure stimulus. Global demand will more than make up for the 2020 losses due to a strong rebound in China, although demand in the rest of the world is not expected to recover fully before 2022. Globally, zinc consumption is forecast to grow by 1.6% during the 2021-2040 period.

On the smelter front, smelters in China continue to be affected by structural problems, including a shortage of coal and low water levels in places like Yunnan.

A key factor to consider is that zinc treatment charges are now rising, which confirms that the market is becoming better supplied.

Long-term zinc consumption growth will be partly supported by government stimulus to facilitate the energy transition.

19.1.3 Lead

After a year of high volatility, investors' interest in lead is again rising, which may result in a high price for the metal in the coming 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.

In terms of supply, lead mine production will not fully rebound to pre-pandemic levels for a couple of years. 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.

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.

19.1.4 Silver

The factor that has the biggest impact on the silver price at any given time is not industrial or jewellery demand but investment demand.

Silver is used in a variety of industrial applications, while jewellery 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 President Biden's 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.

19.1.5 Gold

Historically, gold is a haven for investors during turbulent times. In 2020, the high level of uncertainty in the global economy due to the coronavirus outbreak fuelled demand for the yellow metal. In 2021, the gold price has gradually fallen as uncertainty has decreased, but volatility is still high.

The expectations for an economic recovery due to the rollout of the vaccinations against the coronavirus point to a decline in the gold price, but the high degree of volatility and uncertainty may put pressure on the price, as low-to-negative interest rates and loose monetary policies persist.

19.1.6 Tin

Tin has been one of the better performing commodities on the London Metals Exchange with robust long-term fundamentals.

In recent years, the tin market has been faced with a consistent supply deficit, which is forecast to continue into 2022 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, given that it is widely used in lead-acid batteries 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.

Ownership of the tin supply is fairly concentrated, as China remains the world's largest producer. 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 that will likely cause the price to rise.

19.2 PROJECTIONS

The metal prices, payable metal, treatment and refining charges, and freight costs were provided by Ascendant. Quadrante has reviewed these assumptions, which are aligned with its internal guidelines.

19.2.1 METAL PRICES

The metal prices assumed for the PEA are summarised in Table 22-6.

19.2.2 PAYABLE METAL

The payable metal assumed for the PEA are summarised in Table 22-5.

19.2.3 TREATMENT AND REFINING CHARGES

The TC/RCs assumed for the PEA are summarised in Table 22-3.

19.2.4 FREIGHT COSTS

The freight costs assumed for the PEA are summarised in Table 22-4.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

The environmental scope and approach used in the present document is based on the applicable Portuguese legislation in force. The following aspects have been analysed and will be described in this chapter:

- Environmental permitting of the Lagoa Salgada Project.
- Environmental approach and relevant potential areas of impact.
- Community engagement and status of local agreements.
- Mine closure.

20.2 ENVIRONMENTAL PERMITTING FRAMEWORK IN PORTUGAL

According to the Portuguese legislation in force, to be able to operate, industrial projects must obtain a unique environmental permit (Licenciamento Único Ambiental – LUA). As a result of the LUA process, a single environmental title (Título Único Ambiental – TUA) will be issued by the Portuguese Environmental Authority (Agência Portuguesa de Ambiente, I.P. – APA). The TUA is a document that brings together all the requirements for the construction, operation, monitoring and deactivation/closure of a specific project, both in terms of the environmental and the administrative authorisations required by all environmental regimes to which the project is subject.

The TUA will be issued upon the initial environmental approval of the Project.

Obtaining a TUA may involve different and complementary stages of approval as established in the LUA regime set out in Decree-Law n. 75/2015, of May 11th which defines the proceedings for obtaining a TUA under consideration of all applicable legal regimes.

The LUA regime includes, among others, the obligations under the following regimes:

- Legal Regime for Environmental Impact Assessment (RJAIA)
- 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)
- European Union Emissions Trading Scheme (CELE)
- General Solid Waste Management regime (SWM)
- Legal Regime for Mining Waste Management (RJGRM)
- Legal Regime for the Use of Water Resources (RJURH)

The Portuguese legal framework for Environmental Impact Assessment, relies on the “RJAIA – Regime Jurídico de Avaliação de Impacte Ambiental” established by Decree-Law n. 151-B/2013, of October 31st amended and republished in Decree-Law n. 152-B/2017, of December 11th.

Under RJAIA rules, the Lagoa Salgada Project, is subject to an Environmental Impact Assessment in line with Article 1st, n. 3 paragraph a) and paragraph b), subparagraph i) because it can be framed into item b) and e) of n.2 of Annex II and n.9 of Annex I of RJAIA, as follows:

- Anexo II, n.º 2, alínea b) "Extração subterrânea" (the underground mine itself);
- Anexo II, n.º 2, alínea e) "Instalações industriais de superfície para a extração e tratamento de hulha, petróleo, gás natural, minérios e xistos betuminosos (the surface support industrial infrastructures like mineral processing plant);
- Anexo I, n.º 9 "Instalações destinadas à incineração (D10), valorização energética (R1), tratamento físico-químico (D9) ou aterro de resíduos perigosos (D1)" (the tailings disposal facility).

Regarding the Lagoa Salgada Project, the first stage of the AIA process is the scope definition proposal (Proposta de Definição de Âmbito – PDA). This document defines the proposed scope of the environmental impact study, establishes the importance of each environmental factor, determining which of them are more relevant for the environmental assessment of the Project, and establishes how deeply each environmental component will be developed.

The approval of the PDA guarantees the commitment of the authorities to the scope of the environmental impact study (Estudo de Impacte Ambiental – EIA) and the level of analysis to be considered in the EIA for each environmental factor.

The next steps in the AIA process will be the following.

Next steps of AIA proceeding will be:

- **EIA -> DIA**
- **RECAPE -> DCAPE**
- **LA -> TUA**

These processes will evolve as follows:

- The EIA will assess the early stages of the Project (i.e., a pre-feasibility study, feasibility study or base project);
    - The EIA approval statement (Declaração de Impacte Ambiental – DIA) approves the location of the Project and its main characteristics and establishes some constraints that need to be considered in the detailed project design, while requiring a second environmental impact assessment, after the development of the detailed project design.
- The environmental conformity of the project to be executed (Relatório de Conformidade Ambiental do Projeto de Execução – RECAPE) will assess the Project at the detailed design stage.
    - The RECAPE approval statement (Declaração de Conformidade Ambiental do Projeto de Execução – DCAPE) allows the construction of the Project at the location evaluated and approved in the EIA; the DCAPE may establish some constraints for construction and/or the operation phases as well as a monitoring plan.

- The environmental license (Licença Ambiental – LA) that is required for projects under the regime of integrated prevention and pollution control of industrial emissions (Regime de Emissões Industriais – REI) established by Decree-Law no. 127/2013. As a result of the REI application, the applicant must fill in several fields in the SILiAmb platform (APA's electronic platform for environmental issues) to request an environmental license for the Project. The information required for filling out the platform fields is specified in Ordinance no. 399/2015.
    - Once the process has been approved by APA, an environmental permit will be issued, and its constraints will be registered in the Project's single environmental title (Título Único Ambiental – TUA). In this case, the LA cannot be issued without a DCAPE having previously been issued.

All constraints and obligations associated with the environmental permit of the Lagoa Salgada Project will be registered in its TUA, to be issued by APA, the Portuguese Environmental Authority

20.3 Permitting Background and Present Status of Lagoa Salgada Project

A PDA for the Lagoa Salgada Project has already been prepared and submitted to the authorities, and the APA issued a favourable statement on 28 June 2019, in AIA Proceeding no. 213. However, the validity of that statement expired in June 2021.

The Project under analysis refers to the mineral deposit of Lagoa Salgada, a volcanogenetic deposit of massive sulphides (VMS), with mineralisation of zinc, copper, lead, gold, silver and tin.

The Evaluation Committee issued an opinion on the submitted PDA, with some specific comments on the following aspects of the Project:

- The proposed area of the mining concession application, which is of significant size, i.e., greater than the combined size of the currently active local mining concessions – Aljustrel (Almina) and Neves-Corvo (Somincor – Lundin Mining).
- The characteristics of the area under evaluation.
- The lack of definition of the areas where the future infrastructure will be located – in particular those from which more severe environmental impacts are expected (for instance, the tailings disposal facility, the waste rock deposits and the mineral processing plants).
- The fact that there are still open options for additional mineral processing.

Due to this fact, Redcorp has decided not to ask for an extension of the PDA's validity. Instead, given that the Project's characteristics have changed, a new, updated PDA can now be prepared and submitted.

Nevertheless, the fact that the Lagoa Salgada Project already had an approved PDA in the past permits the conclusion that it complies with the minimum requirements for passing the first stage of the AIA process.

The approval of the PDA statement will be issued within 30 to 40 working days after submission (depending on whether there will be a public consultation or not). Once this period has elapsed without approval being granted, the applicant will obtain tacit approval for the scope of the Project (administrative silence).

