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Technical Report

Technical Report and PEA for the Lagoa Salgada Property, Setúbal District, Portugal

Ascendant Resources Inc.

In accordance with the requirements of National Instrument 43-101 "Standards of Disclosure for Mineral Projects" of the Canadian Securities Administrators

Qualified Persons:
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AMC Project 719037
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1 Summary

1.1 Terms of reference

AMC Mining Consultants (Canada) Ltd. (AMC) has 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 produced by Micon International Limited (Micon).

1.2 Property description and location

The Project is located approximately 80 kilometres (km) south-east of Lisbon, Portugal's capital; and approximately 120 km by road. It is located approximately 50 km south-east of Setúbal, the regional administration 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 within a single exploration permit covering 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) which is 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 (DGEG), or General Directorate for Energy and Geology in 2017 as a renewal and an application for a mining concession was made in April 2019 for which 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% held subsidiary of TH Crestgate GmbH (TH Crestgate), a Swiss investment company and a 25% held subsidiary of Ascendant, a Canadian company listed on the Toronto Stock Exchange.

1.4 Geology and mineralization

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 mineralization and about 250 Mt of stockwork mineralization (Oliveira et al. 2005, 2006; Tornos 2006). The IPB hosts more than 90 MS deposits. Ten deposits are in Portugal where currently only Neves Corvo and Aljustrel are being exploited.

1.4.2 Property geology and mineralization

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 LS Project. 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 8.0 km long and over 1 km wide.

The LS Project currently has three known deposits: the North, Central, 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 North deposit is further offset by an east-west-trending Alpine-age fault in the north, with a 50 m downthrow of the northern block (Far North Target), but whose horizontal amount and sense of displacement is unknown (Matos et al. 2000).

The mineralization comprises MS and semi-massive sulphide lenses and sulphide veins and veinlets and is mainly hosted by a thick (up to 250 m) and strongly chloritized quartz-phyric rhyodacite unit. Currently, the mineralization is known to extend continuously over a cumulative strike length of 1.7 km in a north-northwesterly to south-southeasterly direction.

1.4.3 Deposit types

The LS Project deposit is a polymetallic, volcanogenic massive sulphide (VMS) deposit. VMS deposits are a type of metal sulphide deposit that 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 as a result 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 observed on the LS Project suggests that hydrothermal activity was prolonged and that additional lenses associated with separate alteration zones may exist.

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, have successfully demonstrated that mineralization on the LS Project 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.

1.4.5 Mineral Resource estimate

The Mineral Resources estimated for the LS Project are summarized in Table 1.1. All resource parameters are disclosed in Section 14 of this Report. The effective date of the estimate is 5 September 2019.

Table 1.1 Summary of the Mineral Resource estimate for the LS Project, dated 5 September 2019

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 (I)	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
					Average grade							**Contained metal**						
Deposit	**Category**	**Min zones**	**Cut-off CuEq (%)**	**Tonnes (kt)**	**Cu (%)**	**Zn (%)**	**Pb (%)**	**Sn (%)**	**Ag (g/t)**	**Au (g/t)**	**CuEq (%)**		**Cu (kt)**	**Zn (kt)**	**Pb (kt)**	**Sn (kt)**	**Ag (koz)**	**Au (koz)**
Central	Inferred	Str	0.9	1,707	0.15	0.16	0.06	0	12	2.22	1.66		2.5	2.7	1.0	-	635.2	121.9
South	Measured	Str/Fr																
	Indicated	Str/Fr	0.9	2,473	0.47	1.53	0.83	0.00	19	0.06	1.54		11.5	37.9	20.6	0.0	1,484.7	4.7
	M & I	**Str/Fr**	**0.9**	**2,473**	**0.47**	**1.53**	**0.83**	**0.00**	**19**	**0.06**	**1.54**		**11.5**	**37.9**	**20.6**	**0.0**	**1,484.7**	**4.7**
	Inferred	Str/Fr	0.9	6,085	0.40	1.34	0.80	0.00	17	0.05	1.37		24.6	81.6	48.7	0.0	3,285.2	10.0

Notes:

- The Mineral Resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definitions and Standards (2014).
- Mineralized Zones: GO, MS, Str=Stringer, Str/Fr=Stockwork.
- 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))/67.24.
- 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.
- 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.
- Densities: GO=3.12, MS=4.76, Str=2.88, Str/Fr=2.88.

1.4.6 Mineral Resource comparison

The significant growth (70% increase in Measured and Indicated Resources) at the LS Project is attributed to the success of Ascendant / Redcorp's recent infill and step-out drilling directed mainly at the North deposit. The richest part of the LS Project coincides with the Measured Resource area close to the northern extremity of the North deposit.

Currently, the greatest contribution to the Mineral Resources is from the North deposit. However, all deposits have the potential to delineate more resources with additional drilling. The stringer / fissure type mineralization of the South and Central deposits appears to be more amenable to metallurgical processing than the massive mineralization of the North deposit and future priority drilling will depend on progress in metallurgical testwork.

Micon's Qualified Person (QP) considers that the resource estimate for the LS Project has been reasonably prepared and conforms to the CIM Definition Standards (2014) for reporting Mineral Resources.

1.5 Mining

The mine design is based on a single decline access from surface at a 12.6% gradient. Decline access is via a 30 m deep boxcut. Stopes are accessed from level access drives in the north and the south of the deposit. Interlevel spacing varies between 24 m and 35 m. All mineralized material and waste development are mined with a 4.5 m by 4.5 m end profile.

The deposit is planned to be mined using transverse sub-level open stoping (SLOS) with paste fill at a production rate of approximately 1 million tonnes per annum (Mtpa). Crosscuts will access the deposit with drives developed laterally across the mineralization. Drives in mineralization will be placed 12.5 m apart along strike, with stopes approximately 25 m to 35 m high, 12.5 m wide and 25 m in length. Stope heights in the Gossan (GO) tend to be generally less, approximately 20 m high. A slot will be cut at the end of the mineralization and consecutive rings blasted in a retreating fashion over the full stope length back to the crosscut. Uphole drill rings from the existing drives in the MS will be drilled to extract the mineralization from the overlying GO deposit. Ore and waste will be hauled to surface using 30 tonne (t) trucks.

Unplanned dilution due to the extraction of the stope was assumed to be 8% for the GO zone and 5% for the MS zone. Mining recovery of 90% was assumed for the GO and 93% for the MS.

Approximately 55% of tailings (up to 540,000 tonnes per annum (tpa) at a dry bulk density of 1.4) will be placed underground as paste fill to meet an annual demand of 400,000 cubic metres (m^3) of void and the remaining tailings placed in the dry stack Tailings Storage Facility (TSF). The paste plant will have an annual utilization of just below 50%. Paste fill will be transported underground using a combination of pumping and gravity via boreholes and high-pressure pipelines to the stopes.

1.6 Infrastructure

There is currently access to the mine via paved roads to Cilha do Pascoal, followed by 4 km of gravel roads to the mine site. Some improvement to the roads to the mine site may be required to accommodate heavy construction traffic.

The site will require an office, changeroom, shop and warehouse as well as storage for fuel, laydown areas, site fencing, and security building. An allowance for a total of 2,600 metres squared (m^2) of building space has been included in the PEA.

Total power requirement for the mine and mill is estimated to be 15 megawatts (MW). There is ample opportunity to connect to the national grid with both 400 kilovolts (kV) and 30 kV transmission lines operating within 7 km of the project site. However, for this study, a conservative allowance has been made to run a 30 kV, 20 mega volt amperes (MVA) transmission line from the existing sub-station at Grândola.

Tailings and waste rock will be disposed of through the use of a dry-stack facility. Total tailings for life-of-mine (LOM) are estimated at 7.5 Mt with a further 0.7 Mt of waste rock. Approximately 55% of tailings will be disposed of in the mined-out stopes via the paste fill system. The remaining 4.1 Mt of tailings and waste must be accommodated in the dry stack facility. The base of the facility will be lined, and a low perimeter berm and ditch will capture any precipitation run off during the LOM. Run off will be collected in a settling pond for use by the mine as service water.

Regional precipitation averages 700 millimetres (mm) per year, and it is anticipated that the site will have a net neutral water balance once the initial dewatering of the mine is complete. All water from the mill will be reused.

Total annual water gain through precipitation and mine dewatering is estimated to be approximately 325,000 m^3. Loss to the tailings is estimated at 250,000 m^3 per year with evaporation accounting for the remaining loss. A complete climate and water balance study is required.

It is anticipated that any make-up water that may be required will be obtained via local wells on site. Should this not be adequate, water can be obtained from the Sado River approximately 5 km from the project site.

A settling pond with capacity of 100,000 m^3 will be established to hold precipitation run-off during the rainy season as well as mine and mill water discharge.

AMC has assumed ground water inflow of 5 litres per second (L/s). Water will be discharged via a staged pump system with pumps located on 3 levels staging to surface.

The ventilation system is designed as a 'pull' system with primary exhaust fans located on surface at the top of each primary exhaust raise. Fresh air is delivered into the mine from the decline and fresh air raise. Internal return air raises carried with the production ramps connect to a dedicated return airway and the exhaust raises to surface. Based upon benchmark data for similar mines, the required airflow would be in the range of 340 m^3/s to 380 m^3/s.

Escape ways will be raised between sublevels and equipped with manways to allow emergency egress from level to level. The main decline will be the escape route to surface with an equipped ventilation raise providing second egress.

1.7 Mineral processing

The metallurgical work completed to date is of a reconnaissance nature and no firm conclusions can be drawn therefrom. Detailed testwork is in progress.

Scoping level Metallurgical Studies completed by Grinding Solutions Ltd. (GSL) indicated that a conventional polymetallic processing flowsheet will recover copper, lead and zinc sulphides. Limited test work has been undertaken on oxidized material. The final tailings will be cyanide leached to recover additional gold and silver.

The metal recoveries and concentrate grades need to be improved / optimized to produce saleable products. The metal recoveries were estimated based on limited test work and experience working with similar deposits. Additional test work should be undertaken to confirm the assumed recoveries as well as production of saleable products.

1.8 Mine development and production schedule

Initial mine development is planned over a two-year period. Included in the mine development is the box cut and initial capital access decline development as well as infrastructure, processing plant, backfill plant and dry stacked tailings facility. In Table 1.2 and Table 1.3, development is shown as commencing in year 1 and production ramp up occurring in Year 2.

Waste development over the LOM consists of 5,573 m of lateral and ramp development as well as 721 m of vertical development. Approximately 2,766 m of development will be in mineralized material.

Development quantities are reported directly from the mine design and are summarized in Table 1.2.

Table 1.2 Development schedule

Description	Unit	Total	Year					
			1	2	3	4	5	6
Box cut	t	466,897	466,897					
Horizontal development waste	m	5,573	720	1,720	1,252	649	701	531
Vertical development waste	m	721		436	189			96
Development in mineralization	m	2,766		858	664	576	436	232
Total development		**9,060**	**720**	**3,014**	**2,105**	**1,225**	**1,137**	**859**

The production rate is estimated based on Taylors rule of thumb and maximum vertical extraction per year. The selected production rate of 1.0 Mtpa is well supported by the production schedule. All ore and waste will be hauled to surface using 30 t trucks. Haul routes are short, and the majority of the resource is located over three production levels.

A summary of the production schedule for the LOM plan is shown in Table 1.3. Approximately 7.25 Mt of mineralized material is planned to be mined at an average production rate of 1 Mtpa with an average zinc equivalent (ZnEq) of 7.9%.

Table 1.3 LOM production schedule

Description	Unit	Year								
		2	3	4	5	6	7	8	9	Total
Mineralization	kt	579	1,032	1,016	1,033	1,001	1,001	996	595	7,251
Cu	%	0.50	0.26	0.35	0.39	0.50	0.36	0.26	0.10	0.34
Zn	%	2.06	3.85	2.99	3.09	2.54	2.72	0.83	0.40	2.44
Pb	%	2.82	2.55	2.70	2.78	3.31	2.57	3.29	2.72	2.85
Sn	%	0.15	0.13	0.13	0.13	0.14	0.14	0.27	0.26	0.16
Au	g/t	0.71	0.71	0.80	0.68	0.82	0.76	0.72	0.79	0.75
Ag	g/t	104.1	65.6	77.9	62.7	84.2	75.9	51.4	38.9	69.8
ZnEq	%	8.39	8.56	8.29	8.25	8.86	7.92	6.86	5.26	7.91

The PEA is preliminary in nature, it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

1.8.1 Mobile equipment

Table 1.4 indicates the estimated equipment required to support a 1.0 Mtpa mine plan with associated mineralized material and waste development. Equipment number estimates allow for some redundancy to accommodate break downs and ramp congestion.

Table 1.4 Estimated equipment requirements

Primary equipment required	Number
LHD, matched to trucks	4
Trucks (30 t)	4
Development drill	3
Longhole drill rig	2
Rock support drill rig	2
Cable bolter	1
Scalers	3
Rock breaker	1
Support trucks, high-lifts	4
Total fleet	**24**

1.8.2 Manpower

The total underground manpower envisaged for the 1.0 Mtpa mine is shown in Table 1.5. The mine will be operated on a three shift per day, eight hours (hrs) per shift basis, Monday to Friday. Only essential services will be run over the weekends.

Table 1.5 Underground manpower

Description	Number
Total operators	72
Maintenance team	30
Services team	10
Mining manager	1
Engineering manager	1
Technical services	8
Supervisors	12
Total manpower (Mine)	**134**

1.9 Environmental

In terms of Environmental Licensing, an Environmental Scoping Proposal (PDA) has already been prepared and submitted to the Environmental authorities for the start of the Environmental Impact Assessment (EIA), in accordance with Portuguese regulations, which has already been approved by the Environmental Impact Assessment Authority (APA).

1.10 Capital and operating costs

The total capital cost estimate is $182.9 million (M), as summarized in Table 1.6. Initial capital expenditure (Capex) of $162.7M with $20.2M in sustaining capital and four-year payback period.

Table 1.6 Annual capital costs summary

Capital costs	Unit	Year									LOM total
		1	2	3	4	5	6	7	8	9	
Mining fleet	$(000)	14,200									14,200
Ramp box cut	$(000)	1,000									1,000
Waste development	$(000)	2,520	7,328	4,706	2,273	2,452	2,180				21,460
Maintenance	$(000)	1,000	4,500	250	250	250	250	250			6,750
Backfill plant	$(000)		12,000								12,000
Process plant	$(000)	25,000	35,000	100	100	100	100	100	100		60,600
TSF	$(000)	4,000	8,000	50	50	50	50	50	50		12,300
Infrastructure and services	$(000)	7,893	2,727	50	50	189	1,067	50	50		12,076
Contingency and closure	$(000)		37,551							5,000	42,551
Grand total	**$(000)**	**55,613**	**107,106**	**5,156**	**2,723**	**3,041**	**3,647**	**450**	**200**	**5,000**	**182,937**

The LOM unit operating costs are estimated to be $49.43/t milled as summarized in Table 1.7. Costs are based on benchmark data from other local operations and local labour costs. Mining is estimated to be $16.84/t milled, Processing $29.17/t milled and General and Administration (G&A) $3.42/t milled.

Table 1.7 Unit operating costs summary

Operating costs	$/t milled
Mining	16.84
Processing	29.17
G&A	3.42
Grand total	**49.43**

1.11 Economic analysis

The project shows robust economic results with a pre-tax net present value (NPV) at 8% of $137M and an IRR of 37%, and an after tax NPV at 8% of $103M and IRR of 31%. Key aspects of the project economics are provided in Table 1.8.

Table 1.8 Summary of economic results

	Unit	Base case
Cash cost	$/lb payable ZnEq	0.44
Gross sales revenue	$M	825
Royalties	$M	33
Site operating costs (mining, processing, G&A)	$M	358
EBITDA	$M	433
Total taxes	$M	48
Revenue split by commodity	% Cu	1%
	% Zn	27%
	% Pb	28%
	% Sn	3%
	% Ag	20%
	% Au	21%
Pre-tax LOM cash flows (undiscounted)	$M	250
Pre-tax NPV at 8%	$M	137
Pre-tax IRR	%	37%
Pre-tax payback period	Years	3.9
After-tax LOM cash flows (undiscounted)	$M	202
After-tax NPV at 8%	$M	106
After-tax IRR	%	31%
After-tax payback period	**Years**	**4.1**

Note: EBITDA = Earnings before interest, tax, depreciation, and amortization.

1.11.1 Sensitivity

Sensitivity analyses were performed for variations in metal prices, capital costs, and operating costs to determine their relative importance as project value drivers. Post-tax NPV is most sensitive to changes in the zinc and lead prices. The NPV is moderately sensitive to changes in operating costs and capital costs, and in the price of gold and silver. Changes in the price of tin and copper have the least impact on NPV. The NPV remains positive for all drivers within the sensitivity range of ±15%.

1.12 Interpretation and conclusions

1.12.1 Geology

The LS Project is located within the north-western portion of the IPB. The IPB is one of the most prolific European metallic provinces, hosting one of the largest concentrations of MS in the Earth's crust.

The entire property 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 Project. The Tertiary sedimentary rocks unconformably overlie rocks of the Volcano-Sedimentary Complex of the IPB.

The LS Project currently has three known deposits: the North, Central, 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. The North deposit is further offset by an east-west-trending Alpine-age fault in the north, with a 50-m downthrow of the northern block but whose horizontal amount and sense of displacement is unknown.

The mineralization comprises MS and semi-massive sulphide lenses and sulphide veins and veinlets and is mainly hosted by a thick (up to 250 m) and strongly chloritized quartz-phyric rhyodacite unit. Currently, the mineralization is known to extent continuously over a cumulative strike length of 1.7 km in a north-northwesterly to south-southeasterly direction.

LS Project 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. 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 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 greatest contribution to the Mineral Resources is from the North deposit. However, all deposits have the potential to delineate more resources with additional drilling. The stringer / fissure type mineralization of the South and Central deposits appears to be more amenable to metallurgical processing than the massive mineralization of the North deposit and future priority drilling will depend on progress in metallurgical testwork.

1.12.2 Mining

The geometry of the deposit is relatively simple, and the deposit is fairly shallow. AMC considers the use of SLOS mining method to be suitable for the deposit and will result in high productivities. The average haulage distance is approximately 1,600 m which is relatively short and will ensure low cost haulage and high productivity. Mining takes place primarily over three sub-levels spaced 25 m to 35 m apart. The mine is very compact and as such equipment use will be highly efficient and manpower numbers low. The mine will operate on a five-day week, three eight-hour shifts per day, with essential services manning only, on weekends.

The mine design leads to a low-cost mining operation with relatively high production throughput of 1.0 Mtpa that is well supported by the production schedule. The mine will use paste fill to backfill the stopes reducing the size of the surface tailings facility significantly. Paste fill will be delivered underground using a combination of pumping and gravity via boreholes and high-pressure pipelines to the stopes.

1.12.3 Infrastructure

In general, the Project is well supported with good local infrastructure. Road access to the site is good with potential upgrades required when construction commences. Local power is close to site and the mine can readily connect to the local grid system. Indications are that the mine will be water neutral, and that any make-up water that may be required will be obtained via local wells on site. Should this not be adequate, water can be obtained from the Sado River approximately 5 km from the project site.

It is proposed that administrative buildings will be uncomplicated, and that the local workforce will live in the local towns near the mine site. No mine camp is proposed. With the local community of Grândola 12 km and Lisbon, Portugal's capital, 120 km by road, and both easily accessible, the project will be well supported locally with supplies as well as personnel.

Approximately 55% of all tails will be placed underground as paste fill, this will significantly reduce the size of the TSF. A dry stacked facility is proposed.

1.12.4 Processing

The PEA assumes a processing rate of 3,000 tonnes per day (tpd) of polymetallic ore. The following conclusions can be drawn from the scoping level metallurgical study:

- The deposit has two major ore types, namely, massive sulphide and Gossan. The Gossan consists of oxidized material.
- The metals of interest in the deposit are copper, lead, zinc, gold, silver, and tin.
- The MS has tin which may not be recoverable whereas Gossan has uneconomic quantities of zinc. Hence, the process flowsheet needs to be flexible to recover these minerals.
- The mineralogy of the deposit indicates that very fine grind will be required to produce saleable concentrates.
- A typical polymetallic process flowsheet may produce copper, lead, and zinc concentrates with some gold and silver. Copper concentrate is not considered in this study.
- The tailings from the sulphide circuit and the Gossan can be cyanide leached to recover the majority of gold and silver.

1.12.5 Economics

The project shows robust economic results with a pre-tax NPV at 8% of $137M and an IRR of 37%, and an after tax NPV at 8% of $103M and IRR of 31%. The economics are highly dependent on the assumptions for metal recovery. AMC recommends that more metallurgical testwork is undertaken to firm up these assumptions and validate them.

Costs have been determined at a PEA (±35%) level of detail. Additional work is required to better define these costs.

1.13 Recommendations

1.13.1 Geology

Micon makes the following recommendations:

- Ascendant should continue to expand the Mineral Resources systematically.
- The immediate focus in the short to medium-term 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.
- The second priority should be infill drilling the gaps separating the North and Central deposits and the gap separating the Central and South deposits.
- Models of the deposits should continue to be refined / updated as more information becomes available.
- Geophysical investigations to the eastern and south-eastern areas of the LS deposit should be continued.
- Subject to satisfactory results, the same exercise should be implemented to the north of the North deposit, targeting the area immediately beyond the major east-west Alpine fault.

1.13.2 Mining

AMC makes the following recommendations:

- Additional geotechnical work should be undertaken in order to support the mine design and provide suitable ground support recommendations.
- A hydrogeological study that aims to determine and predict ground water inflows should be undertaken.

- Following the creation of a hydrology model, additional work should be undertaken to determine the dewatering strategy and ensure sufficient pumping arrangements are considered.
- A more detailed design is required for the paste fill distribution. Exact locations for proposed boreholes and the backfill plant location are also required.
- The cost assumptions are mostly based on benchmark information. Additional work is recommended to develop a first principles cost estimate and that comparative quotes from contractors be obtained.
- A detailed production schedule should be developed that carefully considers the primary and secondary stope mining sequence from level to level.

1.13.3 Infrastructure

- Additional work is required to better define and cost the project infrastructure.
- Ascendant should engage with the local power supplier to better understand the availability and capacity of the power from the local substation. Power line detail and electrical equipment requirements can be better defined and more accurately costed.
- A complete climate and water balance study is required in order to better define the water requirements for the operation.
- A tailings specialist should be engaged to determine the exact design, layout and cost of the proposed dry stack tailings facility.

1.13.4 Processing

Optimum metallurgical recoveries are key to the success of the LS Project. AMC recommends detailed metallurgical investigations that should be prioritized over additional drilling to expand the Mineral Resource.

- The project warrants advancement to the next level of study based on the results of the PEA.
- A systematic bench-scale study should be undertaken to optimize the recoveries and concentrate grades for Cu, Pb, and Zn from both fresh and oxidized ores.
- Develop a process for recovery of tin from the MS final tailings.
- Optimize the cyanidation leach circuit for gold and silver extraction.
- Generate process data for proper sizing of equipment.

1.13.5 Economics

- AMC recommends that more metallurgical testwork is undertaken to firm up the metal recovery assumptions and validate them.
- Costs have been determined at a PEA (±35%) level of detail, additional work is required to better define these costs.

1.13.6 Additional work

- Prepare a proposed 2020 exploration / development program and budget.
- Conduct follow up work to confirm the favourable geophysics results obtained during the 2019 exploration program.
- 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 mine design.

- Implement a hydrogeological study to better define and predict ground water inflow.
- Complete site-based climate and water balance.
- On completion of the above consider undertaking a Pre-Feasibility study (PFS).
- Continue Environmental permitting.

In order to complete the planned 2020 exploration / development work, AMC has proposed a budget of approximately $4.7M broken down as summarized in Table 1.9.

Table 1.9 Proposed work program and budget for the LS Project for 2020

Program	**Activity**	**Cost ($)**
Drillhole IP Survey (North, Central, & South deposits)	Interpretation / modelling	30,000
Metallurgical testwork drilling	4 drillholes (1,200 m)	240,000
Detailed metallurgical testwork	Optimizing recoveries	350,000
North deposit exploration drilling (expanding inferred)	4 drillholes (1,400 m) + assays + modelling	420,000
Central / South deposits + other targets exploration drilling	14 to 16 drillholes (5,700) + assays + modelling	1,710,000
Geotechnical work to support PFS level design	Stope modelling empirical modelling, design criteria, ground support recommendations.	350,000
Hydrological modelling	Field testwork, modelling, ground water impact establishment	350,000
Site based water balance	Field testwork and measurements, modelling	150,000
Environmental permitting	Field studies, base line data collection, Mining Plan, RECAPE	300,000
PFS	Complete study	750,000
All activities	**Grand total**	**4,650,000**

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2 Introduction

2.1 Terms of reference

AMC Mining Consultants (Canada) Ltd. (AMC) has been retained by Ascendant Resources Inc. (Ascendant) to complete a Preliminary Economic Assessment (PEA) for the Lagoa Salgada (LS) Property (Property) 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) "Standards of Disclosure for Mineral Projects" of the Canadian Securities Administrators. 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 produced by Micon International Limited (Micon). The Lagoa Salgada Project (LS Project or Project) is located within the Property.

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

2.2 Qualified Persons

The names and details of persons who prepared, or who have assisted the Qualified Persons (QPs), in the preparation of this Technical Report are listed in Table 2.1. The QPs meet the requirements of independence as defined in NI 43-101.

Table 2.1 Persons who prepared or contributed to this Technical Report

Qualified Persons responsible for the preparation of this Technical Report						
Qualified Person	**Position**	**Employer**	**Independent of LS**	**Date of last site visit**	**Professional designation**	**Sections of Report**
Mr C Murahwi	Senior Geologist	Micon International Limited	Yes	28-31 May 2019	P.Geo., Pr.Sci.Nat., FAusIMM	Sections 4 to 12 and 14, Parts 1, 25, 26, and 27
Mr G Methven	Underground Manager / Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	No visit	P.Eng. (BC)	Sections 2, 3, 15, 16, 20, 22, 23, and 24, Parts of 1, 21, 25, 26, and 27
Mr G Zazzi	Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	No visit	P.Eng. (BC)	Section 18, Parts of 1, 21, 25, 26, and 27
Mr D Malhotra	Principal Metallurgist	Pro Solv, LLC	Yes	No visit	SME Registered Member	Sections 13, 17, and 19, Parts of 1, 21, 25, 26, and 27
Other Experts who have assisted the Qualified Persons						
Expert	**Position**	**Employer**	**Independent of VTT**	**Visited site**	**Professional designation**	**Sections of Report**
Mr Neil Ringdahl	COO	Ascendant Resources Inc.	No	Yes		All
Mr Joao Barros	Managing Director	Redcorp	No	Yes		Sections 4, 20, and 23

Mr Charley Murahwi, P.Geo. representing Micon conducted site visits to the property from 16 to 19 October 2018, from 13 to 17 November 2018 and from 28 to 31 May 2019. During these visits, Micon discussed the geological model, verified some of the drillhole collar positions, witnessed downhole survey measurements, examined drill cores, reviewed drillhole logs, reviewed mineralization types, and reviewed / discussed the Quality Assurance / Quality Control (QA/QC) protocols / results of the on-going drilling programs.

2.3 Sources of information

This 2019 PEA makes extensive use of the previous Technical Report, dated 5 November 2019 with an effective date of 5 September 2019, and titled "Technical Report of the Resource Estimate Update for the Lagoa Salgada Project, Setúbal District Portugal" (2019 Micon Technical Report).

Other principal sources of information for this Report are:

- Previous NI 43-101 Technical Reports on the LS Project filed on the System for Electronic Document Analysis and Retrieval (SEDAR).
- Drillhole databases supplied by Redcorp.
- Observations made during the site visits by Micon, represented by Mr Charley Murahwi, P.Geo., FAusIMM.
- Redcorp internal exploration assessment reports and copies of reports submitted to the Government of Portugal.
- Geophysical reports prepared for Redcorp by Christopher J. Hale, P.Geo. of Intelligent Exploration (IE).
- Discussions with Redcorp management and staff familiar with the property.
- Mineralogical and metallurgical reports supplied by Redcorp.
- Mine design and wireframe files from Ascendant.
- Supporting excel spreadsheets for the estimation of costs, production schedules, and productivities from Ascendant.

2.4 Units of measure and abbreviations

All currency amounts are stated in US dollars (US$). Quantities are generally stated in metric units, the standard Canadian and international practice, including metric tons (tonnes, t) or kilograms (kg) for weight, kilometres (km) or metres (m) for distance, hectares (ha) for area, grams (g), and grams per metric tonne (g/t) for gold and silver grades (g/t Au, g/t Ag). Wherever applicable, Imperial units have been converted to Système International d'Unités (SI) units for reporting consistency. Precious metal grades may be expressed in parts per million (ppm) and their quantities may also be reported in troy ounces (ounces, oz), a common practice in the mining industry. A list of abbreviations and acronyms are provided in Table 2.2.

Table 2.2 List of abbreviations and acronyms

Abbreviations & acronyms	Description
$	United States dollar
€	Euro
%	Percentage
>	Greater than
<	Less than
°	Degree
°C	Degrees Celsius
µ	Micron
µm	Micrometre
2D	Two-dimensional
3D	Three-dimensional
ABS	Acrylonitrile butadiene styrene
Ag	Silver
AMC	AMC Mining Consultants (Canada) Ltd.
APA	Environmental Impact Assessment Authority
Ascendant	Ascendant Resources Inc.
Au	Gold
AuEq	Gold equivalent
BHIP	Borehole Induced Polarization
Capex	Capital expenditure
CCDR	Alentejo Regional Coordination and Development Commission
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	Centimetre
Coeff. V / CoV	Coefficient of variation
CRM	Certified reference material
Cu	Copper
CuEq	Copper equivalent
$CuSO_4$	Copper sulphate
D_{80}	80% particle size distribution
DGEG	Direção Geral de Energia e Geologia / General Directorate of Energy and Geology
dmt	Dry metric tonne
Drillcon	Drillcon Iberia S.A.
DTM	Digital terrain model
E	East
EDM	Empresa de Desenvolvimento Mineiro S.A.
EIA	Environmental Impact Assessment
EPCM	Engineering, Procurement and Construction Management
EPDM	Empresa de Perfurção e Desenvolvimento Mineiro, S.A.
Fe	Iron
g	Gram
G&A	General and Administration
g/t	Grams per tonne
GO	Gossan
GSL	Grinding Solutions Ltd.
ha	Hectare

Abbreviations & acronyms	Description
Hg	Mercury
hr(s)	Hour(s)
I	Indicated
ID	Identification
ID^2	Inverse distance squared
ID^3	Inverse distance cubed
IE	Intelligent Exploration
IGCP	Instituto Geográfico e Cadastral de Portugal
IGM	Instituto Geológico e Mineiro
IGT	International Geophysical Technology
IP	Induced polarization
IPB	Iberian Pyrite Belt
IRIS	IRIS Instruments
ISO	International Organization for Standardization
JV	Joint venture
k	Thousand
kg	Kilogram
kg/m^3	Kilogram per cubic metre
kg/t	Kilogram per tonne
kL	Kilolitre
km	Kilometre
koz	Thousand ounces
kt	Thousand tonnes
kV	Kilovolts
kVA	Kilovolt-Ampere
kW	Kilowatts
kW-hr	Kilowatt-hour
L	Litre
L/s	Litres per second
lb	Pound
LCT	Locked-cycle test
LHD	Load-haul-dump
LIS	Lisbon International Airport
LNEG	Laboratório Nacional de Energia e Geologia
LOM	Life-of-mine
LS	Lagoa Salgada
LVU	Lower Volcanic Unit
M	Million; Measured
m	Metre
m/month	Metre per month
m^2	Metre squared
m^3	Cubic metre
m^3/hr	Cubic metre per hour
m^3/s	Cubic metre per second
Masl	Metre above sea level
Max	Maximum

Abbreviations & acronyms	Description
MIBC	Methyl isobutyl carbinol
Micon	Micon International Limited
Min	Minute; minimum
MLA	Mineral Liberation Analysis
mm	Millimetre
Mm^3	Million cubic metres
Mn	Manganese
MS	Massive sulphide
Mt	Million tonnes
Mtpa	Million tonnes per annum
MVA	Mega volt amperes
mV/V	Millivolts per volt
MW	Megawatt
N	North
NaCN	Sodium cyanide
NI 43-101	National Instrument 43-101
NN	Nearest neighbour
NNW	North-north-west
NPV	Net present value
NSR	Net smelter return
NW	North-west
OK	Ordinary kriging
Opex	Operating expenditure
oz	Troy ounce
P_{80}	80% Passing
Pb	Lead
PDA	Environmental Scoping Proposal
PEA	Preliminary Economic Assessment
PFS	Pre-Feasibility Study
pH	pH is a measure of hydrogen ion concentration; a measure of the acidity or alkalinity of a solution
Portex	Portex Minerals Inc.
ppm	Parts per million
Project / LS Project	Lagoa Salgada Project
Property	Lagoa Salgada Property
PVC	Polyvinyl chloride
QA/QC	Quality Assurance / Quality Control
QP	Qualified Person as defined by NI 43-101
Redcorp	Redcorp - Empreendimentos Mineiros, Lda
Redcorp Ventures	Redcorp Ventures Ltd.
Report	Technical Report
RQD	Rock quality designation
RTZ	Rio Tinto Zinc
S	South
SEDAR	System for Electronic Document Analysis and Retrieval
SFM	Serviço de Fomento Mineiro

Abbreviations & acronyms	Description
SIPX	Sodium Iso-Propyl Xanthate
SLOS	Sub-level open stoping
SMU	Selective mining unit
Sn	Tin
SSE	South-south-east
st	Short ton
ST; Str/Fr	Stockwork
Std Dev	Standard deviation
Str	Stringer
S_{TOTAL}	Total Sulphur
t	Tonne
TEM	Transient electromagnetic
TH Crestgate	TH Crestgate GmbH
tonne	Tonne = 1,000 kg
tpa	Tonnes per annum
tpd	Tonnes per day
tph	Tonnes per hour
tpvm	Tonnes per vertical metre
TSF	Tailings Storage Facility
US	United States
US$	United States dollar
UTM	Universal Transverse Mercator
UVU	Upper Volcanic Unit
V	Volt
VMS	Volcanogenic massive sulphide
W	Watt, West
Wt	Weight
Zn	Zinc
ZnEq	Zinc equivalent
$ZnSO_4$	Zinc sulphate

3 Reliance on other experts

This Report has been prepared by AMC for Ascendant. The information, conclusions, opinions, and estimates contained herein, for which the named QPs take responsibility, are based on:

- Information available to AMC, including from the AMC database, at the time of preparation of this Report.
- Assumptions, conditions, and qualifications as set forth in this Report.
- Data, reports, and other information supplied by Ascendant and from other sources.

The QPs have relied, in respect of legal aspects, upon the work of the Expert listed below. To the extent permitted under NI 43-101, the QPs disclaim responsibility for the relevant section of the Report:

- Mr Joao Barros, Managing Director, Redcorp.
- Report, opinion or statement relied upon information on mineral tenure and status, title issues, and mining concessions. Map supplied by the General Directorate for Energy and Geology.
- Extent of reliance: full reliance following a review by the QPs.
- Portion of Technical Report to which disclaimer applies: Section 4.

The QPs have relied, in respect of environmental aspects, 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 section of the Report:

- Mr Joao Barros, Managing Director, Redcorp.
- Report, opinion or statement relied upon information on environmental studies and permitting.
- Extent of reliance: full reliance following a review by the QPs.
- Portion of Technical Report to which disclaimer applies: Section 20.

The QPs have relied, in respect of taxation and royalty aspects, 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 section of the Report:

- Neil Ringdahl, Chief Operating Officer, Ascendant.
- 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: Section 22.

4 Property description and location

4.1 Description and location

The Property consists of a single exploration permit described in Section 4.2 and contains the project that is the subject of the PEA.

Geographically, the LS Project is located 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.

The Property is located approximately 80 km south-east of Lisbon, Portugal’s capital; and approximately 120 km by road. It is located approximately 50 km south-east of Setúbal, the regional administration centre, 12 km north-east of the municipality of Grândola and approximately 3 km north of the village of Cilha do Pascoal. See Figure 4.1.

Figure 4.1 Location map for the Property

Source: Daigle 2012.