20.4 Environmental Permitting – Next Steps

The next steps towards the environmental approval of the Lagoa Salgada Project will be the following:

- **PDA -> PDA approval**
- **EIA -> DIA**
- **RECAPE -> DCAPE**
- **LA -> TUA**

20.4.1 General Information

The Project information required for preparing the new PDA and EIA includes, among others:

- An updated topographic survey of the concession area and its immediate (100m) surroundings at a scale of 1:2000, including any paths, walls, poles, houses, rivers, and streams.
- The procedural background of the Lagoa Salgada Project.
- The mining annexes characteristics, including industrial and waste management surface infrastructures.
- Information on the site water balance, considering the water origins, the water supply infrastructure, residual water drainage, reuse, treatment and discharge, and the rainwater catching and diversion systems.
- Information on other site infrastructure such as that related to electricity, compressed air, fuel and lighting, among other information considered relevant for the development of the proposed works.
- The internal site and external access roads.
- The mining plan (and associate plans).
- The mine closure plan.

20.4.2 Mine Plan ("Plano de Lavra")

This Plan, to be submitted to the General Directorate of Energy and Geology ("DGEG – Direção-Geral de Energia e Geologia") for approval, should contemplate all activities to be executed inside the concession area and all mining annexes ("anexos mineiros"), whether they are inside or outside of the concession area and must be compliant with the Best Available Techniques (BAT), ensure adequate operational economy, and respect all the applicable safety and environmental protection measures.

According to the Decree-Law 30/2021 of May 7th that regulates the mining Law n. 54/2015 of June 22nd, among other detailed technical information on the resources, mine facilities, environmental constraints in the plan area, extraction methodologies, and equipment, the mining plan must incorporate the following plans:

- Waste Management Plan ("PGR – Plano de Gestão de Resíduos");
- Health and Safety Plan ("PSS – Plano de Segurança e Saúde");
- Environmental and Landscape Recovery Plan ("PARP – Plano Ambiental e de Recuperação Paisagística");
- Mine Closure Plan ("PEE – Plano de Encerramento da Exploração").

DGEG will approve the mining plan after consultation with the competent authorities. In the case of Lagoa Salgada project, the consultation will take place within the scope of the AIA proceeding.

20.4.3 Waste Management Plan

The Waste Management Plan must be developed accordingly to the established in Decree-Law n. 10/2010 of February 4th, that approves the Mining Waste Management Legal Regime, altered by Decree-Law 31/2013 of February 22nd.

20.4.4 Health and Safety Plan

The Health and Safety Plan must follow the Portuguese legislation in force regarding this typology of plans, namely:

- Decree-Law N. 324/95 of November 29th – minimum safety and health requirements to be applied in extractive industries either by open cut or underground drilling;
- Decree-Law N. 162/90 of May 22nd – "Regulamento Geral de Segurança e Higiene no Trabalho nas Minas e Pedreiras" (General Regulation of Safety and Health at Work in Mines and Quarries).

20.4.5 Environmental and Landscape Recovery Plan

The Environmental and Landscape Recovery Plan is a dynamic plan that will follow the evolution of the of the operation's activities and must be reviewed every five years.

The PARP should, preferably be implemented along the operation activities development, namely through the application of reposition measures since they are technically viable. A failure to comply with the plan constitutes a serious environmental infraction.

20.4.6 Mine Closure Plan

The Mine Closure Plan, a compulsory document for every mining site, must be developed in accordance with Decree-Law N. 30/2021 of May 7th which regulates the mining Law n. 54/2015 of June 22nd. The PEE establishes technical measures regarding the closure of the mine as well as mitigation measures to reduce the social, economic and environmental impacts of the end of operations on the mining site and its surroundings, and it must be reviewed every five years. Failure to comply with this plan constitutes a serious environmental infraction.

Where possible and technically viable, the PEE measures should be implemented in parallel with the operational activities.

Even after termination of the concession, the concession owner must be compliant with all requirements regarding environmental protection and landscape recovery.

20.5 RELEVANT POTENTIAL AREAS OF IMPACT

20.5.1 RELEVANT POTENTIAL AREAS OF IMPACT OF THE LAGOA SALGADA PROJECT AND SUSTAINABLE RESPONSES

Biodiversity Conservation Management

Even though every mining operation has an environmental impact on biodiversity during its construction, operation and closure, these impacts can be mitigated through the adoption of an adequate "mitigation hierarchy", which includes the following aspects:

1. Avoidance: measures taken to avoid creating any negative impact on biodiversity from the outset of a project – the Lagoa Salgada layout considers the land and its characteristics and takes advantage of the existing natural conditions (for instance, by locating the tailings facility in a dry, salty area with no vegetation, by avoiding interference with protected trees and valuable habitats, and by avoiding disturbance of the latter at critical times, such as the breeding season of migratory birds);
2. Minimisation: measures taken to reduce the duration, extent and intensity of negative impacts on biodiversity at every opportunity (by reducing the number of protected trees and/or the area of affected valuable habitats, by reducing noise and dust pollution to avoid impairing the behaviour of local species due to the presence of the mine and its infrastructure).
3. Rehabilitation and Restoration: measures taken to improve degraded or removed ecosystems that it was impossible to avoid being negatively impacted (through remediation measures that, if applicable will be set in the EIA so that the affected habitats and species can be restored, for instance, through soil and vegetation replacement to allow the local biodiversity and ecosystems to regenerate).
4. Offset: measures taken to compensate for any residual, adverse impacts that could not be completely avoided or minimised (through compensation measures that, if applicable, will be set out in the EIA to restore these degraded habitats).

To control the implementation of the "mitigation hierarchy", a biodiversity conservation plan will be developed to assess the mine's performance at all stages. As a result of the assessment of the Project area during the AIA process, this plan will establish site-specific conservation objectives for significant biodiversity aspects, to guarantee the conservation of biodiversity throughout the life of mine and to restore the local ecosystems once the mining activities have concluded.

The biodiversity conservation plan will adopt a regional approach to biodiversity conservation and will consider other potential measures foreseen by surrounding projects and/or by the municipalities or other interested parties.

Waste Management

The waste management will include the following:

- The management of mining waste according to the PGR referred to chapter 20.4.3, namely the management of the waste and tailings disposal facilities as mentioned in chapter 18.4 and 18.5;
- Other waste produced on site that does not result directly from mining operations and, is consequently subject to the General Solid Waste Management Regime (SWM) established under Decree-Law n. 102-D/2020 of December 10th.

From among all mining waste to be produced at Lagoa Salgada, tailings are the type of waste that most worries the local and regional communities because it may pose a danger if bad practices are followed and may also harm the environment if not properly managed.

The Project's tailings management system will be compliant with the best available technology (BAT) for tailings management to ensure the safety of both the surrounding communities and the environment. All legally applicable safeguards will be used, and a risk analysis will be performed to prevent risks and allow for the identification and implementation of appropriate safety measures.

Permanent monitoring of the tailings facility will be required, including legally required monitoring measures and analyses of the monitoring results, to allow for informed decision-making processes and the verification that all performance and risk management objectives have been met. Taking into account that the tailings disposal method will be dry stack disposal, the controls should be less complex compared to the controls needed for a paste-disposal or sub-aqueous tailings facility.

Each tailings facility is unique, reflecting site-specific environmental and physical characteristics that determine the most appropriate approach to performance and risk management for the facility in question. The Project's TSF should be conceived in a manner consistent with the need for responsible management, as defined by comprehensive assessments of the physical and chemical risks associated with the facility, with the aim of evaluating the potential health, safety, environmental, societal, business, economic and regulatory impacts, as well as the implementation of appropriate controls to effectively manage those risks.

Water Management

As mentioned in chapter 18.6, a water balance was prepared to understand the main origins and destination of fresh water, residual water, rainwater and runoff water flows and to support the decision makers in the establishment of sustainable and effective water management measures.

The Project's water management will consider the following aspects:

- Water supply – identification and quantification.
- Impacts of water abstraction/diversion on local water resources/users.
- Permits for the use of the public water domain.
- Water supply, storage and treatment (design and construction).
- Wastewater treatment and disposal.
- Site stormwater management.
- Acid rock drainage management.

At a later stage, the Project will establish the expected water balance, including identification of all inputs and outflows, an inventory of pond and interstitial water, and the principles for site water management under normal operating conditions as well as in the event of abnormal runoff or severe precipitation events.

Superficial and underground water resources may be harmed if a mine site's water management system is not effective and efficient because its water needs can potentially induce lowering of the surrounding groundwater levels. Similarly, the quality of surface water courses may be affected by the chemicals used in industrial ore processing plants; for these reasons, a water management system based on BAT and BAP is crucial to prevent environmental impacts on water resources.

The tailings management facilities and the associated water management systems are integral components of mining and ore processing operations and must be managed throughout their life cycle to ensure safe and responsible management. This includes the prevention of adverse impacts on human health and safety, the environment and the infrastructure.

The water management activities must be operated, maintained and monitored on a daily basis. This includes the collection of qualitative and quantitative data, data analysis and communication to inform decision makers whether the performance and risk management objectives are being met and the characteristics and availability of natural resources are being preserved.

Energy Use and Greenhouse Gas Emissions Management

The sustainable approach foreseen for the Lagoa Salgada Project implies, above all, a reduction of energy consumption to the indispensable minimum and the efficient use of the necessary energy. The feasibility of implementing measures to increase the energy efficiency should be examined. The potential measures include the following:

- The construction of a Photovoltaic Plant.