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, Redcorp renewed the permit, in 2017, at which time, 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 the completion of the first three years, a company may apply for a renewal, for an additional two years and submit a reduction of the permit area of up to 20%. The exploration permit may be renewed a maximum of two times. During this time, a 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 which is 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 - 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 renewed to 20 June 2019. Table 4.1 summarizes the original exploration permit information along with renewal information. 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

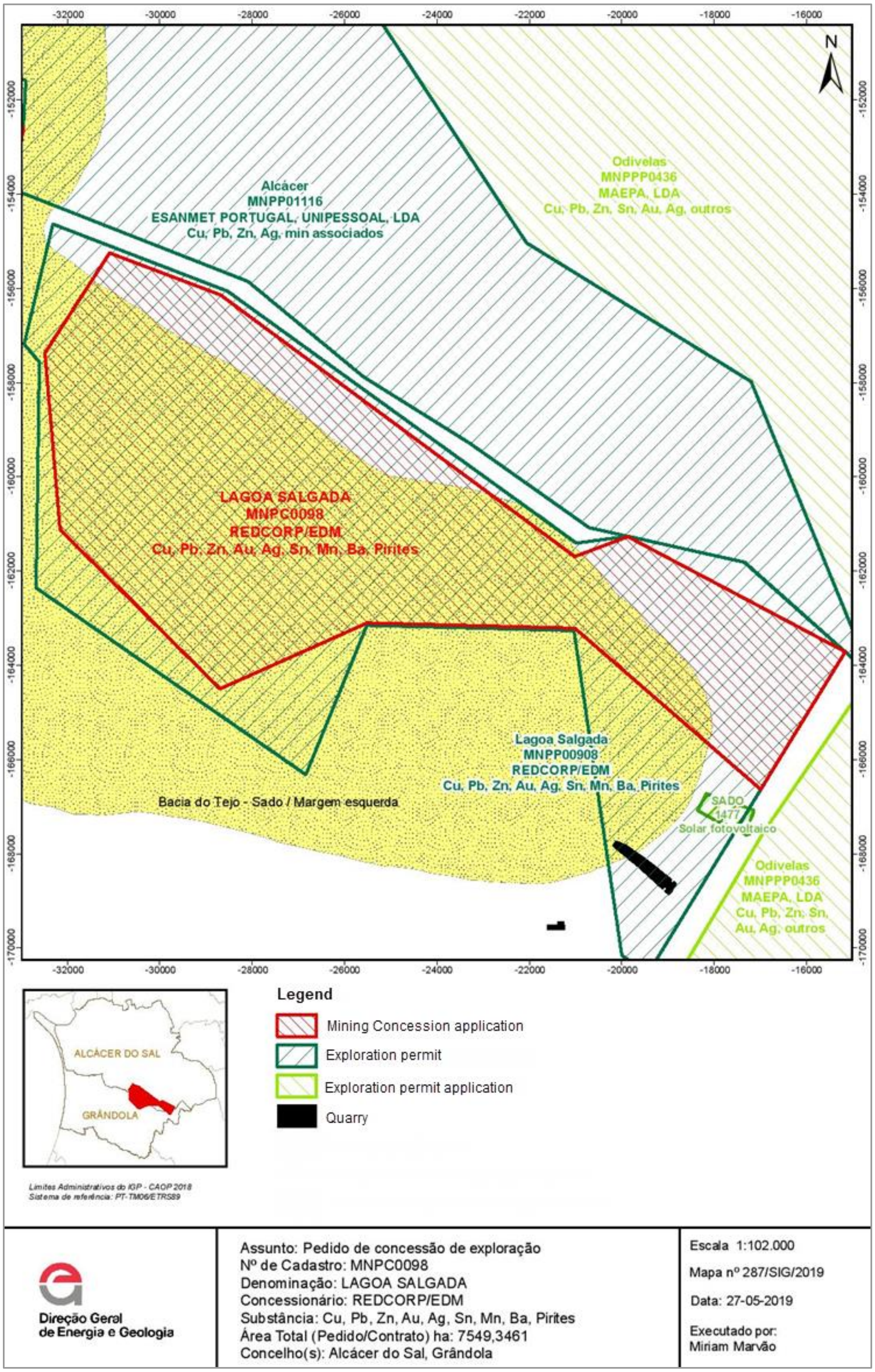
Name	Exploration permit	Expiry date	Area (ha)
Lagoa Salgada	Contrato MN/PP/009/08	20 June 2017	13,333.9
Lagoa Salgada	Contrato MN/PP/009/08	20 June 2019	10,700.0

In April 2019, Redcorp and EDM applied to the Portuguese Government, through the Secretary of State for Energy, for the definitive mining concession for LS, for an area of approximately 7,500 ha (Figure 4.2). The application has been accepted and an official publication was made on 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.

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.

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 Project 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.

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 has the right to earn a further effective 25% interest via staged payments and funding obligations as outlined below:
  - 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.
  - 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 earn an additional 30%, totalling an 80% interest in Redcorp, the operating subsidiary, by completing a feasibility study within 54 months and making a further payment of $2.5M to TH Crestgate.
- Ascendant will fund all development and future construction costs and recoup TH Crestgate's share of investment through cash flow until 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 has the right to earn into 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 Project 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 located 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 QP's knowledge, all of the required permits and permissions to access and conduct exploration activities have been obtained from the holders of the surface rights. All exploration activities conducted on the Property do not require additional permits; however, proposed exploration programs are subject to approval by the DGEG.

4.4.3 Environmental liabilities

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

4.5 Royalties

A 4% royalty is applied to the gross revenue.

5 Accessibility, climate, local resources, infrastructure, and physiography

5.1 Accessibility

The Property is located approximately 80 km south-east of Lisbon, capital of Portugal. By road, the distance from the Lisbon International Airport (LIS) to the property is approximately 120 km. The property is easily accessible by national highways and roads.

Access to all parts of the property is conducted by truck / utility vehicle or 4 x 4 vehicle, through various unpaved all-weather dirt roads. The dirt roads are maintained, and some may be accessed by car.

The city of Lisbon is serviced by regular scheduled international and domestic flights with the drive from LIS to the LS Project typically 1.5 hours (hrs) in duration. Highway A-2 crosses through the western portion of the property.

5.2 Physiography

The Property is relatively flat with gentle to moderate relief and with shallow valleys running through potions of the property. Topographic elevations range between 20 m and 100 m above sea level (masl).

Vegetation is typical of dry Mediterranean climates, consisting of scrub brush, tall grass, and pine trees. The land on the which the Project operates is privately owned and used primarily for the cultivation of cork trees with some olive and pine nut tree plantations. The soil in the area of the Project is sandy with limited exposure of the Tertiary sedimentary bedrocks.

Figure 5.1 shows a general view of the nature of the soil and surrounding topography at the edge of one of the drill sites.

Figure 5.1 A view of the topography surrounding a drill site

Source: Photo taken by Micon, November 2018.

5.3 Climate

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

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

Exploration activities can be conducted year-round with an occasional halt due to extreme weather conditions.

5.4 Local resources and infrastructure

The closest town of any size to the Project is Grândola, population 14,000, which serves the agricultural industry in the area. Setúbal, the district capital, has a population of approximately 90,000 and lies midway between the Project and Lisbon. Setúbal was once a manufacturing and fish canning center, however, these industries are currently in decline. Most basic services and supplies may be sourced from either of these towns. Heavy equipment contractors are available in Grândola.

As the LS Project is located 50 km from the Aljustrel Mine (zinc / lead) and 85 km from the Neves Corvo Mine (zinc / copper), there is access to experienced mining personnel. Unskilled labour may be sourced from the nearby towns and villages.

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

The LS Project is located near most major infrastructure including, roads, railway, electric power lines, ports, and airports.

There is power available from the national grid on the property. The LS deposit is located approximately 7 km from the nearest 400 kilovolt amperes (kVA) high tension power lines, that power the electric railway.

The nearest deep-water port is in the town of Sines, located approximately 50 km south-west of the property with the closest airport being LIS in Lisbon. The railway is located roughly 7 km to the west with the nearest railhead located in Grândola, 12 km south-west of the Project.

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

6 History

6.1 Prior ownership / ownership changes

The prior ownership and ownership changes of the LS Project 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, then the Serviço de Fomento Mineiro (SFM). In 1993, the SFM became the Instituto Geológico e Mineiro (IGM); which later became incorporated into the Laboratório Nacional de Energia e Geologia (LNEG). The IGM completed 17 drillholes in and around the LS Project 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. The discovery hole, LS-04, intersected massive sulphide (MS) from 126.8 to 203.7 m (Wardrop 2007).

6.2.2 Rio Tinto Zinc, 1994 – 2000

In 1994, the area was awarded to a mining consortium composed of RTZ and 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 Project. 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 Project 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 Project (see Figure 4.2) 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 Micon.

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.

6.2.4 Portex Minerals Inc., 2009-2012

In 2009, Portex acquired a 100% interest in Redcorp Ventures to develop the North (formerly LS-1) 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 in order 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 Project, 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 Project.

7 Geological setting and mineralization

7.1 Preamble

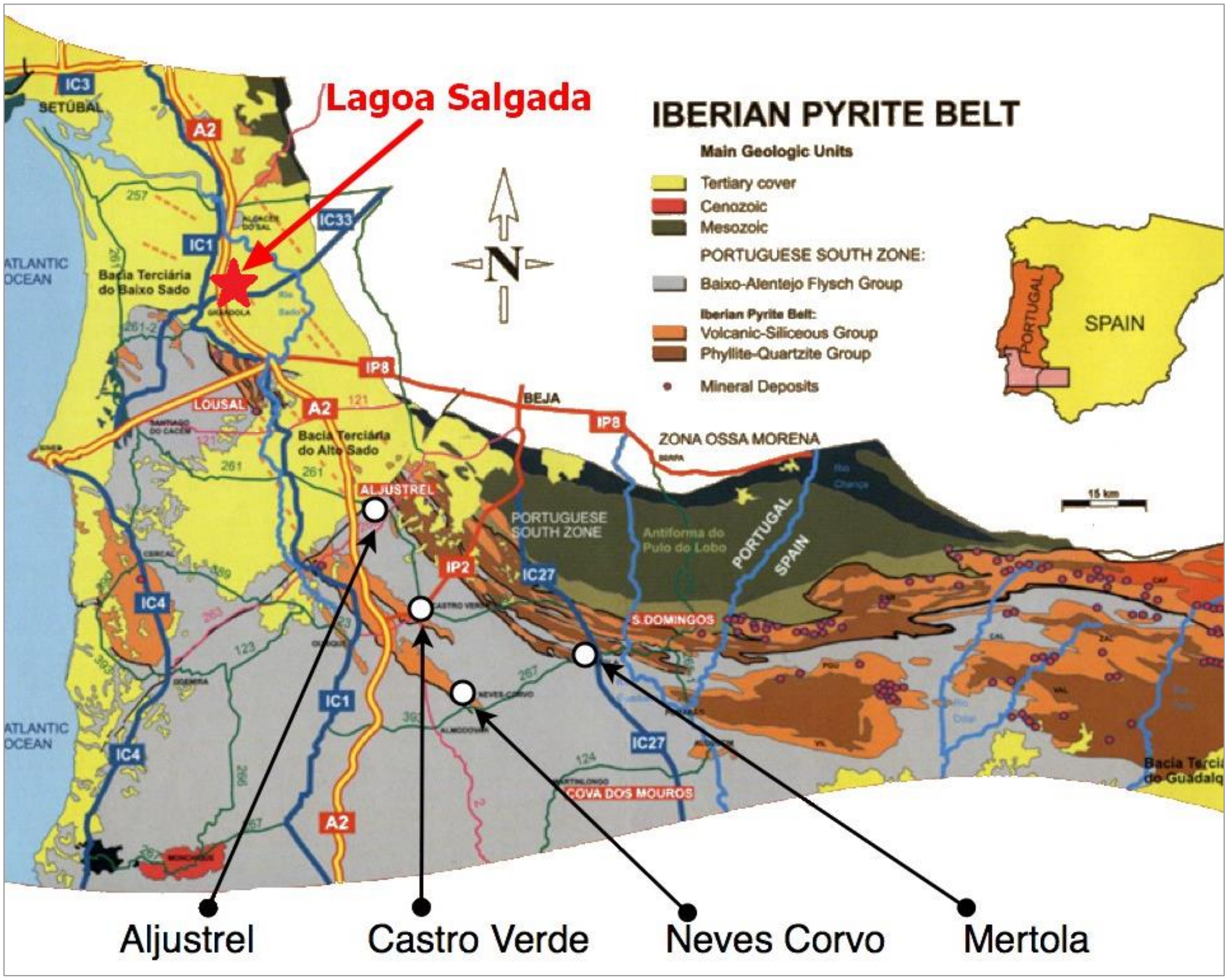
The Property is located within the north-western portion of the Iberian Pyrite Belt (IPB). The IPB is one of the most prolithic European ore provinces, hosting one of the largest concentrations of MS in the Earth's crust; it contains more than 1,600 million tonnes (Mt) of MS ore and about 250 Mt of stockwork 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).

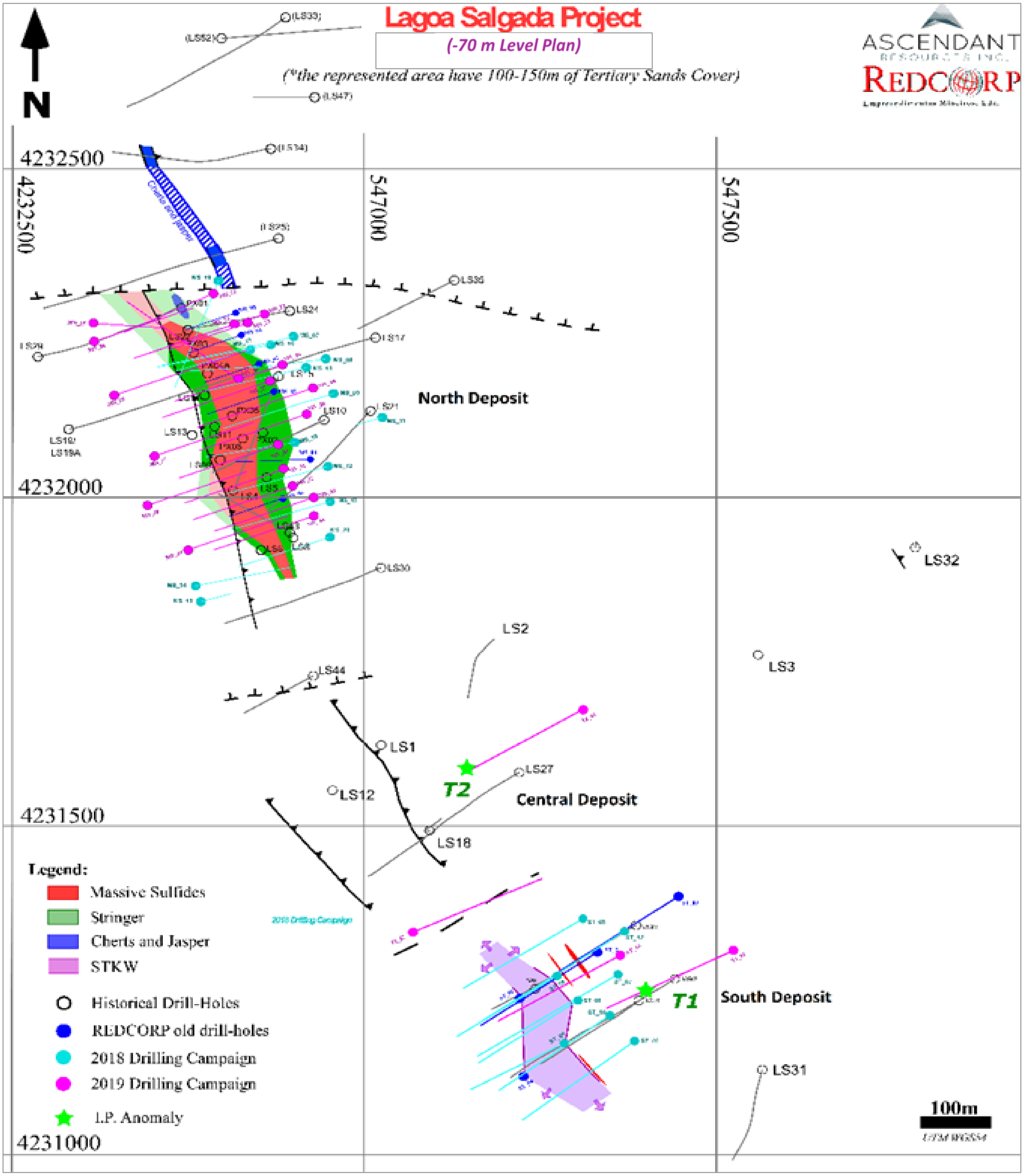
Deformation of the IPB stratigraphic sequence during the Variscan orogeny is characterized by south- to southwest-verging folds, corresponding to a thin-skinned foreland fold and thrust belt (Silva et al. 1990). Low-grade regional metamorphism displays a northward increase from zeolite facies in the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

7.3 Property geology

The entire Property 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 9.8). 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 Project – alpine fault in the North



The mineralization comprises MS and semi-massive sulphide lenses and sulphide veins and veinlets and is mainly hosted by a thick (up to 250 m) and strongly chloritized quartz-phyric rhyodacite unit.

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 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-centimetre (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 (Matos et al. 2000; Figure 7.2).

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).

7.3.3 Mineralization

There are four types of mineralization at the LS Project: 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 a distance of 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 hangingwall 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 as a result 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 as a result 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.

9 Exploration

From the discovery period to the present, exploration on the LS Project 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 the 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. A number of polished sections were selected for examination using scanning electron microscopy and x-ray microanalysis (MEV-EDS) to confirm the identities of certain minerals; while others were selected to perform quantitative microanalysis at the electron microprobe.

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 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 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 the majority of 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) extends with N160 E azimuth along o 500 m approximately, it 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 IE), 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 Project 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 fairly 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.

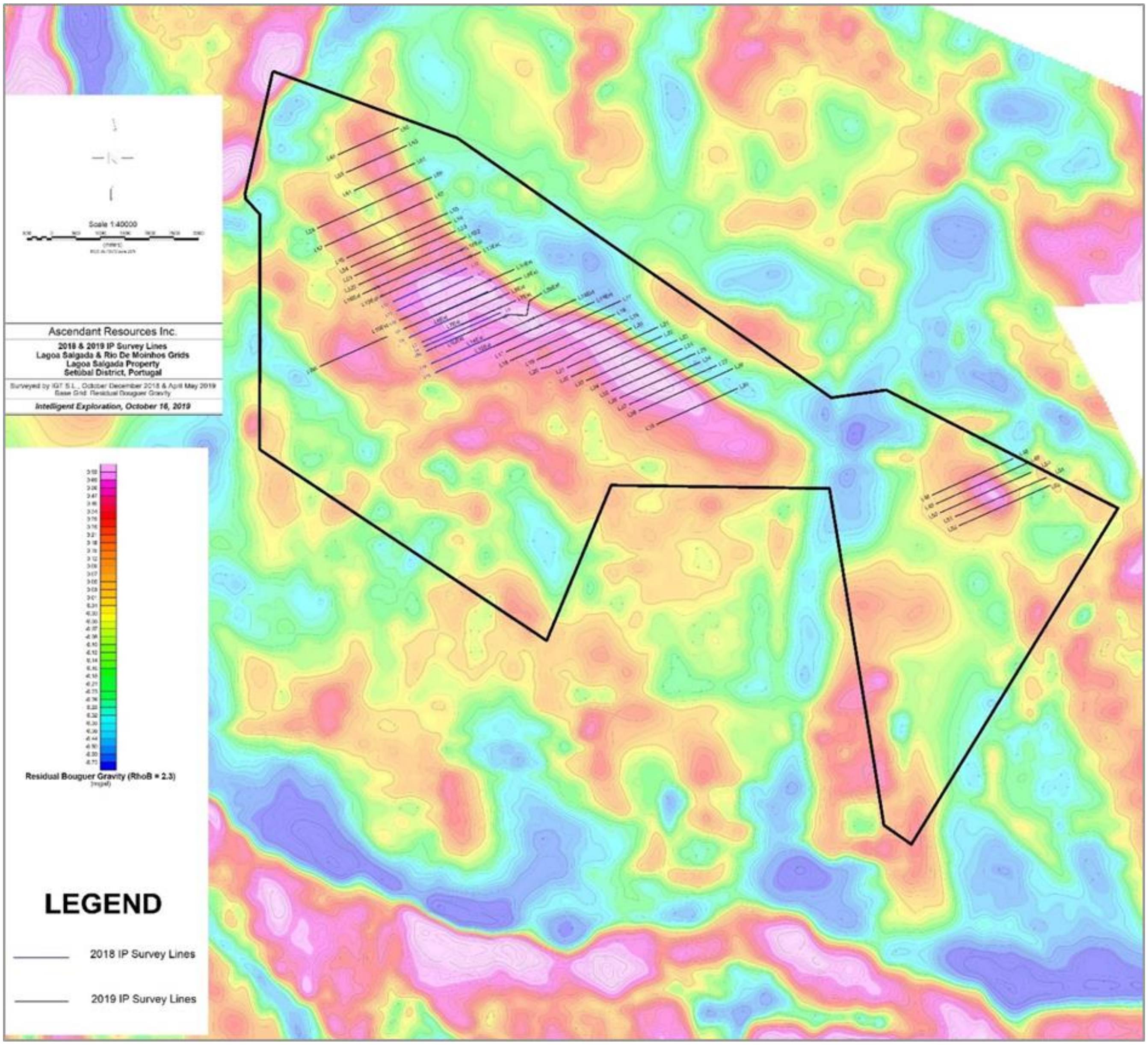
9.3.2 Gravity at LS

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



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 south-west to north-east 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/m^3 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.

Table 9.1 IP / Resistivity survey coverage

	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
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**

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.

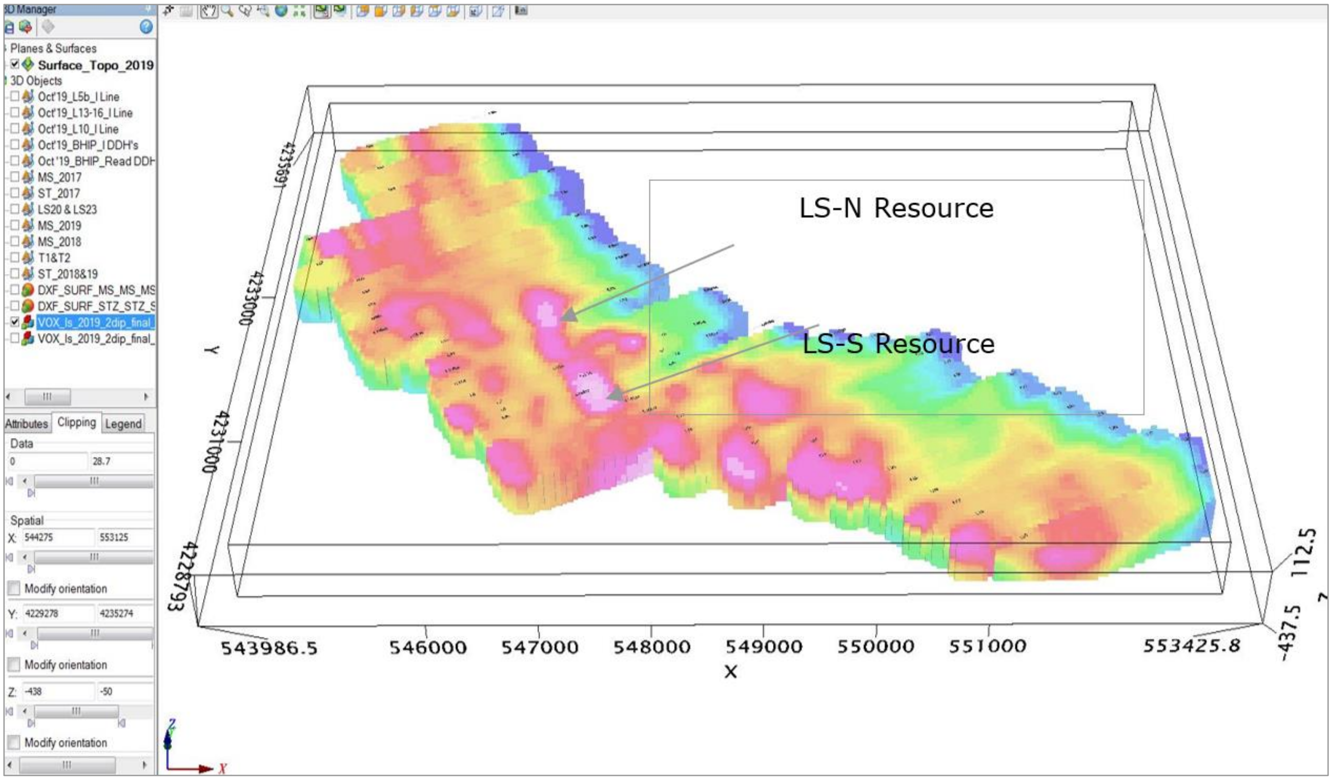
Data files were imported into Geosoft Oasis Montaj® databases (.gdb). Separate data channels were created to store the apparent Resistivity and average IP value (Mx Chargeability) of the middle time slices (~500 to 1,000 msec). Four panel pseudosections with Apparent Resistivity, Chargeability (Mx), 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

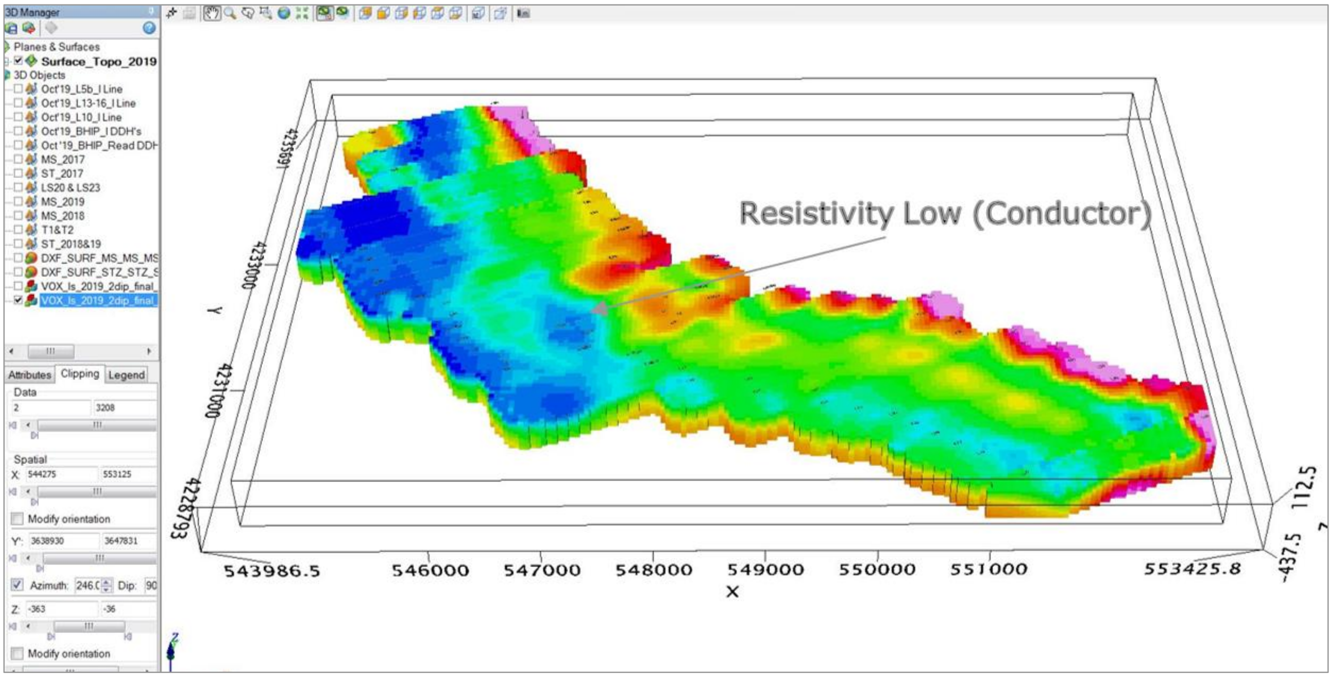
Figure 9.2 and Figure 9.3 show the chargeability and resistivity models respectively, composed from the 2D inverse models calculated for each surface line. The block model has been sectioned at an altitude of -50 m (relative to sea level), approximately 140 m below the surface. This level plan indicates 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: Ascendant.

Figure 9.3 Model Resistivity at -50 m (relative to sea level, about 140 m below the surface)



Source: Ascendant.

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.

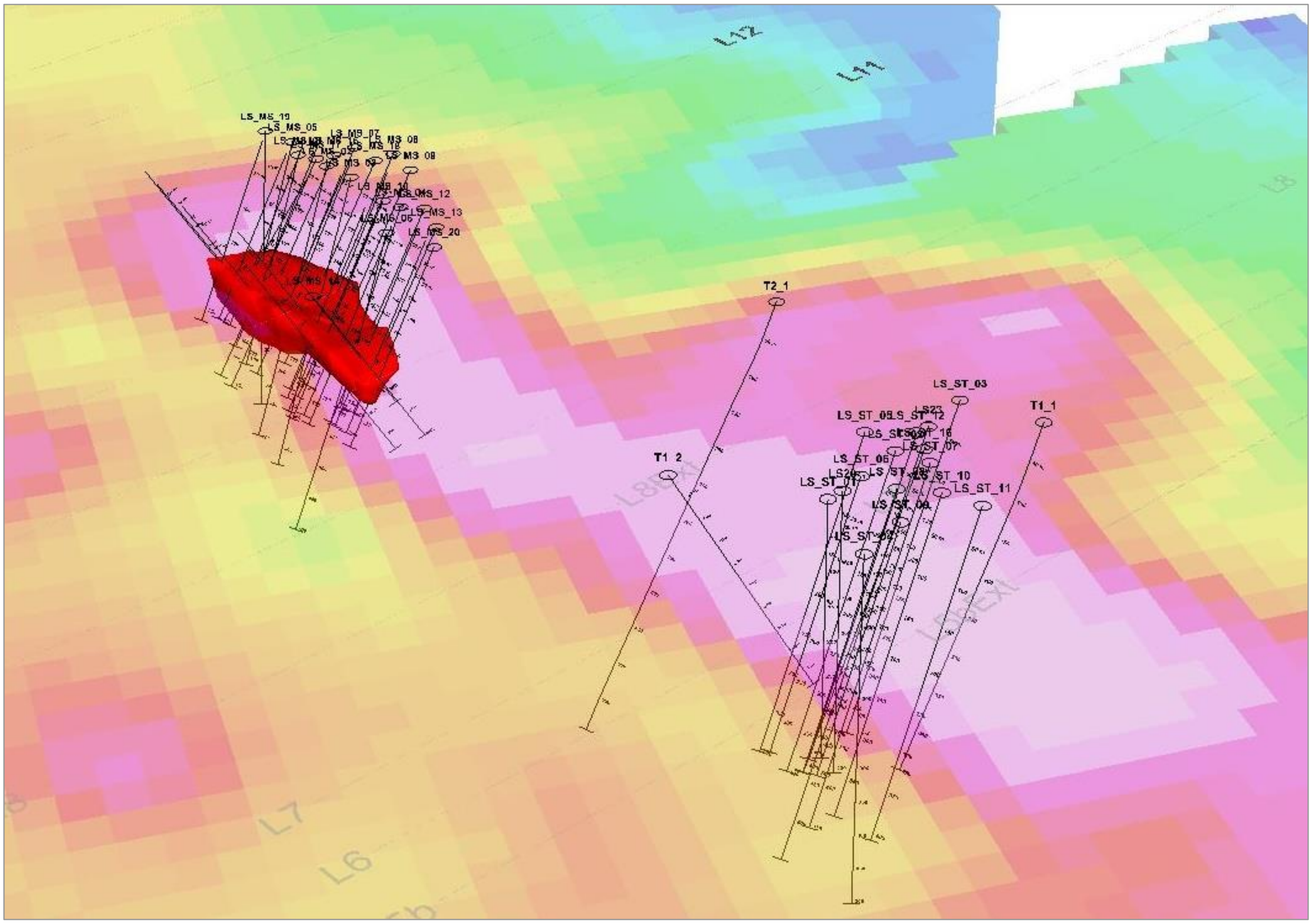
A chargeability anomaly on the apparent east limb of the LS-Rio do Moinhos folded structure, was recommended for drill follow-up early in 2019. This recommendation is still valid in view of the clearer anomaly presented by the 2019 data.

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 Borehole induced polarization results

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 9.4 Model chargeability at -108 m (about 200 m below the surface)

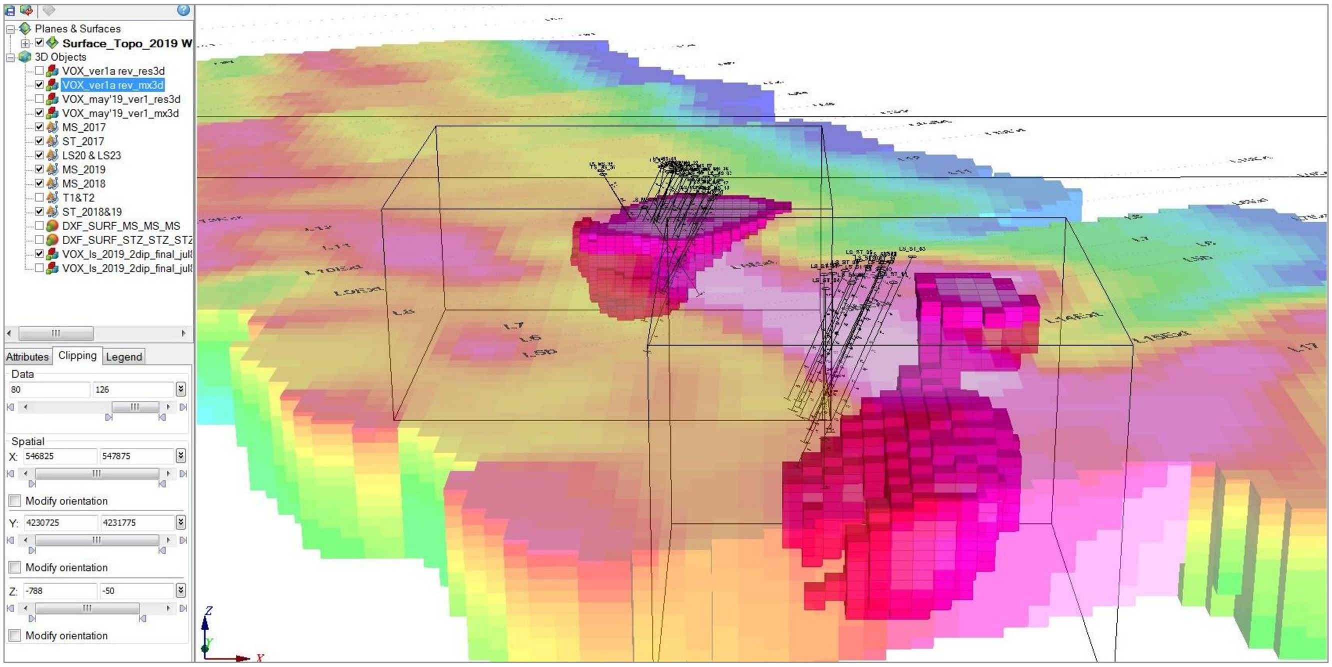


Source: Ascendant.

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.

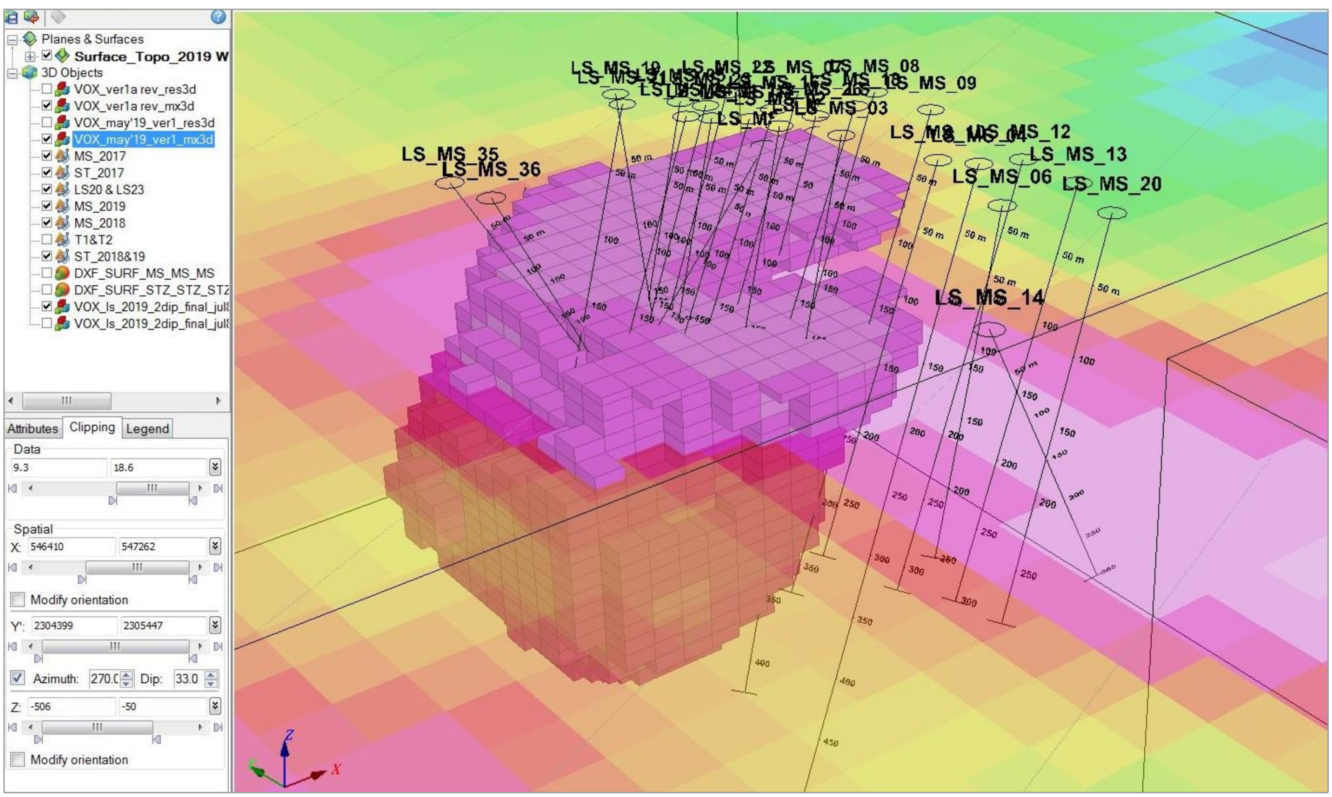
Figure 9.6 shows a closer view of the BHIP model for the North Resource, looking toward the north-east. The top of both the surface IP model and the BHIP has been set to roughly the top of the volcano-sedimentary rocks, about 140 m below the surface. The chargeability determined from BHIP measurements 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 north west 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: Ascendant.