- On-demand ventilation of the underground workings.
- Usage of battery powered mining equipment.
- Usage of autonomous mining equipment.

Soil and Landscape

As a result of the typology of the Project, any impacts on the soil will be mainly due to the implementation and operation of the industrial surface infrastructure. Being an underground mine, the actual mining activities will not affect the land surface to a great extent, unlike an open-pit mine. Nevertheless, a plan for monitoring the soil before the start of construction (and throughout the mine's life cycle) should be developed.

Social Component

Regarding the social component, two scales of analysis will be considered:

- Surrounding area – Consists to a macro level of analysis, considering as object of study the territorial units at the level of the region, sub-region, municipalities, and parishes, where the project is located. This analysis aims to contextualize and frame the existing reality in the territorial units considered, so the required approach is centered on a set of dimensions that allow the perspective of the Project's interaction with the dynamics of occupation of the territory, seeking, simultaneously, to obtain a notion of the structure and specialization of economic activities and employment.
- Study area – The area in which some of the most relevant direct impacts of the project can essentially be felt, aiming as the ultimate objective, to identify the various functionalities existing in the area to find out if and how they will be disturbed.

20.6 COMMUNITY ENGAGEMENT AND THE STATUS OF LOCAL AGREEMENTS

As the Lagoa Salgada Project will be subject to the AIA process, the EIA assessment by the AIA authority (the APA in this case) will include a public consultation where the EIA documents (including the description of the Project elements) will be made available to the public. As part of this process, local/regional public meetings related to the disclosure and clarification of the Project will take place, promoted by APA, the national environmental authority, with the participation of the Project applicant and the environmental specialists responsible for completing the EIA.

During these public meetings, anyone can pose questions that will be then answered during the meeting and/or complemented in the public consultation report to be prepared by APA, which will be disclosed together with the DIA.

All relevant and applicable questions raised by the community will be assessed by APA and will be considered in establishing the measures that will form part of the DIA statement.

20.7 MINE CLOSURE

As mentioned earlier, according to the Decree-Law 30/2021 of May 7th that regulates the mining Law n. 54/2015 of June 22nd, all mining sites must have a Mine Closure Plan (PEE). The PEE will establish technical measures regarding the closure of the mine and mitigation measures to reduce social, economic, and environmental impacts of the end of operation in the mining site and its surroundings and must be reviewed every five years. A failure to comply with the plan constitutes a serious environmental infraction.

The closure plan includes the commitment to restore the affected lands to their natural state upon completion of the mining activities. For that purpose, the Lagoa Salgada Project should provide a guarantee of the financial resources necessary to close and reclaim the mine site before construction begins. This financial guarantee is reflected in the concession contract between Redcorp and DGEG.

Where possible and technically viable, the applicable PEE measures should be implemented in parallel with the development of the operational activities. Even after expiry of the concession, the concessionaire must continue to be compliant with all requirements regarding environmental protection and landscape recovery.

21 CAPITAL AND OPERATING COSTS

21.1 INTRODUCTION

The capital and operating cost estimates have been prepared by Quadrante. All costs are exclusive of Portuguese value-added tax (IVA – "Imposto sobre o Valor Acrescentado").

The accuracy of the capital and operating costs estimates is expected to be in the range of ±35% which is consistent with industry standards for a PEA. All costs are expressed in 2021 US$ and uses an exchange rate EUR:US$ of 1.2 where applicable.

21.2 CAPITAL COSTS

The capital costs for the Lagoa Salgada Project are estimated at 230 MUS$ and are summarised in Table 21-1.

Table 21-1 – Estimated capital costs

	Unit	TOTAL	
Production (feed to mill)			
Waste Development	kUS$	$	80 332
Mobile Equipment	kUS$	$	31 550
Processing Plant	kUS$	$	66 950
Pastefill Plant	kUS$	$	10 650
Dry Stack Facility	kUS$	$	5 700
Support Infrastructure	kUS$	$	12 220
Other Capex / Studies	kUS$	$	4 750
Closure Costs	kUS$	$	6 000
Sub-Total Capex	**kUS$**	**$**	**218 153**
Contingency	kUS$	$	12 028
Total Capex	**kUS$**	**$**	**230 180**

The initial capital costs include all costs incurred during the construction period (1.9 years). This initial capital costs have been estimated at 109.6 MUS$. The breakdown of total capital cost during the LOM is presented in Table 21-2.

Table 21-2 – Breakdown of capital costs

Capex			
Initial Capex	kUS$	$	109 570
Sustaining Capex	kUS$	$	102 583
Closure Costs	kUS$	$	6 000
Contingency	kUS$	$	12 028
Total Capex	**kUS$**	**$**	**230 180**

21.2.1 WASTE DEVELOPMENT

The waste development costs include the following three components:

- Initial boxcut excavation and support;
- Horizontal capital development;
- Vertical capital development.

The boxcut excavation volume has been estimated to be around 260 000 m^3. The unit cost of excavation and support of the boxcut has been estimated to be 3.85 US$/$m^3$. The estimated total cost for the boxcut construction is 1.0 MUS$.

Both the horizontal and the vertical capital development meters are outputs of the mining plan exercise. During the LOM, a total of 25 491 m of horizontal development, and 1 780 m of vertical development are planned. The unit costs have been estimated by benchmarking active contracts with local operations and applied to these meters. The benchmarked unit costs are 3 000 US$ for horizontal development, and 1 606 US$/m for vertical development.

The total estimated cost for waste-related capital development during the LOM is 80.3 MUS$.

21.2.2 MOBILE EQUIPMENT

A list of the mobile equipment considered for the operations, as well as the purchase unit cost is shown in Table 21-3.

Table 21-3 – List of mobile equipment for operation

Equipment	Unit cost (kUS$)	# Units in Full Production
Charging Platform	$ 350	1
Underground Loaders	$ 750	4
Scalers	$ 175	1
Production Jumbos	$1 000	2
Haul Trucks	$ 800	8
Cable Bolters	$ 1 000	1
Auxiliary Platforms	$ 200	2
Auxiliary Underground Loaders	$ 450	1
Remote Control Rockbreaker	$ 450	1
Front-end-loaders	$ 650	2
Light Vehicles	$ 45	15

The implementation of a fleet replacement plan has been considered for the period between Year 6 and Year 10.

It has been assumed that the horizontal development (capital and operational) will be executed by a mining contractor. As a result, the following equipment has not been considered in the estimate of the owner's capital cost:

- Development Jumbos.
- Charging Platforms for development.
- Scalers except small maintenance activities.
- Underground Loaders for development.
- Support Bolters.
- Underground Mixer Trucks.
- Shotcrete Robots.

The acquisition and depreciation of this equipment are included in the contractor's unit costs.

The total estimated capital costs for mobile equipment during the LOM is 31.6 MUS$.

21.2.3 PROCESSING PLANT

As discussed previously, the metallurgical testwork needs to continue during future stages of activity. This will allow for a detailed definition of the flowsheet and, consequently, a detailed definition of the mechanical, electrical and instrumentation equipment needed for the processing of the Lagoa Salgada ores.

The capital costs have been estimated based on the processing plants of similar operations in the IPB. These benchmarks have been scaled to the estimated production rate of the Lagoa Salgada Project.

The total capital cost estimate for the processing plant during the LOM is 67.0 MUS$.

21.2.4 PASTEFILL PLANT

The cost of building a paste-fill system with a production capacity rate that is compatible with the production plan has been estimated. The following aspects have been considered:

- Mechanical equipment costs.
- Electrical and instrumentation equipment costs.
- Construction costs.
- Piping and installation costs of the reticulation system.
- Engineering projects.
- QA/QC.
- Spare equipment.

The mechanical equipment includes a paste-fill pump, assuming that part of the paste-fill deposition will need to be pumped to the stopes. The proportion of paste fill that will need to be pumped will need to be confirmed once the final location of the backfill plant has been selected.

The total pastefill plant construction costs during the LOM are estimated at 10.7 MUS$. This estimation is in line with two other paste-fill systems built for IPB operations with similar production rates.

21.2.5 Tailings Storage Facility

As explained in Chapter 18.5, the tailings storage facility of the Lagoa Salgada Project, will be a dry stack disposal facility. Unit costs have been developed for the following aspects:

- Site preparation.
- Earthworks with compaction.
- Perimetral water collection.
- Impermeabilisation with HDPE geomembrane (1.5 mm).
- Installation of the geomembrane.
- Engineering projects.
- QA/QC.
- Geotechnical testwork.

The total capital costs for TSF construction during the LOM amount to 5.7 MUS$. This includes the initial construction cost and future expansions.

21.2.6 Support Infrastructure

The support infrastructure includes all infrastructure described in Chapter 18, with some additions, such as the batch plant and main silo. The TSF is not included, as this area has already been described above. The costs for this infrastructure have been based on Quadrante's extensive experience in the Portuguese construction and infrastructure industry, which has facilitated the development of an extensive unit cost database.

The capital costs for the support infrastructure are summarised in Table 21-4.