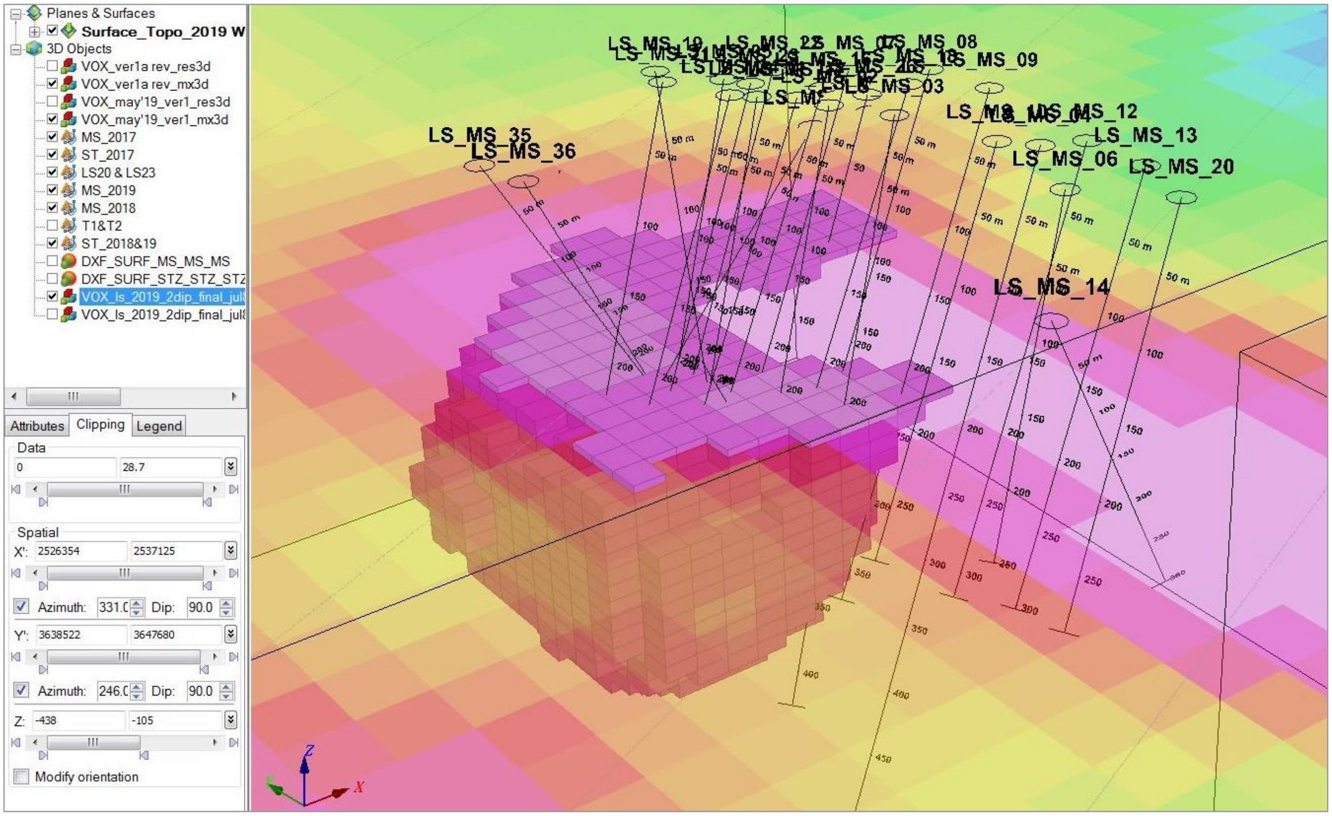
Figure 9.6 BHIP chargeability model for North Resource (looking NE)



Source: Ascendant.

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 north-west 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 north-west. 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: Ascendant.

9.3.7 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 in the Central Zone 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 in the Central Zone 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 (Figure 9.1) 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.

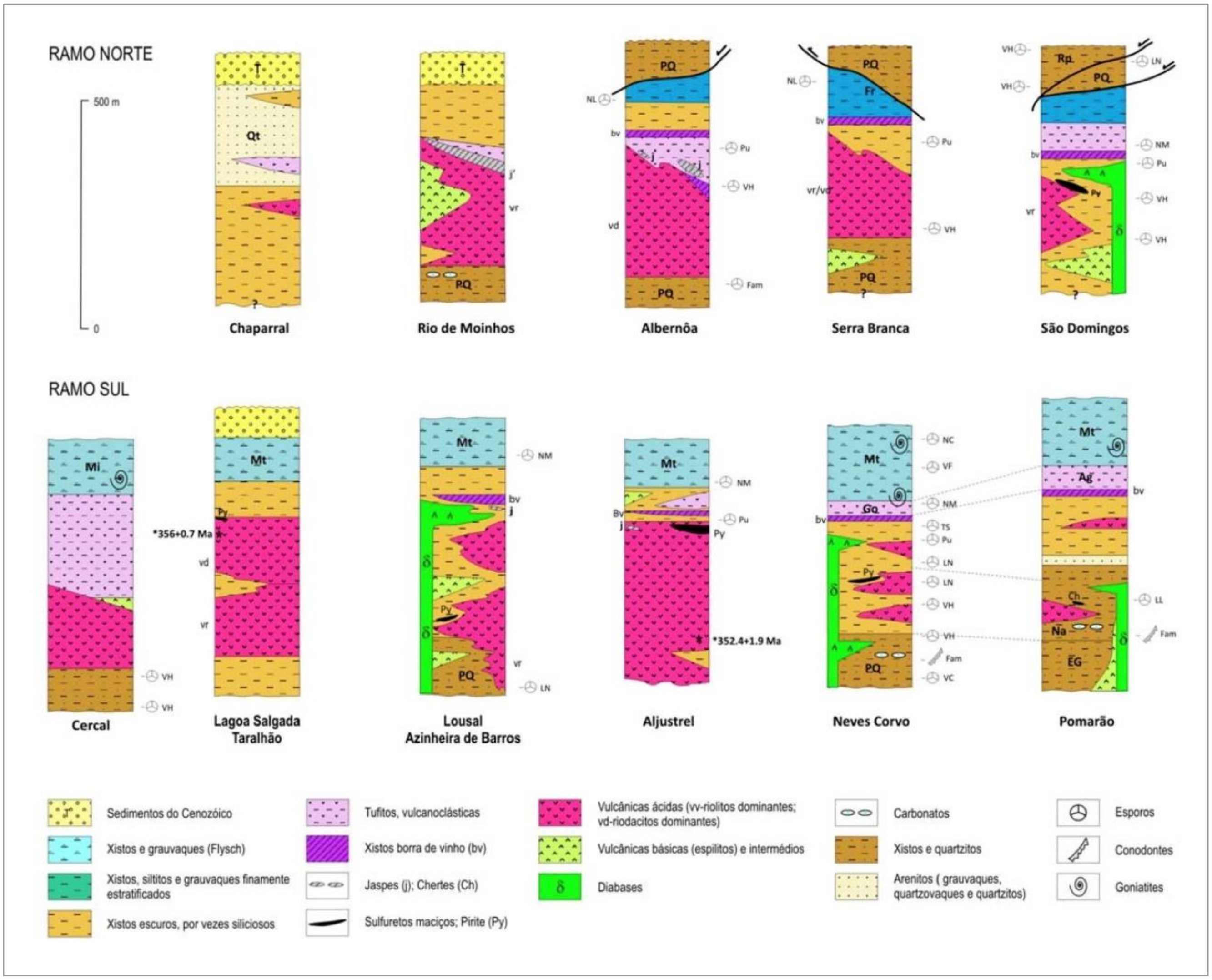
9.4 Structural modelling

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

9.5 QP comments

The QP has reviewed the exploration programs to date and find 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.

Figure 9.8 Stratigraphy and physical vulcanogeny, geochemistry, and mineralizations of the IPB



Source: Ascendant.

LS is further offset by an east-west–trending Alpine-age fault in the north, with a 50-m downthrow of the northern block (Figure 7.2), but whose horizontal amount and sense of displacement is unknown (Matos et al. 2000).

The LS deposits (Figure 7.2) 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 in close proximity to 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.

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 Project 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.

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
Total	**1995 to 2019**	**125**	**49,519**	**HQ**

Of the total, 61 drillholes intersect the North deposit, 4 drillholes intersect the Central Deposit, and 13 drillholes intersect the South deposit.

10.2 Ascendant / Redcorp drilling program

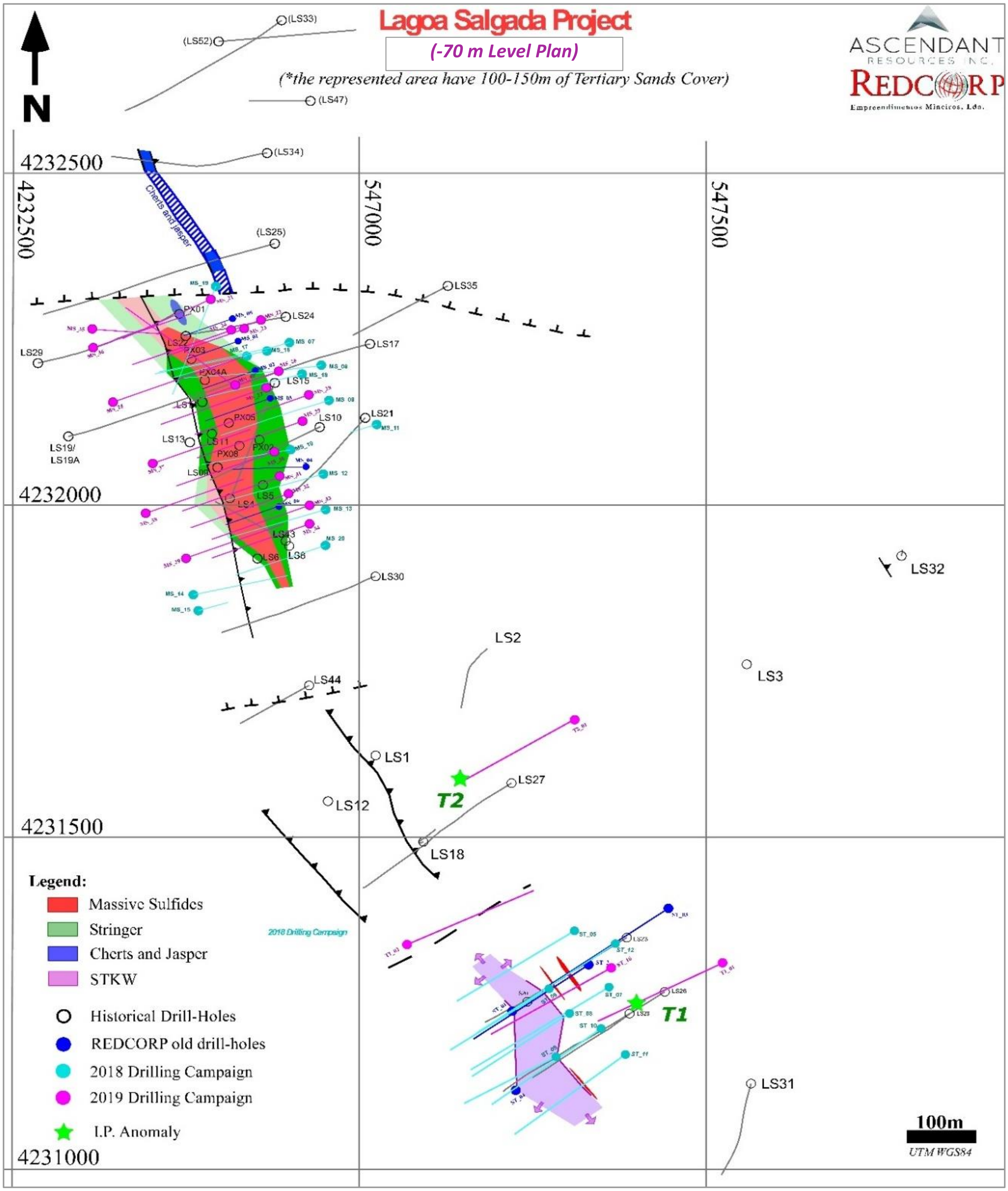
Ascendant / Redcorp have so far conducted two drilling campaigns on the LS Project as shown in Table 10.1 above. In both instances, Drillcon was contracted to undertake the drilling.

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 2019.

Once the drillhole was completed, an NQ size ABS PVC pipe was inserted down the entire length of the drillhole. This was conducted in order 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 Project 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.

The Central deposit profile / mineralization style is similar to the South deposit.

10.2.3 2019 summary of drilling results

The 2019 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.

10.3 QP comments

The drilling results summarized on the above sections demonstrate that the drilling campaigns have progressively yielded encouraging results. However, down dip and lateral extensions still remain to be fully tested for each of the three 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 North deposit

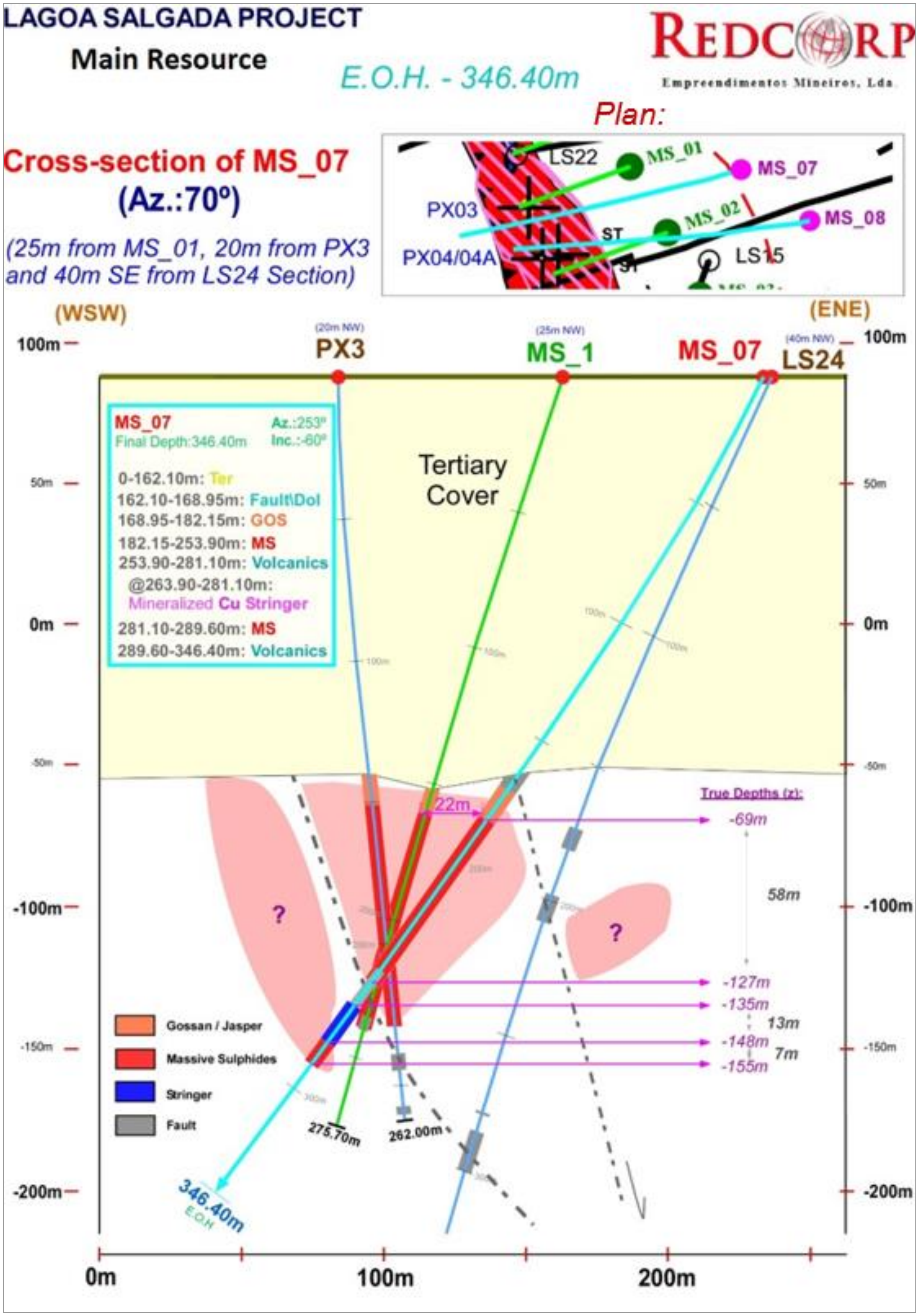
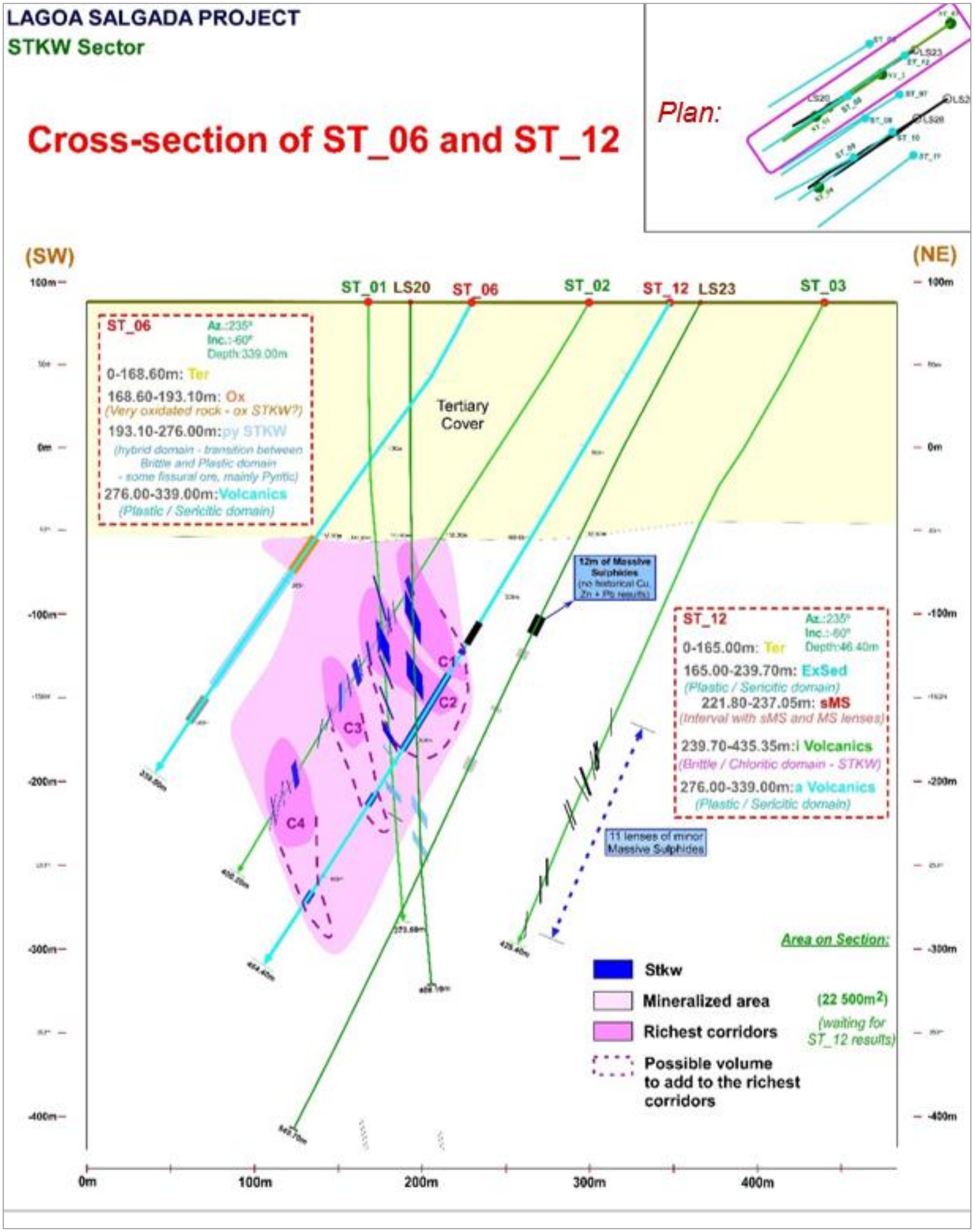
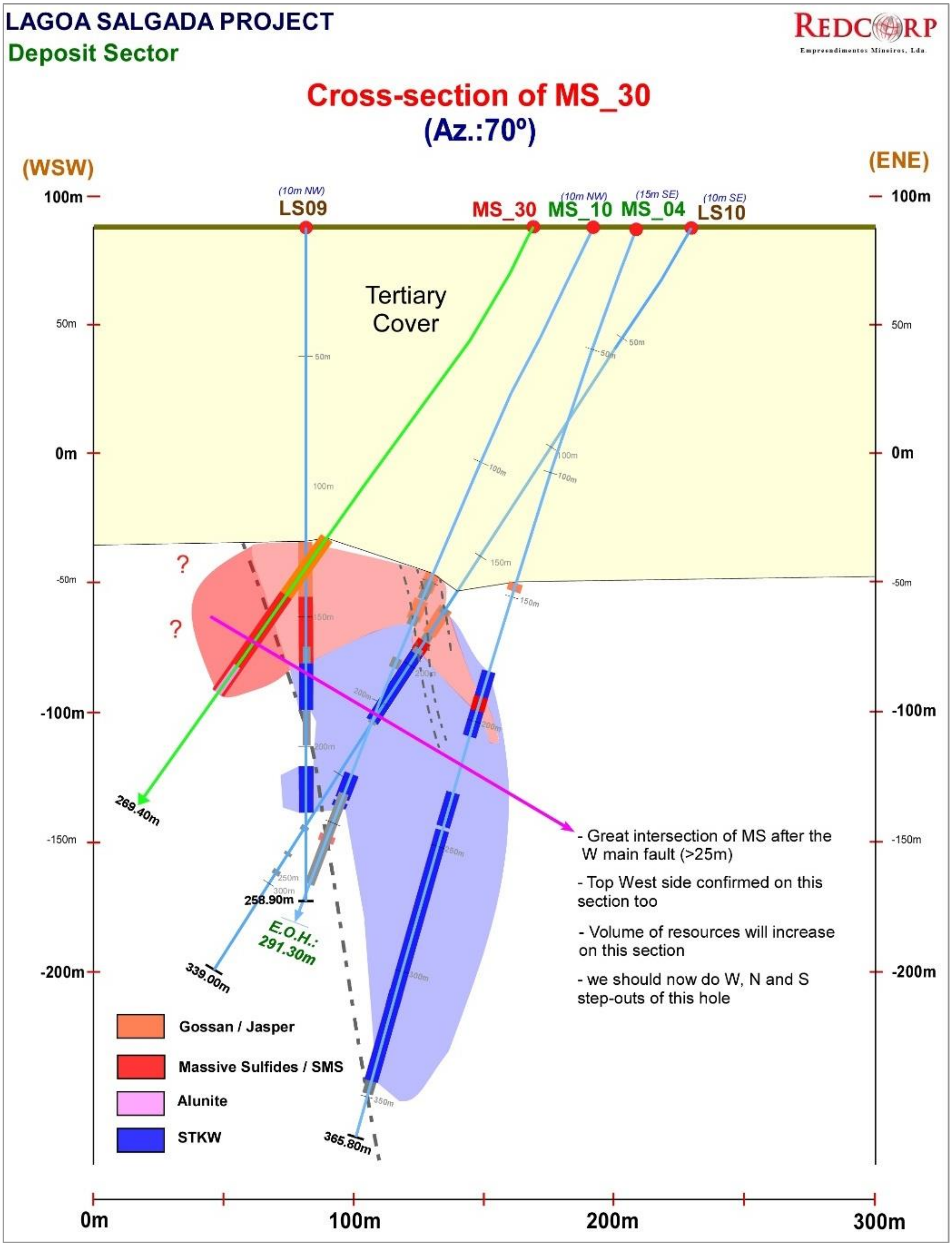


Figure 10.3 Drill section through the South deposit



Source: Micon.

Figure 10.4 Section demonstrating the effects of infill drillhole MS_30



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. 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).

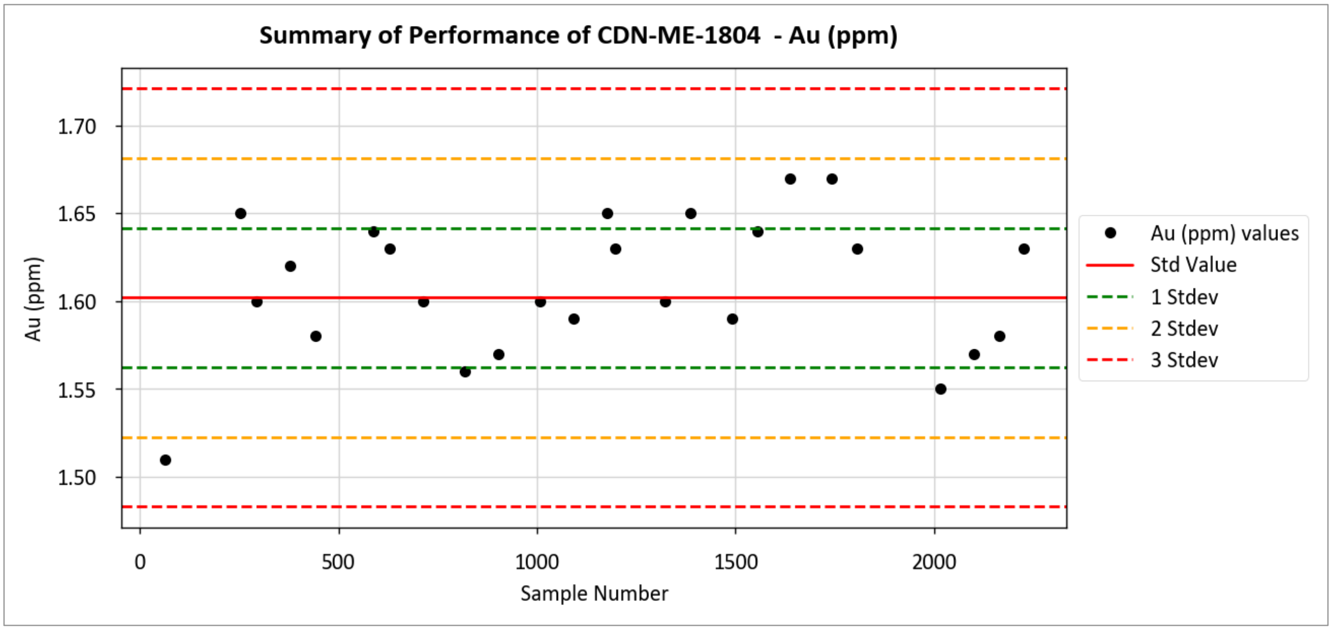
11.2 Bulk density

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

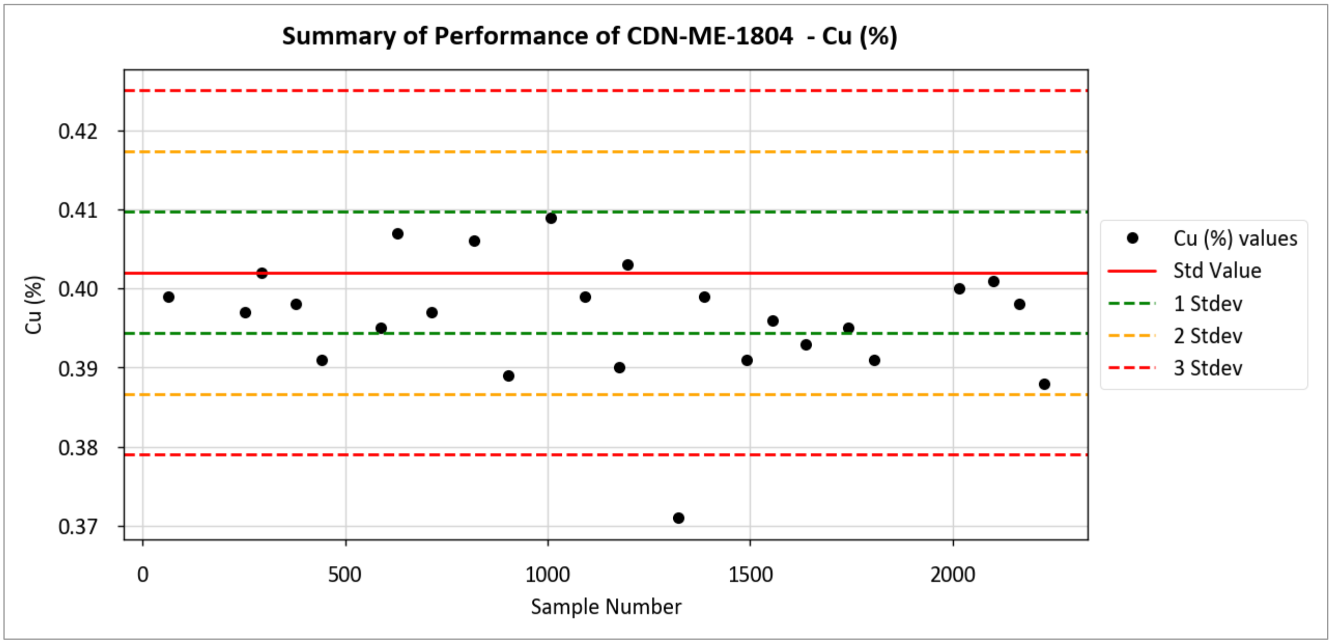
All assays are reported directly to Redcorp via e-mail to designated personnel. Signed assay certificates are sent via courier or post. The monitoring of the performance of the QA/QC samples is conducted immediately after the assay results are received. The assay results for control samples were plotted upon receipt of the initial assays. Certified reference materials (CRM) / standards were considered a failure if the assay was 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. On the whole, the performance of all control samples (blanks and standards) for analytical work has been satisfactory. As examples the performance 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



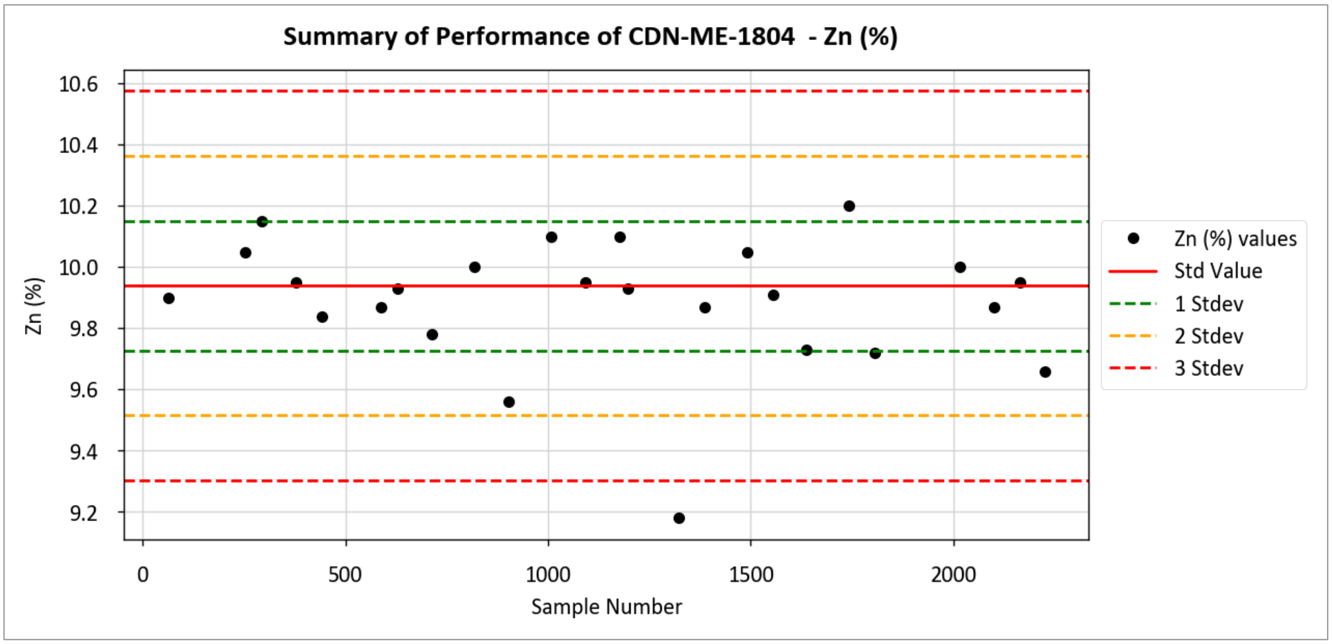
Source: Micon.

Figure 11.2 Summary of performance of CRM CDN-ME-1804: Cu (%)



Source: Micon.

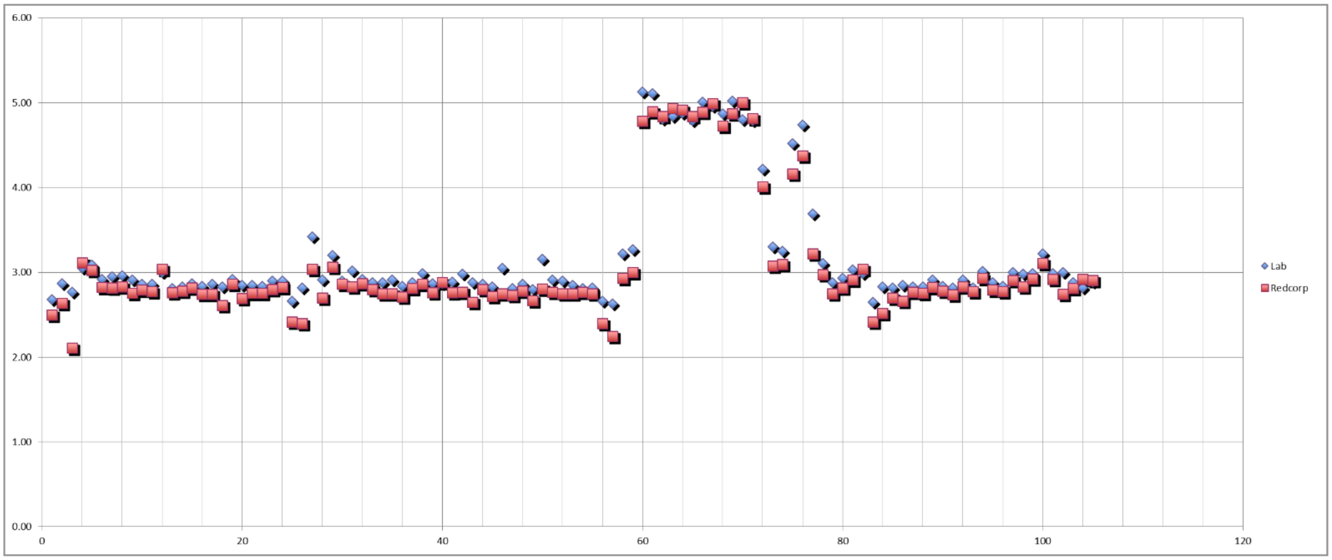
Figure 11.3 Summary of Performance of CRM CDN-ME-1804: Zn (%)



Source: Micon.

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.

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 Project during the site visits, as the mineralization is easily identified in drill cores with the unaided eye.

12.1 Site visit

Micon senior geologist, Charley Murahwi, P.Geo., FAusIMM, visited the LS Project 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 (Redcorp Managing Director) 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, Central and South deposits is arbitrary, being based on the existing drill pattern. In reality, all three deposits coalesce into a single zinc-rich VMS system manifesting / displaying its macro-genetic features from secondary GO to primary massive to primary 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 a distance of 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 North deposit 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.

Figure 12.1 Calibration details for Reflex Ez-Sot used at the LS Project

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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** Reference Magnetometer Serial Number: **12EJB121772**

Issued on: **13 December 2017**

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.