Table 21-4 – Capital cost of the support infrastructure

Area	Capital Cost (kUS$)	Comments
Roads and accesses	$ 470	Internal roads
Stockpiles	$ 520	ROM pad
Water management	$ 1 650	Includes treatment and supply
Buildings and other infrastructure	$ 2 950	
Power and electrical	$ 5 030	

Area	Capital Cost (kUS$)	Comments
Mine main ventilation	$ 1 600	2 main fans

The total estimated capital costs for the support infrastructure during the LOM among to 12.2 MUS$.

21.2.7 Other Capex/Studies

An allowance of 4.8 MUS$ has been considered for other smaller capital projects and studies that may be required until the construction of the Project has been completed.

21.2.8 Closure Costs

The closure costs of the Lagoa Salgada Project are estimated at 6.0 MUS$. This represents around 5% of the initial capital costs of the Project.

Due to the dry stack deposition method, the closure costs of the TSF are lower than if other deposition method were used.

21.2.9 Contingency Costs

A contingency equivalent to 10% of the total capital costs incurred in the first two years has been considered, amounting to a total of 12.0 MUS$.

21.3 Operating Costs

21.3.1 Introduction

The operating costs have been estimated using external databases, refined with benchmark costs from other operations in the IPB. These costs have been scaled to the estimated production rates and the labour costs in Portugal.

The estimate includes the underground mining, processing and G&A operating costs. It excludes costs associated with escalation beyond 2021, currency fluctuations, offsite costs, interest charges and taxes. No contingency has been included in the operating costs.

The operating costs during the LOM are summarised in Table 21-5 and described in the next chapters.

Table 21-5 – Breakdown of operating costs

Opex			
Mining Costs	kUS$	$	498 805
Processing Costs	kUS$	$	414 286
G&A Costs	kUS$	$	91 245
Total Opex	**kUS$**	**$**	**1 004 336**

21.3.2 Mining Costs

The mining costs estimate assumes that the operational development will be executed by a mining contractor. All other costs are based on owner mining.

The mine operating costs include the following:

- Stope preparation.
- Mining cycle (drilling, charging, mucking, scaling, and support).
- Backfill.
- Trucking to surface.
- Mining equipment maintenance.
- Ancillary services (water, air, dewatering).
- Power.
- Labour (direct, supervision, and technical services).

The mineral resources are planned to be mined using two different mining methods: the C&F method for the majority of the oxide ore and LHOS for the remaining ore. The estimated unit costs for each of these mining methods reflect the considerable difference in productivity between them.

- C&F – 33.68 US$/t
- LHOS – 18.53 US$/t

The total mining costs during the LOM are estimated to be 498.8 MUS$ which corresponds to a mining unit cost of 19.13 US$/t.

21.3.3 Processing Costs

The processing costs have been estimated using benchmark exercises based on operations that process similar ores from the IPB. These costs have been adapted to the specific characteristics of the Lagoa Salgada Project that differentiate it from the benchmarked operations, i.e., the cyanide leach circuits and tin flotation.

For the Lagoa Salgada Project, an operating processing cost estimate of 15.89 US$/t has been used. The total processing costs for the LOM are estimated to be 414.3 MUS$.

21.3.4 General and Administration Costs

A benchmark exercise with other operations with similar size in the IPB has been performed. An average price of 3.5 US$/t has been considered for the Project which corresponds to total G&A costs of 91.2 MUS$ during the LOM.

22 ECONOMIC ANALYSIS

22.1 Cautionary Statements

Certain information and statements contained in this section and in the Report are “forward looking” in nature. Forward-looking statements include, but are not limited to:

- Statements with respect to the economic and study parameters of the Project.
- Mineral Resource estimates.
- The cost and timing of any development of the Project.
- The proposed mine plan and mining methods.
- Dilution and extraction recoveries.
- Processing method and rates and production rates.
- Projected metallurgical recovery rates.
- Infrastructure requirements.
- Capital, operating and sustaining cost estimates.
- The projected life of mine and other expected attributes of the Project.
- The net present value (NPV), the internal rate of return (IRR) and the payback period of capital.
- Capital requirements.
- Future metal prices.
- The timing of the environmental assessment process.
- Changes to the Project configuration that may be requested as a result of stakeholder or government input to the environmental assessment process.
- Government regulations and permitting timelines.
- Estimates of reclamation obligations.
- Requirements for additional capital.
- Environmental risks.
- General business and economic conditions.

The production schedules and the annualised financial analysis of the cash flow are presented with the projected years shown. The years shown in these tables are indicative only. If additional mining, technical and engineering studies are conducted, these may alter the Project assumptions discussed in this Report and may result in changes to the timelines presented.

The mine plan is partly based on inferred mineral resources that are considered too speculative geologically for the application of the economic considerations that would enable them to be categorised as mineral reserves, and there is no certainty that the PEA based on these mineral resources will be realised. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

22.2 Methodology Used

An engineering financial model has been developed to estimate the annual pre-tax and post-tax cash flows and sensitivities of the Project, based on an 8% discount rate. It should be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operation of the mine; as such, the after-tax results are only approximations. Sensitivity analysis was performed to assess the impact of variations in metal prices and operating costs. The capital and operating cost estimates have been developed specifically for this Project and are summarised in Section 21 of this Report.

All monetary amounts are presented in constant 2021 US$. For discounting purposes, cash flows are assumed to occur at the end of each period. Revenue is recognised at the time of production.

22.3 FINANCIAL MODEL PARAMETERS

22.3.1 MINERAL RESOURCE AND PRODUCTION SCHEDULE

The production schedule used in the financial model is the PEA Plan presented in Chapter 16. As explained, this plan was based on the latest Mineral Resource update for the Lagoa Salgada Project.

22.3.2 METALLURGICAL RECOVERIES

Based on the results/conclusions from the two phases of testwork undertaken at Grinding Solutions (2019 & 2021), and the author's previous experience while working at the nearby Neves-Corvo Mine (a similar deposit to Lagoa Salgada), the following recovery assumptions have been adopted.

Metallurgical recoveries and concentrate grades and moisture content are summarised in Table 22-1 and Table 22-2 respectively.

Table 22-1 – Metallurgical recoveries assumed for financial model

Metallurgy	
Oxide	
Copper Recovery	0%
Zinc Recovery	0%
Lead Recovery	65%
Silver Recovery	66%
Gold Recovery	86%
Tin Recovery	40%
Massive Sulphide	
Copper Recovery	25%
Zinc Recovery	80%
Lead Recovery	65%
Silver Recovery (in Lead Concentrate)	35%
Silver Recovery (in Zinc Concentrate)	20%
Silver Recovery (Leach)	20%
Gold Recovery (in Lead Concentrate)	10%
Gold Recovery (Leach)	65%
Tin Recovery	30%
Stockwork	
Copper Recovery	80%
Zinc Recovery	80%
Lead Recovery	75%
Silver Recovery (in Lead Concentrate)	40%
Silver Recovery (in Zinc Concentrate)	20%
Silver Recovery (Leach)	20%
Gold Recovery (in Lead Concentrate)	10%
Gold Recovery (Leach)	65%
Tin Recovery	0%

Table 22-2 – Concentrate grades and moisture content

Concentrate		
Cu Concentrate Grades		
Oxides	% Cu	25%
Massive Sulphide	% Cu	25%
Stockwork	% Cu	25%
Zn Concentrate Grades		
Oxides	% Zn	48%
Massive Sulphide	% Zn	48%
Stockwork	% Zn	48%
Pb Concentrate Grades		
Oxides	% Pb	45%
Massive Sulphide	% Pb	45%
Stockwork	% Pb	45%
Sn Concentrate Grades		
Oxides	% Sn	30%
Massive Sulphide	% Sn	30%
Stockwork	% Sn	30%
Copper Concentrate Moisture Content		9%
Zinc Concentrate Moisture Content		9%
Lead Concentrate Moisture Content		9%
Tin Concentrate Moisture Content		9%

22.3.3 SMELTING AND REFINING TERMS

The smelting and refining terms estimated for the concentrate produced during operations are summarised in Table 22-3. The freight costs incurred for the concentrate shipment are presented in Table 22-4.

Table 22-3 – TC/RC assumed for financial model

Treatment and Refining Charges			
Copper - TC	US$/dmt conc	$	60,00
Copper - RC	US$/lb	$	0,06
Zinc - TC	US$/dmt conc	$	175,00
Lead - TC	US$/dmt conc	$	150,00
Silver - RC	US$/oz	$	1,00
Gold - RC	US$/oz	$	5,00
Tin - TC	US$/dmt conc	$	450,00

Table 22-4 – Freight costs assumed for financial model

Freight Costs			
Cu Concentrate - Oxides	US$/dmt conc	$	60,00
Zn Concentrate - Oxides	US$/dmt conc	$	60,00
Pb Concentrate - Oxides	US$/dmt conc	$	60,00
Sn Concentrate - Oxides	US$/dmt conc	$	70,00
Cu Concentrate - MS	US$/dmt conc	$	60,00
Zn Concentrate - MS	US$/dmt conc	$	60,00
Pb Concentrate - MS	US$/dmt conc	$	60,00
Sn Concentrate - MS	US$/dmt conc	$	70,00
Cu Concentrate - Stockwork	US$/dmt conc	$	60,00
Zn Concentrate - Stockwork	US$/dmt conc	$	60,00
Pb Concentrate - Stockwork	US$/dmt conc	$	60,00
Sn Concentrate - Stockwork	US$/dmt conc	$	70,00

The payability factors that have been applied are summarised in Table 22-5.