12.1.3 Examination of drill cores

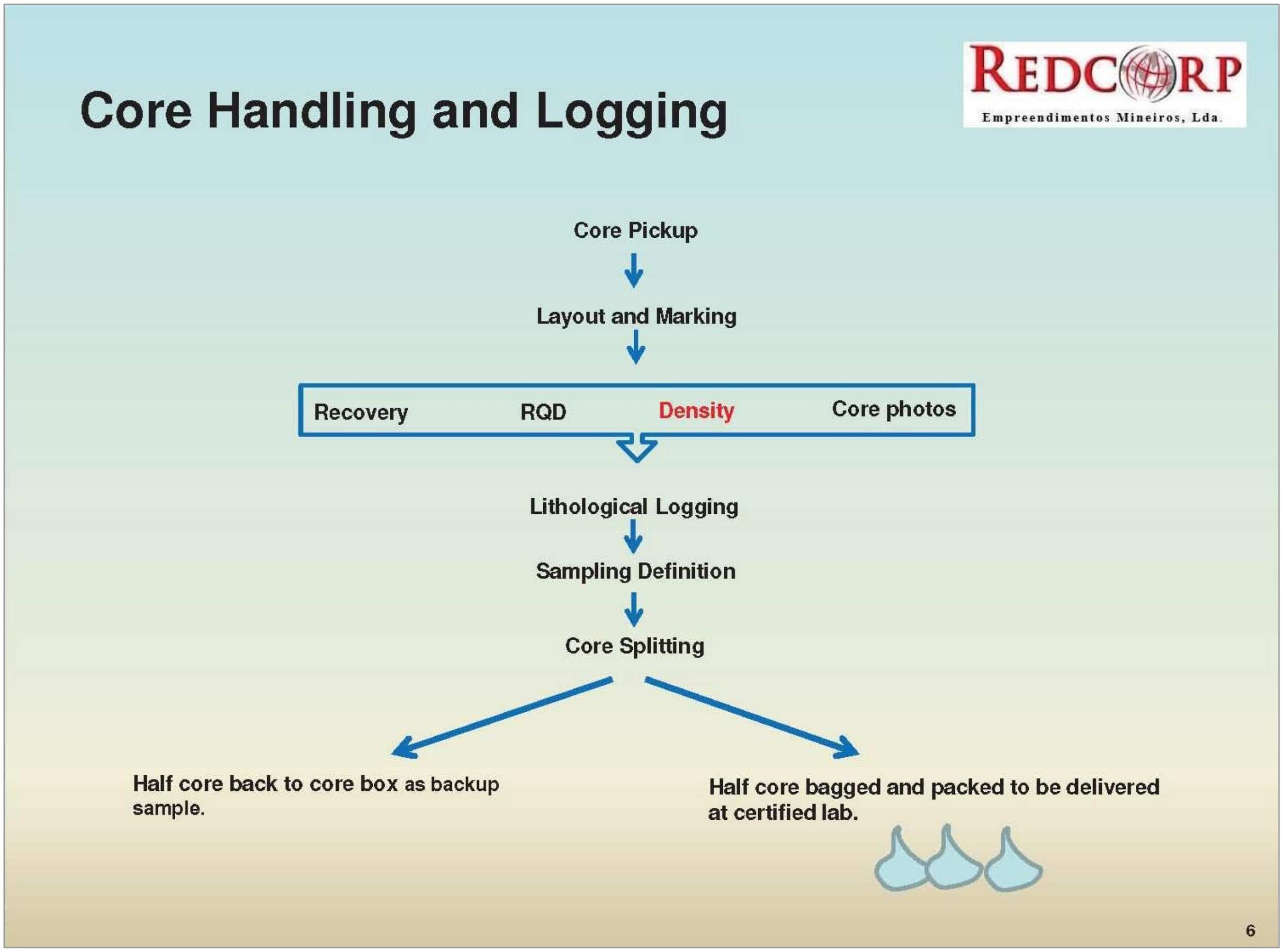
Most of the drilling on the LS Project was conducted using HQ-size core, yielding good core recoveries, and in turn, representative samples. Micon examined diamond drill cores from 6 holes of the North deposit and three holes of the South deposit. All the major mineralization and alteration styles described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

In a number of 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.

Figure 12.2 Redcorp protocols for core handling and logging



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

12.2 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 principle 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 Table 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_C&S	150	2.88

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, thus the database of the LS Project is considered to have been generated in a credible manner and to be sufficiently error-free to support Mineral Resource estimates.

13 Mineral processing and metallurgical testing

Several scoping level metallurgical and mineralogical studies have been undertaken for the LS Project. The following documents were reviewed to prepare this section of the PEA report:

- Empresa de Perfurção e Desenvolvimento Mineiro, S.A. (EPDM) Mineralogical Study Reports, 14 and 23 February 2019.
- Scoping Metallurgical Study on Lagoa Salgada Deposit (Grinding Solutions Ltd), 22 May 2019.
- Lagoa Salgada Metallurgical Testwork (Micon), 14 June 2019.
- Wardell Armstrong memorandums on Falcon and Coarse Ore Bottle Roll Tests.

13.1 Historical metallurgical testwork

The historical test work consisted mostly of mineralogical studies. Anamet Services undertook test work in 1995 on one heavily mineralized sample composited from a continuous interval of approximately 8 m taken from drillhole LS-22. This sample does not constitute a representative sample of the mineralogy or potential recoveries for the deposit.

In 2017, Redcorp arranged for a comprehensive mineralogical definition program to be conducted at the EPDM laboratory in Portugal. The work was completed using Mineral Liberation Analysis (MLA) equipment to provide modal mineralogy, mineral association, and liberation data on selected samples from the LS Project.

The samples submitted were from six drillholes and were selected from the GO, MS, and Stringer domains of the North deposit, and from the Stockwork domain of the South deposit. The information for each sample is summarized in Table 13.1.

The samples submitted were assay rejects and had been pulverized to 95% passing 75 micrometres (μm) prior to mineralogical evaluation. A comprehensive mineralogical characterization of each sample was carried out to give modal mineralogy, liberation data, and mineral associations. A summary of the modal mineralogy is given in Table 13.2.

Table 13.1 Redcorp samples sent for comprehensive mineralogical definition

Drillhole ID	Sample No.	From	To	Interval (m)	Domain
LS_MS_01	MS_01/159	159	160	1.0	GO
LS_MS_01	MS_01/169	169	170	1.0	MS
LS_MS_01	MS_01/227	227	228	1.0	MS
LS_MS_01	MS_01/238	238	239	1.0	MS
LS_MS_02	MS_02/171	171	172	1.0	MS
LS_MS_02	MS_02/208	159	160	1.0	MS
LS_MS_02	MS_02/219	219	220	1.0	MS
LS_MS_03	MS_03/177	177	178	1.0	MS
LS_MS_03	MS_03/179	179	180	1.0	MS
LS_MS_03	MS_03/231	231	232	1.0	Stringer
LS_MS_03	MS_03/249	249	250	1.0	Stringer
LS_ST_01	ST_01/200	200	201	1.0	Stockwork
LS_ST_01	ST_01/211	211	212	1.0	Stockwork
LS_ST_02	ST_02/329	229	229	1.0	Stockwork
LS_ST_04	ST_04/385	385	386	1.0	Stockwork

Source: Table taken from 2018 Technical Report.

Table 13.2 Modal mineralogy 2017 samples

Modal data	Weight (%)														
	MS-01/159	**MS-01/169**	**MS-01/227**	**MS-01/238**	**MS-02/171**	**MS-02/208**	**MS-02/219**	**MS-03/177**	**MS-03/179**	**MS-03/231**	**MS-03/249**	**ST-01/200**	**ST-01/211**	**ST-02/329**	**ST-04/385**
Domain	**GO**	**MS**								**Stringer**		**Stockwork**			
Sphalerite	0.0	4.0	5.8	22.7	3.6	20.1	8.5	3.9	3.1	2.1	4.6	10.1	19.7	16.6	6.0
Chalcopyrite	0.0	0.2	0.2	0.1	0.2	0.1	0.2	0.0	0.0	7.8	2.7	2.2	1.3	15.3	5.4
Galena	0.2	3.0	9.8	11.3	4.3	17.2	4.2	5.1	3.9	0.2	0.0	4.9	3.1	12.1	5.0
Pyrite	0.7	78.6	78.5	63.0	84.5	54.5	81.4	87.6	86.3	25.1	9.1	26.0	4.1	4.9	5.7
Tetrahedrite	0.0	0.0	0.1	0.1	0.0	0.0	0.1	0.1	0.0	0.1	0.0	0.1	0.0	0.0	0.0
Enargite		0.1	0.1	0.1	0.0	0.0	0.0	0.0	0.0	0.2	1.3	0.0	0.0	0.0	0.0
Arsenopyrite		10.9	1.8	0.1	5.4	6.5	2.7	1.1	3.7	0.3	0.0	0.0	0.0	0.0	0.0
Chalcocite	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.5	1.2	1.2	0.9	0.1	0.0
Quartz	37.8	2.1	0.6	1.0	0.6	0.3	0.6	0.7	1.2	1.7	14.1	4.5	3.0	17.6	29.7
Smithsonite	6.0	0.0	0.0	0.0	0.0	0.1	0.0	0.0	0.0	0.5	0.6	23.4	3.2	3.2	2.3
Dolomite		0.0	0.1	0.0	0.0	0.0	0.0	0.0	0.1	0.1	0.1	0.2	0.4	2.8	20.0
Chlorites	12.8	0.2	0.2	0.4	0.4	0.4	0.7	0.2	0.4	59.8	62.7	25.5	61.0	21.8	22.7
Phyllosilicates	0.9	0.2	0.4	0.5	0.4	0.2	0.8	0.4	0.6	0.3	2.6	0.8	1.4	4.7	1.5
Other Silicates	0.5	0.1	2.1	0.2	0.2	0.1	0.5	0.5	0.4	0.2	0.3	0.4	0.9	0.5	0.9
Rutile	0.1	0.0	0.0	0.0	0.1	0.0	0.0	0.0	0.0	0.9	0.3	0.2	0.6	0.2	0.4
Cassiterite	2.3	0.3	0.1	0.3	0.1	0.3	0.2	0.1	0.2	0.0	0.1	0.0	0.0	0.0	0.0
Goethite	37.3	0.0	0.0	0.3	0.0	0.0	0.0	0.0	0.1	0.1	0.1	0.0	0.1	0.0	0.1
Phosphates	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.3	0.3	0.1	0.1
Others	1.3	0.2	0.2	0.0	0.1	0.1	0.0	0.1	0.1	0.0	0.0	0.1	0.0	0.1	0.2
Total	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**

Note: MS = massive sulphide and ST = stockwork.
Source: Table taken from 2018 Technical Report.

13.2 2019 Ascendant / Redcorp metallurgical testwork

Grinding Solutions Ltd. (GSL) completed a scoping level flotation test work on four samples from the LS deposit. The composite samples consisted of fresh and oxidized material from two ore types, namely, MS and stockwork. The head analyses of the composites are given in Table 13.3.

Table 13.3 Head analyses of composite samples

Element (%)	Samples			
	Fresh MS	Oxidized MS	Fresh ST	Oxidized ST
Cu	0.33	0.25	2.01	2.16
Pb	3.25	5.46	5.16	5.19
Zn	3.13	6.46	7.79	9.01
Fe	36.17	33.80	14.81	13.69
Sn	0.16	0.19	0.04	0.05
S_{Total}	48.21	46.68	9.78	12.11
$S_{sulphide}$	48.05	46.49	9.74	12.06
$S_{sulphate}$	0.16	0.19	0.04	0.05

Majority of the test work, consisting of mineralogy, open-circuit, and locked-cycle flotation tests (LCT), were performed on the fresh samples. A few open-circuit flotation tests were performed on the oxidized material.

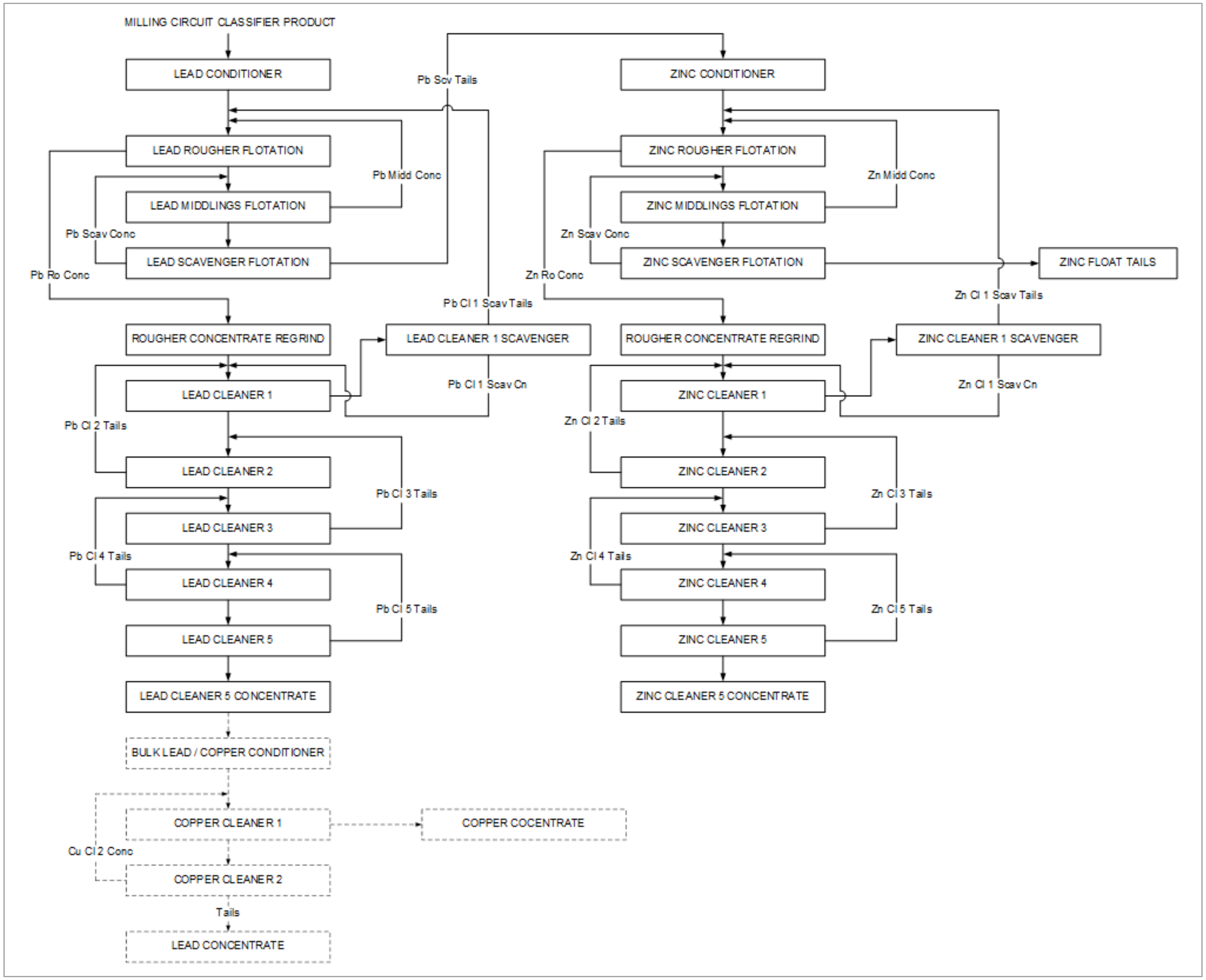
13.2.1 Massive sulphide

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.

LCT was performed on fresh composite using the flowsheet given in Figure 13.1 and Figure 13.2 following open-circuit rougher and cleaner flotation tests. The test conditions are given in Table 13.4 and Table 13.5 and the results are summarized in Table 13.6. The detailed concentrate analyses are given in Table 13.7. The test results indicated the following:

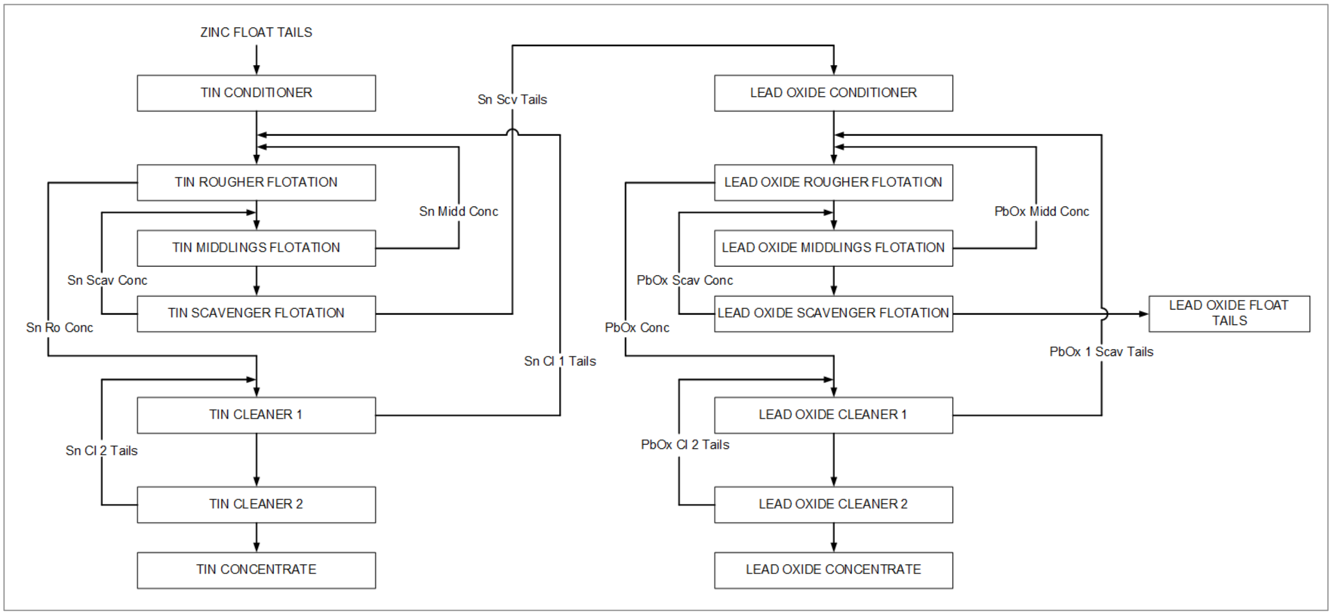
- The final lead concentrate assaying 33.4% Pb, 10.8% Zn, 313.8 g/t Ag, and 1.7 g/t Au, recovered 33.4% of the lead.
- The final zinc concentrate, assaying 42.4% Zn, 5.3% Pb, 144.1 g/t Ag, and 0.8 g/t Au, recovered 60.7% of the zinc.
- The final tailings contained 76.6% of silver, 89.9% of gold, 93.8% of tin, 76.9% of copper, 55.7% of lead, and 27.1% of zinc.
- The cassiterite, which reported to the zinc tailing, was well liberated and can possibly be concentrated by flotation. However, the tailing is too fine (P_{80} of 29 µm) for the gravity concentration.
- The smelter penalty analyses indicated elevated levels of arsenic, antimony, and mercury in the concentrates.

Figure 13.1 MS lead and zinc flotation circuit



Source: Ascendant.

Figure 13.2 MS tin and lead oxide flotation circuit



Source: Ascendant.

Table 13.4 Flotation reagent conditions for MS LCT

	Grind size	Aeration	pH	SMBS	$ZnSO_4$	A3418A	DF250	Lime	$CuSO_4$	SIPX
Pb Rougher	D_{80} 29 µm	22 min	8-9			45	28	1,700		
Pb Regrind	D_{80} 5.5 µm		8-9	250	250					
Pb Cleaners			8-9			3.5	2.5	50		
Zn Rougher			11					3,000	300	105
Zn Regrind	D_{80} 8.5 µm		11							
Zn Cleaners			11					4,100		11
Total				**250**	**250**	**48.5**	**30.5**	**8,850**	**300**	**116**

Note: Reagent dosages are in g/t.

Table 13.5 Flotation residence times for the LCT

Stream	Time (min)	Stream	Time (min)
Pb Rougher	14	Zn Rougher	16
Pb Cleaner 1	20	Zn Cleaner 1	19.5
Pb Cl. Scavenger	9.5	Zn Cl. Scavenger	5
Pb Cleaner 2	11.5	Zn Cleaner 2	14
Pb Cleaner 3	8	Zn Cleaner 3	12
Pb Cleaner 4	9	Zn Cleaner 4	7
Pb Cleaner 5	6	Zn Cleaner 5	7

Table 13.6 Locked-cycle MS test balance

Selected cycles				Assay (%, g/t)								Distribution (%)							
	Cycle	**Wt (g)**	**Wt (%)**	**Cu**	**Pb**	**Zn**	**Fe**	**S**	**Sn**	**Au**	**Ag**	**Cu**	**Pb**	**Zn**	**Fe**	**S**	**Sn**	**Au**	**Ag**
Pb Conc	3-6	32.2	3.2	1.1	33.4	10.8	18.1	33.0	0.1	1.7	313.8	10.1	36.9	12.2	1.8	2.4	2.2	6.3	14.8
Zn Conc	3-6	40.7	4.1	1.1	5.3	42.4	8.6	33.7	0.2	0.8	144.1	13.1	7.3	60.7	1.1	3.0	4.0	3.9	8.6
Zn Cl Sc Tail / Ro Tail	3-6	894.4	89.2	0.3	1.8	0.9	35.6	47.8	0.2	0.9	58.4	76.9	55.7	27.1	97.2	94.6	93.8	89.9	76.6
Total		**967.3**	**100.0**	**0.3**	**2.9**	**2.8**	**32.7**	**45.0**	**0.2**	**0.9**	**68.0**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**

Table 13.7 Detailed concentrate analysis for MS LCT

Element (%)	Pb concentrate	Zn concentrate
Cu	1.20	1.03
Pb	31.80	5.14
Zn	10.59	41.23
As	1.23	1.49
SiO_2	0.21	0.22
Al	0.14	0.18
Fe	18.19	10.42
Cr	0.02	0.03
Ca	0.10	0.49
S	32.80	33.61
Cd	0.064	0.239
Ni	0.006	0.004
Co	0.003	0.002
Mn	0.014	0.029
Bi	0.015	0.003
Sb	0.130	0.066
Hg	0.107	0.225
Te	0.002	0.001
Se	0.039	0.008
Mg	<0.001	0.02
Au, g/t	1.26	0.65
Ag, g/t	337.5	156.9

13.2.2 Stockwork

The dominant phase in the Stockwork sample was mica group minerals. The main ore 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.

Following the open-cycle rougher and cleaner flotation tests, an LCT was performed on fresh composites using the flowsheet given in Figure 13.3 and Figure 13.4. The test conditions are given in Table 13.8 and Table 13.9 and the results are summarized in Table 13.10. the detailed concentrate analyses are given in Table 13.11. The test results indicate the following:

- The copper concentrate, assaying 24.7% Cu, 16.4% Pb, 12.1% Zn, 321.6 g/t Ag, and 0.4 g/t Au, recovered 61.6% of copper.
- The lead concentrate, assaying 49% Pb, 6.9% Cu, 15.8% Zn, 460.9 g/t Ag, and 0.2 g/t Au, recovered 74.7% of the lead and 55.7% of silver.
- Majority of the gold (>70%) reported to the final tailings.
- The deportment of zinc in all the products is likely due to solubilization of copper minerals in the grinding process which causes pre-activation.
- The concentrates did not appear to have any deleterious elements.

Table 13.8 Flotation reagent conditions for Stockwork LCT

	Grind size	Aeration	pH	SMBS	$ZnSO_4$	Na_2S	NaSil	A3418A	MIBC	Lime	$CuSO_4$	SIPX
Cu Rougher	D_{80} 37 µm	5 min	11.5		250	250	20	13	18	1,300		
Cu Regrind	D_{80} 20 µm		8	500	250	250						
Cu Cleaners			8					11	8			
Pb Rougher			9	2,250				30	8			
Pb Regrind	D_{80} 22 µm		8-9	500	500							
Pb Cleaners			8-9					9	8			
Zn Rougher			11						8	1,700	450	40
Zn Regrind	D_{80} 25 µm		11							500		
Zn Cleaners			11.5						10	150		7
Total				**3,250**	**1000**	**500**	**20**	**63**	**60**	**3,650**	**450**	**47**

Note: Reagent Dosages are in g/t.

Table 13.9 Flotation residence times for the Stockwork LCT

Stream	Time (min)	Stream	Time (min)	Stream	Time (min)
Cu Rougher	10.5	Pb Rougher	14	Zn Rougher	19
Cu Cleaner 1	17	Pb Cleaner 1	11	Zn Cleaner 1	12
Cu Cl. Scav.	4	Pb Cl. Scav.	3	Zn Cl. Scav.	5
Cu Cleaner 2	12	Pb Cleaner 2	10	Zn Cleaner 2	11
Cu Cleaner 3	10	Pb Cleaner 3	7	Zn Cleaner 3	10

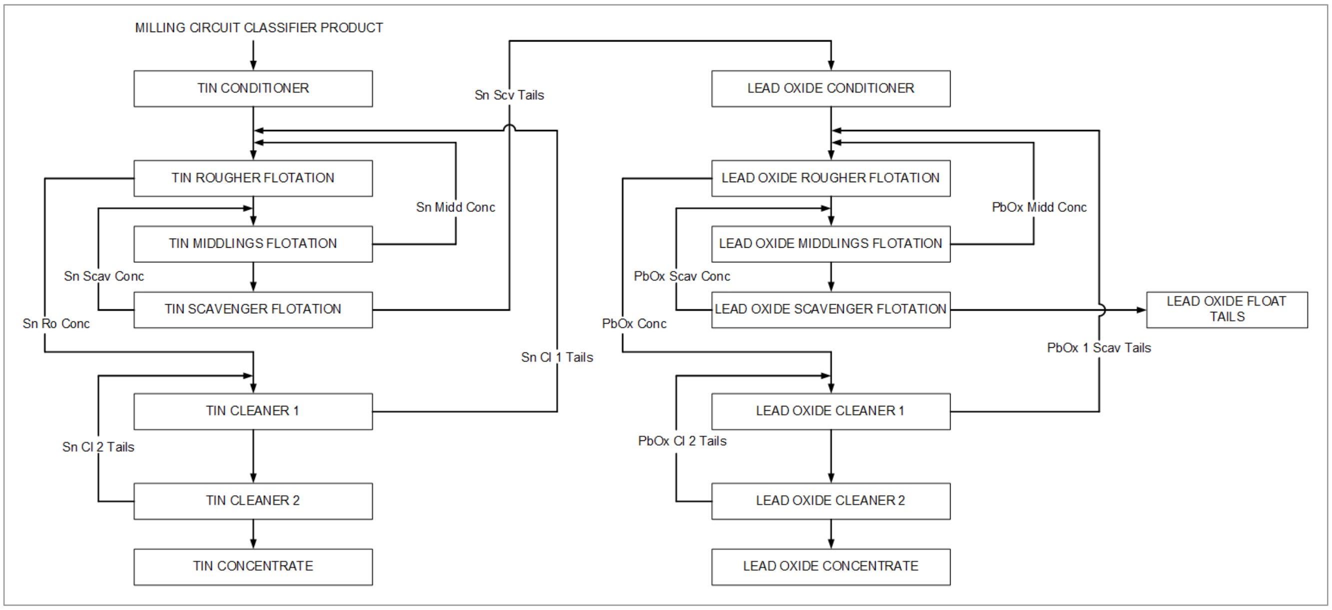
Table 13.10 Locked-cycle Stockwork test balance

Selected cycles				Assay (%, g/t)							Distribution (%)						
Product	**Cycle**	**Mass (g)**	**Wt (%)**	**Cu**	**Pb**	**Zn**	**Fe**	**S**	**Au**	**Ag**	**Cu**	**Pb**	**Zn**	**Fe**	**S**	**Au**	**Ag**
Cu Conc	4-6	43.7	4.5	24.7	16.4	12.1	15.7	28.8	0.4	321.6	61.6	14.6	8.0	4.8	14.0	11.4	22.6
Pb Conc	4-6	75.1	7.8	6.9	49.0	15.8	5.8	22.4	0.2	460.9	29.6	74.7	18.0	3.1	18.8	10.0	55.7
Zn Conc	4-6	74.9	7.8	1.0	1.8	53.2	3.7	29.4	0.1	84.0	4.3	2.7	60.5	1.9	24.6	6.2	10.1
Zn Cl Sc TI	4-6	61.7	6.4	0.3	1.2	3.5	17.6	7.7	0.2	27.6	1.1	1.5	3.3	7.6	5.3	8.9	2.7
Zn Ro Tail	4-6	708.4	73.5	0.1	0.4	0.9	16.6	4.7	0.1	7.7	3.4	6.4	10.2	82.6	37.3	63.5	8.8
Total	**4-6**	**963.8**	**100.0**	**1.8**	**5.1**	**6.8**	**14.8**	**9.3**	**0.2**	**64.4**	**100**	**100**	**100**	**100**	**100**	**100**	**100**

Table 13.11 Detailed concentrate analysis for Stockwork LCT

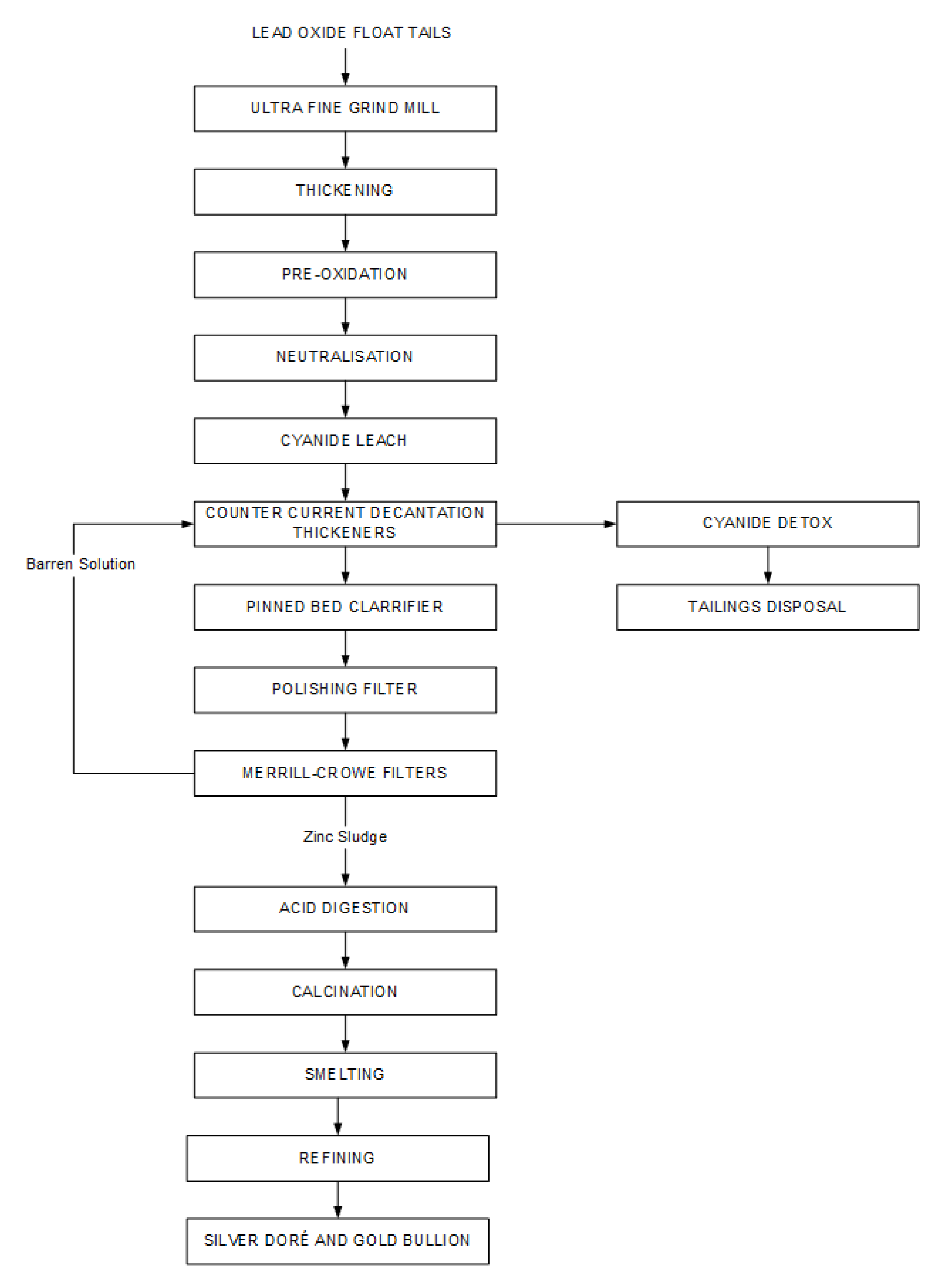
Element (%)	Cu concentrate	Pb concentrate	Zn concentrate
Cu	23.88	6.89	1.05
Pb	15.21	45.71	1.75
Zn	11.96	15.77	53.95
As	0.040	0.017	0.012
Cd	0.065	0.0925	0.3417
Ni	<0.001	<0.001	0.004
Co	0.007	0.005	0.010
Mn	0.046	0.045	0.110
Bi	0.0152	0.0547	0.0018
Sb	0.0241	0.0309	0.0039
Hg	0.0049	0.0069	0.0278
Te	<0.0001	<0.0001	<0.0001
Se	0.0434	0.1538	0.0063
SiO_2	1.10	0.48	2.81
Al	0.34	0.12	0.81
Fe	15.71	6.37	4.27
Mg	0.22	0.09	0.38
Cr	0.01	0.01	0.03
Ca	0.20	0.14	0.37
S	29.16	23.36	29.50
Au, g/t	0.34	0.16	0.12
Ag, g/t	304.0	432.9	79.2

Figure 13.3 GO tin and lead oxide flotation circuit



Source: Ascendant.

Figure 13.4 Silver and gold leach circuit



Source: Ascendant.

13.2.3 Cyanide leaching of GO sample

Bottle roll cyanidation leach tests were run on GO sample assaying 0.095% Cu, 2.88% Pb, 0.97% Zn, 48.05% Fe, 0.57 g/t Au, 11.3 g/t Ag, and 0.76% S_{Total}. The tests were run for 72 hrs at varying grind sizes. The test results are summarized for the leach tests in Table 13.12. The test results indicate the following:

- Both gold and silver extraction are relatively fast. The leach recoveries for 24 hrs leach time indicate ±80% of gold extraction and over 90% of silver extraction at P_{80} of ±45 µm.
- The lime consumption was reasonable at 1.0 kg to 1.3 kilograms per tonne (kg/t) and the maximum sodium cyanide (NaCN) consumption will be ±2 kg/t. The values reported in Table 13.12 are for 72 hrs of leach time.

Table 13.12 Bottle roll cyanide leach test results for GO sample

Test No.	P_{80} µm	Extraction, % (24 hrs)			Cal head (g/t)			Consumption, kg/t (72 hrs)	
		Au	Ag	Cu	Au	Ag	Cu	NaCN	Lime
LT-1	150	73.6	61.2	3.6	0.57	11.25	877.9	2.71	1.10
LT-2	106	83.1	56.9	3.4	0.58	11.25	891.4	2.59	1.15
LT-3	75	82.6	66.7	4.0	0.57	11.05	920.5	2.34	1.33
LT-4	45	78.1	99.4	97.8	0.60	7.73	35.3	2.52	1.29
LT-5	-	80.6	98.4	98.3	0.51	28.48	578.1	3.56	1.08

Note: LT-5 is for flotation tailing sample.

13.2.4 Summary of metallurgical results

Insufficient metallurgical test work has been completed to date to allow accurate forecast of metallurgical performance for the deposit.

13.3 Recommendations

The scoping level study indicates that it is possible to recover copper, lead, zinc, gold, and silver from the deposit. No test work has been undertaken to determine the viability of recovering tin as a by-product so far.

The authors recommend that composite samples (MS, Stockwork, GO, and Central Zone) representing each ore type of the deposit be collected and a systematic process development study be undertaken for the next level of study.

14 Mineral Resource estimates

14.1 Exploratory data analysis

14.1.1 Database description

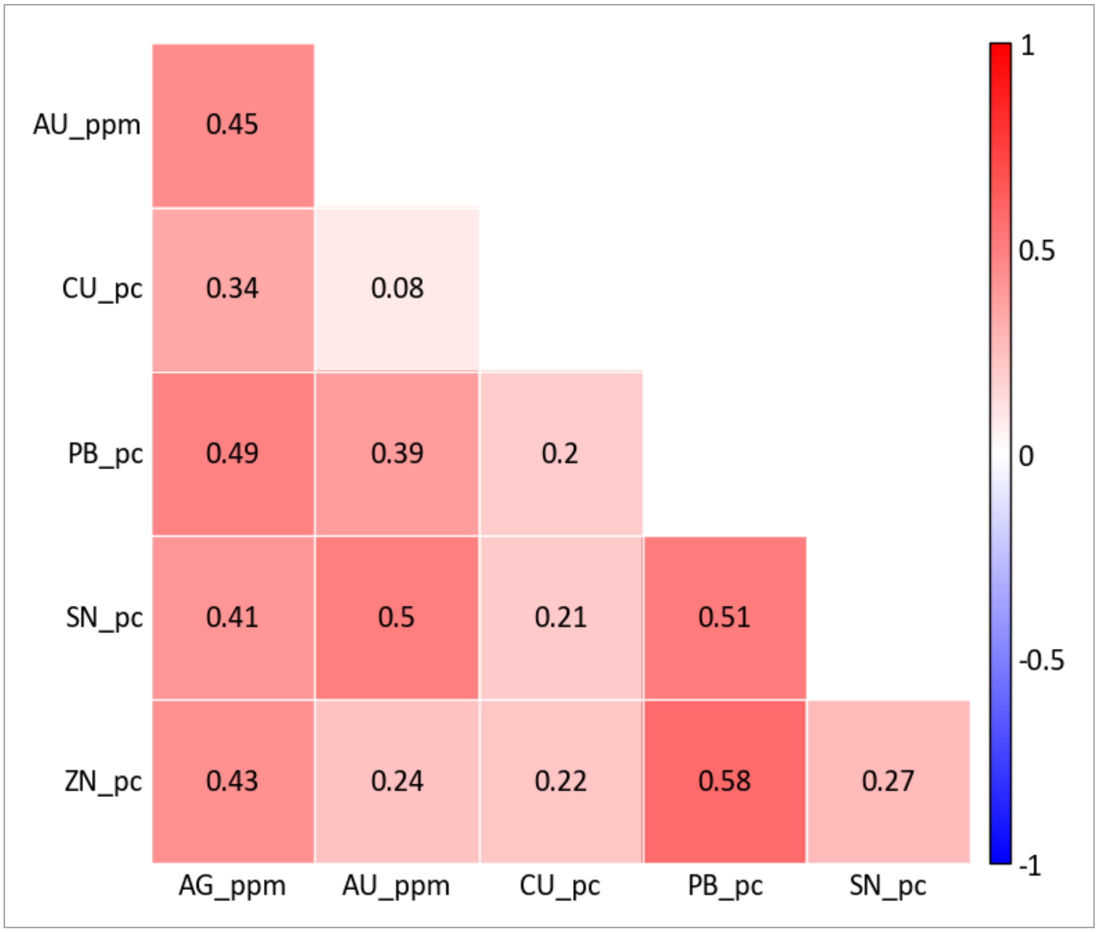
The LS Project deposits have been tested by diamond drilling over a cumulative strike length of approximately 1.6 km and down to a vertical depth of about 600 m. The resource database is derived from 76 surface diamond drillholes, all of which were utilized in the resource estimation. Original assay 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 Project 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.

14.2 Overview of estimation methodology

Following the completion of the database validation as outlined in Section 12 above, Micon has estimated the LS Project 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 the ED50 co-ordinate reference system (CRS) and projection to UTM Zone 37N.

14.3 Geological interpretation

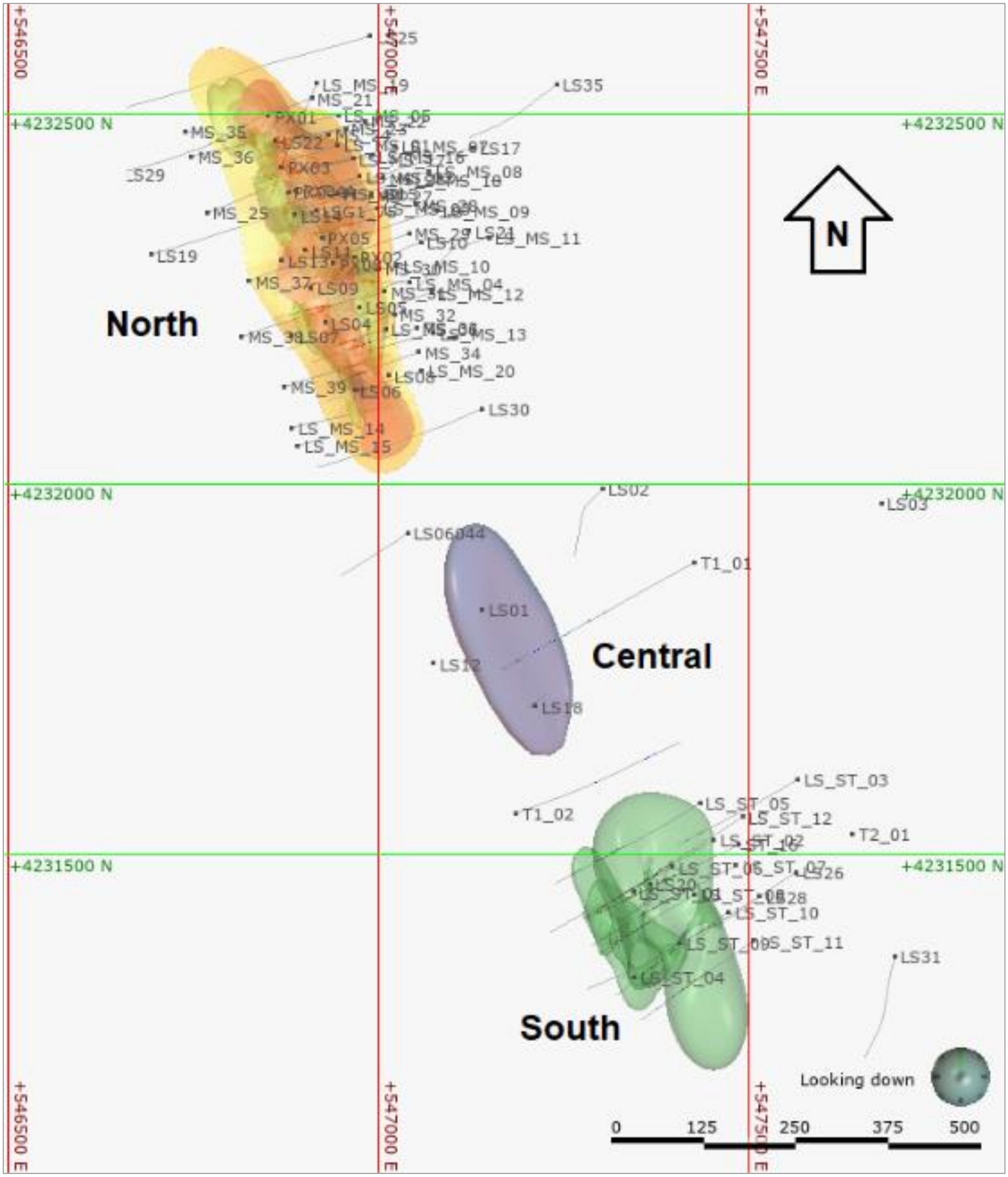
The mineralization extends continuously beneath Tertiary cover rocks over the entire drilled strike length of about 1.6 km. However, three deposits are recognized; these are the North deposit (formerly LS-1 deposit), the Central deposit and the South deposit (formerly LS-1 Central deposit). The respective locations are shown in Figure 14.2.

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

In contrast, the Central and South deposits are characterized by stringer / fissure / stockwork type mineralization.

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 Map showing the location of the currently known deposits on the LS Project



Source: Micon.

14.4 Selection and modelling of estimation domains

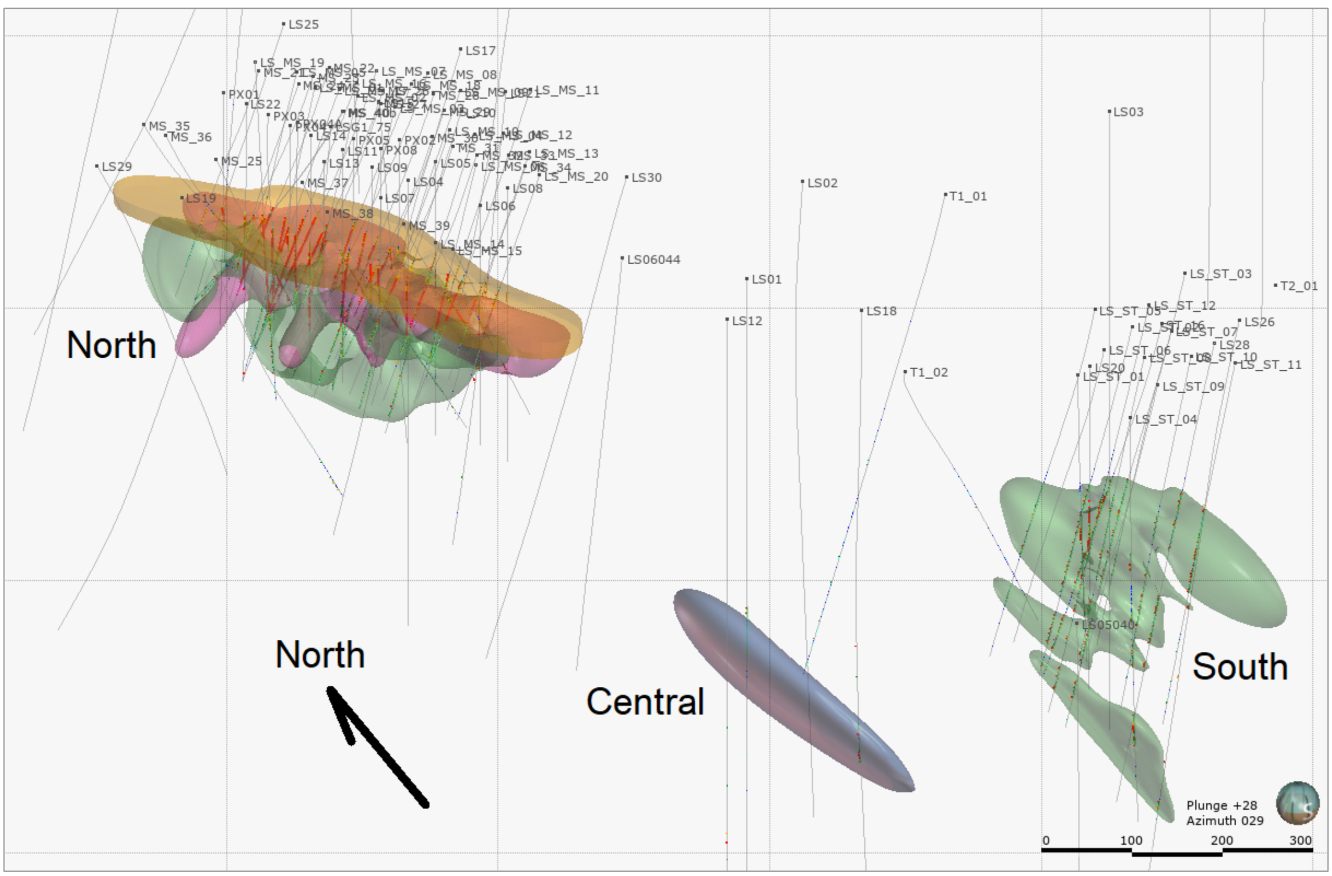
Micon's estimation domain selection criterion is based purely on geology for the GO and MS domains and on the zinc equivalent (ZnEq) threshold value for the stringer / fissure / stockwork domains. To obtain the ZnEq% threshold value for modelling, the threshold grade for each metal in the stringer zones 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 South and Central was established as 0.95% ZnEq.

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 deposit domains



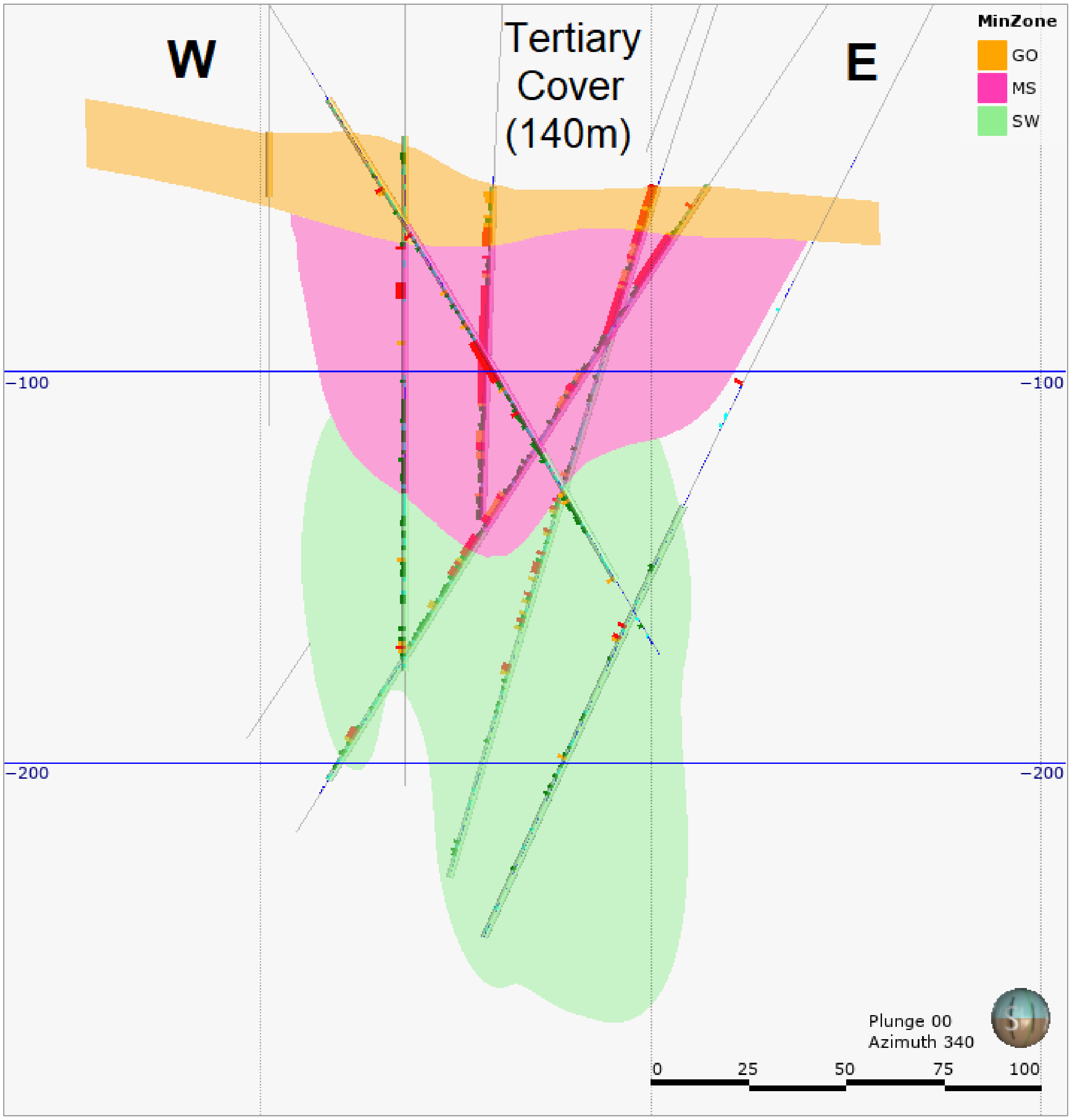
Source: Micon 2019.

In summary, the selected domains are as follows:

- GO (brown / yellow) – North deposit.
- MS (purple) – North deposit.
- Stringer zone (green) – North deposit.
- Stringer / fissure / stockwork (grey) – Central deposit.
- Stringer / fissure / stockwork (green) – South deposit.

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.

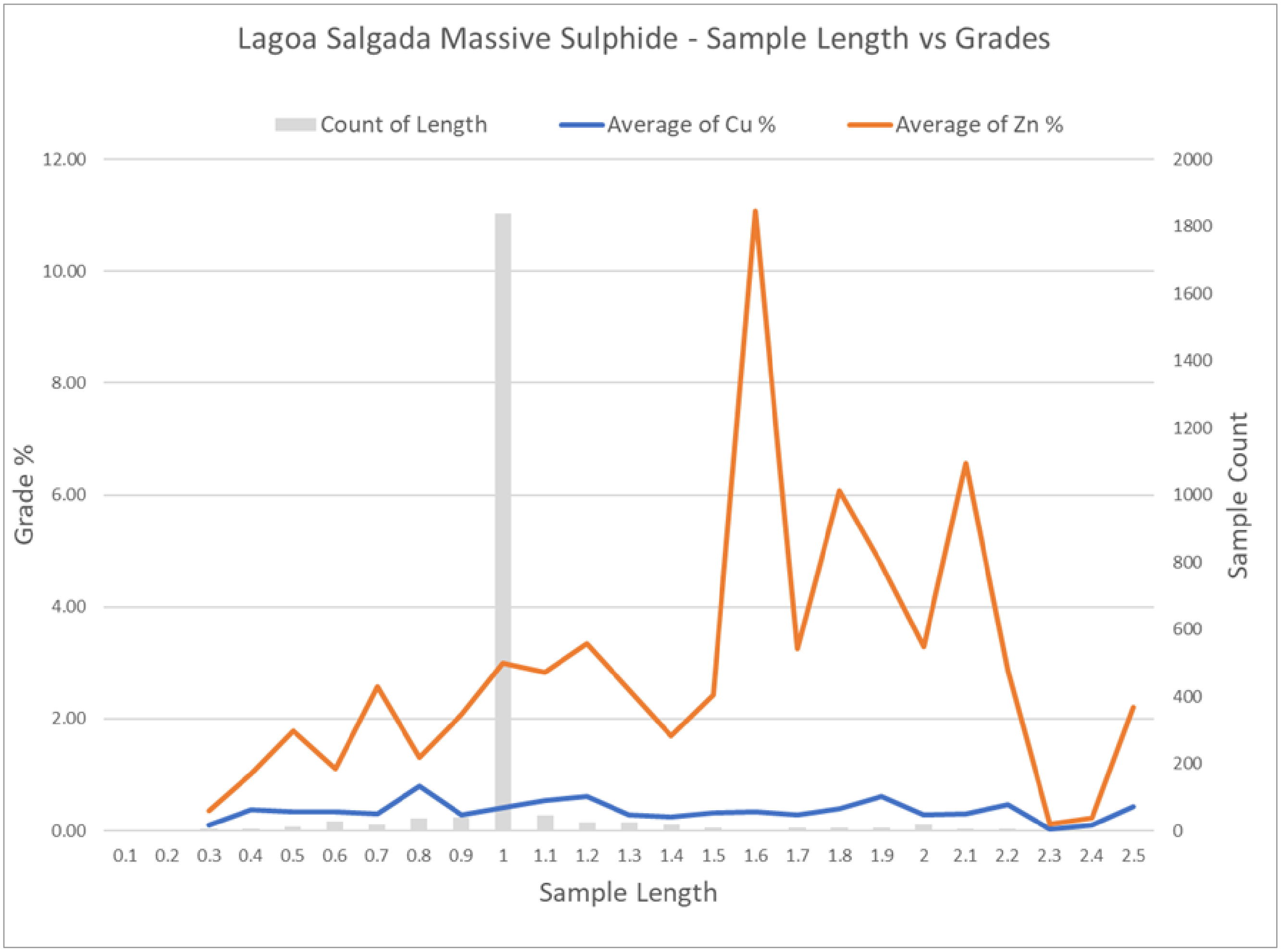
The stockwork domains of the Central and South deposits, although currently separated, may eventually merge into one body with further infill drilling.

14.5 Grade capping, compositing, statistics, and variography

14.5.1 Grade capping and compositing

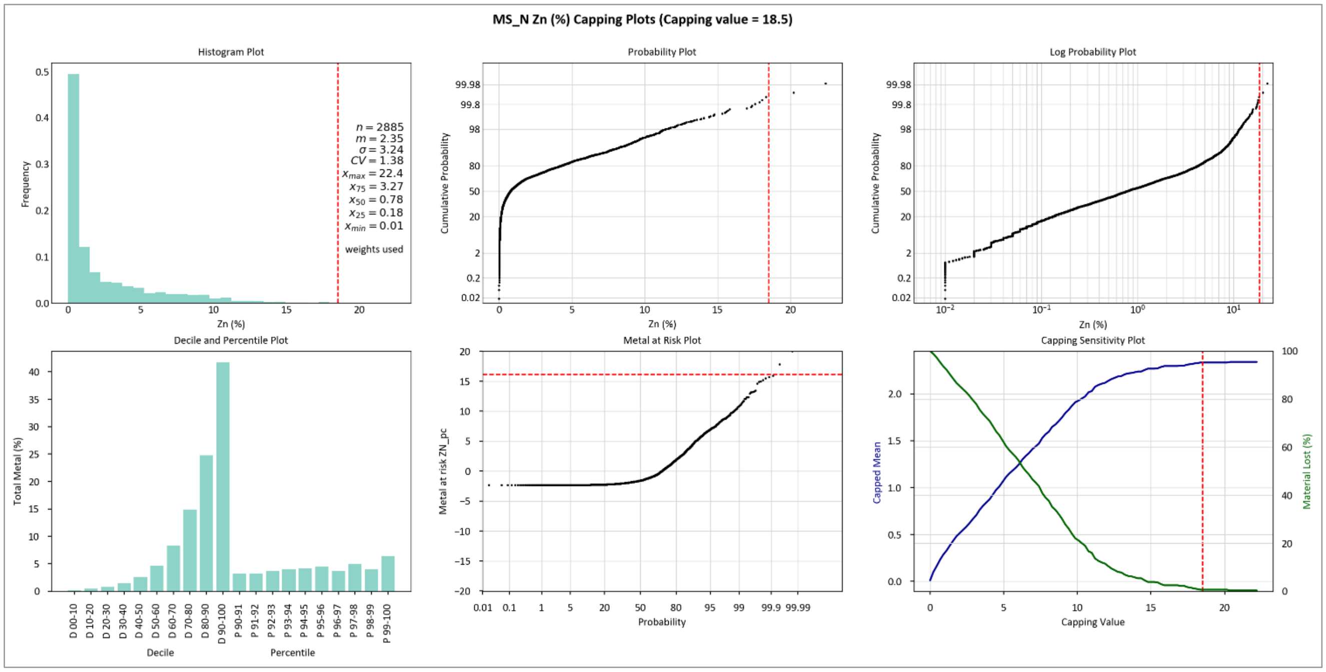
Micon investigated the relationship between sample length and grade and established that a considerable number of high grades were associated with lengths greater than the mode of the sample lengths of 1 m as illustrated in Figure 14.5. Thus, the determination of grade capping threshold values was conducted 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 mineralized domain) are shown in Figure 14.6 to Figure 14.11. Grade capping values are indicated in red on the plots for each element. The same procedure was followed for the other domains.

Figure 14.5 Grade versus sample length in the MS domain



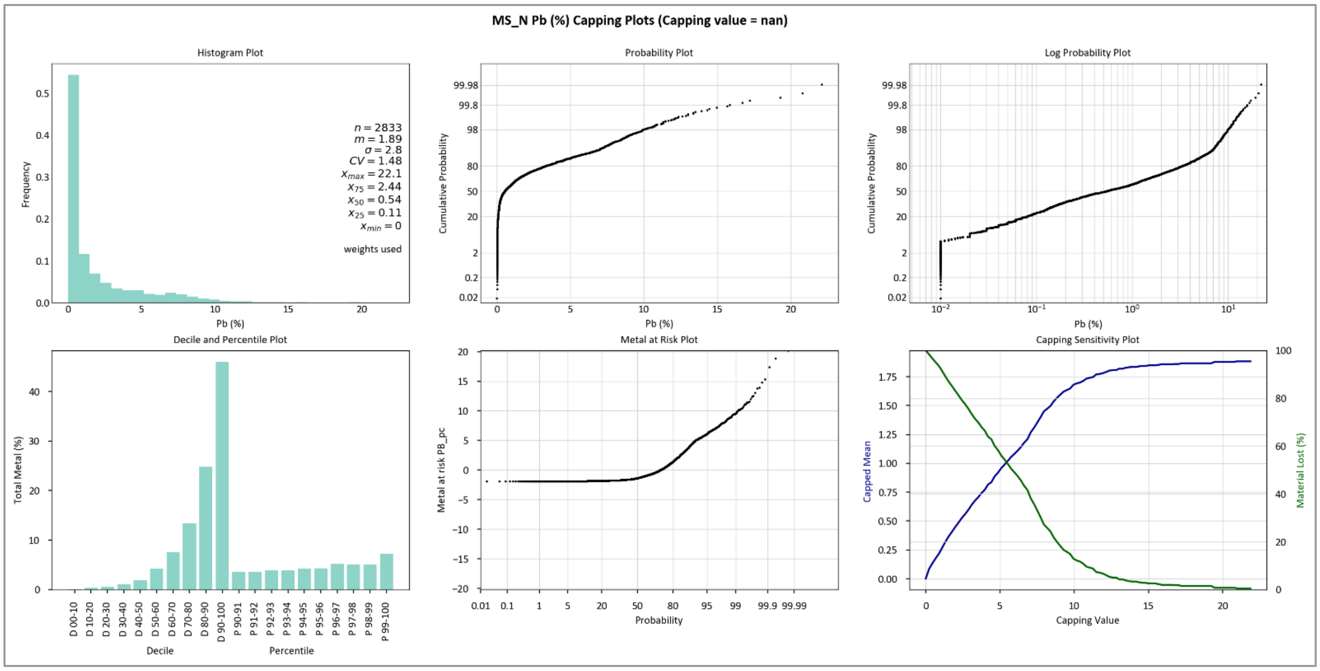
Source: Micon.

Figure 14.6 Domain MS summary statistics and probability / log-probability plot for Zn



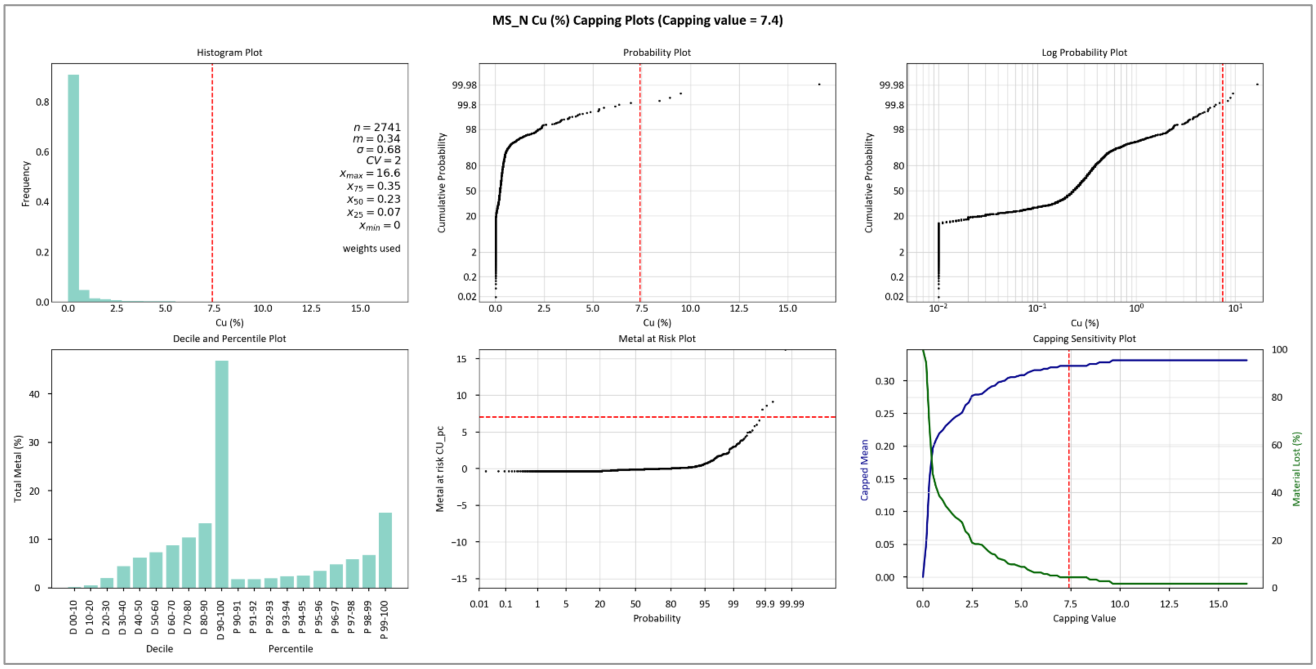
Source: Micon.

Figure 14.7 Domain MS summary statistics and log-probability plot for Pb



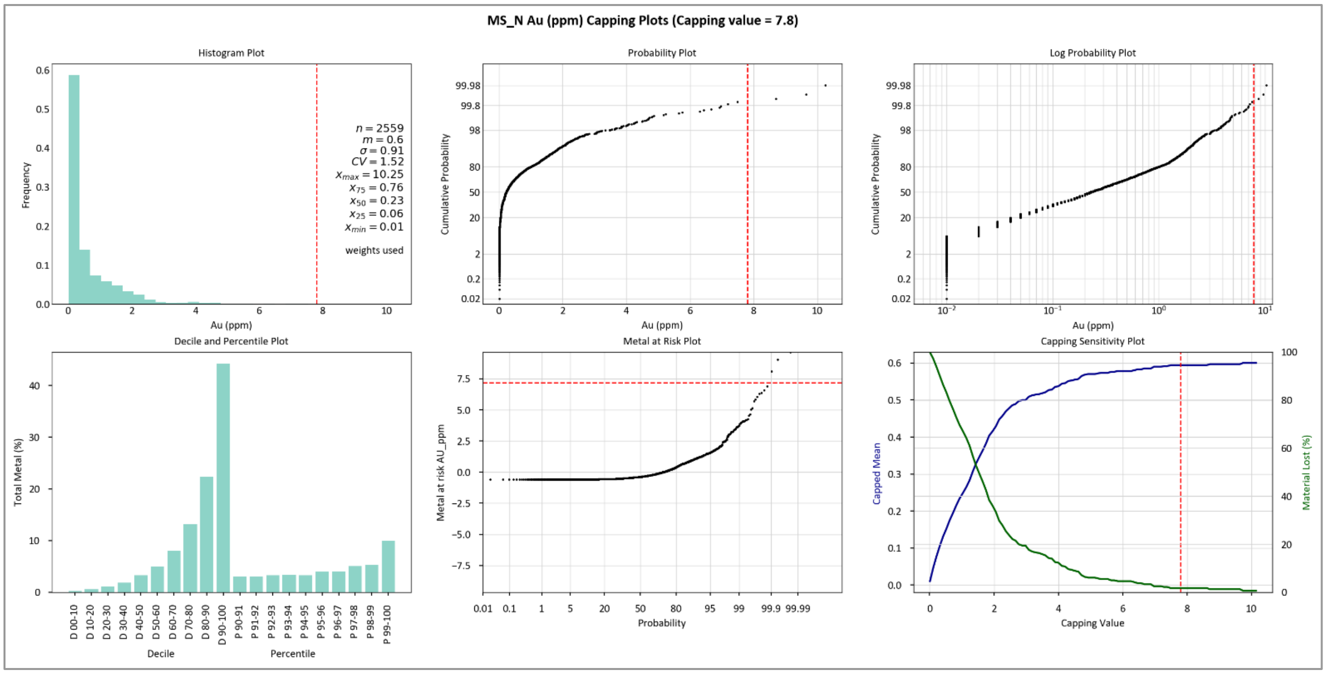
Source: Micon.

Figure 14.8 Domain MS summary statistics and log-probability plot for Cu



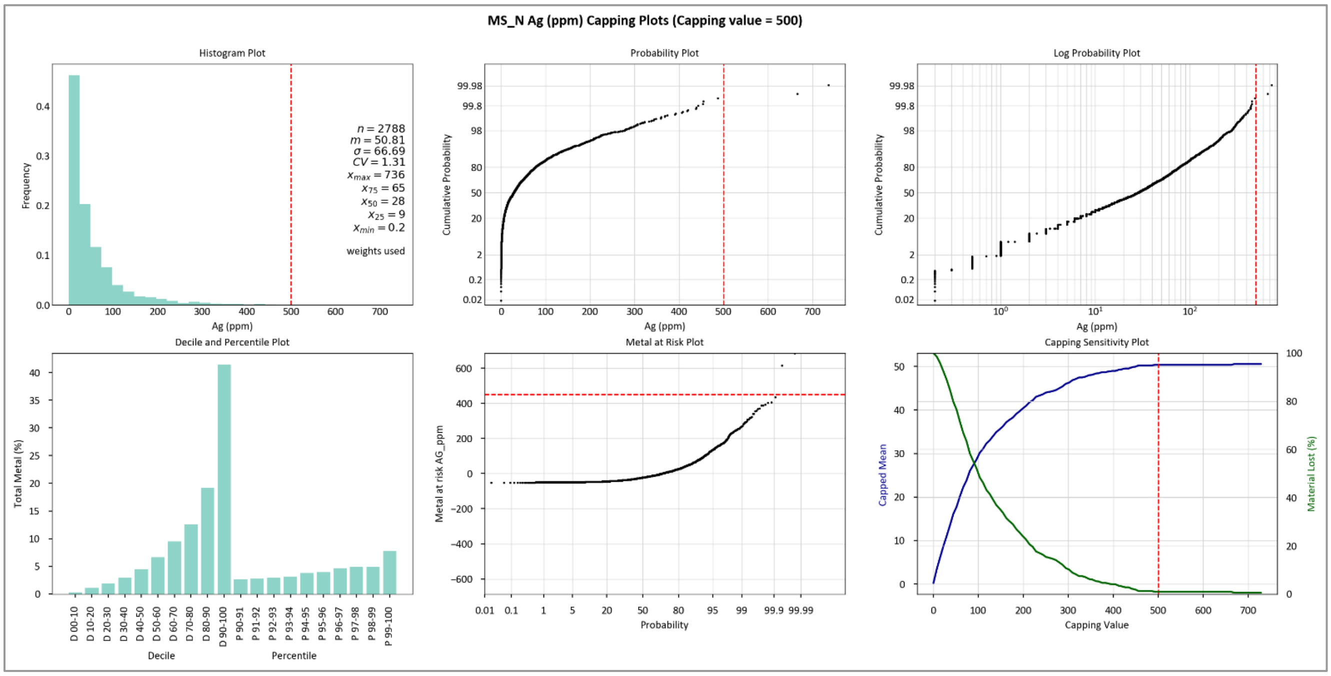
Source: Micon.

Figure 14.9 Domain MS summary statistics and log-probability plot for Au



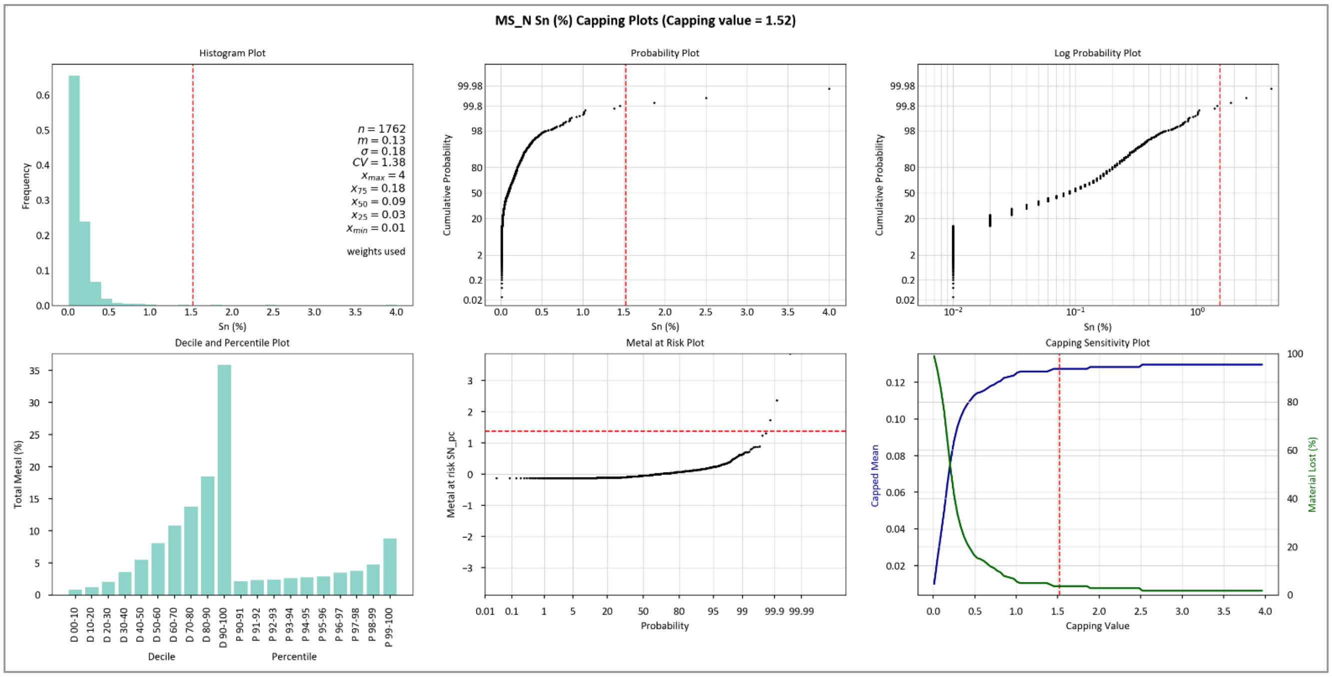
Source: Micon.

Figure 14.10 Domain MS summary statistics and log-probability plot for Ag



Source: Micon.

Figure 14.11 Domain MS summary statistics and log-probability plot for Sn



Source: Micon.

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

Precision in spatial analysis / variography is directly proportional to the quality of the sampling pattern. Due to the subvertical / steeply dipping nature of the LS Project deposits, all drillholes from surface intersect the mineralization at high oblique angles, culminating in an unrepresentative sampling pattern. Thus, variographic results are not truly representative of the spatial continuity / distribution patterns of the mineralization at the LS Project. Nonetheless, Micon completed a geostatistical analysis of all domains in an attempt to potentially determine the optimum grade interpolation parameters.

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
GO	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

Notwithstanding the weakness highlighted above, the variography generally confirms the following:

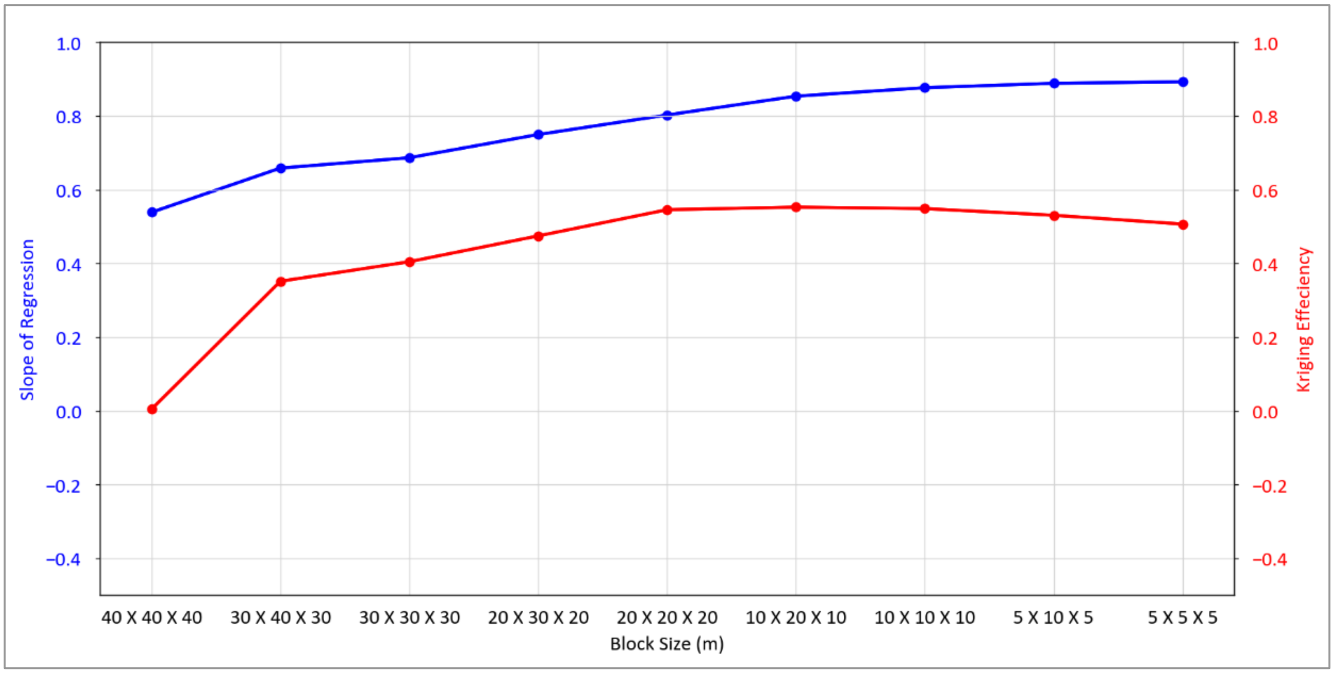
- The isotropic nature of the GO and MS domains in major and semi-major axes.
- The apparent major continuity down dip / plunge and along strike in the SW (stringer) and SW2 (stockwork) domains.