Table 22-5 – Payables assumed in the financial model

Payables	
Oxide Production	
Copper	0%
Zinc	0%
Lead	95%
Silver	99%
Gold	99%
Tin	95%
Massive Sulphide Production	
Copper	25%
Zinc	85%
Lead	95%
Silver (in Lead Concentrate)	95%
Silver (in Zinc Concentrate)	70%
Silver (Leach)	99%
Gold (in Lead Concentrate)	95%
Gold (Leach)	99%
Tin	95%
Stockwork Production	
Copper	95%
Zinc	85%
Lead	95%
Silver (in Lead Concentrate)	95%
Silver (in Zinc Concentrate)	70%
Silver (Leach)	99%
Gold (in Lead Concentrate)	95%
Gold (Leach)	99%

22.3.4 Metal Prices

The metal prices used in the financial model are summarised in Table 22-6 below.

Table 22-6 – Metal prices assumed in the financial model

Metal Prices		
Copper price	US$/lb	$ 3,25
Zinc price	US$/lb	$ 1,20
Lead price	US$/lb	$ 1,05
Silver price	US$/oz	$ 20,00
Gold price	US$/oz	$ 1 650,00
Tin price	US$/lb	$ 12,00

22.3.5 Capital Costs

The capital cost assumptions are those presented in Chapter 21. A construction period of 1.9 years has been considered, with Years -2 and -1 earmarked for the construction period and Year 1 as the first full year of production. The capital costs are presented without inflation or other increments.

22.3.6 Operating Costs

The operating cost assumptions are those presented in Chapter 21. For the purpose of the PEA, it has been assumed that development (CAPEX and OPEX) will be done by a mining contractor, while stoping and trucking will be carried out by the owner. The operating costs are presented without inflation or other increments.

22.3.7 Royalties and Taxes

Royalties are estimated to be the highest value of the following:

- 10% of the net profit.
- 2.5% of the NSR.

The royalties that will be paid to the Portuguese government are calculated on a yearly basis.

Companies in Portugal are subject to three main taxes:

- Income tax – A fixed rate of 21% is applied to the taxable income.
- Local tax (Derrama) – This tax depends on the municipality where the company operate. The Lagoa Salgada Project is located in the municipality of Grândola, which charges a fixed rate of 1.4% on the taxable income.
- State surtax (Derrama Estadual) – This tax varies for different intervals of the taxable income (yearly) as follows:
    - The taxable income portion up to 1.5M€ (1.8MUS$) are not taxed.
    - The taxable income portion between 1.5M€ and 7.5M€ (9MUS$) is taxed 3%.
    - The taxable income portion between 7.5M€ and 35M€ (42MUS$) is taxed 5%.
    - The taxable income portion above 35M€ is taxed 9%.

A tax benefit for companies that execute investments in the Portuguese productive sector has been considered in the financial model. The eligibility of the Lagoa Salgada Project for this benefit has been confirmed by a PWC study to which the author was given access. This benefit has a maximum value of 14% of the initial investment and is discounted from the yearly income tax (up to this maximum).

The royalties and taxes are summarized in Table 22-7.

Table 22-7 – Royalties and taxes

Royalties	
Net Profit Royalty Rate	10,0%
NSR Royalty Rate	2,5%
Taxes	
Income Tax	21,0%
Local Tax (Derrama) - Grândola	1,4%
State Surtax (Derrama Estadual) NP <= 1.5M€	0,0%
State Surtax (Derrama Estadual) NP <= 7.5M€	3,0%
State Surtax (Derrama Estadual) NP <= 35M€	5,0%
State Surtax (Derrama Estadual) NP > 35M€	9,0%

22.3.8 Closure Costs and Salvage Value

A provision of 6MUS$, equivalent to ~2,5% of the total capital cost, has been included to account for closure costs.

No salvage value has been considered.

22.3.9 Financing

This PEA is based on 100% equity financing.

22.3.10 Inflation

No escalation or inflation has been applied. All amounts are given in real (constant) terms.

22.4 Economic Analysis

The economic analysis that has been performed assumes an 8% discount rate. The main pre-tax economic indicators are the following:

- NPV(8%) – 342 MUS$
- IRR – 68 %
- Payback – 1.3 years

The post-tax economic indicators, are as follows:

- NPV(8%) – 247 MUS$
- IRR – 55 %
- Payback – 1.5 years

The estimated post-tax cashflow of the Project is shown in Figure 22-1.

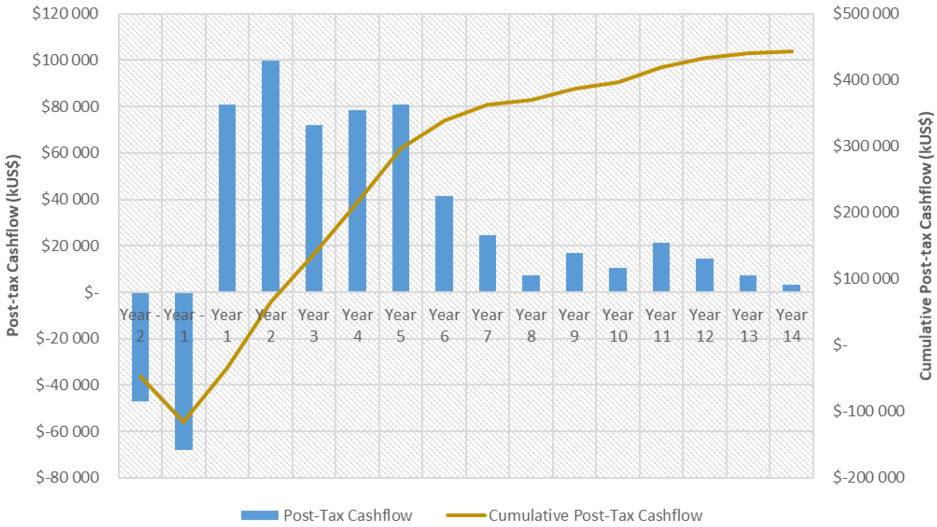


Figure 22-1 – Post-tax cashflow for the Lagoa Salgada project

A summary of the economic indicators of the Project is included in Table 22-8 through to Table 22-13 below.

Table 22-8 – General and production plan indicators

Parameter	Unit	LOM Total / Avg.
General		
Cu price	US$/lb	$ 3,25
Zn price	US$/lb	$ 1,20
Pb price	US$/lb	$ 1,05
Ag price	US$/oz	$ 20,00
Au price	US$/oz	$ 1 650,00
Sn price	US$/lb	$ 12,00
Exchange Rate (EUR:USD)	-	$ 1,20
Mine Life	Years	16
Total Mill Feed Tonnes - Oxides	kt	1 101
Total Mill Feed Tonnes - Massive Sulphides	kt	7 838
Total Mill Feed Tonnes - Stockwork	kt	17 131
Total Mill Feed Tonnes - Total Ore	kt	26 070
Total Waste Tonnes Mined	kt	7 342

Table 22-9 – Metal production indicators

Parameter	Unit	LOM Total / Avg.
Production Summary		
Cu Payable	klb	90 747
Zn Payable	klb	556 383
Pb Payable	klb	459 513
Ag Payable (in Pb Concentrate)	koz	8 490
Ag Payable (in Zn Concentrate)	koz	3 425
Ag Payable (in Leach)	koz	5 746
Ag Payable (Total)	koz	17 661
Au Payable (in Pb Concentrate)	koz	18
Au Payable (in Zn Concentrate)	koz	-
Au Payable (in Leach)	koz	149
Au Payable (Total)	koz	167
Sn Payable	klb	8 938
Zn Payable Equivalent	**Mlb**	**1 818**
Zn Payable Equivalent	**kt**	**825**

Table 22-10 – Unit operating costs indicators

Parameter	Unit	LOM Total / Avg.	
Operating Costs			
Mining	US$/t	$	19,1
Processing	US$/t	$	15,9
G&A	US$/t	$	3,5
Total Operating Costs	**US$/t**	**$**	**38,5**
Treatment & Refining Charges	US$/t	$	8,6
Royalties	US$/t	$	1,8
Total Cash Costs	**US$/t**	**$**	**48,9**
Total Cash Costs	**US$/lb Zn eq.**	**$**	**0,70**
Sustaining Capital	US$/t	$	3,9
All-in Sustaining Costs (AISC)	**US$/t**	**$**	**52,8**
All-in Sustaining Costs (AISC)	**US$/lb Zn eq.**	**$**	**0,76**

Table 22-11 – Capital cost indicators

Parameter	Unit	LOM Total / Avg.
Capital Costs		
Initial Capital Costs	kUS$	$ 109 570
Sustaining Capital Costs	kUS$	$ 102 583
Closure Costs	kUS$	$ 6 000
Contingency	kUS$	$ 12 028

Table 22-12 – Pre-tax and post-tax indicators

Parameter	Unit	LOM Total / Avg.
Financials		
Pre-Tax NPV (8%)	MUS$	$ 341,6
Pre-Tax IRR	%	68,2%
Pre-Tax Payback	Years	1,3
Post-Tax NPV (8%)	**MUS$**	**$ 246,7**
Post-Tax IRR	**%**	**54,9%**
Post-Tax Payback	**Years**	**1,5**

Table 22-13 – Summary of the annualised economic indicators

	Unit	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	TOTAL
Production (feed to mill)																		
Oxide	kt	-	4	8	8	24	3	12	10	77	84	199	23	199	199	174	77	1 101
Massive Sulphide	kt	-	266	1 684	1 834	1 206	1 008	1 295	417	123	1	1	1	1	1	0	-	7 838
Stockwork	kt	-	7	157	156	758	966	687	1 573	1 799	1 915	1 800	1 953	1 799	1 718	1 333	509	17 131
Total Ore Processed	**kt**	**-**	**277**	**1 850**	**1 999**	**1 989**	**1 977**	**1 994**	**2 000**	**1 999**	**2 000**	**2 000**	**1 977**	**1 999**	**1 918**	**1 507**	**586**	26 070
Metal Recovered																		
Copper	kt	-	0	1	2	3	3	3	4	5	5	4	5	4	4	3	1	48
Zinc	kt	-	5	38	39	31	29	30	19	16	15	14	16	15	15	12	4	297
Lead	kt	-	4	23	24	19	23	26	15	13	10	14	9	14	12	11	3	219
Silver	koz	-	278	2 234	2 766	2 158	2 621	2 117	1 768	1 169	834	781	713	780	687	519	210	19 633
Gold	koz	-	4	25	26	19	19	22	11	8	6	8	4	8	6	2	2	170
Tin	kt	-	0	1	1	0	0	0	0	0	0	0	0	0	0	0	0	4
Concentrate Production																		
Cu Concentrate	kdmt	-	1	5	7	11	13	11	17	20	20	18	19	17	16	13	5	193
Zn Concentrate	kdmt	-	10	80	82	64	60	63	39	33	31	30	33	31	31	25	9	619
Pb Concentrate	kdmt	-	9	50	53	43	50	57	34	28	23	32	20	31	27	24	6	488
Sb Concentrate	kdmt	-	0	2	2	2	1	1	1	1	0	1	0	1	1	0	0	14
Total Concentrate Shipped	**kdmt**	**-**	**20**	**137**	**144**	**120**	**124**	**133**	**91**	**81**	**74**	**80**	**73**	**80**	**74**	**62**	**20**	**1 314**
Net Smelter Return																		
Revenue	kUS$	$ -	$ 29 740	$ 209 659	$ 228 579	$ 186 219	$ 197 040	$ 199 678	$ 141 755	$ 120 530	$ 106 835	$ 113 722	$ 96 643	$ 115 828	$ 103 630	$ 78 987	$ 29 490	$ 1 958 335
Commercialization Costs	kUS$	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Concentrate Freight Costs	kUS$	$ -	$ 1 185	$ 8 257	$ 8 691	$ 7 205	$ 7 478	$ 7 995	$ 5 490	$ 4 879	$ 4 450	$ 4 786	$ 4 352	$ 4 805	$ 4 470	$ 3 710	$ 1 219	$ 78 972
NSR	**kUS$**	**$ -**	**$ 28 555**	**$ 201 401**	**$ 219 888**	**$ 179 014**	**$ 189 562**	**$ 191 683**	**$ 136 265**	**$ 115 651**	**$ 102 385**	**$ 108 937**	**$ 92 291**	**$ 111 023**	**$ 99 160**	**$ 75 277**	**$ 28 271**	**$ 1 879 363**
Royalties																		
Royalties	kUS$	$ -	$ 714	$ 5 035	$ 5 497	$ 4 475	$ 4 739	$ 4 792	$ 3 407	$ 2 891	$ 2 560	$ 2 723	$ 2 307	$ 2 776	$ 2 479	$ 1 882	$ 707	$ 46 984
Opex																		
Mining Costs	kUS$	$ -	$ 5 130	$ 34 272	$ 37 039	$ 36 848	$ 36 636	$ 36 946	$ 37 056	$ 38 222	$ 38 349	$ 40 089	$ 36 991	$ 40 072	$ 38 567	$ 30 563	$ 12 025	$ 498 805
Processing Costs	kUS$	$ -	$ 4 400	$ 29 392	$ 31 765	$ 31 601	$ 31 419	$ 31 684	$ 31 779	$ 31 767	$ 31 783	$ 31 781	$ 31 412	$ 31 767	$ 30 477	$ 23 950	$ 9 310	$ 414 286
G&A Costs	kUS$	$ -	$ 969	$ 6 473	$ 6 996	$ 6 960	$ 6 920	$ 6 978	$ 6 999	$ 6 997	$ 7 000	$ 7 000	$ 6 918	$ 6 997	$ 6 712	$ 5 275	$ 2 050	$ 91 245
Total Opex	**kUS$**	**$ -**	**$ 10 499**	**$ 70 138**	**$ 75 800**	**$ 75 408**	**$ 74 975**	**$ 75 608**	**$ 75 834**	**$ 76 986**	**$ 77 131**	**$ 78 870**	**$ 75 322**	**$ 78 835**	**$ 75 756**	**$ 59 788**	**$ 23 385**	**$ 1 004 336**
EBITDA	**kUS$**	**$ -**	**$ 17 342**	**$ 126 229**	**$ 138 591**	**$ 99 130**	**$ 109 848**	**$ 111 283**	**$ 57 025**	**$ 35 773**	**$ 22 694**	**$ 27 343**	**$ 14 662**	**$ 29 412**	**$ 20 925**	**$ 13 606**	**$ 4 179**	**$ 828 043**
Capex																		
Initial Capex	kUS$	$ 42 934	$ 66 636	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 109 570
Sustaining Capex	kUS$	$ -	$ 10 707	$ 26 256	$ 12 113	$ 7 657	$ 9 897	$ 6 242	$ 3 460	$ 3 950	$ 11 410	$ 4 950	$ 2 235	$ 750	$ 1 098	$ 1 107	$ 750	$ 102 583
Closure Costs	kUS$	$ -	$ -	$ -	$ -	$ 1 000	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1 000	$ 1 000	$ 3 000	$ -	$ 6 000
Contingency	kUS$	$ 4 293	$ 7 734	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 12 028
Total Capex	**kUS$**	**$ 47 227**	**$ 85 078**	**$ 26 256**	**$ 12 113**	**$ 8 657**	**$ 9 897**	**$ 6 242**	**$ 3 460**	**$ 3 950**	**$ 11 410**	**$ 4 950**	**$ 2 235**	**$ 1 750**	**$ 2 098**	**$ 4 107**	**$ 750**	**$ 230 180**
Depreciation																		
	kUS$	$ -	$ 9 445	$ 26 461	$ 31 712	$ 34 135	$ 35 866	$ 28 400	$ 12 633	$ 8 074	$ 6 441	$ 6 992	$ 6 002	$ 5 201	$ 4 859	$ 4 489	$ 3 028	$ 223 738
Taxable Income																		
Applicable Tax Losses	kUS$	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Loss Carryforward	kUS$	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Taxable Income	kUS$	$ -	$ 7 896	$ 99 768	$ 106 879	$ 64 996	$ 73 982	$ 82 883	$ 44 392	$ 27 700	$ 16 253	$ 20 352	$ 8 659	$ 24 211	$ 16 066	$ 9 118	$ 1 151	$ 604 304
Taxes																		
Total Taxes	kUS$	$ -	$ 293	$ 18 937	$ 26 756	$ 18 495	$ 21 316	$ 24 111	$ 12 025	$ 7 356	$ 4 219	$ 5 342	$ 2 145	$ 6 400	$ 4 168	$ 2 264	$ 258	$ 154 087
FCF																		
Pre-Tax Cashflow	kUS$	$ -47 227	$ -67 736	$ 99 972	$ 126 478	$ 90 474	$ 99 951	$ 105 041	$ 53 565	$ 31 823	$ 11 284	$ 22 393	$ 12 427	$ 27 662	$ 18 826	$ 9 499	$ 3 429	$ 597 862
Post-Tax Cashflow	kUS$	$ -47 227	$ -68 029	$ 81 035	$ 99 721	$ 71 979	$ 78 635	$ 80 930	$ 41 540	$ 24 468	$ 7 065	$ 17 051	$ 10 281	$ 21 262	$ 14 658	$ 7 235	$ 3 171	$ 443 775

22.5 Sensitivity Analysis

A sensitivity analysis has been performed to consider variations in metal prices and operating costs on the post-tax NPV (8%), IRR, and payback. The result of this analysis is presented in Figure 22-2, Figure 22-3, and Figure 22-4.