14.6 Estimation

14.6.1 Block size analysis

Sensitivity analysis on block size was performed to select the most appropriate block size for the estimation. The method involved running multiple ordinary kriging (OK) estimations on zinc in the 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 14.12 below.

Figure 14.12 Block model sensitivity analysis



Source: Micon.

14.6.2 Resource block model definition

The block model definition is 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 drillhole spacing, the envisaged selective mining unit (SMU) and the geometry of the deposit.

Table 14.2 LS Project Mineral Resource block model definition

Item	X	Y	Z
Origin coordinates	547200	4231080	3
Extents	445	1600	655
Block size (parent m)	5	10	5
Rotation (degrees)	20 degrees anti-clockwise		

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 (stringer zone) = 2.88
- SW2 (stockwork) = 2.88

14.6.4 Search parameters

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 14.3.

Table 14.3 Summary of search parameters

Domain	Element	Pass *	Interpol. method	Y (m)	X (m)	Z (m)	Dip (°)	Dip Az. (°)	Pitch (°)	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
Central (MS2)	Au	1	ID^2	200	200	30	59	79	171	2	18	3
	Ag	1	ID^2	200	200	30	59	79	171	2	18	3
	Cu	1	ID^2	200	200	30	59	79	171	2	18	3
	Zn	1	ID^2	200	200	30	59	79	171	2	18	3
	Pb	1	ID^2	200	200	30	59	79	171	2	18	3
Stockwork (SW2)	Au	1	OK	80	60	50	55	84	124	9	18	3
	Ag	1	OK	100	50	50	55	84	124	9	18	3
	Cu	1	OK	100	50	50	55	84	124	9	18	3
	Zn	1	OK	100	60	50	55	84	113	9	18	3
	Pb	1	OK	100	50	40	55	84	124	9	18	3
All	All	2	OK	P1x2	P1x2	P1x2	As P1	As P1	As P1	1	12	3

Notes: Y = Major axis (north – south); X = Semi-major axis (east – west); Z = Minor axis (vertical).

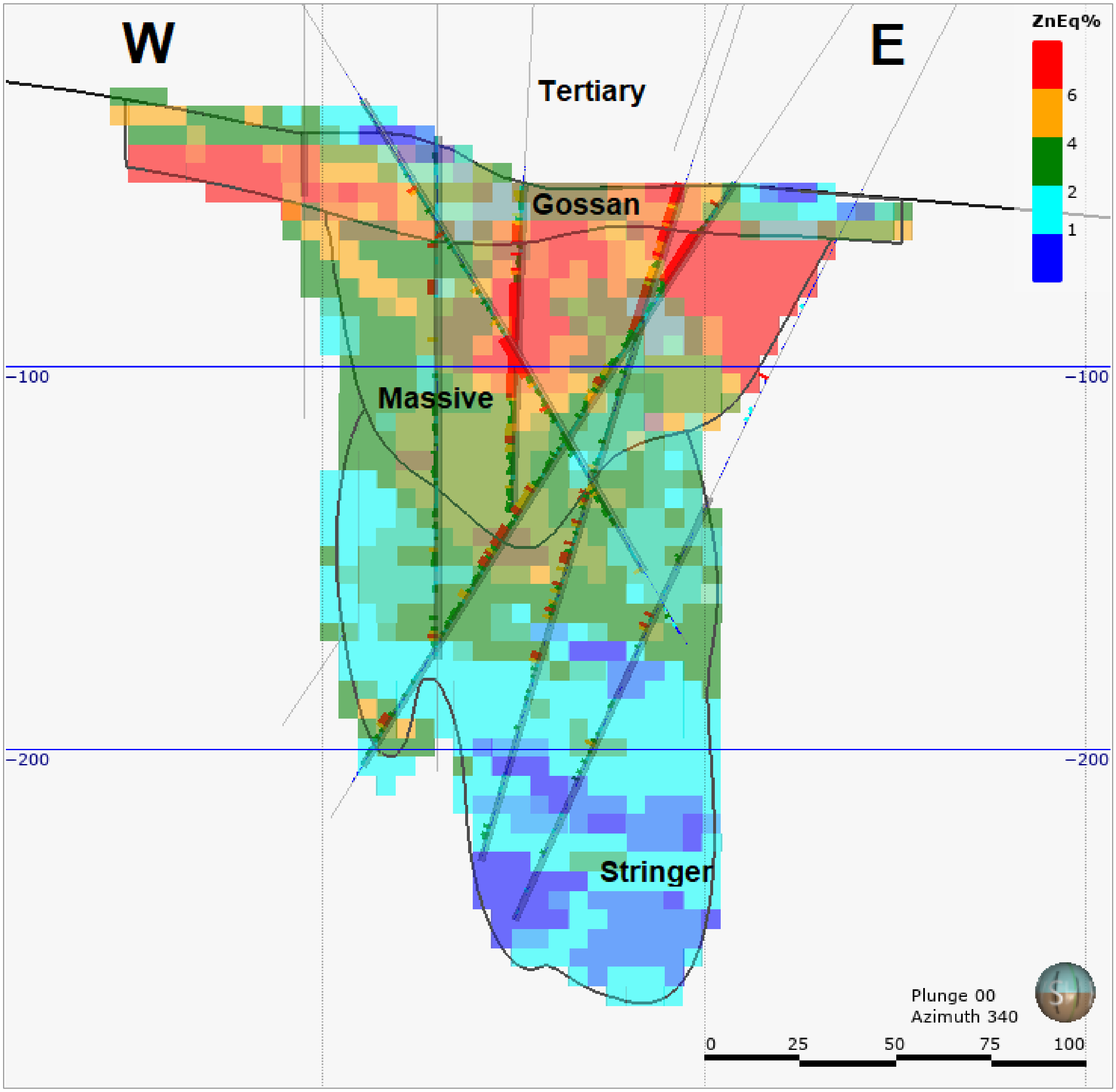
14.6.5 Grade interpolation and validation

OK was used for grade interpolation for all of the North and South deposit domains. The inverse distance squared (ID^2) technique was used for the Central deposit domain due to very limited drilling. 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 LS Project deposits was found to be satisfactory. An example is given in Figure 14.13.

Figure 14.13 Section through the MS domain showing the match between block and composite grade

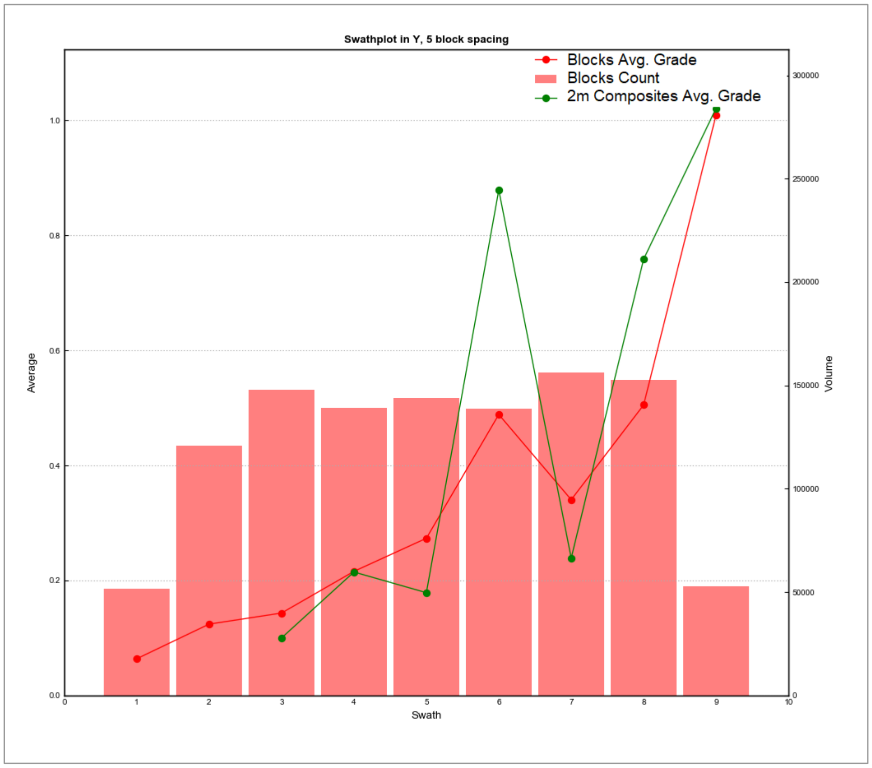


Source: Micon.

14.6.5.2 Validation by swath plots

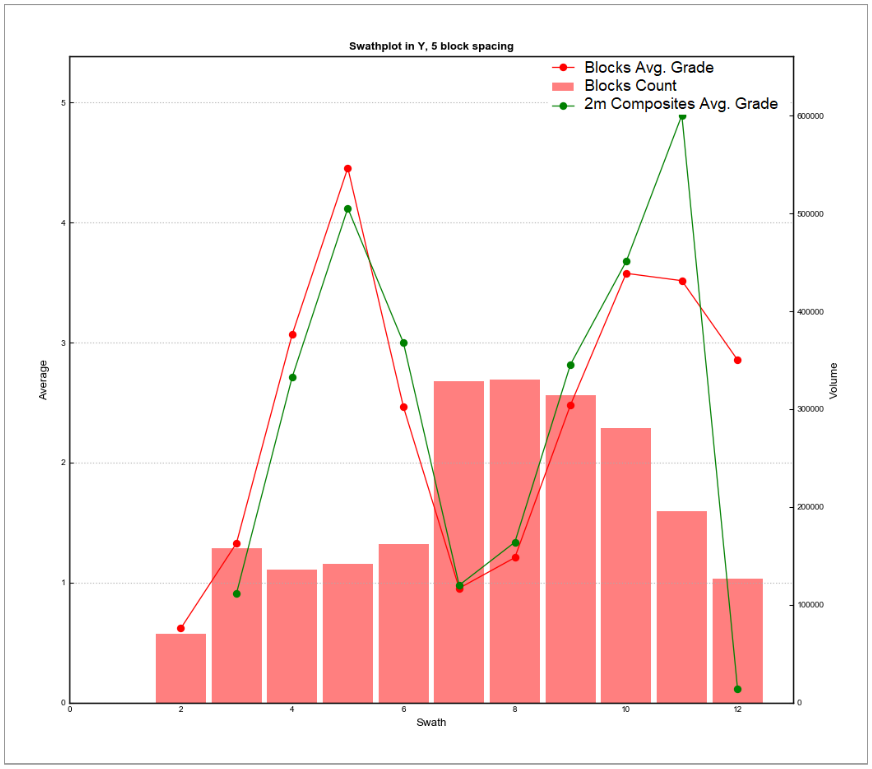
Validation using swath plots produced satisfactory results. Examples are given in Figure 14.14 to Figure 14.18. In all cases, a satisfactory overall match is reflected between block grades and composites.

Figure 14.14 North deposit GO domain Au swath plot



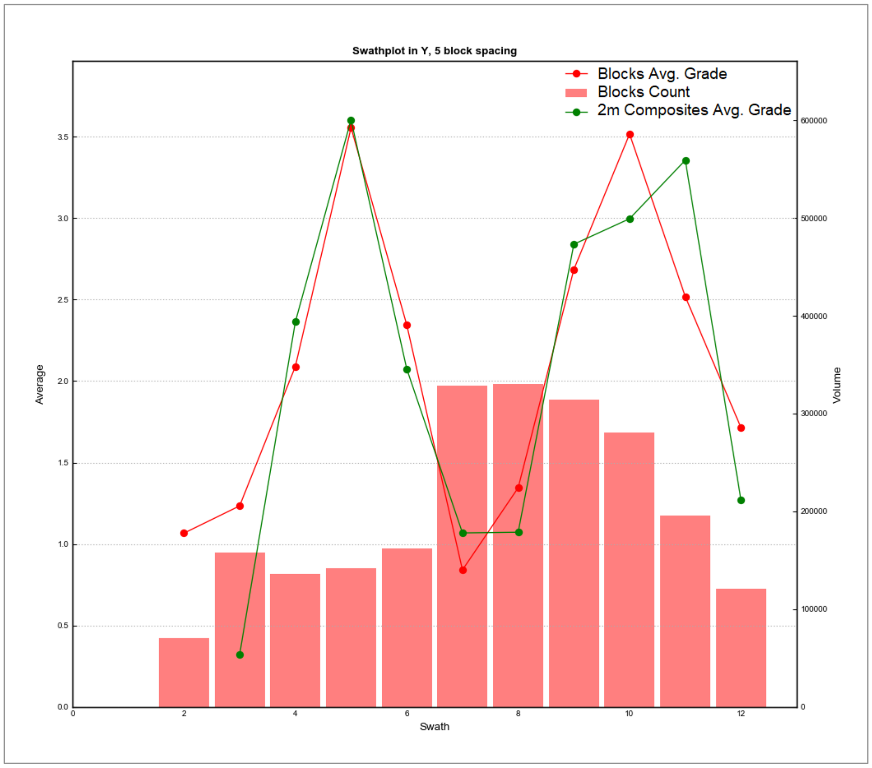
Source: Micon.

Figure 14.15 North deposit MS domain Zn swath plot



Source: Micon.

Figure 14.16 North deposit MS domain Pb swath plot

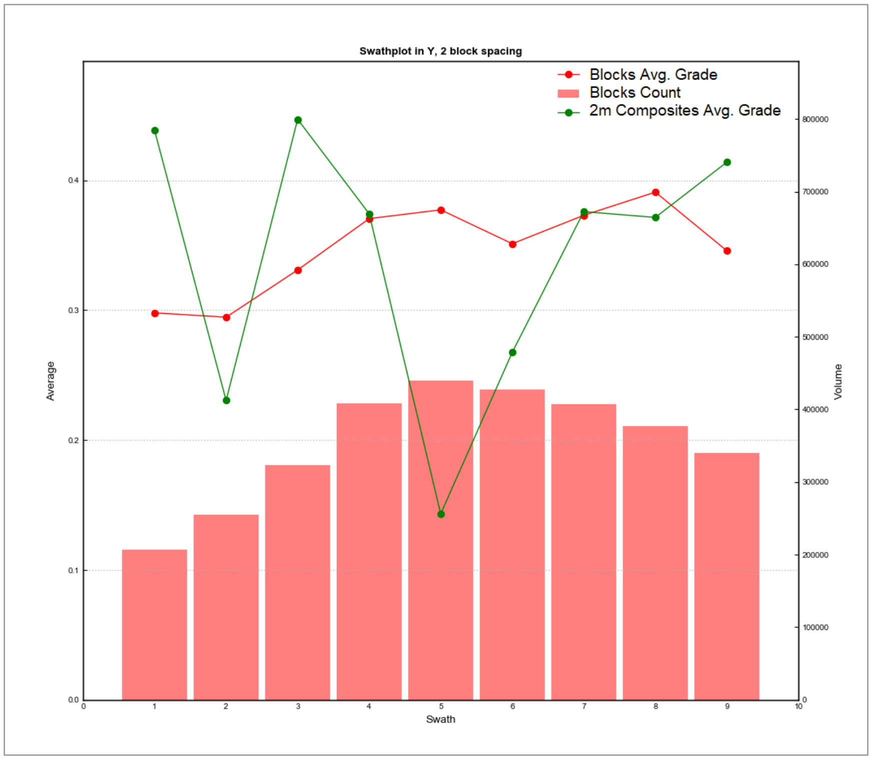


Source: Micon.

Figure 14.17 North deposit MS domain Cu swath plot

Source: Micon.

Figure 14.18 South deposit SW2 domain Cu swath plot



Source: Micon.

14.6.5.3 Validation by different estimation technique

The final validation was conducted by utilizing the inverse distance cubed (ID^3) and nearest neighbour (NN) estimation techniques on the MS domain, which contains the majority of the resource. The results for zinc shown in Table 14.4 below indicate a favourable match with the original OK method used for the estimate.

Table 14.4 Comparison between ID^3, NN, and OK estimation results for the MS domain

Technique	Blocks count	Mean	Std Dev	Coeff. V	Median	Max
OK	9027	2.27	2.01	0.89	1.63	13.46
NN	9052	2.39	2.11	0.89	1.59	11.78
ID^3	9052	2.32	2.25	0.97	1.46	18.44

14.6.5.4 Overall comments

All of the three methods used to validate block grade estimation supported the estimation results. Table 14.5 presents the mineral inventory's sensitivity to cut-off grade for ZnEq and Table 14.6 presents the mineral inventory's sensitivity to cut-off grade for copper equivalent (CuEq).

Table 14.5 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**
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 14.6 LS Project – South deposit global mineralized tonnes at various CuEq cut-off grades

Category	CuEq cut-off	Tonnes (kt)	Average grade							
			CuEq (%)	Cu (%)	Zn (%)	Pb (%)	Sn (%)	Ag (g/t)	Au (g/t)	AuEq (g/t)
South deposit	6.0%	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	5.5%	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	5.0%	1	5.09	1.94	3.86	2.01	0.00	91.18	0.26	8.51
	4.5%	2	5.01	1.85	4.12	2.05	0.00	81.19	0.24	8.38
	4.0%	18	4.29	1.82	3.03	1.68	0.00	68.55	0.18	7.17
	3.5%	56	3.89	1.60	2.93	1.59	0.00	58.34	0.15	6.50
	3.0%	146	3.47	1.26	3.01	1.60	0.00	48.77	0.11	5.80
	2.5%	382	3.00	1.04	2.74	1.44	0.00	39.84	0.09	5.01
	2.0%	982	2.52	0.82	2.44	1.29	0.00	31.23	0.07	4.21
	1.5%	2,683	2.01	0.63	1.98	1.05	0.00	25.25	0.06	3.36
	1.0%	7,175	1.51	0.45	1.48	0.85	0.00	18.79	0.05	2.53
	0.9%	8,559	1.42	0.42	1.40	0.81	0.00	17.33	0.05	2.37
	0.5%	13,195	1.18	0.34	1.18	0.69	0.00	13.69	0.05	1.97
	0.3%	14,121	1.13	0.32	1.13	0.66	0.00	13.06	0.05	1.88
	0.0%	14,206	1.12	0.32	1.12	0.66	0.00	13.00	0.05	1.88
	Total	**14,206**	**1.12**	**0.32**	**1.12**	**0.66**	**0.00**	**13.00**	**0.05**	**1.88**

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 follow:

*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*

Metals 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 / Central deposits) than for the MS (North deposit). The South and Central resources are reported at a copper equivalent grade of 0.9% 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 14.7 summarizes the underground economic assumptions upon which the resource estimate for the LS deposits are based.

Table 14.7 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	65%

14.7.2 Classification of the Mineral Resource

Micon has classified the Mineral Resource estimate at the LS Project in the Measured, Indicated, and Inferred categories.

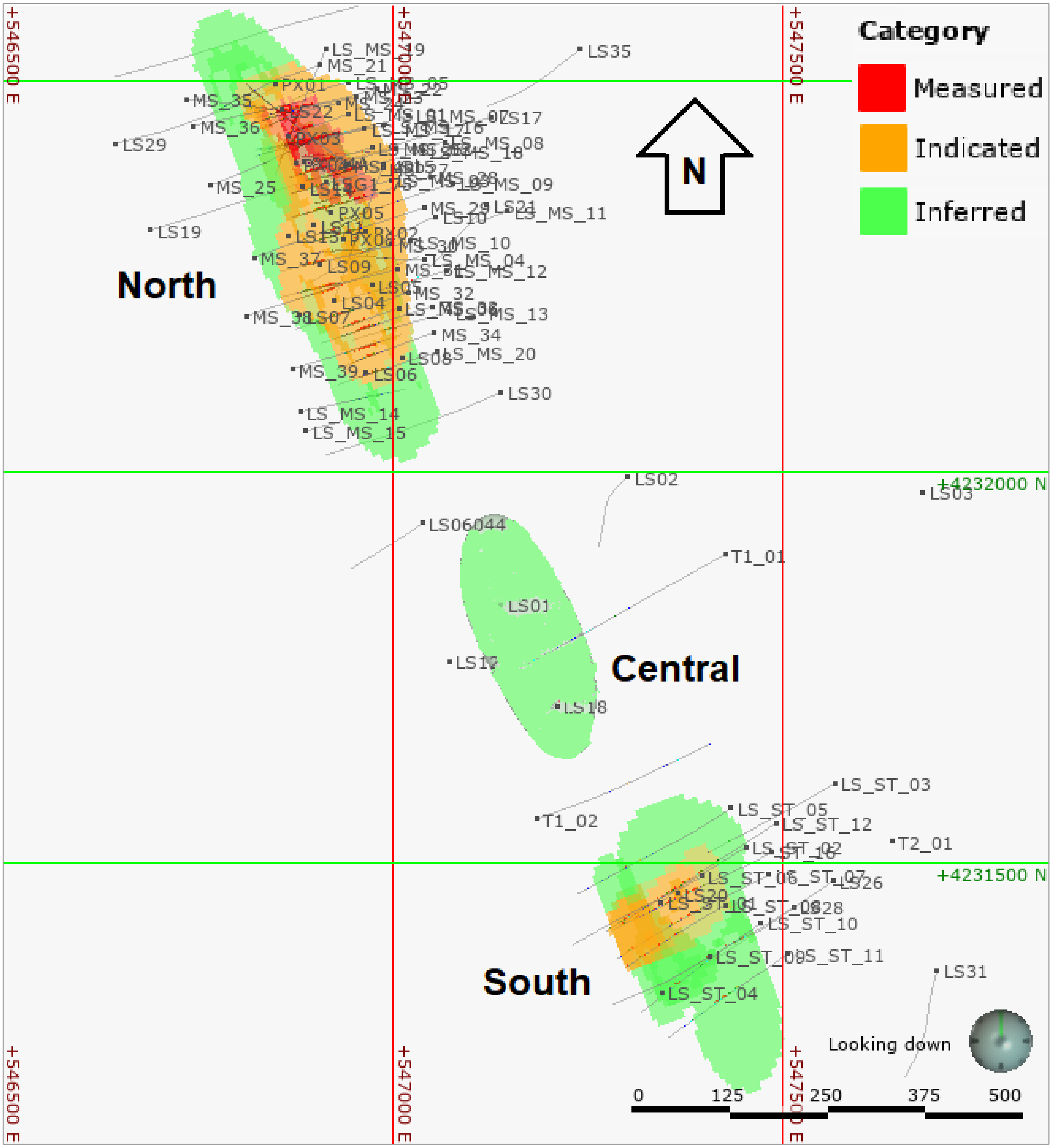
The approach used to categorize the Measured Resource was to select those blocks informed by more than 4 drillholes and within a 20 - 30 m distance from the closest composite. The approach used to categorize the Indicated Resource was to select those blocks informed by more than 3 drillholes and within 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. A plan view of the resource categorization is shown in Figure 14.19.

14.7.3 Mineral Resource statement

The Mineral Resource statement for the LS project is summarized in Table 14.8.

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.

Figure 14.19 Plan view of the LS Project showing Mineral Resource categorization



Source: Micon.

Table 14.8 LS Project Mineral Resource estimate as of 5 September 2019

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
					Average grade							**Contained metal**						
Deposit	**Category**	**Min Zones**	**Cut-off CuEq (%)**	**Tonnes (kt)**	**Cu (%)**	**Zn (%)**	**Pb (%)**	**Sn (%)**	**Ag (g/t)**	**Au (g/t)**	**CuEq (%)**		**Cu (kt)**	**Zn (kt)**	**Pb (kt)**	**Sn (kt)**	**Ag (koz)**	**Au (koz)**
Central	Inferred	Str	0.9	1,707	0.15	0.16	0.06	0	12	2.22	1.66		2.5	2.7	1.0	-	635.2	121.9
South	Measured	Str/Fr	0.9	0	—	—	—	—	—	—	—							
	Indicated	Str/Fr	0.9	2,473	0.47	1.53	0.83	0.00	19	0.06	1.54		11.5	37.9	20.6	0.0	1,484.7	4.7
	M & I	**Str/Fr**	**0.9**	**2,473**	**0.47**	**1.53**	**0.83**	**0.00**	**19**	**0.06**	**1.54**		**11.5**	**37.9**	**20.6**	**0.0**	**1,484.7**	**4.7**
	Inferred	Str/Fr	0.9	6,085	0.40	1.34	0.80	0.00	17	0.05	1.37		24.6	81.6	48.7	0.0	3,285.2	10.0

Notes:
- The Mineral Resources were estimated using the CIM Definitions Standards (2014).
- Mineralized Zones: GO, MS, Str=Stringer, Str/Fr=Stockwork.
- 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))/67.24.
- 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.
- 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.
- Densities: GO=3.12, MS=4.76, Str=2.88, Str/Fr=2.88.

15 Mineral Reserve estimates

There are currently no Mineral Reserves reported for the Property.

16 Mining methods

16.1 Mineral Resource

Micon produced a Mineral Resource estimate and this was reported in the 2019 Micon Technical Report. This AMC Technical Report is for a PEA for the Project and only considers those Mineral Resources for the North deposit. The economic analysis in this Report is preliminary in nature and is based, in part, on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the results of the PEA will be realized.

Measured and Indicated Resources of 10.3 Mt at 9.06% Zn Eq are considered in this study as well as 2.5 Mt at 5.93% ZnEq of Inferred Resource, both for the North Zone. The LS Project is a polymetallic, VMS deposit and the grades of the individual metals and modifying factors are detailed in Table 14.8. Only the MS and GO (oxide) mining zones are considered in the PEA.

16.2 Geotechnical

Minimal work has been completed on the geotechnical aspects of the deposit and it is recommended that sufficient geotechnical work is undertaken in the next phase of studies to adequately support underground mine design assumptions.

16.3 Mining assumptions

In order to convert the Mineral Resource into an inventory of mineralized material for the PEA, stope wireframes were developed for each mining zone. These stope wireframes were designed to include any internal dilution necessary to generate a practical mining shape (planned dilution). In addition, unplanned dilution due to the extraction of the stope was assumed to be 8% for the GO zone and 5% for the MS zone. Mining recovery of 90% was assumed for the GO and 93% for the MS.

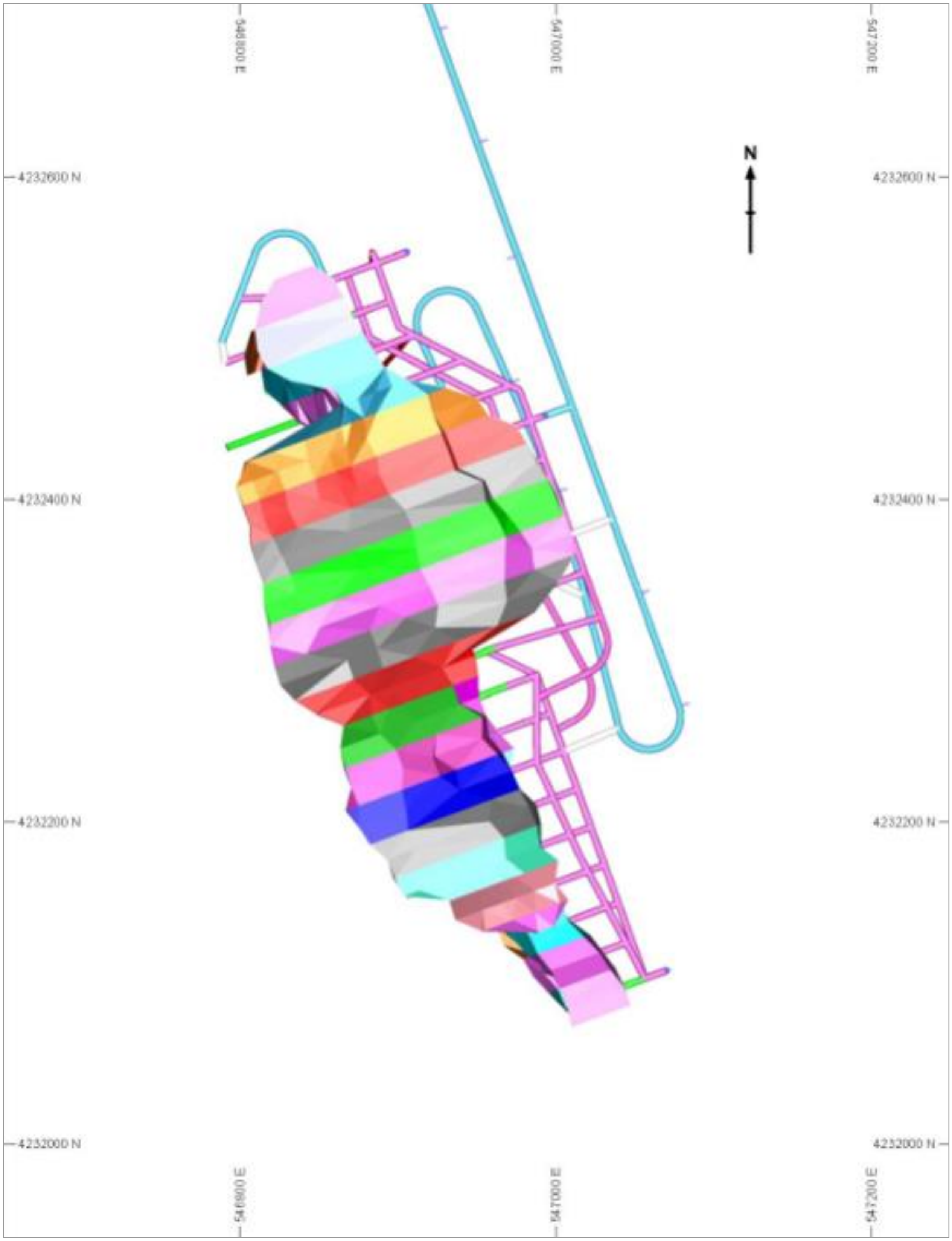
16.4 Mining method and design

The mine design is based on a single decline access from surface (83 RL) at a 12.6% gradient. Decline access is via a 30 m deep boxcut, 250 m in length. Stopes are accessed from level access drives in the north and the south of the deposit. Level elevations vary between the north and the south access drive, Level 1 being located on -67 RL in the north and -55 RL in the south. Interlevel spacing varies between 24 m and 35 m. All mineralized material and waste development are mined with a 4.5 m by 4.5 m end profile.

The deposit is planned to be mined using transverse sub-level open stoping (SLOS) with paste fill at a production rate of approximately 1 million tonnes per annum (Mtpa). Crosscuts will access the deposit with drives developed laterally across the mineralization. Drives in mineralization will be placed 12.5 m apart along strike, with stopes approximately 25 m to 35 m high, 12.5 m wide, and 25 m in length. Stope heights in the GO tend to be generally less, approximately 20 m high. A slot will be cut at the end of the mineralization and consecutive rings blasted in a retreating fashion over the full stope length back to the crosscut. Uphole drill rings from the existing drives in the MS will be drilled to extract the mineralization from the overlying GO deposit.

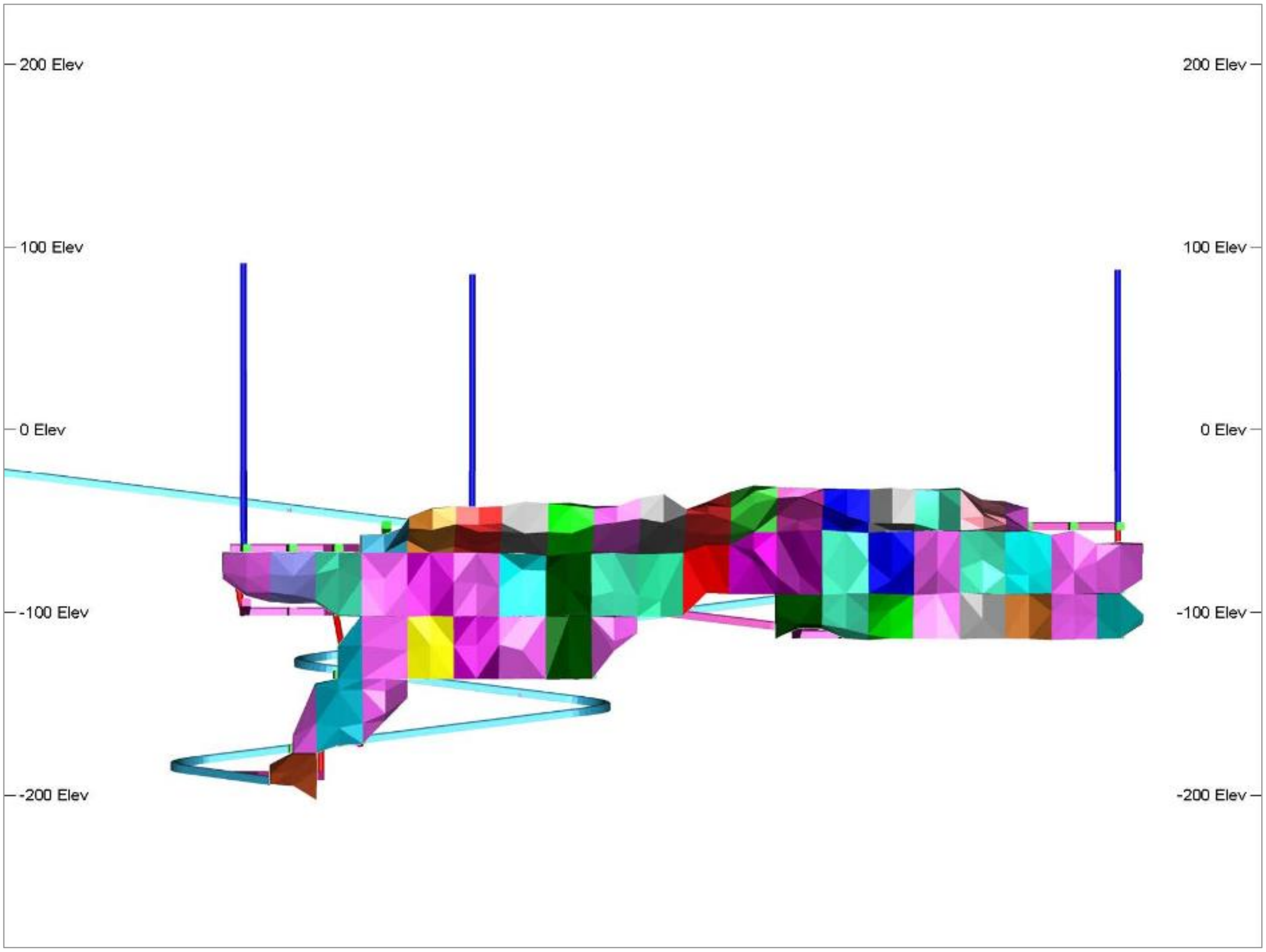
The mine design is shown in Figure 16.1 (plan view) and Figure 16.2 (long section).

Figure 16.1 Plan view of the mine design



Source: AMC.

Figure 16.2 Long section view of the mine design



Source: AMC.

16.5 Stope optimization

A Net Smelter Return (NSR) value was determined for each block of mineralization based on the expected metal recoveries, selected metal prices and tonnes and grades. The NSR value is based on the assumptions reported in Table 16.1.

The NSR for the MS is calculated using the formula:

NSR = Zn% grade x 2,646 x 0.68 + Pb% grade x 2,315 x 0.6175 +Cu% grade x 5,952 x 0.0625 + Sn% grade x 16,535 x 0.2850 + Ag g/t grade x 0.58 x 0.6075 + Au g/t grade x 45.0 x 0.7385.

The NSR for the GO is calculated using the formula:

NSR = Pb% grade x 2,315 x 0.6175 + Sn% grade x 16,535 x 0.38 + Ag g/t grade x 0.58 x 0.6534 + Au g/t grade x 45.0 x 0.8514.

Stope wireframes were generated for each stope using a cut-off NSR value $40/t of mineralized material for the GO and $49/t for the MS stopes. The selected mining factors for dilution and mining recovery were then used to determine an inventory of tonnes and grade. NSR was recalculated to ensure mineralization remains economically viable and a final check was made to remove any outlying stopes that would not be economic when the cost of access development was included.