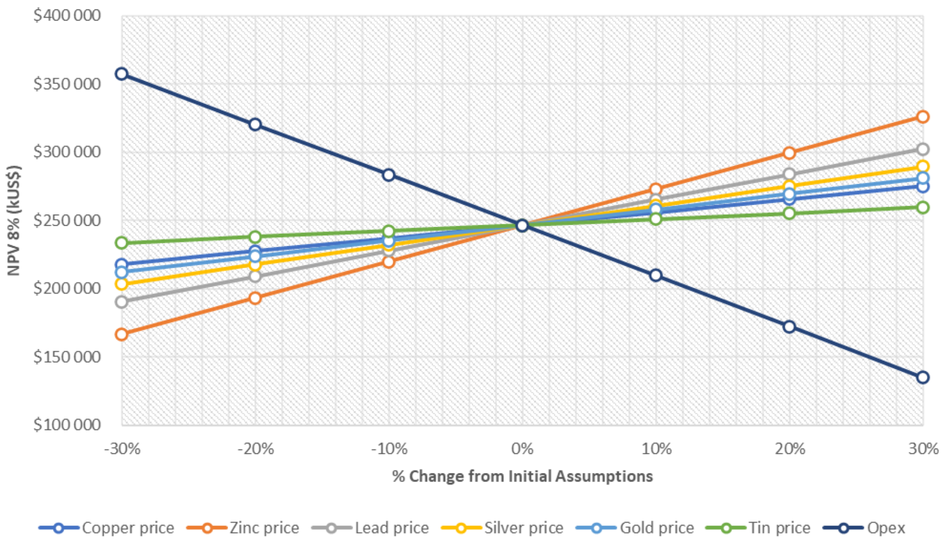


Figure 22-2 – Sensitivity analysis of post-tax NPV (8%)

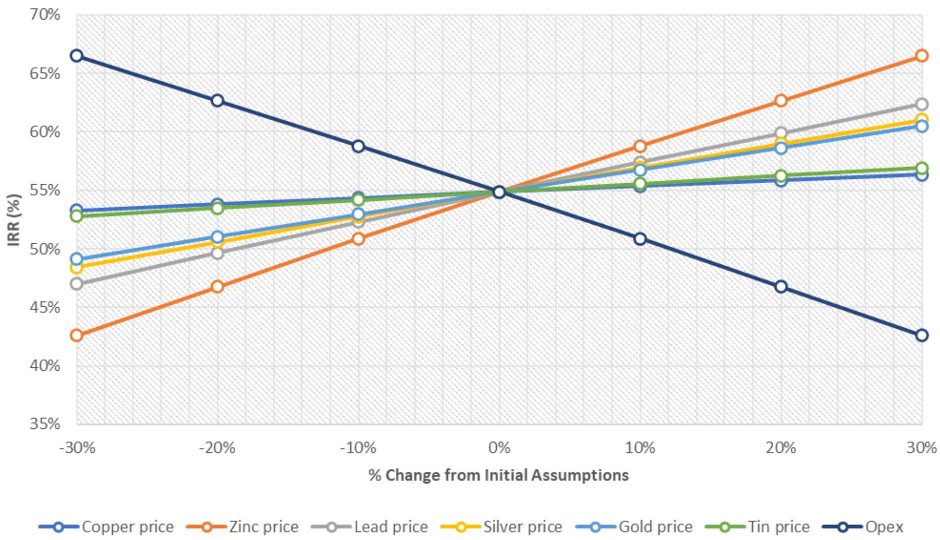


Figure 22-3 – Sensitivity analysis of post-tax IRR

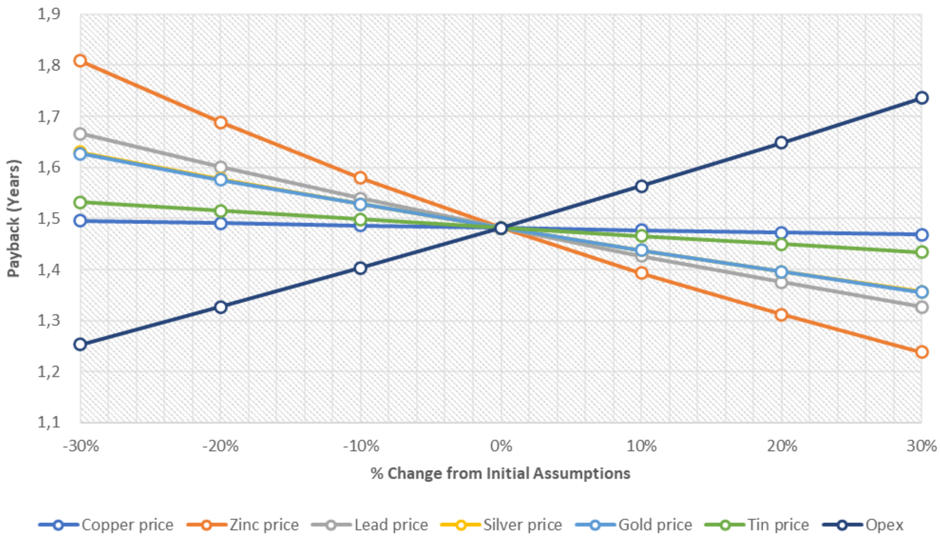


Figure 22-4 – Sensitivity analysis of post-tax payback

From an NPV perspective, it is clear that the Project is most sensitive to changes in operating costs, followed by fluctuations in the zinc price. It is less sensitive to fluctuations in the price of lead, silver, gold and copper, and it is least sensitive to the tin price.

The economic indicators of the Lagoa Salgada Project are robust across all the analyses that have been performed. This is confirmed when calculating the NPV for several discount rates as shown in Table 22-14.

Table 22-14 – Sensitivity analysis on NPV for different discount rates

Discount Rate	NPV (kUS$)
2%	$ 381 048
4%	$ 328 541
6%	$ 284 273
8%	$ 246 702
10%	$ 214 621
12%	$ 187 072
14%	$ 163 293

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.

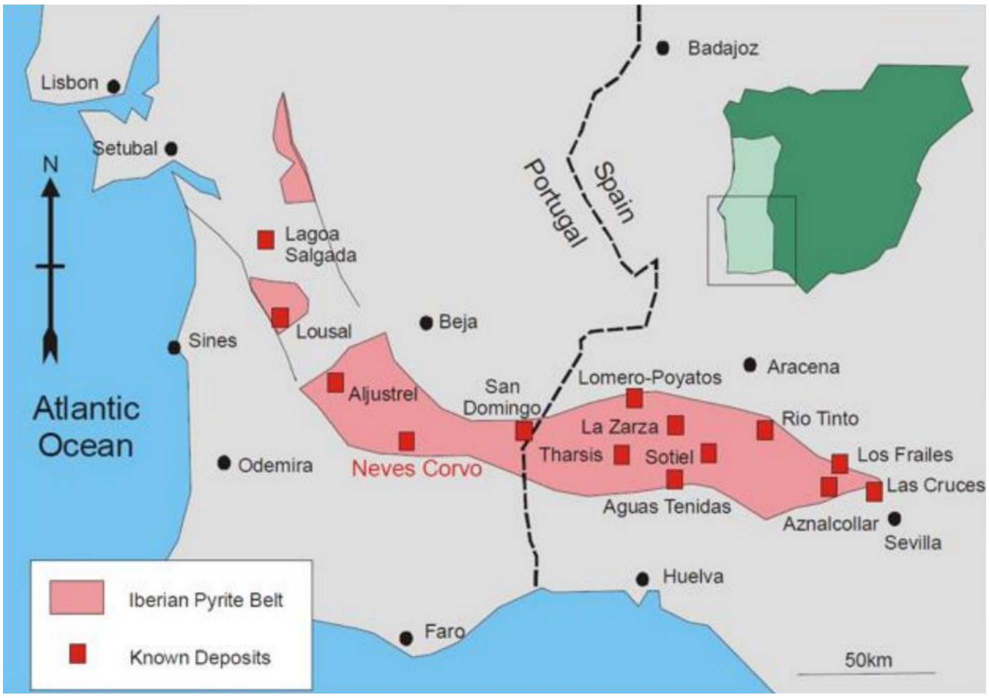


Figure 23-1 – Current and past mining operations in the IPB (source: lundinmining, 2019)

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 mining tailings.

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

24 OTHER RELEVANT DATA AND INFORMATION

To the authors’ knowledge, there is no additional information or data beyond that presented in this Report that would affect its completeness, clarity and correctness.

25 INTERPRETATION AND CONCLUSIONS

25.1 PROPERTY DESCRIPTION AND OWNERSHIP

The company currently has the concession contracted to perform all scheduled production in current LOM. The land needed to build the Project infrastructure is currently under rental contract and with option to buy if required by the company.

25.2 GEOLOGY/MINERALISATION AND RESOURCES

The subdivision of the LS property into the Venda Nova North and South deposits is arbitrary, being based on the existing drill pattern. With further concerted systematic drilling, the two deposits are likely to coalesce into a single zinc-lead-copper VMS system, manifesting / displaying its macro-genetic features from secondary GO to primary MS and ending with peripheral primary / secondary stringer / fissure type mineralization in the waning phases of volcanic activity. This interpretation is backed by geophysics which shows that all two deposits lie on a continuous coincidental IP chargeability anomaly with an estimated geological strike length of 1.7 km in an SSE to NNW direction from the South deposit to beyond the North deposit and terminating against the Alpine fault. Thus, the LS property's two deposits are components of one polymetallic, volcanogenic massive sulphide (VMS) deposit.

Potential to expand the resource is realistic as the LS property mineralization remains open in all directions although with a stronger signature on the eastern side of the currently drilled / known linear trend of about 1.7 km. The geometry of the MS domain of the North deposit appears to suggest that the main vent of the volcanic activity that gave rise to the LS property deposit may be located at the north-western end where the plunge swings westwards. However, this remains speculative until proven by additional drilling.

Currently, the contribution to the LS mineral resources is split as follow: North deposit = 48%, South deposit = 52%. However, both deposits have the potential to delineate more resources with additional drilling. The stringer / fissure type mineralization of the South deposit appears to be more amenable to metallurgical processing than the massive mineralization of the North deposit and future priority drilling for additional resources will depend on progress in metallurgical testwork.

The MS intersections observed in drillholes LS 23 and LS-ST 12 on the eastern side of the South deposit suggest the possibility of another volcanic vent.

25.3 MINING AND MINERAL RESERVE ESTIMATE

No mineral reserves have been estimated for the Project. The available data indicate that mining is possible with a sustained production rate of 2.0 Mtpa, using mainly a sublevel stoping method, with cut&fill for the oxide ore. The 2.0 Mtpa is reached from year 2 to year 11. There is a ramp-up of less than two years, and a ramp-down of 3 years.

25.4 METALLURGY AND PROCESSING

The preliminary oxidation assessment suggested that the samples were oxidized and not truly representative of fresh mineralization that would feed a mineral processing facility. Therefore, the flotation recoveries and concentrate grades produced from the 2021 GSL programme were not optimal. 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. Grinding Solutions Ltd. is 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.

Gravity separation tests on the MS sample gave no selectivity, recoveries were proportional to mass pull.

Intensive cyanide leach tests on flotation pyrite concentrates gave metal extractions after 48 hours of between 9% and 25% for gold, between 40% and 65% for silver, between 62% to 82% for copper, and between 13% and 24% for zinc.

The flotation locked cycle test (LCT) completed using the MS composite sample resulted in Pb concentrate grade of 22% with 43% recovery and a Zn concentrate grade of 35% at 66% recovery. Separate Cu and Pb concentrates were not recovered for the MS tests.

The LCT completed for the SW sample resulted in Cu concentrate grade of 25% at 69% recovery, Pb concentrate grade of 28% at 16% recovery and Zn grade of 44% at 54% recovery.

A LCT using a blend of MS and SW resulted in Cu concentrate grade of 24% at 53% recovery, Pb concentrate grade 12% at 5% recovery Zn concentrate grade 28% at 64% recovery.

The process plant is designed for a total of 2Mtpa or 250tph throughput to produce two concentrates. Based on this, the plant LOM is expected to be 14 years. The crushing plant is designed at 65% availability, while the main plant will operate at 92% availability. Based on head grades of 0.31% for Cu, 1.44% for Zn, and 1.22% for Pb he designed commodity recoveries are 80% for the Cu SW and 25% for the Cu MS, 80% for the Zn, and 75% for the Pb respectively.

It is to be noted that due to fluctuations of the head grades of the different minerals from the initial years of operation through the LOM, the plant has been designed for the range of head grades of each mineral, rather than on the average grades presented in Table 17-1. Same logic has been used as part of the equipment selection and sizing.

The conceptual methodology process flowsheet envisioned for this deposit consists of primary, secondary, and tertiary crushing; coarse material stockpiling; primary ball mill grinding; secondary ball mill grinding and cyclone classification; Lead and Zinc flotation, concentrates thickening and filtration; final concentrates stockpiling; and tailings disposal.

25.5 INFRASTRUCTURE

The estimated support infrastructure for the Lagoa Salgada Project does not require complex engineering. Other than mine and processing plant infrastructure, all engineering can be performed by local companies that may already develop projects in the area.

The planned dry stack tailings disposal method that has been estimated for this project has also reduced considerably the complexity of the engineering and permitting processes.

25.6 PROJECTED ECONOMIC INDICATORS

The valuation results of the Lagoa Salgada Project indicate that it has a post-tax Net Present Value (NPV) of approximately 247 MUS$. Based on an 8% discount rate. The operation is expected to have negative cashflows during the two construction years (years -2 and -1), with payback expected by year 2.

Life of mine is projected to end of year 14 resulting in a total production of 297 thousand tonnes of zinc, 219 thousand tonnes of lead, 19.6 million ounces of silver, 48 thousand tonnes of copper, 170 thousand ounces of gold, and 4 thousand tonnes of tin.

The estimated All-in Sustaining Costs (AISC), including sustaining capital, is 0.76 US$/lb Zn eq.

26 RECOMMENDATIONS

26.1 INTRODUCTION

The recommended work is divided into two phases.

The first phase consists of the following:

- Exploration and infill drilling.
- Metallurgical testwork.
- Geotechnical testwork.
- Mine studies.
- A hydrological study.
- A hydrogeological study.
- Water treatment testwork.
- Detailed OPEX and CAPEX detailed studies.
- Engineering studies.

The second phase consists of:

- Completion of a pre-feasibility / feasibility study.

26.2 GEOLOGY AND RESOURCES

A systematic program to upgrade and expand the resources is recommended as follows:

- The immediate priority should be drilling the gap separating the North and South deposits.
- The second priority should be infill drilling to upgrade the Inferred resources for the North and South deposits. This is necessary as Inferred resources cannot be included in advanced economic studies, i.e., prefeasibility/feasibility studies.
- The third priority should be drilling directed at the north-west end of the North deposit to define the geometry / extent of the plunge and at the same time increase the resource.
- Models of the deposits should continue to be refined / updated as more information becomes available.
- Geophysical investigations of the LS deposits should be sustained and continued to cover the area to the north of the North deposit, targeting the area immediately beyond the major east-west Alpine fault.

26.3 PROCESSING

Metallurgical recoveries and concentrate grades assumed for the PEA should take into account metallurgical results presented in Section 13.0. Although the samples used for the GSL 2021 testwork programme were oxidized, these flotation test results, together with other historical work, suggest that the poly-metallic mineralization Lagoa Salgada is complex, and it will be challenging to produce high quality concentrate products with high recoveries.

Although detailed final concentrate characterization was not included in the most recent program of metallurgical testwork, historical results suggest that the concentrates produced may include deleterious elements (to be confirmed with additional test work). Depending on the additional test work results, penalties may need to be applied in future techno-economic studies.

A new program of testwork is recommended using fresh drill core samples that represent the different lithologies found within the mineral resources. Grinding Solutions Ltd. is 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.

Further studies will be required to confirm the plant design and production capacity. Test programs at selected equipment suppliers locations are recommended to facilitate and support the equipment selection and sizing.

26.4 MINING

A geotechnical drilling programme is recommended to support the design of the mine at the pre-feasibility/feasibility stage. These drill holes will be completed as part of the infill drill programme.

Detailed studies should be undertaken to review the capital and operating costs included in the NSR cut-off calculations. The mine design should be reviewed using the new cut-off values.

Further hydrogeological and hydrological investigations are required to define the mining sequence and to anticipate potential issues. This information will also be required for the design of the underground water management system.

Trade-off studies for the operation of the mine should be developed to examine the impact of a contractor vs. owner-operated equipment fleet, a leased vs. purchased equipment fleet, the various equipment class sizes, and the utilisation of electrically driven mine equipment over diesel driven units.

It is recommended that a dedicated trade-off study is performed regarding the hoisting method for the Lagoa Salgada Project. This trade-off should evaluate several methods, such as, conveyor belts from underground, the construction of a production shaft to hoist ore/waste, and the option selected at this PEA level (underground mine trucks).

26.5 INFRASTRUCTURE

A geotechnical field investigation should be completed to confirm the locations of the Project facilities and the infrastructure design. Once geotechnical data are available, trade-off studies should be conducted to determine the optimal location of the infrastructure installations.

26.6 TSF

The location and design of the TSF should be further evaluated. The use of the Lagoa Salgada area for dry stack tailings should be subject to a thorough risk assessment. Other areas are available tailings disposal, namely that of the South Deposit where the installation of the surface infrastructure is currently planned.

Other recommended activities specific to the TSF include the following:

- Geotechnical site investigations at the preferred site to investigate the foundation conditions. For PEA purposes, it is assumed that the area is appropriate for a TSF.
- In-situ permeability tests of the overburden soil and bedrock.
- Tailings assessment to examine the geotechnical properties in order to understand the deposition characteristics.
- Further evaluation of the water management aspects that may impact the TSF management.

26.7 Economics

The economic model should be updated with the detailed estimations from all areas of the project. The economic indicators at the PFS/FS level will be more accurate due to all the additional work that will be performed on the next level of study.

If new information is available regarding the marketing of the products, this should also be updated in the economic model and, if needed, a restructure of this model should be performed to reflect the most updated information.

26.8 Pre-Feasibility / Feasibility Study

In a second phase, the recommendation is to advance to a pre-feasibility (PFS)/feasibility study. This should be initiated once the maximum resource thresholds/classifications have been achieved and the requisite metallurgical and geotechnical data are available.

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