Table 16.1 Assumptions used to generate NSR field

Parameter field	Unit	Parameter value
Gold price	$/oz	1,400
Silver price	$/oz	18
Copper price	$/lb	2.70
Tin price	$/lb	7.50
Lead price	$/lb	1.05
Zinc price	$/lb	1.20
Massive sulphides		
Pb recovery	%	65% (45% Pb in concentrate)
Zn recovery	%	80% (48% Zn in concentrate)
Cu recovery	%	25
Au recovery to Pb concentrate	%	10
Au recovery from leached concentrate tails	%	65
Ag recovery to Pb concentrate	%	35
Ag recovery to Zn concentrate	%	20
Ag recovery from leached concentrate tails	%	20
Sn recovery to concentrate	%	30
Payable Pb	%	95
Payable Zn	%	85
Payable Cu	%	25
Payable Au	%	45 in Pb con, 99 from leach
Payable Ag	%	81
GO / oxides		
Pb recovery	%	65
Sn recovery	%	40
Au leached to doré	%	86
Ag leached to doré	%	66
Payable Pb	%	95
Payable Sn	%	95
Payable Au	%	99
Payable Ag	%	99
Charges and freight		
Lead concentrate treatment charge	$/dmt	180
Zinc concentrate treatment charge	$/dmt	210
Tin concentrate treatment charge	$/dmt	450
Silver refining cost	$/oz	1.00
Gold refining cost	$/oz	15.00
Freight and shipping of oxide concentrates	$/dmt	80
Freight and shipping of sulphide concentrates	$/dmt	60 - 80
NSR cut-off value (GO)	$/t	40
NSR cut-off value (MS)	$/t	49

The MS has drives mined transversely above the stope to provide access for paste fill. These drives will also be used to mine the GO using uphole fan drilling. As such the cost for mining the GO is lower than the MS.

16.6 Production rate

In order to initially arrive at an appropriate production rate that can be supported by the deposit, AMC has used a combination of Taylor's rule of thumb and tonnes per vertical metre (tpvm) to project production ranges.

Production rate based on Taylor's rule of thumb is estimated at 0.7 Mtpa.

Annual Production Rate = 5 * 'Mineralized material' $^{0.75}$

Most successful SLOS operations do not exceed 30 to 40 vertical metres per year. Based on the mineralization by level, this would be equivalent to 1.3 Mtpa to 1.8 Mtpa.

Given that the majority of the resource is located over three production levels, AMC considers a rate of 1 Mtpa to be appropriate. This production rate is well supported by the production scheduling extracting stopes on a primary / secondary basis and paste filling.

16.7 Development schedule

The development schedule is summarized in Table 16.2. Ramp development was scheduled at a rate of 90 metres per month (m/month) and where multiple headings were available a maximum of 180 m/month. Development quantities are reported directly from the mine design.

Table 16.2 Development schedule

Description	**Unit**	**Total**	**Year**					
			1	**2**	**3**	**4**	**5**	**6**
Box cut	t	466,897	466,897					
Horizontal development waste	m	5,573	720	1,720	1,252	649	701	531
Vertical development waste	m	721		436	189			96
Development in mineralization	m	2,766		858	664	576	436	232
Total development	m	**9,060**	**720**	**3,014**	**2,105**	**1,225**	**1,137**	**859**

16.8 Production schedule

A summary of the production schedule for the life-of-mine (LOM) plan is shown in Table 16.3. Approximately 7.25 Mt of mineralized material is planned to be mined at an average production rate of 1 Mtpa with an average ZnEq of 7.9%.

Table 16.3 LOM production schedule

Description	Unit	Year								
		2	3	4	5	6	7	8	9	Total
Mineralization	kt	579	1,032	1,016	1,033	1,001	1,001	996	595	7,251
Cu	%	0.50	0.26	0.35	0.39	0.50	0.36	0.26	0.10	0.34
Zn	%	2.06	3.85	2.99	3.09	2.54	2.72	0.83	0.40	2.44
Pb	%	2.82	2.55	2.70	2.78	3.31	2.57	3.29	2.72	2.85
Sn	%	0.15	0.13	0.13	0.13	0.14	0.14	0.27	0.26	0.16
Au	g/t	0.71	0.71	0.80	0.68	0.82	0.76	0.72	0.79	0.75
Ag	g/t	104.1	65.6	77.9	62.7	84.2	75.9	51.4	38.9	69.8
ZnEq	%	8.39	8.56	8.29	8.25	8.86	7.92	6.86	5.26	7.91

The PEA is preliminary in nature, it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

16.9 Mobile equipment

Table 16.4 indicates the estimated equipment required to support a 1.0 Mtpa mine plan with associated mineralized material and waste development. Equipment number estimates allow for some redundancy to accommodate break downs and ramp congestion.

Table 16.4 Estimated equipment requirements

Primary equipment required	Number
LHD, matched to trucks	4
Trucks (30 t)	4
Development drill	3
Longhole drill rig	2
Rock support drill rig	2
Cable bolter	1
Scalers	3
Rock breaker	1
Support trucks, high-lifts	4
Total fleet	**24**

16.10 Manpower

The total underground manpower envisaged for the 1.0 Mtpa mine is shown in Table 1.5. The mine will be operated on a three shift per day, eight hours per shift basis, Monday to Friday. Only essential services will be run over the weekends.

Table 16.5 Underground manpower

Description	Number
Total operators	72
Maintenance team	30
Services team	10
Mining manager	1
Engineering manager	1
Technical services	8
Supervisors	12
Total manpower (Mine)	**134**

16.11 Ventilation

The function of the ventilation system is to dilute / remove airborne dust, diesel emissions, gases from explosives, and to maintain temperatures at levels necessary to ensure safe production throughout the LOM.

The ventilation system is designed as a 'pull' system with primary exhaust fans located on surface at the top of each primary exhaust raise. Fresh air is delivered into the mine from the decline and fresh air raise. Internal return air raises carried with the production ramps connect to a dedicated return airway and the exhaust raises to surface.

The main intake air is via the decline and a centrally located ventilation raise 3.5 m in diameter. Return air to surface is via two 3.5 m diameter raises located at the extremities of the north and south blocks.

Based upon benchmark data for similar mines, the required airflow would be in the range of 340 m^3/s to 380 m^3/s. With a cross sectional area for the decline of 4.5 m by 4.5 m it would only be able to accommodate 120 m^3/s at a 6 m/s velocity (best practice maximum velocity). This means the fresh air raise is required to supply between 220 m^3/s to 260 m^3/s. A 3.6 m diameter intake raise is required together with the decline. The two 3.6 m diameter return air raises are sufficient to handle the return air (approximately 190 m^3/s per raise).

The distribution of the required airflow from the primary intakes to the working areas will be controlled by a combination of regulators and fans. Auxiliary fans will be used to deliver fresh air to the production levels. Drop-board regulators in the return air raises on each level access will ensure that the required amount of air is supplied to the auxiliary fans.

To ensure the primary airflow is available in sufficient quantities to the active working places, diligent airflow monitoring, and control will be required as development in mineralization and waste access drives progresses to breakthrough.

16.12 Emergency egress

Escape ways will be raised between sublevels and equipped with manways to allow emergency egress from level to level. The main decline will be the escape route to surface with an equipped ventilation raise providing second egress.

17 Recovery methods

Since only preliminary scoping level metallurgical study has been done, it is difficult to accurately forecast the metallurgical recoveries of the economic metals of interest. However, assumptions have been made to estimate the recoveries.

17.1 Mineral processing assumptions

There are essentially two major ore types, namely MS and stockwork. Each ore type has both oxidized and fresh ore.

The processing flowsheet will have to be flexible to handle the various ores because of the following reasons:

- The test work indicates that it may not be economical to recover copper from MS ore, a low recovery is assumed for the PEA.
- The stockwork and GO ore does not have sufficient tin values for economic recovery.

The following assumptions were made to estimate the recoveries of valuable minerals:

- Better separation of copper, lead, and zinc minerals can be achieved in additional planned test work.
- LCT work was done using sequential copper, lead, and zinc flotation. However, bulk copper / lead flotation followed by zinc flotation process is envisioned for the deposit. The latter process scheme is more flexible than the former, for the deposit.
- The oxidized ores (GO) should be sulphidized prior to flotation in order to increase lead and zinc recoveries.
- The final tailings from the polymetallic flotation circuit will be sent to a cyanidation leach circuit for recovery of gold and silver.
- The leached residue from the cyanidation circuit will be subjected to flotation to recover cassiterite.

The metal recoveries were estimated based on limited test work, recoveries from similar ore deposits and extensive experience of processing polymetallic and precious metals. The projected recoveries are given in Table 17.1 and Table 17.2. The payable metal values are given in Table 17.3 and Table 17.4.

17.2 Conceptual process flowsheet

The conceptual process flowsheet envisioned for this deposit consists of three stage crushing followed by ball milling of the ore. The ground product will be conditioned with depressant for zinc and bulk lead-copper rougher concentrate floated. The tailing will be sent to the zinc flotation circuit. When processing oxide ore, sulphidization reagent will be added in the mill.

The lead / copper or lead concentrate will be reground and cleaned three times to produce a concentrate which can then be sent to lead and copper separation circuit or filtered as final lead concentrate. The copper concentrate can be cleaned once or twice to produce saleable copper concentrate.

The zinc concentrate will be reground and cleaned three times to produce saleable zinc concentrate. The final tailing will be thickened and sent to cyanidation leach circuit for extraction of gold and silver. The tailings will be subjected to cyanide destruction. Then they will be sent to the tin flotation circuit or to the TSF.

Table 17.1 Process plant recoveries for MS

Recovered metal for MS	Recovery
Pb recovery to concentrate	65% (45% Pb in concentrate)
Zn recovery to concentrate	80% (48% Zn in concentrate)
Cu recovery to concentrate	25%
Au recovery to Pb concentrate	10%
Au recovery from leached concentrate tails	65%
Ag recovery to Pb concentrate	35%
Ag recovery to Zn concentrate	20%
Ag recovery from leached concentrate tails	20%
Sn recovery to concentrate	30% (10% Sn in concentrate)

Table 17.2 Process plant recoveries for oxides

Recovered metal for oxides	Recovery
Pb recovery into a sulphidized concentrate	65% (60% Pb in concentrate)
Sn recovery to concentrate	40% (10% Pb in concentrate)
Au leached into a doré	86%
Ag leached into a doré	66%

Assumptions for the metal payabilities for the MS and the oxides are summarized in Table 17.3 and Table 17.4, respectively.

Table 17.3 Payable metal for MS

Payable metal for MS	Payability
Pb	95%
Zn	85%
Cu in Pb concentrate	25%
Au in Pb concentrate	45%
Au from leached concentrate tails	99%
Ag in Pb concentrate	88%
Ag in Zn concentrate	50%
Ag from leached concentrate tails	99%
Sn	95%

Table 17.4 Payable metal for oxides

Payable metal for oxides	Payability
Pb	95%
Sn	95%
Au	99%
Ag	99%

18 Project infrastructure

LS is a green fields site. The land is currently owned by two local residents and permission will be required prior to mining. Complete site infrastructure will be required for mining. With the local community of Grândola (population around 15,000), 12 km and Lisbon, Portugal's capital, 120 km by road, and both easily accessible, the project will be well supported locally with supplies as well as personnel.

There are some existing cork trees located on the project site. Infrastructure will be designed to ensure that these trees are not disturbed for the LOM.

18.1 Site access

There is currently access to the Property via paved roads to Cilha do Pascoal, followed by 4 km of gravel roads to the mine site. Some improvement to the roads from the EM-543 intersection to the mine site may be required to accommodate heavy construction traffic.

Site roads will be gravel surfaced.

18.2 Structures

Minimal structures will be required with some services being covered by the local community at Grândola or in Lisbon.

The site will require an office, changeroom, shop and warehouse as well as storage for fuel, laydown areas, site fencing, and security building. An allowance for a total of 2,600 metres squared (m^2) of building space has been included in the PEA.

The office will accommodate direct mine and mill management as well as technical staff. Clerical staff, payroll, and other administrative positions may be located in the local community of Grândola.

A shop is required to service the underground and surface mobile and fixed equipment, primarily the mining fleet. The shop space allocated will be adequate to accommodate daily maintenance, preventative maintenance and servicing, and repairs of equipment. Rebuilds and major overhauls can be best accommodated off-site.

Warehouse facilities will be required to maintain stock of consumables such as ground control, mill reagents, and equipment spares.

Change room facilities are required for workers to wash after each shift.

18.3 Electrical

Total power requirement for the mine and mill is estimated to be 15 megawatts (MW). There is ample opportunity to connect to the national grid with both 400 kilovolts (kV) and 30 kV transmission lines operating within 7 km of the project site. However, for this study, a conservative allowance has been made to run a 30 kV, 20 mega volt amperes (MVA) transmission line from the existing sub-station at Grândola. The possibility of interconnection closer to the mine site must be confirmed with Redes Energéticas Nacionais, SGPS, S.A. (the National Energy Network).

Site power will be at 11 kV with distribution both to the mill and into the underground mine. Local power will be at 690 V.

Initial power and emergency generation will be provided by diesel generator sets.

18.4 Backfill

At an annual production rate of 1.08 Mtpa, the mill, at 92% utilization, will process 3,500 tonnes per day (tpd) of ore with 8% mass pull to concentrates. Tailings will be produced at up to 135 tonnes per hour (tph) and dewatered in the tailings filtration plant to a cake with 10% to 15% moisture content.

On demand, as required by the mine, filtered tailings cake, cement binder and process water will be combined in a continuous mixer to produce cemented paste fill slurry at a nominal 73% solids by weight at up to 110 m^3/hr. Subject to future test work, cement binder addition rates will range between 2% and 10% by weight and average around 4% for the LOM operations. Approximately 55% of tailings (up to 540,000 tonnes per annum (tpa) at a dry bulk density of 1.4) will be placed underground as paste fill to meet an annual demand of 400,000 cubic metres (m^3) of void and the remaining tailings placed in the dry stack Tailings Storage Facility (TSF). The paste plant will have an annual utilization of just below 50%. At 95% saturation of the paste fill, approximately 180,000 kilolitres (kL) of water will be retained in the paste fill each year.

Paste fill will be transported underground using a combination of pumping and gravity via boreholes and high-pressure pipelines to the stopes. The paste fill will be retained in stopes by structural arched shotcrete barricades designed to enable continuous filling of each stope without pausing for plug curing. Filling will be remotely monitored from the surface paste plant by means of pressure sensors and video cameras and automated line switching will be considered.

Where tight filling of stopes is required additional monitoring will be applied to ensure over pressure events are eliminated. Backfill operations will be directed by a Backfill Management Plan and co-ordinated by a responsible person who will ensure that all design and operations requirements of mine production, geomechanics, and milling are fully integrated to ensure safe backfill operations.

18.5 Tailings

Tailings and waste rock will be disposed of through the use of a dry-stack facility. Total tailings for LOM are estimated at 7.5 Mt with a further 0.7 Mt of waste rock. Approximately 55% of tailings will be disposed of in the mined-out stopes via the paste fill system. The remaining 4.1 Mt of tailings and waste must be accommodated in the dry stack facility. The dry stack facility will be approximately 200,000 m^2 in area with a capacity of 2.9 million cubic metres (Mm^3). The base of the facility will be lined, and a low perimeter berm and ditch will capture any precipitation run off during the LOM. Run off will be collected in a settling pond for use by the mine as service water.

Tailings will be dewatered in a filter plant located at the mill to a final water content of 10% to 15%. Dewatered tailings will be transported to the dry-stack and placed by dozer. Waste will also be placed in the dry-stack and blended with the tailings.

18.6 Water and dewatering

Regional precipitation averages 700 mm per year and it is anticipated that the site will have a net neutral water balance once the initial dewatering of the mine is complete. All water from the mill will be reused.

Total annual water gain through precipitation and mine dewatering is estimated to be approximately 325,000 m^3. Loss to the tailings is estimated at 250,000 m^3 per year with evaporation accounting for the remaining loss. A complete climate and water balance study is required.

It is anticipated that any make-up water that may be required will be obtained via local wells on site. Should this not be adequate, water can be obtained from the Sado River approximately 5 km from the project site.

A settling pond with capacity of 100,000 m^3 will be established to hold precipitation run-off during the rainy season as well as mine and mill water discharge. It will also have sufficient capacity to accommodate the 100-year storm event.

Mine water inflows are unknown at this time; AMC has assumed an inflow of 5 litres per second (L/s) based other operations in the area. Water will be discharged via a staged pump system with pumps located on 3 levels staging to surface.

19 Market studies and contracts

There are no offtake contracts in place for the zinc, lead, and tin concentrates which will be produced at LS, which is reasonable given the project stage.

Discussions with potential strategic off-takers have been undertaken, indicating interest in long term offtake commitments. While negotiations will ultimately need to be undertaken, it is expected that 100% of the production will be absorbed by the market when production commences and future offtaker engagement will likely lead to mutually agreed modifications to the Sn concentrate specification.

As Europe is expected to remain an important buyer of zinc and lead concentrates, it is assumed that the ultimate sales profile will involve shipping the majority of the zinc and lead concentrate production to Europe, complemented by some sales to Asia. Conversely, due to the absence of European primary tin smelting capacity, it is expected that the tin concentrate production will be allocated to Asian custom smelters, potentially complemented by South American sales.

19.1 Markets

19.1.1 Zinc

The current market for zinc concentrate market is oversupplied, as reflected in prevailing spot treatment charges, and the fact that spot treatment charges are currently higher than benchmark, which is unusual from a historical perspective. The very strong prices and low treatment charges which prevailed until relatively recently have stimulated mine production, contributing to the current oversupply. Compounding this issue was reduced Chinese smelting consumption, due to a combination of environmental pressures and reduced metal demand.

However, the concentrate market will shift to greater balance in the medium term, and ultimately additional mine supply will be required to satisfy global demand. The combination of current metal prices and treatment charges may force additional mine closures in the short term, at a time when Chinese smelter consumption is improving, contributing to a supply gap in the medium term. Further, mine development globally is currently being deferred in light of depressed prices and elevated treatment charges. Accordingly, a tighter concentrate market, and lower treatment charges, will prevail in the medium term corresponding with the start of production from LS provided that the development timeline continues to advance. This market shift to greater tightness could happen sooner than many industry participants expect, particularly if mine production is curtailed for strategic reasons by major producers in the face of weak market conditions. There is precedent for such a response.

Zinc metal prices are currently under pressure, in part reflecting reduced demand from the automotive sector and a protectionist stance on trade between select influential governments. While LME stocks are at critically low levels after four consecutive years of metal deficit, it is expected that the current concentrate oversupply will ultimately become a metal oversupply, in part as Chinese smelting capacity improves, contributing to a metal market surplus over the next few years. However, the price required to incentivize new mine investment is significantly higher than current prices. Both zinc metal market fundamentals and pricing are expected to improve through the medium term.

19.1.2 Lead

As with zinc, the short-term market weakness being experienced in Pb is not expected to prevail in the medium term.

The recent strong zinc prices referenced above contributed to new mine production, with many of these mines producing both zinc and lead concentrates.

Accordingly, the current concentrate market has recently shifted from severe deficit to oversupply, as reflected in a sharp rise in spot treatment charges. The recent restart of production at the Penasquito mine in Mexico, and the recent announcement of the permanent closure of the Belledune smelter in Canada, offset the impact of the announced restart of the Port Pirie smelter, such that the concentrate market is currently well supplied, with TC's at a three year high.

However, the concentrate market will moderate through the medium term, exerting downward pressure on treatment charges.

With regards to lead metal, weak demand from the auto sector as well as increased lead concentrate availability has pressured current pricing. Further, global metal surpluses are forecast to be maintained for the next few years. However, beyond that, additional mine supply will be required, with improved prices expected to stimulate such longer-term supply.

19.1.3 Tin

Market weakness earlier this year prompted various smelters in China and Indonesia to reduce production, as prices were trading at 3-year lows, in part due to strong tin exports from Indonesia, and lower demand due to a trend in miniaturization of electronics.

Consequently, idled capacity exists at both mines and smelters in China and Indonesia, such that improved metal demand will not immediately be reflected in improved pricing.

However, demand is expected to improve significantly in the medium and longer term. The recent trend toward electronic miniaturization is coming to an end. More generally, increased demand is expected in the medium and longer term due to the electronic revolution, and tin's application as a battery metal. A recent MIT study identified tin as the metal to be most impacted by emerging technologies. Perhaps for this reason, the US Government has designated tin as a strategic metal.

As noted above, current idled capacity can accommodate incremental short-term metal demand. However, in the medium and long term, the project pipeline is such that supply increases will be constrained, due in part to a lack of mine development in the West, contributing to a structural supply challenge for the industry. This is expected to contribute to increased metal prices, and improved commercial terms for miners selling tin concentrate to custom smelters.

19.2 Assumptions for the PEA regarding smelter charges

19.2.1 Copper terms

- Average copper payability of 25%.

19.2.2 Zinc terms

- Treatment costs of $210 per dry tonne of concentrate, with standard offtake terms.
- Average zinc payability of 85%, subject to a minimum deduction of 8 units.

19.2.3 Lead terms

- Treatment costs of $180 per dry tonne of sulphide concentrate, with standard offtake terms.
- Average lead payability of 95%, subject to a minimum deduction of 3 units.

19.2.4 Tin terms

- Treatment costs of $450 per dry tonne of concentrate, with standard offtake terms.
- Average tin payability of 95%.

19.2.5 Gold and silver terms

- Gold payable in oxide 99%.
- Gold payable in lead concentrate approximately 45%.
- Gold payable in sulphide leach 99%.
- Silver payable in oxide 99%.
- Silver payable in lead concentrate approximately 88%.
- Silver payable in zinc concentrate approximately 50%.
- Silver payable in sulphide leach 99%.

19.2.6 Transport costs

- Base case is transport of concentrate by truck either to European smelters or to the ports of Sines and Setubal.
- Transport charges $60 per dry tonne of concentrate for zinc.
- Transport charges $60 per dry tonne of concentrate for lead.
- Transport charges $80 per dry tonne of concentrate for tin.

19.3 Doré sales contracts

No off-take agreements have been made with potential doré off-takers at the time of preparation of this Technical Report, however several indicative non-binding proposed term sheets have been received from European and global refiners of doré metal.

The terms quoted are in line with the current market.

19.3.1 Doré treatment terms

- Payable gold oxide doré: 99%
- Payable silver oxide doré: 99%
- Payable gold sulphide doré: 99%
- Payable silver sulphide doré: 99%

19.4 Contracts

There are no major contracts in place given the stage of the project.

20 Environmental studies, permitting, and social or community impact

20.1 Permitting status

In terms of Environmental Licensing, an Environmental Scoping Proposal (PDA) has already been prepared and submitted to the Environmental authorities for the start of the Environmental Impact Assessment (EIA), in accordance with Decree-Law No. 151-B/2013 of 31 October, as amended and republished by Decree-Law No. 152-B/2017 of 11 December, which has already been approved by Environmental Portuguese Agency – Environmental Impact Assessment Authority (APA).

Redcorp have received the formal answer from APA with the guidelines for the execution of the EIA for licensing the LS mining concession.

To comply with legislation, Redcorp plans to execute the following works and services, in conjunction with an experienced environmental engineering services company:

1. Preparation of the Mining Plan (under Previous Study Phase), pursuant to Law No. 54/2015 of 22 June, which defines the basis of the legal regime for the disclosure and use of geological resources and Decree-Law no. 88/90, of 18 March.
2. Elaboration of the EIA for the Mining Plan (under Previous Study Phase), according to Decree-Law No. 151-B/2013, of 31 October, amended and republished by Decree-Law 152-B/2017, of 11 December.
3. Technical support and representation to the project promoter, from public and private entities that may interfere in the process, namely, the DGEG, the APA and the Alentejo Regional Coordination and Development Commission (CCDR Alentejo).

In the evaluation of a project under Previous Study Phase, the Mining Plan will not have all the details but rather quantitative ranges and approximate values, being mandatory to present project alternatives (to be chosen by APA). The EIA will assess the impacts expected by the actions of the Mining Plan. In the case of approval (favourable Environmental Impact Statement decision on EIA procedure (DIA), a detailed Mining Plan (execution phase) and RECAPE (compliance report of the execution project), will be prepared in order to verify if the impacts predicted in the previous study phase are confirmed with the detailed project. In view of the environmental compliance of the (APA) execution project to RECAPE, the licensing process with DGEG may be instructed.

20.2 Information required for execution of the EIA

20.2.1 General information

- Updated topographic survey of the areas covered by the proposal (concession), and 100 m from the surroundings, at a scale of 1: 2000, with paths, walls, poles, houses, rivers and streams, etc.
- Information about the background of the process.
- Information on all supporting and industrial infrastructures to be installed.
- Information on the supply of electricity, water, compressed air, fuels, lighting, among others.
- Other relevant information for the development of the proposed works.

20.2.2 Information for mining plan and EIA

- Information regarding Redcorp (history, financial capacity, technical capacity, etc.).
- Existing documents, licenses, and contracts (land and land register).
- Exploration reports.
- Geological information of the mineral deposit in the various areas to be exploited and surroundings.

- Geological information treated in 3D CAD (or Surpac) format with the model of mineralized orebodies and the distribution of grades along the mineralized orebodies, as well as the definition of the mining geometry.
- Resource and Reserve data (refer to whether international codes were used), contents, etc.
- Data and reports of mineralogical and metallurgical tests.
- Description and quantification of:
  - Labour (mine, mill, and construction phase).
  - Mine lifetime.
  - Mine construction phases, mill construction phases, and infrastructure construction phases.
  - Daily and weekly working hours (mine and mill).
  - Underground work area (size, number of underground levels, etc.).
  - Expected water consumption in the mine and mill.
  - Expected quantities of waste and tailings.
  - Areas for implementation of the mill and other surface infrastructure.
- Operational and geotechnical sizing of underground structures (blasting faces, ramps, galleries, shafts, etc.).
- Geotechnical information and data, including testing, analysis, etc., of the rock mass and the materials to be deposited (waste and tailings).
- Dimensioning of waste and tailings landfills (with geotechnical, waterproofing, and drainage).
- Information relating to the mill, including processes, installations, mass balances, reagents, consumables, effluents, wastes, etc., from ore treatment, as well as chemical analysis of all components, including waste, ore, fresh water, concentrate, tailings, and wastewater.
- HSE safety rules, standards, and regulations for the ore processing facility (mill).
- Detail of surface infrastructures projects such as buildings, landfills, basins, roads and access, electricity, water supply, sewage, compressed air, etc. (deployment area, height, building materials, colour, etc.).
- Description of accesses to be used between the mine and the mill and for the dispatch from the mill.
- Health and safety information (policy, objectives, company rules and emergency systems, among others).
- Information on other similar mills and related environmental studies.
- Construction stages (phases) for the mill and other surface infrastructure.
- Logistics data, namely origin and quantity of equipment, materials and consumables, as well as accesses, means, and destination of the concentrate.
- Destination of concentrate (locations and customers).
- Study of traffic (roads, ports) and possible improvements of these infrastructures (agreements with entities).
- Pre-Feasibility Study (PFS).

The expected timeline to complete the EIA (final execution project) and obtain approval is approximately 30 months, timelines are suspended whenever continuity is dependent on the issuing of opinions or authorizations by the public or private entities involved in the process, or the evaluation and supply of elements by Redcorp. The schedule is summarized in Table 20.1.

Table 20.1 Schedule to obtain approval for the EIA

Task	Entity	Duration
Execution of the EIA	VISA / Redcorp	11 months
EIA Conformity Assessment	APA and other entities	1.5 months
Request for Additional elements	VISA / Redcorp	3 months
Public Consultation and Evaluation	APA and other entities	3.5 months
Preparation of the Final Project Execution	Redcorp / VISA	4 months
RECAPE	VISA / Redcorp	3 months
Conformity Assessment and Public Consultation	APA and other entities	3 months
Project Environmental Compliance Decision	APA	-

After the above approvals, and before beginning mining, the company also has to:

- Own or rent the land.
- Submit the industrial plant (mill) project, including architecture, civil engineering, electric project, sewers, water, emergency plans, etc., which are obtained in the DGEG (industrial license).
- Obtain specific environmental authorizations (water discharge, water collection, etc.).
- Obtain Environmental Licence of the industrial plant (mill), that will define environmental objectives and will present a monitoring plan for environmental performance.

21 Capital and operating costs

21.1 Capital costs

21.1.1 Summary

Capital costs for the underground mine and processing plant are deemed to be of a level of accuracy consistent with industry standards for a PEA. The costs are presented in US$ and are based on price estimates determined by AMC and Pro Solv for the Mineral Processing plant cost.

The LOM capital costs are estimated to a total of $183M, as summarized in Table 21.1.

Table 21.1 Annual capital costs summary

Capital costs	Unit	1	2	3	4	5	6	7	8	9	LOM total
Mining fleet	$(000)	14,200									14,200
Ramp box cut	$(000)	1,000									1,000
Waste development	$(000)	2,520	7,328	4,706	2,273	2,452	2,180				21,460
Maintenance	$(000)	1,000	4,500	250	250	250	250	250			6,750
Backfill plant	$(000)		12,000								12,000
Process plant	$(000)	25,000	35,000	100	100	100	100	100	100		60,600
TSF	$(000)	4,000	8,000	50	50	50	50	50	50		12,300
Infrastructure and services	$(000)	7,893	2,727	50	50	189	1,067	50	50		12,076
Contingency and closure	$(000)		37,551							5,000	42,551
Grand total	**$(000)**	**55,613**	**107,106**	**5,156**	**2,723**	**3,041**	**3,647**	**450**	**200**	**5,000**	**182,937**

Details on the capital costs are discussed in the following sub-sections.

21.1.2 Mining fleet

The total cost for the mobile mining equipment fleet is $14.2M. Details on the quantities and cost of the mining fleet is shown in Table 21.2. An additional $6.8M, is allocated for replacement and maintenance of equipment including rebuilds.

Table 21.2 Mining fleet quantities and costs

Equipment	Unit	Quantity	Unit cost	Total cost
LHD	$M	4	0.8	3.2
Truck	$M	4	0.9	3.6
Development drill	$M	3	0.8	2.4
Longhole drill rig	$M	2	0.8	1.6
Rock support drill rig	$M	2	0.8	1.6
Cable bolter	$M	1	0.8	0.8
Scalers	$M	3	0.15	0.5
Rockbreakers	$M	1	0.1	0.1
Support trucks, highlifts	$M	4	0.1	0.4
Grand total	**$M**	**24**	**-**	**14.2**

21.1.3 Portal construction

The box cut for the portal construction is estimated to a total cost $1M. The box cut has a volume of 259,437 m³ and an average density of 1.8t/m³, which result in 466,897 t of overburden and rock to remove. Unit rate for mining the portal is $1.80/t, which totals to $841k. An allowance of $159k has been established for the portal ground support.

21.1.4 Underground development

Waste development will be operated by the owner. Waste development cost is estimated to be a total of $21.5M. Based on the benchmark costs for European mining operations, AMC estimated the unit rates for the lateral development to be $3,500/m. Unit rates for the drop raising is $1,500/m, while for raiseboring it is $3,800/m.

21.1.5 Processing plant

Since insufficient test work has been completed so far, it is not possible to develop the processing plant costs from equipment sizing and costing principles. The cost estimate was developed from cost models published by Mining Cost Services.

The processing plant cost is estimated as a total of $60M. The initial capital cost for the plant is $25M in Year 1 and then $35M in Year 2. The breakdown of the cost is as follows:

- Three product polymetallic plant: $50M.
- Cyanide leach circuit / Gold Recovery Plant: $7M.
- Tin flotation circuit: $3M.

The sustaining capital cost for the process plant is estimated to be $100K a year for Year 3 to Year 8.

21.1.6 Backfill plant

Backfill plant cost is estimated to a total of $12M. There is no sustaining capital cost for the backfill plant.

21.1.7 Tailings storage facility

The TSF is estimated to be a total of $12.3M. The initial capital for the TSF is $4M in Year 1 and then $8M in Year 2, which totals to $12M. Sustaining capital cost for the TSF is estimated to $50k a year from Year 3 to Year 8.

21.1.8 Infrastructure and services

The cost for infrastructure and services is estimated to a total of $12.1M. In the first two years, infrastructure and services is estimated to $9.2M, which includes the supply and installation of the main ventilation fans, auxiliary ventilation fans, portable refuge stations, air compressor, electrical substation and distribution, dewatering pump and pipes, service water supply, and surface structures (office, shop, dry, warehouse). As the mine gets deeper between Year 5 and 6, additional cost for the dewatering pumps and electrical substation is estimated to be $963k. Sustaining capital cost for the infrastructure and services is estimated to $50k a year from Year 3 to Year 8. Engineering, procurement, and construction management (EPCM) cost is estimated to be 20% of the capital cost of the infrastructure and services.

21.1.9 Contingency and closure

A contingency of 30% has been applied to the capital cost for the first two years. Total contingency cost is $37.6M. Closure and reclamation costs have been estimated at $5M and are assumed to be incurred at the end of the mine life.

21.2 Operating costs

21.2.1 Summary

Operating costs for the underground mine and processing plant are deemed to be of a level of accuracy consistent with industry standards for a PEA. The costs are presented in US$ and are based on price estimates from AMC and Pro Solv for processing.

The costs cover the mining of ore underground and processing of mineralized material. This includes the sections covering crushing, milling, flotation, dewatering, concentrating handling and trucking, site services (power, air, and water), and administration costs.

The LOM operating costs are estimated to a total of $358M, as summarized in Table 21.3.

Table 21.3 Annual operating costs summary

Operating costs	Unit	1	2	3	4	5	6	7	8	9	LOM total
Mining	$(000)		12,121	19,156	18,667	18,516	15,448	14,750	14,674	8,758	122,090
Processing	$(000)		17,360	30,959	30,465	30,974	30,025	30,035	26,829	14,862	211,510
G&A	$(000)		1,979	3,530	3,473	3,531	3,423	3,424	3,406	2,033	24,800
Grand total	**$(000)**		**31,460**	**53,645**	**52,605**	**53,021**	**48,896**	**48,209**	**44,909**	**25,654**	**358,400**

The LOM unit operating costs are estimated to be $49.43/t milled, as summarized in Table 21.4.

Table 21.4 Unit operating costs summary

Operating costs	$/t milled
Mining	16.84
Processing	29.17
G&A	3.42
Grand total	**49.43**

21.2.2 Mining costs

Underground operating costs are based on benchmark costs in the European mining operations for development and longhole stoping over a range of annual throughputs. All underground mining will be undertaken by the owner. The operating cost estimates include all labour, material, equipment supply and operation, supervision, administration, and on-site management. Labour costs include all benefits and employment taxes.

The unit rate for the ore drift development is $3,500/m, which totals to $9.7M. Total longhole stoping cost is $41.8M. Other mining costs have a total of $70.6M, which includes cost for mine maintenance, power, technical service, and labour.

The total LOM mining cost is estimated to be $122M, or an average of $16.84/t milled.

21.2.2.1 Other mining costs

Maintenance, power and mine services costs are included in the total mining costs of $16.84/t milled. The basis for the maintenance cost is $1M per year for seven pieces of primary equipment.

Power cost is at rate of $0.063/kW-hr. Main power consumptions for mining only is the ventilation fans, pumps and other equipment.

An allowance of $500k per year is allocated to the technical service costs, which includes software, sampling, and instruments.

Labour is based on the manpower quantities, as shown in Table 21.5.

Table 21.5 Manpower quantities and costs

Labour	**Quantity**	**Unit $/month**	**Total $/month**
Total operators	72	3,520	253,440
Maintenance team	30	3,520	105,600
Services team	10	3,520	35,200
Mining Manager	1	14,300	14,300
Engineering Manager	1	11,000	11,000
Technical services	8	8,800	70,400
Supervisors	12	7,700	92,400
Grand total	**134**		**582,340**

21.2.3 Processing costs

The operating costs were estimated from the cost models published by Mining Cost Services.

The processing cost was estimated at $30/t for MS and $25/t for oxides. Total LOM processing cost is estimated to be $212M or an average of $29.17/t milled. The breakdown of the cost is as follows:

- Processing cost for three product flotation plant (Pb / Zn / Cu): $19.00.
- Cyanide leach circuit / gold recovery plant: $7.00.
- Tin flotation circuit: $4.00.

21.2.4 General and Administration costs

An estimate from other local mine sites for the General and Administration (G&A) costs is $3.1M per year. The G&A costs have been adjusted to reflect the annual throughput at the LS Project compared to the benchmark operation. Total LOM G&A cost is estimated to be $24.8M, or an average of $3.42/t milled.

22 Economic analysis

The PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the results of the PEA will be realized.

An economic model was developed to estimate annual cash flows and sensitivities of the LS Project. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed and are likely to approximate the true value.

22.1 Assumptions

All currency is in US$ unless otherwise stated. As some costs will be in Euro (€), where applicable, an exchange rate of US$1.1 to 1.0 was used. The cost estimate was prepared with a base date of year 1, assumed to be 2020 and does not include any escalation beyond this date. For net present value (NPV) estimation, all costs and revenues are discounted at 8% from the base date. The economic model shows the LS Project to be under construction for two years, which is considered development and then in production for the balance of the projected cash flows, which is considered operating.

A regular income tax rate of 21% and a municipal surtax of 1.5% are applied. No tax planning has been included, all historical tax attributes such as any loss carry forwards, recapture, mineral property, exploration costs, or net tax basis of capital assets are ignored. Taxes are paid in the year they are incurred.

The estimates of capital and operating costs have been developed specifically for this project and are summarized in Section 21 of this Report (presented in 2020 dollars). The economic analysis has been run with no inflation (constant dollar basis). No project financing costs are included.

Project revenue is derived from the sale of copper, lead, zinc, and tin concentrate, and from gold and silver doré into the international market. Details regarding the refining of these concentrates can be found in the market studies presented in Section 19. No allowances for any penalty elements in the metal concentrates have been made.

A discount rate of 8.0% was deemed appropriate for the Project. Discount rates applied to projected cash flows also recognize the time value of money as well the risks and variables associated with the project, such as metal price fluctuation, marketability of the commodity, location of the project, stage of development, and experience of the owner. In selecting an appropriate discount rate, the largest component of risk for the project are the confidence levels of both the PEA and the use, in part, of Inferred Mineral Resources.

It is assumed that concentrate produced in a given year are considered sold in the same period with no inventories of work-in-process or finished goods.

Underground development costs incurred after the commencement of underground production are assumed to relate to mining in the year the costs are incurred and are expensed in the same year.

Reclamation costs are assumed to be incurred at the end of the mine life. A 4% royalty is applied to the gross revenue.

22.2 Economic analysis

AMC conducted a high-level economic assessment of the proposed underground operation of the LS Project. The project is projected to generate approximately $137M pre-tax NPV and $106M post-tax NPV at 8% discount rate, pre-tax IRR (internal rate of return) of 37% and post-tax IRR of 31%, with a payback period of 4.1 years. Key assumptions and results of the economics are provided in Table 22.1. The LOM production schedule, average metal grades, recovered metal, and cash flow forecast is shown in Table 22.2.

Table 22.1 LS deposit – key economic input assumptions and cost summary

	Unit	Value
Total waste mined	kt	305
Total sulphide plant feed	kt	6,047
Total oxide plant feed	kt	1,204
Total ore plant feed	kt	7,251
Cu grade	%	0.34
Zn grade	%	2.44
Pb grade	%	2.85
Sn grade	%	0.16
Au grade	g/t	0.75
Ag grade	g/t	69.8
Cu price	$/lb	$2.70
Zn price	$/lb	$1.20
Pb price	$/lb	$1.05
Sn price	$/lb	$7.50
Ag price	$/oz	$18
Au price	$/oz	$1,400
Payable Cu produced	kt	1.41
Payable Zn produced	kt	112.78
Payable Pb produced	kt	123.31
Payable Sn produced	kt	2.91
Payable Au produced	Moz	0.12
Payable Ag produced	Moz	9.60
Payable ZnEq produced	kt	371
Payable ZnEq produced	Mlbs	819
Tax rate	%	22.5
Discount rate	%	8
Revenue oxide	$M	118
Revenue sulphide	$M	707
Total revenue	$M	825
Royalties	$M	33
Mine operating costs	$M	122
Processing costs	$M	212
G&A costs	$M	25
Capital costs	$M	183
Tax	$M	48

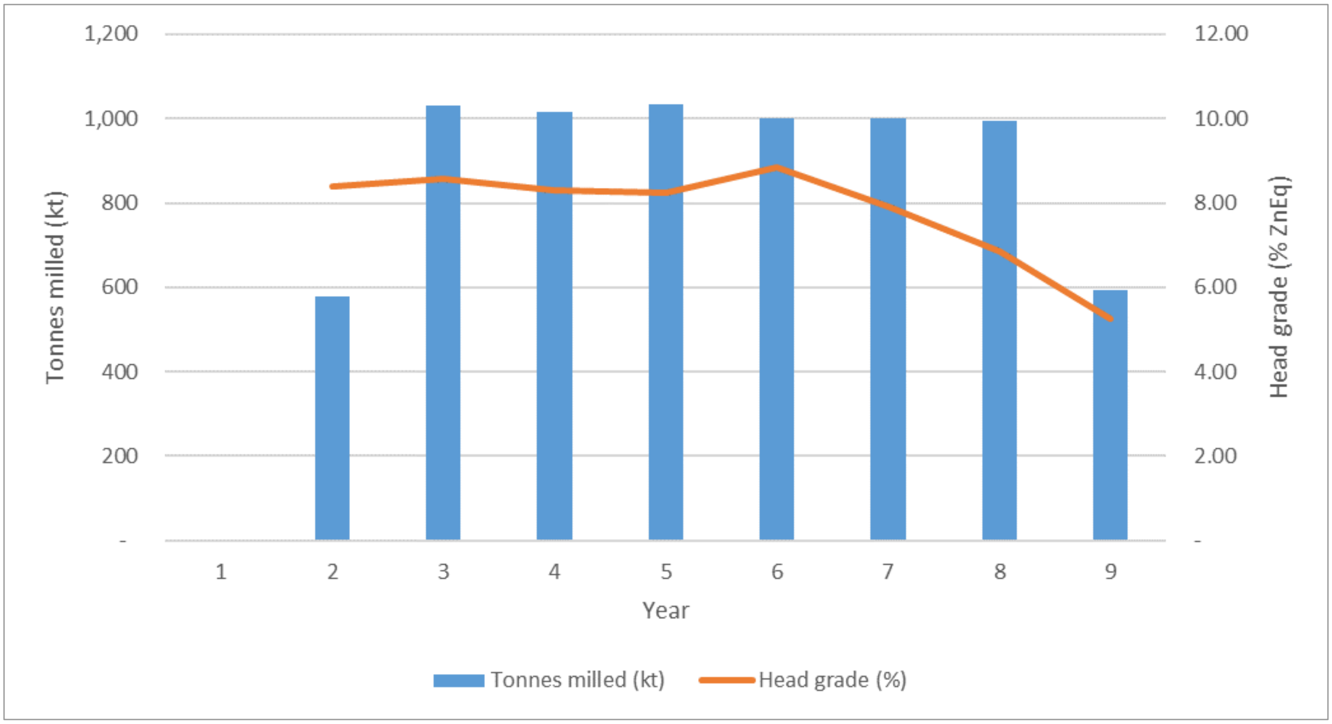
Table 22.2 Summary of economic results

	Unit	Base case
Cash cost	$/lb payable ZnEq	0.44
Gross sales revenue	$M	825
Royalties	$M	33
Site operating costs (mining, processing, G&A)	$M	358
EBITDA	$M	433
Total taxes	$M	48
Revenue split by commodity	% Cu	1%
	% Zn	27%
	% Pb	28%
	% Sn	3%
	% Ag	20%
	% Au	21%
Pre-tax LOM cash flows (undiscounted)	$M	250
Pre-tax NPV at 8%	$M	137
Pre-tax IRR	%	37%
Pre-tax payback period	Years	3.9
After-tax LOM cash flows (undiscounted)	$M	202
After-tax NPV at 8%	$M	106
After-tax IRR	%	31%
After-tax payback period	Years	4.1

Note: EBITDA = Earnings before interest, tax, depreciation, and amortization.

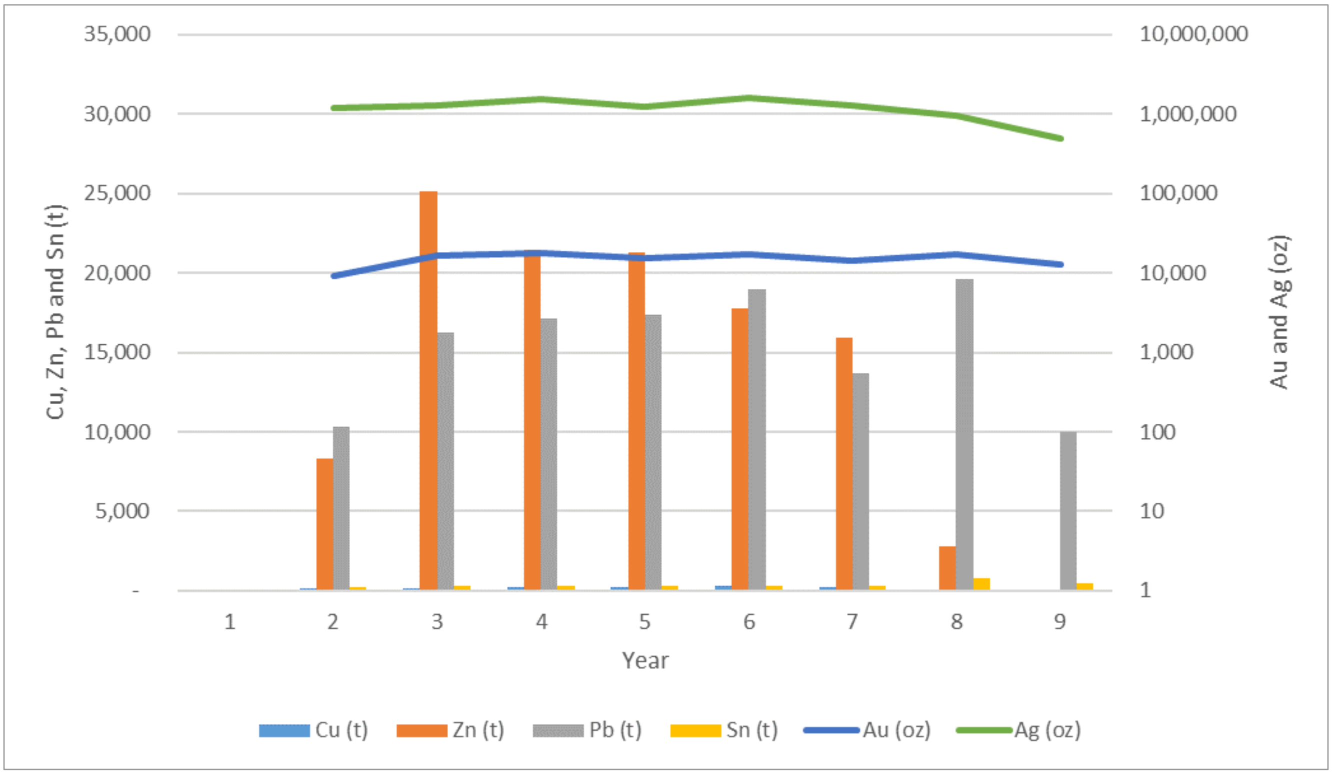
A total payable metal production of 371 kt of ZnEq is projected to be produced during the mine life. Figure 22.1 shows the tonnes milled and head grade by year. Figure 22.2 shows a breakdown of the payable copper, zinc, lead, tin, gold, and silver produced by year during the mine life. Figure 22.3 shows the annual after-tax cash flow.

Figure 22.1 Tonnes milled and head grades by year



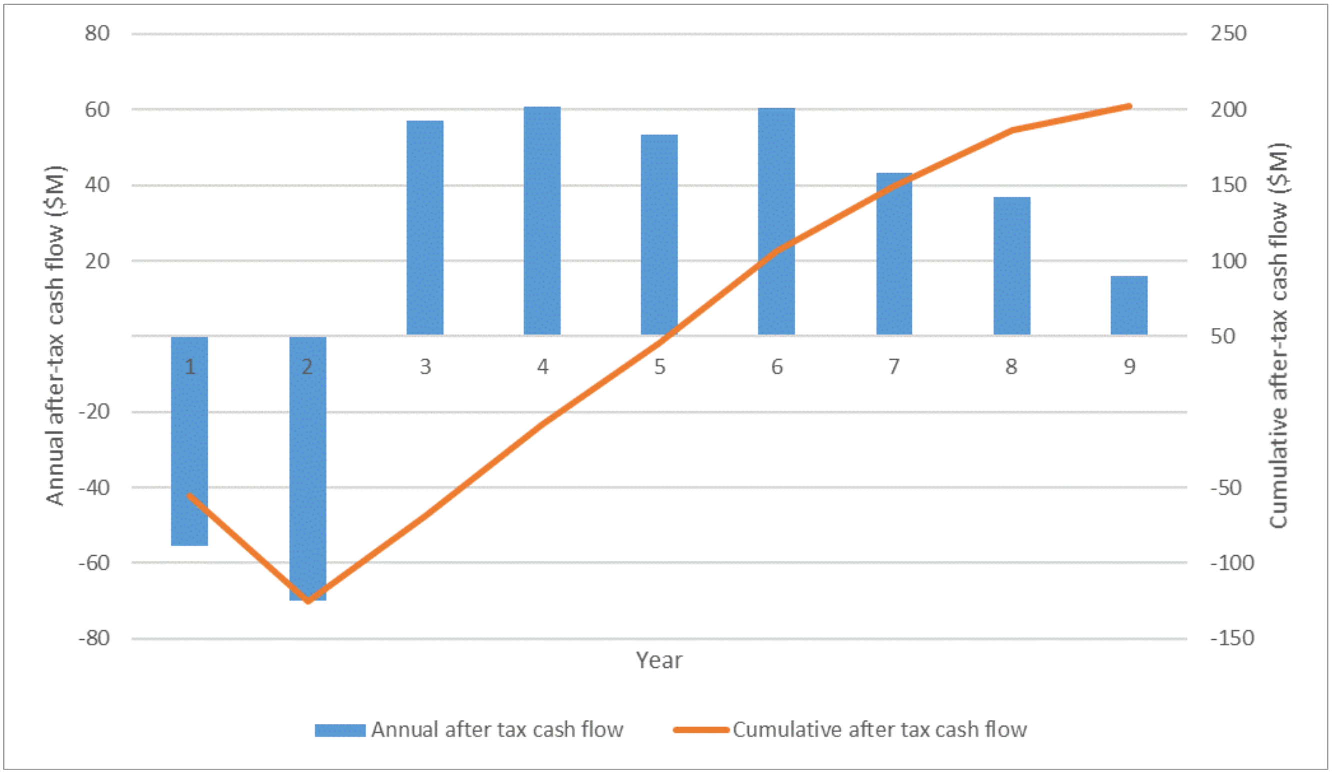
Source: AMC.

Figure 22.2 Metal production by year



Source: AMC.

Figure 22.3 Annual after-tax cash flow



Source: AMC.

22.3 Sensitivity analysis

Sensitivity analyses were performed for variations in metal prices, capital costs, and operating costs to determine their relative importance as project value drivers. The results of the sensitivity analysis are summarized in Table 22.3.

In Figure 22.4, the results show that the post-tax NPV is robust and remains positive for the range of sensitivities evaluated.

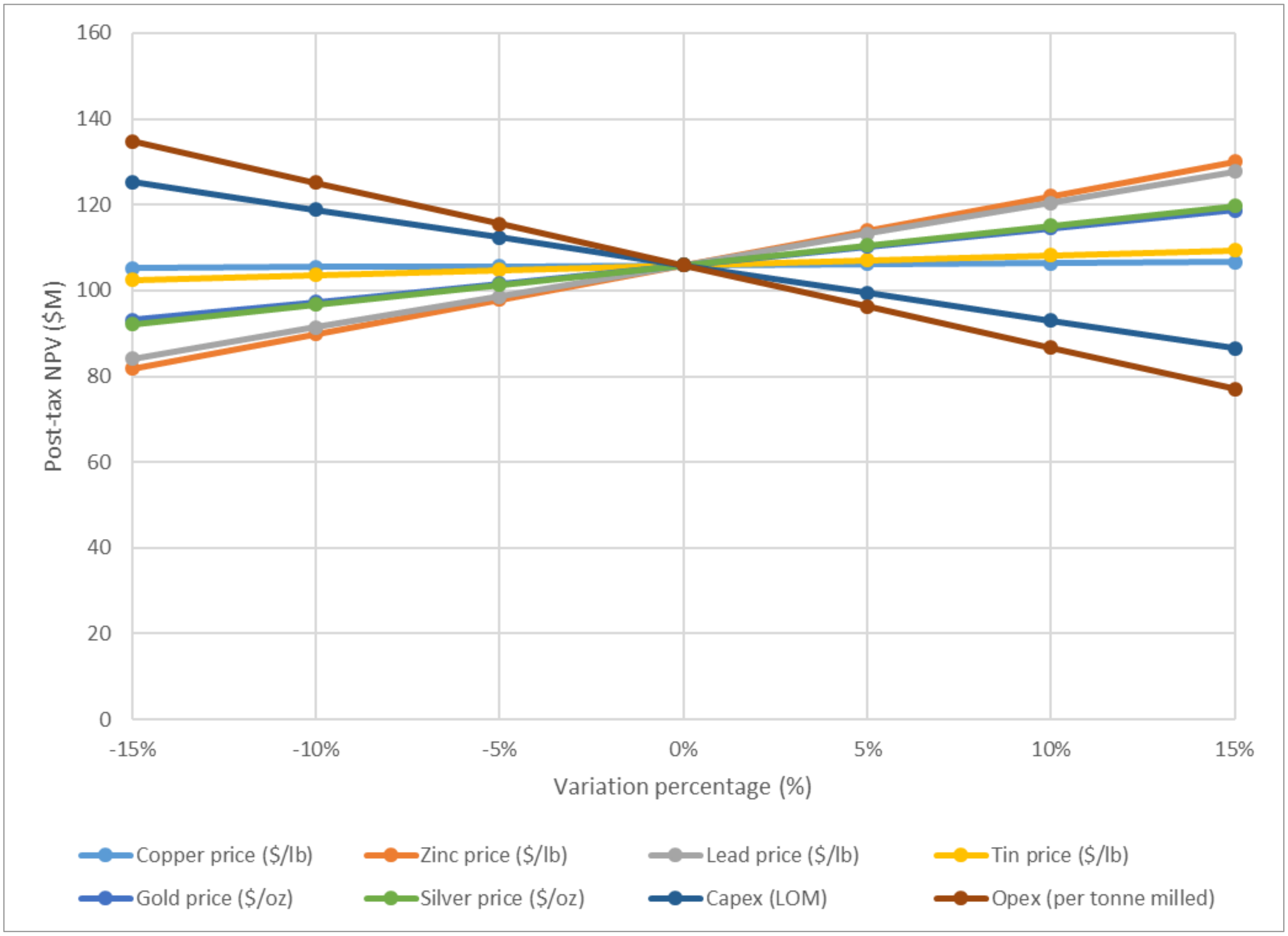
Post-tax NPV is most sensitive to changes in the zinc and lead prices. The NPV is moderately sensitive to changes in operating costs and capital costs, and in the price of gold and silver. Changes in the price of tin and copper have the least impact on NPV.

Lead and zinc metal prices often move in tandem, sensitivities to lead and zinc prices are shown in Table 22.4.

Table 22.3 Project economic sensitivity analysis – post tax

NPV after-tax ($M)	**Sensitivity range**						
	85%	**90%**	**95%**	**100%**	**105%**	**110%**	**115%**
Copper price ($/lb)	105.2	105.5	105.7	105.9	106.1	106.4	106.6
Zinc price ($/lb)	81.9	89.9	97.9	105.9	113.9	122.0	130.0
Lead price ($/lb)	84.2	91.4	98.7	105.9	113.2	120.4	127.7
Tin price ($/lb)	102.4	103.6	104.8	105.9	107.1	108.2	109.4
Gold price ($/oz)	93.1	97.4	101.6	105.9	110.2	114.5	118.8
Silver price ($/oz)	92.2	96.8	101.3	105.9	110.5	115.1	119.7
Capex (LOM)	125.2	118.8	112.4	105.9	99.5	93.0	86.6
Opex (per tonne milled)	134.8	125.2	115.5	105.9	96.3	86.7	77.1

Figure 22.4 Project economic sensitivity analysis – post tax



Source: AMC.

Table 22.4 Lead and zinc price economic sensitivity analysis – post tax

Lead price ($/lb)	**Zinc price ($/lb)**				
	1.02	**1.11**	**1.20**	**1.29**	**1.38**
0.89	60.1	72.1	84.2	96.2	108.2
0.97	71.0	83.0	95.0	107.1	119.1
1.05	81.9	93.9	**105.9**	118.0	130.0
1.13	92.8	104.8	116.8	128.8	140.9
1.21	103.6	115.7	127.7	139.7	151.7

23 Adjacent properties

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

There is an exploration area for Cu, Pb, Zn, Ag, and associated minerals, named "Alcácer" that is held by Esanmet Portugal, Unipessoal, Lda, a 100% Portuguese subsidiary of a Turkish company named ESAN. This exploration area is located North of the LS concession, has no known activity associated with it and is shown in Figure 4.2.

There are two historic lead-zinc mines in regional proximity to the property: the Lousal mine, situated approximately 22 km south of the North deposit; and the Caveira mine situated approximately 13 km south-west of the North deposit.

23.1 Lousal Mine (Mina de Lousal)

The Lousal mine (Mina de Lousal) was first opened in 1882 and operated continuously from 1900 up to 1988 when it was finally closed. Historic production figures for the Lousal mine were not available at the time of writing. The Lousal VHMS mine exploited pyrite primarily as a sulfur source for fertilizer production.

The mining village of Lousal (Grândola, Portugal) is currently an international example of success in socio-economic, environmental and mining heritage rehabilitation. The rehabilitation program resulted from the joint efforts of the Municipality of Grândola and the mine owner company – SAPEC, SA. The program is responsible for the restoration of the mine shafts and conversion of the old power plant into a mining museum. The main warehouse gave place to a regional restaurant, the mine offices were transformed into a handicraft center and the administration house was converted into a rural hotel. A rehabilitation plan directed to reclaim the Lousal contaminated area was defined and promoted by EDM. Some other mining facilities have been used to create the Mine of Science-"Ciência Viva" Centre, which promotes non-formal educational activities devoted to Science and Technology.

23.2 Caveira Mine (Mina da Canal Caveira)

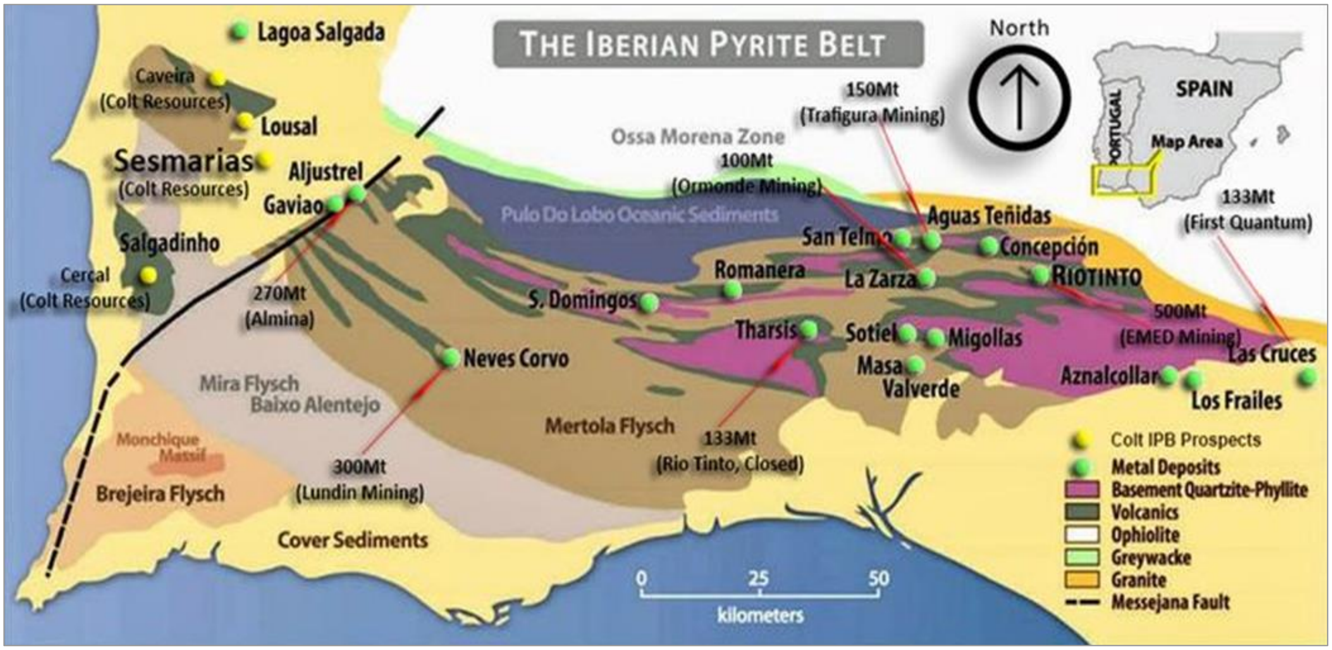
The Caveira mine (Mina da Canal Caveira) operated for 103 years from 1863 to 1966. Historic production figures for the Caveira mine were unavailable at the time of writing.

The Caveira mine located 6 km south-east of the village of Grândola, is one of the most western copper mines of the IPB, geologically identical to the mines of São Domingos and Aljustrel, famous for the Roman finds. Historically, this mine was operated by the Romans along with a number of others.

The mine is known mainly because of the immense slag heap. In 1880, due to the spontaneous combustion of the pyrite, a fire broke out that lasted for three years. The mine consists of three distinct deposits separated by barren host rocks and, due to this configuration, it was decided to concentrate efforts on underground exploitation.

Figure 23.1 shows a plan view of the IPB and the mining operations relative to the Property.

Figure 23.1 Iberian Pyrite Belt



Source: Ascendant.

24 Other relevant data and information

All relevant data and information regarding the LS Project are included in other sections of this Technical Report.

The QP is not aware of any other data that would make a material difference to the quality of this Technical Report or make it more understandable, or without which the report would be incomplete or misleading.

25 Interpretation and conclusions

25.1 Geology and mineralization

The Property is located within the north-western portion of the IPB. The IPB is one of the most prolific European metallic provinces, hosting one of the largest concentrations of MS in the Earth's crust.

The entire property 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 Project. The Tertiary sedimentary rocks unconformably overlie rocks of the Volcano-Sedimentary Complex of the IPB.

The LS Project currently has three known deposits; the North, Central and South deposits. The deposits are folded, faulted, and interpreted to occur mostly on the subvertical-overturned and intensely faulted limb of a southwest-verging anticline. The North deposit is further offset by an east-west-trending Alpine-age fault in the north, with a 50-m downthrow of the northern block but whose horizontal amount and sense of displacement is unknown.

The mineralization comprises MS and semi-massive sulphide lenses and sulphide veins and veinlets and is mainly hosted by a thick (up to 250 m) and strongly chloritized quartz-phyric rhyodacite unit. Currently, the mineralization is known to extent continuously over a cumulative strike length of 1.7 km in a north-northwesterly to south-southeasterly direction.

LS Project 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. 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 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 greatest contribution to the Mineral Resources is from the North deposit. However, all deposits have the potential to delineate more resources with additional drilling. The stringer / fissure type mineralization of the South and Central deposits appears to be more amenable to metallurgical processing than the massive mineralization of the North deposit and future priority drilling will depend on progress in metallurgical testwork.

Geological reasoning suggests that the subdivision of the LS Project into the North, Central and South deposits is arbitrary, being based on the existing drill pattern. With further concerted systematic drilling, the three 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 three 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.

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.

25.2 Mining

The geometry of the deposit is relatively simple, and the deposit is fairly shallow. AMC considers the use of SLOS mining method to be suitable for the deposit and will result in high productivities. The average haulage distance is approximately 1,600 m which is relatively short and will ensure low cost haulage and high productivity. Mining takes place primarily over three sub levels spaced

25 m to 35 m apart. The mine is very compact and as such equipment use will be highly efficient and manpower numbers low. The mine will operate on a five-day week, three eight-hour shifts per day, with essential services manning only on weekends.

The mine design leads to a low-cost mining operation with relatively high production throughput of 1.0 Mtpa that is well supported by the production schedule. The mine will use paste fill to backfill the stopes reducing the size of the surface tailings facility significantly. Paste fill will be delivered underground using a combination of pumping and gravity via boreholes and high-pressure pipelines to the stopes.

25.3 Processing

The metallurgical work completed to date is of a reconnaissance nature and no firm conclusions can be drawn therefrom. Detailed testwork is in progress.

The PEA assumes a processing rate of 3,000 tpd of polymetallic ore. The following conclusions can be drawn from the scoping level metallurgical study:

- The deposit has two major ore types, namely, massive sulphide and Gossan. The Gossan consists of oxidized material.
- The metals of interest in the deposit are copper, lead, zinc, gold, silver, and tin.
- The MS has tin which may not be recoverable whereas Gossan has uneconomic quantities of zinc. Hence, the process flowsheet needs to be flexible to recover these minerals.
- The mineralogy of the deposit indicates that very fine grind will be required to produce saleable concentrates.
- A typical polymetallic process flowsheet may produce copper, lead, and zinc concentrates with some gold and silver. Copper concentrate is not considered in this study.
- The tailings from the sulphide circuit and the Gossan can be cyanide leached to recover the majority of gold and silver.

25.4 Economics

The project shows robust economic results with a pre-tax NPV at 8% of $137M and an IRR of 37%, and an after tax NPV at 8% of $103M and IRR of 31%. The economics are highly dependent on the assumptions for metal recovery. AMC recommends that more metallurgical testwork is undertaken to firm up these assumptions and validate them.

Costs have been determined at a PEA (±35%) level of detail. Additional work is required to better define these costs.

26 Recommendations

26.1 Geology

Micon makes the following recommendations:

- Ascendant should continue to expand the Mineral Resources systematically.
- The immediate focus in the short to medium-term 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.
- The second priority should be infill drilling the gaps separating the North and Central deposits and the gap separating the Central and South deposits.
- Models of the deposits should continue to be refined / updated as more information becomes available.
- Geophysical investigations to the eastern and south-eastern areas of the LS deposit should be continued.
- Subject to satisfactory results, the same exercise should be implemented to the north of the North deposit, targeting the area immediately beyond the major east-west Alpine fault.

26.2 Mining

AMC makes the following recommendations:

- Additional geotechnical work should be undertaken in order to support the mine design and provide suitable ground support recommendations.
- A hydrogeological study that aims to determine and predict ground water inflows should be undertaken.
- Following the creation of a hydrology model, additional work should be undertaken to determine the dewatering strategy and ensure sufficient pumping arrangements are considered.
- A more detailed design is required for the paste fill distribution. Exact locations for proposed boreholes and the backfill plant location are also required.
- The cost assumptions are mostly based on benchmark information. Additional work is recommended to develop a first principles cost estimate and that comparative quotes from contractors be obtained.
- A detailed production schedule should be developed that carefully considers the primary and secondary stope mining sequence from level to level.

26.3 Infrastructure

- Additional work is required to better define and cost the project infrastructure.
- Ascendant should engage with the local power supplier to better understand the availability and capacity of the power from the local substation. Power line detail and electrical equipment requirements can be better defined and more accurately costed.
- A complete climate and water balance study is required in order to better define the water requirements for the operation.
- A tailings specialist should be engaged to determine the exact design, layout, and cost of the proposed dry stack tailings facility.

26.4 Processing

Optimum metallurgical recoveries are key to the success of the LS Project. AMC recommends detailed metallurgical investigations that should be prioritized over additional drilling to expand the Mineral Resource.

- The project warrants advancement to PFS based on the results of the PEA.
- A systematic bench-scale study should be undertaken to optimize the recoveries and concentrate grades for Cu, Pb, and Zn from both fresh and oxidized ores.
- Develop a process for recovery of tin from the MS final tailing.
- Optimize the cyanidation leach circuit for gold and silver extraction.
- Generate process data for proper sizing of equipment.

The cost for the recommended test program is estimated to be $350,000.

26.5 Economics

- AMC recommends that more metallurgical testwork is undertaken to firm up the metal recovery assumptions and validate them.
- Costs have been determined at a PEA (±35%) level of detail, additional work is required to better define these costs.

26.6 Additional work

- Prepare a proposed 2020 exploration / development program and budget.
- Conduct follow up work to confirm the favourable geophysics results obtained during the 2019 exploration program.
- 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 mine design.
- Implement a hydrogeological study to better define and predict ground water inflow.
- Complete site-based climate and water balance.
- On completion of the above consider undertaking a PFS.
- Continue Environmental permitting.

In order to complete the planned 2020 exploration / development work, AMC has proposed a budget of approximately $4.7M broken down as summarized in Table 26.1.

Table 26.1 Proposed work program and budget for the LS Project for 2020

Program	Activity	Cost ($)
Drillhole IP Survey (North, Central & South Deposits)	Interpretation / modelling	30,000
Metallurgical testwork drilling	4 drillholes (1,200 m)	240,000
Detailed metallurgical testwork	Optimizing recoveries	350,000
North deposit exploration drilling (expanding inferred)	4 drillholes (1,400 m) +assays + modelling	420,000
Central / South deposits + other targets exploration drilling	14 to 16 drillholes (5,700) +assays + modelling	1,710,000
Geotechnical work to support PFS level design	Stope modelling empirical modelling, design criteria, ground support recommendations	350,000
Hydrological modelling	Field testwork, modelling, ground water impact establishment	350,000
Site based water balance	Field testwork and measurements, modelling	150,000
Environmental permitting	Field studies, base line data collection, Mining Plan, RECAPE	300,000
PFS	Complete study	750,000
All activities	**Grand total**	**4,650,000**

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28 QP Certificates

CERTIFICATE OF AUTHOR

I, Charley Murahwi, M.Sc., P.Geo., Pr.Sci.Nat., FAusIMM, of Toronto, Ontario, do hereby certify that:

1 I am currently employed as a Senior Geologist with Micon International Limited with an office at 900-390 Bay Street, Toronto, Ontario M5H 2Y2.

2 This certificate applies to the technical report titled "Technical Report and PEA for the Lagoa Salgada Property, Setúbal District, Portugal", with an effective date of 19 December 2019, (the "Technical Report") prepared for Ascendant Resources Inc. ("the Issuer").

3 I am a graduate of the University of Rhodesia in Zimbabwe (Bachelors of Geology in 1979) and Rhodes University in South Africa (Masters of Economic Geology in 1996), and possess a Diplome d˙Ingénieur Expert en Techniques Minières, Nancy, France (1987). I am a member in good standing of the Association of Professional Geoscientists of Ontario (Membership #1618), the Professional Engineers and Geoscientists of Newfoundland & Labrador (Membership #05662), the South African Council for Natural Scientific Professions (Membership #400133/09), and the Australasian Institute of Mining & Metallurgy (FAusIMM) (Membership #300395). I have worked as a geologist in the minerals industry for over 40 years.

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

4 I have visited the Lagoa Salgada Property from 16 to 19 October 2018, from 13 to 17 November 2018, and from 28 to 31 May 2019.

5 I am responsible for Sections 4 to 12 and 14, and parts of 1, 25, 26, and 27 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have had prior involvement with the property that is the subject of the Technical Report in that I was a qualified person for a previous Technical Report on the Lagoa Salgada Property in 2019 (filed 5 November 2019, effective date 5 September 2019).

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 19 December 2019
Signing Date: 24 February 2020

(original signed and sealed by) Charley Murahwi, M.Sc., P.Geo., Pr.Sci.Nat., FAusIMM

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

Senior Geologist

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CERTIFICATE OF AUTHOR

I, Gary Methven, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as Underground Manager and Principal Mining Engineer with AMC Mining Consultants (Canada) Ltd. with an office at 202-200 Granville Street, Vancouver, British Columbia, Canada, V6C 1S4.

2 This certificate applies to the technical report titled "Technical Report and PEA for the Lagoa Salgada Property, Setúbal District, Portugal", with an effective date of 19 December 2019, (the "Technical Report") prepared for Ascendant Resources Inc. ("the Issuer").

3 I am a graduate of the University of Witwatersrand in Johannesburg, South Africa (Bachelors of Mining Engineering in 1993). I am a member in good standing of the Engineers and Geoscientists of British Columbia (License #44471), the Registered Professional Engineers of Queensland (License #06839), and the Australian Institute of Mining and Metallurgy (CP). I have relevant experience in precious and base metal deposits, mine infrastructure, design and planning, mine production and financial evaluation, reserve estimation, technical reviews, and technical studies.

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

4 I have not visited the Lagoa Salgada Property.

5 I am responsible for Sections 2, 3, 15, 16, 20, 22, 23, and 24, and parts of 1, 21, 25, 26, and 27 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

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

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 19 December 2019
Signing Date: 13 February 2020

(original signed and sealed by) Gary Methven, P.Eng.

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Gary Methven, P.Eng.

Underground Manager / Principal Mining Engineer

AMC Mining Consultants (Canada) Ltd.

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I, George Zazzi, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as a Principal Mining Engineer with AMC Mining Consultants (Canada) Ltd. with an office at 202-200 Granville Street, Vancouver, British Columbia, Canada, V6C 1S4.

2 This certificate applies to the technical report titled "Technical Report and PEA for the Lagoa Salgada Property, Setúbal District, Portugal", with an effective date of 19 December 2019, (the "Technical Report") prepared for Ascendant Resources Inc. ("the Issuer").

3 I am a graduate of British Columbia Institute of Technology in Vancouver, Canada (Diploma of Mining Engineering in 1993) and the University of British Columbia in Vancouver, Canada (Bachelors of Metallurgical Engineering in 1989). I am a member in good standing of the Engineers and Geoscientists of British Columbia (License #28636). I have worked as a Mining Engineer for a total of 27 years since my graduation from university and have relevant experience in project management, feasibility studies, and technical report preparations.

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

4 I have not visited the Lagoa Salgada Property.

5 I am responsible for Section 18, and parts of 1, 21, 25, 26, and 27 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

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

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 19 December 2019
Signing Date: 13 February 2020

(original signed and sealed by) George Zazzi, P.Eng.

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George Zazzi, P.Eng.

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CERTIFICATE OF AUTHOR

I, Deepak Malhotra, Ph.D., of Lakewood, Colorado, do hereby certify that:

1 I am currently employed as President of Pro Solv, LLC with an office at 15450 W. Asbury Avenue, Lakewood, Colorado 80228.

2 This certificate applies to the technical report titled "Technical Report and PEA for the Lagoa Salgada Property, Setúbal District, Portugal", with an effective date of 19 December 2019, (the "Technical Report") prepared for Ascendant Resources Inc. ("the Issuer").

3 I am a graduate of Colorado School of Mines in Colorado, US (Masters of Metallurgical Engineering in 1973 and Ph.D. in Mineral Economics in 1978). I am a registered member in good standing of the Association of Society of Mining and Metallurgical Engineers (SME) and a member of the Canadian Institute of Mining and Metallurgy (CIM). I have experience in the area of metallurgy and mineral economics.

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

4 I have not visited the Lagoa Salgada Property.

5 I am responsible for Sections 13, 17, and 19, and parts of 1, 21, 25, 26, and 27 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

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

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 19 December 2019
Signing Date: 11 February 2020

(original signed by) Deepak Malhotra, Ph.D.

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